



# URANIUM EVALUATION AND MINING TECHNIQUES

PROCEEDINGS OF A SYMPOSIUM, BUENOS AIRES, 1-4 OCTOBER 1979  
JOINTLY ORGANIZED BY IAEA, NEA(OECD), OAS-IANEC



INTERNATIONAL ATOMIC ENERGY AGENCY, VIENNA, 1980

The cover pictures are of the Sierra Pintada Uranium District in the Mendoza Province of Argentina. The front cover shows a general view of the mining area, and the back cover installations for heap leaching.

# URANIUM EVALUATION AND MINING TECHNIQUES

STI/PUB/524

## CORRIGENDUM

Paper IAEA-SM-239/41, by M.A. Temnikov

Доклад М.А.Темникова IAEA-SM-239/41

Page 242, Fig.1 and page 248, Fig.3.

Стр. 242, рис. 1 и стр. 248, рис. 3.

The diagram on page 242 should be inverted and interchanged with that on page 248 — the captions, however, remaining as printed.

Чертеж на стр. 242 следует перевернуть и поменять местами со схемой на стр. 248 (подписи к рисункам оставить там, где они напечатаны).





**URANIUM EVALUATION  
AND MINING TECHNIQUES**



PROCEEDINGS SERIES

# URANIUM EVALUATION AND MINING TECHNIQUES

PROCEEDINGS OF AN INTERNATIONAL SYMPOSIUM ON  
URANIUM EVALUATION AND MINING TECHNIQUES  
JOINTLY ORGANIZED BY THE  
INTERNATIONAL ATOMIC ENERGY AGENCY,  
THE OECD NUCLEAR ENERGY AGENCY  
AND THE  
OAS INTER-AMERICAN NUCLEAR ENERGY COMMISSION  
AND HELD IN BUENOS AIRES, 1–4 OCTOBER 1979

INTERNATIONAL ATOMIC ENERGY AGENCY  
VIENNA, 1980

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## FOREWORD

The Agency programmes concerned with uranium resources have primarily been directed towards the exploration for and geology of uranium deposits and the processing of uranium ore. Late in 1976, an Advisory Group met in Rome to review the subject of evaluation of uranium resources. That meeting led to the present symposium, devoted to reviewing the technique of resource evaluation and studying methods of mining uranium.

Not all presently known resources can actually be exploited and, even if they could, production rates from them would be seriously reduced by the mid-1990s. It is therefore most important that exploration for uranium resources be carried out promptly. In this respect, many countries have started national programmes either to determine their uranium potential, or to explore for uranium deposits. The symposium, held in Buenos Aires at the invitation of the Government of Argentina, provides these countries with a report on the latest developments in the various methods of uranium resource evaluation.

The programme of the symposium was divided into five subject areas, the first of which consisted of introductory papers reviewing briefly the problems of ensuring adequate supplies of uranium to meet world demands, and the closely related problem of classifying uranium resources and defining those classifications for international reporting.

The second subject area considered methods of estimating ore reserves. Most notable among these was geostatistics which, although still under development, has gained very wide acceptance because of its versatility. The general availability of computers has made the development of sophisticated methods of estimation such as geostatistics possible, but on the other hand older classical methods, properly applied, are still valid, although less versatile.

The third subject area covered techniques of winning uranium from its several sources. In addition to mining by conventional open-pit or underground methods, in-situ leaching of low grade ores in special environments and of ores left in mines is of growing importance. Further, virtually all marine phosphates contain some uranium that can be recovered as a by-product in the manufacture of phosphoric acid, while uranium is also recovered from copper leach liquors as a by-product, and from gold ores in South Africa as a co-product.

An accurate, comprehensive and understandable appraisal of the world's undiscovered uranium resources is absolutely essential for making meaningful decisions in relation to the future supply of nuclear fuel. The fourth subject area covered methods used to appraise undiscovered uranium resources. Notable

among these methods is one based on interactive genetic models. The goal is to use more geological data and depend less on the intuition and experience of the estimator, thus limiting the amount of subjectivity inherent in most of the currently used appraisal techniques.

The final subject area dealt with production capability. The uncertainty of uranium supply cannot be dealt with only by knowledge of resources. The capability of the industry to produce from those resources at a rate necessary to satisfy the demand must also be considered.

The Agency is grateful to all those who contributed papers and took part in the discussions. Thanks are especially due to the General Chairman, Dr. John A. Patterson, Director, Division of Uranium Resources and Enrichment, US Department of Energy, and the seven session Chairmen. Special thanks are also due to Dr. Felix Rodrigo and his staff at the Comisión Nacional de Energía Atómica, who had the basic responsibility for the symposium arrangements in Argentina. Finally, sincere gratitude is expressed for the invitation to hold the symposium in Buenos Aires and for the general hospitality to the participants and to the Agency staff afforded by the Government of Argentina and the staff of CNEA.

The Agency records with deep regret the death, a few months after the symposium, of James Cameron, a world expert in uranium geology who had first-hand practical knowledge of prospection and mining in many countries. For more than a decade he planned and oversaw all the Agency programmes in uranium geology, mining and resources, and made telling contributions to the search for and development of uranium resources in a large number of Member States. Something of the range of his activities is reflected in the two papers of which he is author and co-author in these Proceedings.



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## **INTRODUCTORY PAPERS**

**(Session I and Session II, Part 1)**





## WORLD URANIUM SUPPLY AND DEMAND

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### Abstract

#### WORLD URANIUM SUPPLY AND DEMAND.

The role of nuclear energy is under increasing scrutiny and uncertainty. None the less, there will be an increasing need for expansion of uranium supply to fuel committed reactors. Longer-term demand projections are very uncertain. Improved knowledge of the extent of world resources and their availability and economics is needed to support planning for reactor development, especially for breeder reactors, and for fuel-cycle development, especially enrichment, and reprocessing and recycle of uranium and plutonium. Efforts to date to estimate world uranium resources have been very useful but have largely reflected the state of available knowledge for the lower cost resources in regions that have received considerable exploration efforts. The IUREP evaluation of world resources provides an initial speculative estimate of world resources, including areas not previously appraised. Projections of long-range supply from the estimated resources suggest that the high-growth nuclear cases using once-through cycle may not be supportable for very long. However, additional effort is needed to appraise and report more completely and consistently on world resources, the production levels attainable from these resources, and the economic and price characteristics of such production.

#### 1. Introduction - Nuclear Energy and the Uranium Question

The future role of nuclear energy is currently a subject of serious discussion and review in many countries. The current commitment to nuclear energy will provide a substantial contribution to future energy supply. Nonetheless, that contribution will be much less than had been expected just a few years ago. One of the uncertainties affecting the future growth of nuclear energy, the types of reactors to be employed, and related fuel cycle practice is the extent of uranium resources, their availability, and their costs.

As concepts of resources are fundamental to many nuclear decisions, it is vital that government and industry planners have available as sound information as is possible. There has been increasing recognition of the inadequacies of available resource assessments as an acceptable basis for decisions involving major national commitments which impact on energy supply for 20 to 50 years ahead. This recognition has in turn increased attention and effort to the questions of how we can better assess world uranium resources and better understand what the availability and economics of that uranium

might be. Such questions are difficult to answer. They involve uncertainties on the geologic nature of uranium deposits and on the nature of unexplored, and in some cases unmapped, areas of the world as well as technical and institutional aspects of discovery, mining, and processing ore from the deposits, and marketing the product.

In addition, there have been increasing political uncertainties, as many governments are getting involved in resource development and utilization. Environmental and safety concerns have added new dimensions to regulation of uranium production making planning much more difficult. New limitations on trading and use of uranium to minimize weapons proliferation have been imposed.

## 2. Uranium Demands - The Uncertainties

The future role of convertor reactors, primarily light water reactors (LWR), is a basic question. There are a number of additional nuclear development issues that impact on uranium demand and in turn are influenced by uranium supply and economics. Enrichment production capacity and mode of operation of existing plants and new capacity, and employment of advanced enrichment technologies, such as the gas centrifuge or laser systems, will influence demand and be influenced by concepts of uranium supply and economics. Improved enrichment technology can reduce costs and allow use of lower enrichment plant tails assays reducing uranium requirements and providing lower cost enriched uranium fuel. This option is particularly attractive if one expects high uranium prices. If the uranium to separative work cost ratio is high enough, retreatment of accumulated stocks of depleted uranium will be attractive.

The question of reprocessing of spent fuel from light water reactors and the recycling of the recovered plutonium and uranium is of continuing concern. It is clear that the economics of reprocessing and recycling are not as attractive as had been thought 4 or 5 years ago, even though uranium prices have increased substantially. Furthermore, increasing concerns about potential proliferation of atomic weapons as a consequence of plutonium availability are raising questions about the wisdom of plutonium recycle and forcing the development of alternate technological approaches and institutional arrangements to minimize proliferation opportunities. Improved confidence in the availability of economic uranium can allow deferral or perhaps bypassing of reprocessing for LWR recycle.

There is general agreement that the long term role of light water reactors is likely to be a limited one. That is, there will be a limitation on how much light water reactor capacity can be economically supported with available uranium. At some time, if we are to continue to use nuclear energy we must move to use of reactors that are most efficient in their use of uranium, and have the ability to use a larger amount of the isotope U<sup>238</sup>. The question of proper timing for transition to advanced types of reactors has been debated since the beginning of the nuclear era. It is still an unresolved question. The large financial commitment involved in developing and employing new reactor systems, the long lead times

(20-30 years or more), and the serious consequences to a nation's energy availability and hence its economic status among nations have made it difficult to develop firm plans that are generally accepted, and that can be adhered to.

These issues and related matters are the subject of a major international review thru the International Nuclear Fuel Cycle Evaluation (INFCE) which involves the active participation of over 50 countries. INFCE is providing an in-depth analysis of basic questions relating to fuel cycle and reactor technology development and employment. The question of proliferation and safeguards is receiving a fresh re-examination. Uranium availability is a principal subject area being evaluated. INFCE is scheduled to complete its work in early 1980. It is clear from what has already been accomplished, that it will have a significant impact in many countries on approaches to development and utilization of nuclear energy. Hopefully, the study will provide a better understanding of the basic issues and a resulting clarification of the future role for nuclear energy.

The subjects that have been discussed regarding the future of nuclear energy all influence future uranium demands. Since there is uncertainty on so many aspects regarding nuclear energy, it is difficult to develop forecasts of future uranium requirements that can be relied on for a long enough time to allow development of plans that can be an effective national commitment to a specific course of action. As a consequence, many reactor and fuel cycle technology development programs are, in effect, for the development of options to be used to the extent and when future conditions warrant. Since there seems to be few options we are ready to abandon, the tendency has been to keep as many viable as possible.

One of the keys to resolving planning uncertainties will be through improvement in our knowledge on the extent of world uranium resources, the economics of those resources, and our ability to discover and produce the uranium on a schedule that would support acceptable and credible scenarios of future energy development.

### 3. Nuclear Growth - The Forecasts

In spite of the uncertainties involved in forecasting demands, forecasts are necessary to provide a basis for understanding the future and for planning. Therefore, it is useful to examine some recent projections of nuclear power growth and related uranium requirements. In Figure 1 are illustrated the nuclear growth projections of the Nuclear Energy Agency (NEA) Working Group on Uranium Demand as published in the report "Nuclear Fuel Cycle Requirements," February 1978. These projections exclude the USSR, Eastern Europe, and China. Two cases were developed, a "Present Trend" case which reflects the future rate of development of nuclear power as it appeared to the Working Group in 1977. A second case, the "Accelerated" case, was developed to indicate the potential for expansion of nuclear capacity, if the need developed and if firm national decisions and appropriate commitments were made. This case is very much higher than

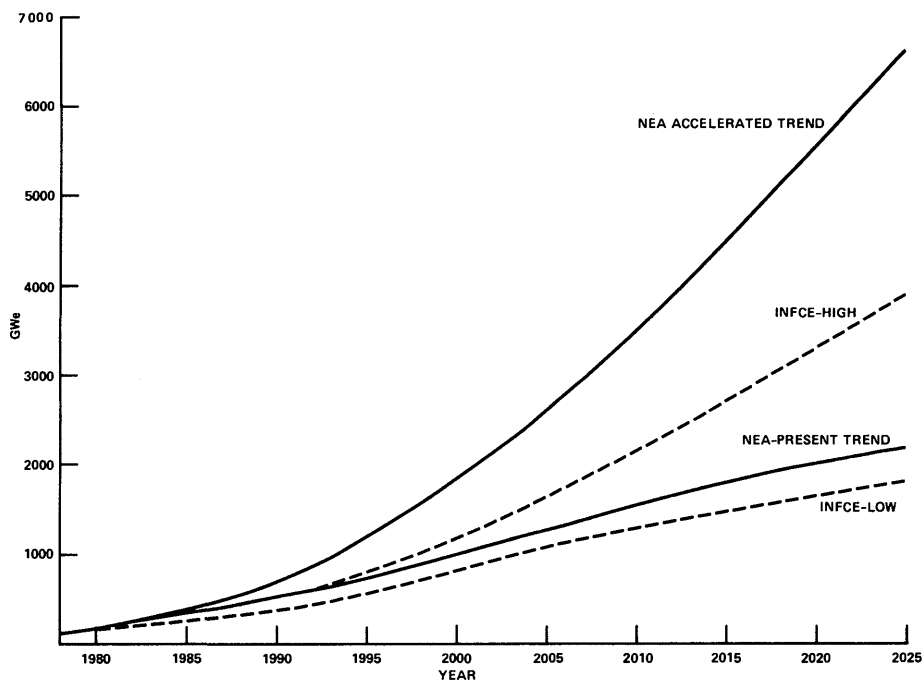


FIG.1. World nuclear power projections (t uranium). (Excluding countries with centrally planned economies).

the Present Trend case. Current indications are that it is very unlikely that such a course could be followed. It appears that the "Present Trend" case may also be somewhat optimistic.

Also shown in Figure 1 are the nuclear growth projections developed by INFCE. The INFCE Low case parallels but is slightly less than the NEA Present Trend case. The INFCE High case while calling for almost twice the nuclear capacity in 2025 as the Low case is still well below the NEA-IAEA Accelerated case. The variations in uranium requirement between these forecasts are considerable. When coupled with possible variations in reactor mixes and fuel cycle practice, the demand variations are even larger.

#### 4. Uranium Demands

The magnitude of uranium requirements thru the rest of the century can be examined thru analysis of the NEA forecast. In Figure 2 are illustrated the annual uranium requirements for the years 1990 and 2000, considering no recycle of uranium and plutonium, and cases assuming these fuels were to be recycled. World requirements for the Present Trend case in 1990 would vary from 85,000-102,000 metric tonnes of

	YEAR		CUMULATIVE
	1990	2000	1977 - 2000
<b>PRESENT TREND CASE</b>			
NO RECYCLE	102,000	178,000	2.3 MILLION
WITH RECYCLE	85,000	125,000	1.9 MILLION
<b>ACCELERATED CASE</b>			
NO RECYCLE	156,000	338,000	3.6 MILLION
WITH RECYCLE	126,000	242,000	2.9 MILLION

*FIG.2. World uranium requirements (t U). (NEA/IAEA Projection)*

	ANNUAL REQUIREMENTS 2025	CUMULATIVE REQUIREMENTS 1977 - 2025
LWR'S ONLY, NO RECYCLE .25% TAILS	350,000	9 MILLION
LWR'S, RECYCLE	210,000	5.8 MILLION
ADVANCED LMFBR LWR	95,000	5.8 MILLION
LWR & HWR (THORIUM CYCLE)	145,000	6.4 MILLION

*FIG.3. Long-range world uranium requirements (t uranium). NEA/IAEA present trend extended case. (2200 GW(e) in the year 2025).*

uranium. In the year 2000 requirements would vary from 125,000-178,000 tonnes of uranium. These levels compare with the approximate 35,000 tonnes of uranium produced in 1978. Cumulative requirements thru the rest of the century would be from 1.9-2.3 million tonnes of uranium.

For comparison the uranium requirement of the Accelerated case are also shown in Figure 2. Requirements in the year 2000 would be almost double the Present Trend requirement. Cumulative requirement would be from 2.9-3.6 million tonnes of uranium.

As shown in Figure 1, nuclear growth beyond the end of the century is expected to be large but the uncertainties are very great. Requirements for the Present Trend extended case of the NEA shown in Figure 3, for 2025 could be as low as 95,000 tonnes if advanced liquid metal fast breeder reactors were in use or could be as high as 350,000 tonnes if only light water reactors were used and there was no recycle of uranium or plutonium. Cumulative uranium requirements over the period 1977-2025 would vary from 5.8 million to 9 million tonnes of uranium.

	REASONABLY ASSURED		ESTIMATED ADDITIONAL	
	\$80/kg U (\$30/lb U <sub>3</sub> O <sub>8</sub> )	\$130/kg U* (\$50/lb U <sub>3</sub> O <sub>8</sub> )	\$80/kg U (\$30/lb U <sub>3</sub> O <sub>8</sub> )	\$130/kg U* (\$50/lb U <sub>3</sub> O <sub>8</sub> )
NORTH AMERICA	<u>702</u>	<u>929</u>	<u>1,181</u>	<u>1,882</u>
U.S.**	530	707	777	1,158
Canada**	172	189	404	706
Mexico**	0	6	0	2
Greenland**	0	27	0	16
AFRICA	<u>592</u>	<u>757</u>	<u>194</u>	<u>311</u>
South Africa**	247	391	54	139
Niger	160	160	53	53
Algeria	28	28	50	50
Gabon	20	20	5	10
C.A.E.**	18	18	8	8
Zaire	2	2	2	2
Somalia	0	5	0	2
Madagascar	0	0	0	2
Namibia**	117	133	30	53
AUSTRALIA	<u>290</u>	<u>299</u>	<u>47</u>	<u>53</u>
EUROPE	<u>60</u>	<u>380</u>	<u>50</u>	<u>90</u>
France	37	52	24	44
Spain	7	7	9	9
Portugal	7	8	1	1
Yugoslavia	5	7	5	21
U.K.	0	0	0	7
Germany	2	2	3	4
Italy	1	1	1	1
Austria	2	2	0	0
Sweden	1	300	3	3
Finland	1	2	0	0
ASIA	<u>20</u>	<u>50</u>	<u>0</u>	<u>20</u>
India**	4	30	1	24
Japan	8	8	0	0
Turkey	4	4	0	0
Korea	0	3	0	0
Philippines	0.3	0.3		
SOUTH AMERICA	<u>85</u>	<u>90</u>	<u>67</u>	<u>72</u>
Brazil**	62	62	58	58
Argentina**	23	29	4	9
Chile	0	0	5	5
Bolivia	0	0	0	0.5
TOTAL (Rounded)	<u>1,749</u>	<u>2,505</u>	<u>1,539</u>	<u>2,428</u>

\* Includes resources at \$80/kg U level

\*\* Revised since OECD report

SOURCE: "Uranium Resources, Production and Demand;" OECD Nuclear Energy Agency and the International Atomic Energy Agency, Dec. 1977.

FIG.4. World uranium resources by continent. (Excluding countries with centrally planned economies). (t U × 10<sup>3</sup>). C.A.E. = Central African Empire.



CONTINENT	NO. OF COUNTRIES	SPECULATIVE RESOURCES (MILLION TONNES U)
AFRICA	51	1.3 - 4.0
AMERICA, NORTH	3	2.1 - 3.6
AMERICA, SOUTH & CENTRAL	41	0.7 - 1.9
ASIA & FAR EAST*	41	0.2 - 1.0
AUSTRALIA & OCEANIA	18	2.0 - 3.0
WESTERN EUROPE	22	0.3 - 1.3
SUBTOTAL	176	6.6 - 14.8
EASTERN EUROPE, USSR, PEOPLES REPUBLIC OF CHINA	9	3.3 - 7.3**
TOTAL	185	9.8 - 22.1

\* Excluding Peoples Republic of China and the eastern part of USSR

\*\* "Estimated Total Potential"

*FIG. 5. International uranium resource evaluation project. (US \$50 speculative resources listed by continent).*

## 5. Uranium Resources

These requirements can be put into perspective by reviewing current information on world uranium resources. The most authoritative information is that published by the NEA-IAEA Working Party on Uranium Resources. In Figure 4 are tabulated data from their December 1977 report with some revisions to reflect updated data available since publication date. The table indicates estimated reasonably assured resources at the cost level of \$130/kilogram of uranium (which corresponds to \$50/lb U<sub>3</sub>O<sub>8</sub>) of around 2.5 million tonnes of uranium. There is also an estimated additional resource of 2.4 million tonnes at the \$130/kilogram cost level.

Most of these estimated resources are in only a few countries. There are many countries for which current estimates indicate very minor resources. There are many more countries for which no estimates of resources are available at all.

The incomplete nature of the world assessment has led to the organization of efforts to produce a more complete geographic appraisal of world resources. A major effort is the International Uranium Resource Evaluation Project (IUREP) which is also carried out under the auspices of the NEA-IAEA. The first phase of IUREP is complete and has resulted in an order of magnitude estimate of resources for some 185 countries based on expert review of the geology of those countries, past exploration efforts, and known uranium occurrences.

Uranium resources, in addition to the Reasonably Assured and Estimated Additional Resources, that could be discovered and exploited at costs of less than \$130/kilogram as estimated in IUREP, by continents, are shown in Figure 5. A total of 6.6-14.8 million tonnes of uranium were estimated, most of

which are in the same continents as those containing the Reasonably Assured and Additional Estimated Resources. Estimates were also made for countries of eastern Europe, Russia, and China. A total 3.3-7.3 million tonnes were estimated, for this group of countries. This includes Reasonably Assured and Estimated Additional Resources, since separate estimates are not available. This project is proceeding to future phases of work designed to develop improved knowledge of uranium favorability and resources in certain countries. This program will take many years to complete but should lead to much improved assessment of world uranium resources.

#### 6. By-Product

In addition to these "conventional" type resources, an additional source of uranium is as a by-product from phosphoric acid and copper leach solutions. Production from these sources will be largely dependent on the growth of phosphoric acid production and copper leach activity. Appraisal of these resources, therefore, involves some special problems. While by-product uranium can make a significant contribution to world supply, perhaps 10,000 tonnes per year or so by the end of the century, it is likely to be a small percentage of annual needs.

#### 7. Higher Cost Intermediate Grade Resources

Uranium producible at costs higher than \$130/kilogram is of economic interest as fuel in light water reactors and could be more economical than certain fuel cycle or reactor options. Therefore, there is increasing interest in appraising the extent and availability of such resources. There are known additional higher cost resources associated with the ore deposits included in current \$130/kilogram ore reserve estimates. For example, in the USA, it is estimated there are some 300,000 tonnes of uranium, in known deposits, that is at a higher grade than 0.01 percent  $U_3O_8$  and which would require costs of over \$130/kilogram to produce. These U.S. resources are, for the most part, in sandstone type deposits. Certain other known types of deposits, such as the Pre-Cambrian conglomerates of Canada and South Africa are also known to have substantial associated lower grade resources. Appraisal of the extent of such resources is hampered by the lack of past commercial interest in them, and hence low levels of exploration and development work and incomplete data. Nonetheless, there is a need to better appraise such high cost resources if we are to improve long nuclear energy planning.

#### 8. Shales

Carbonaceous shales are of particular interest because of their large size and the potential co-products that may be recovered. The principal shales deposits are in Sweden and the USA. However, other shales are also possible uranium sources. The Swedish shales at 300 ppm are of interest at current uranium prices. The U.S. Chattanooga shale at a

grade of 60 pp would require higher uranium prices. However, recent studies indicate that the Chattonooga shale might be a source of uranium at prices of \$180-200 per kg with recovery of the petroleum, minor metals, sulfur, and ammonia. Such prices could make it an economically attractive source of fuel for light water reactors. While much remains to be worked out in the technical aspects of uranium, petroleum, and metal recovery, the work to-date is sufficiently promising that serious consideration must be given to the shales as a potential source of supply.

#### 9. Improving Resource Estimates

There is extensive information available about uranium resources. Probably more effort has gone into appraisal of uranium resources in the last 20 years than for any other of the other metals or fuels. Nonetheless, there are many unanswered questions. We must have more complete field information and the involvement of more countries. We must improve techniques for appraisal of the extent and availability of resources. There is still much to be done to raise the quality of estimates and to assure consistency between various workers and countries so aggregated values can be meaningful.

There is considerable confusion about the significance of the terms that are used to define and classify resources and the economic criteria used for evaluating them. There is confusion amongst estimators. The confusion is greater for those who are not conversant with the arts and practices of resource appraisal and mineral industry concepts. It will require a continuing effort by appraisers of resources to improve their criteria and standards and to explain them more clearly. We must clarify the concepts and assumptions that have been employed in the appraisals, the significance of the resource estimates, and better communicate how they can be used, or should not be used.

#### 10. Uranium Availability

In addition to appraising the extent of resources, it is vital to understand the limitations that may exist on the discoverability and producibility of the estimated resources in light of projected future uranium needs. There is no set timetable by which estimated resources can be expected to be discovered or produced. History has shown exploration in a favorable area may require many years of intensive effort before a significant deposit is found. Additional time is needed before production is obtained.

There are, in addition, inherent limitations on the production levels from a specific deposit or area that can be reached because of the physical and geologic nature of the deposit and its geographic setting, and because of policies in the countries in which the resources occur.

Projections of attainable production levels can be made quite simply if one deals with the near-term and draws on only reserves or reasonably assured resources. Such projections are largely concerned with current production capacity and

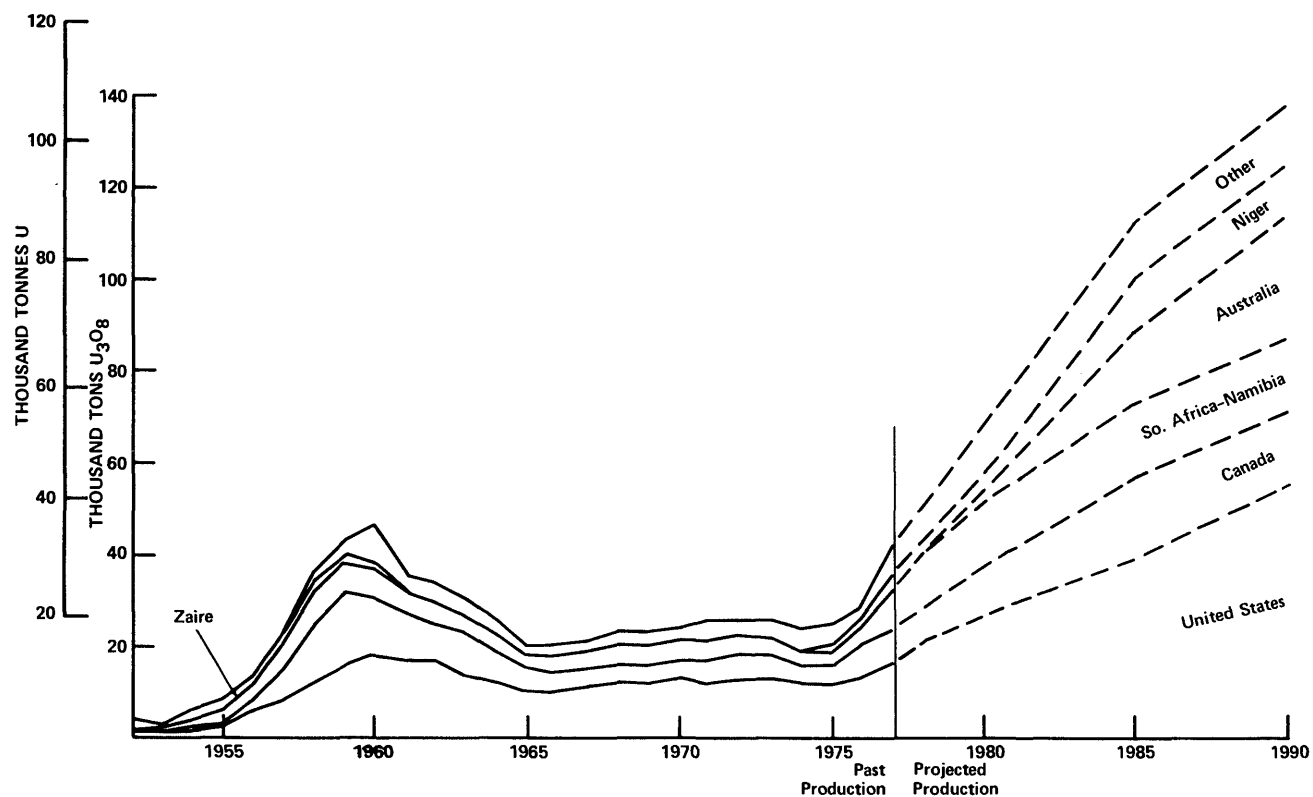


FIG.6. Annual world uranium production. (Excluding countries with centrally planned economies).

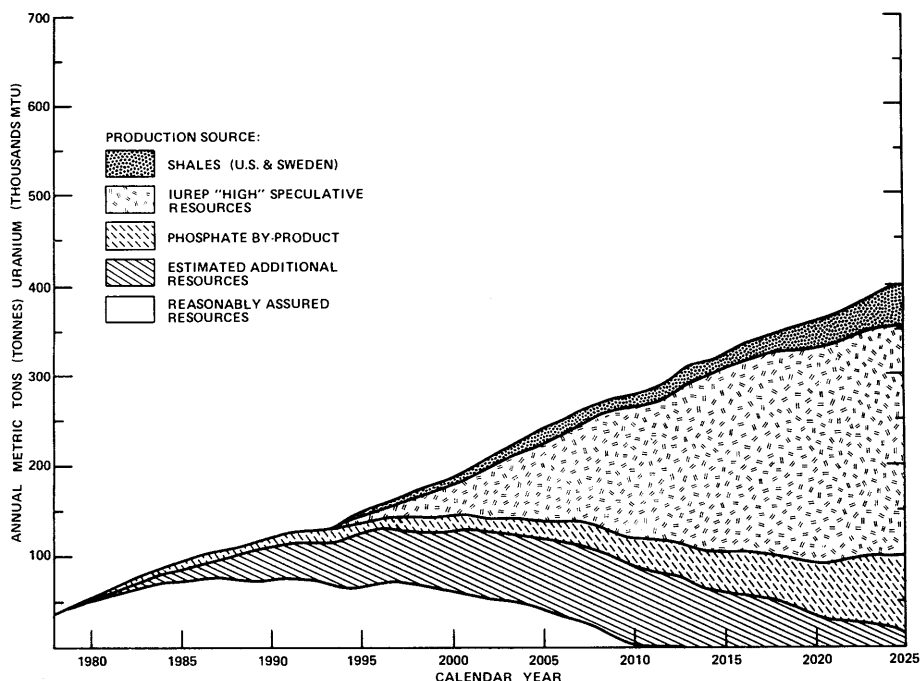


FIG.7. INFCE study – total projected uranium production capability based on 'optimistic' development scenario and IUREP 'high' resources.

firm plans for additional facilities. Analysis of production attainable becomes more difficult and tenuous when considering the estimated additional and speculative resources, and the longer term.

Projections of attainable short-term production capability as developed by the NEA/IAEA Working Party on Uranium Resources, are shown in Figure 6 together with historic production levels. World production capacity has been growing at a rate of around 6,000 tonnes of uranium per year over the last few years. This is close to the growth rate experienced in the 1950's. A similar rate of growth is projected to be attainable at least through the mid-1980's, largely based on production from the USA., Canada, South Africa, Namibia, Australia, and Niger.

In an attempt to develop some idea of potential long range uranium supply, we have analyzed possible production levels attainable from the estimates of Reasonably Assured and Estimated Additional Resources, IUREP estimates of speculative resources, and phosphates and shales. The analysis of these resources considered exploration lead times, mine and mill construction lead times, and typical production capacity expansion rates. These lead times and rates were

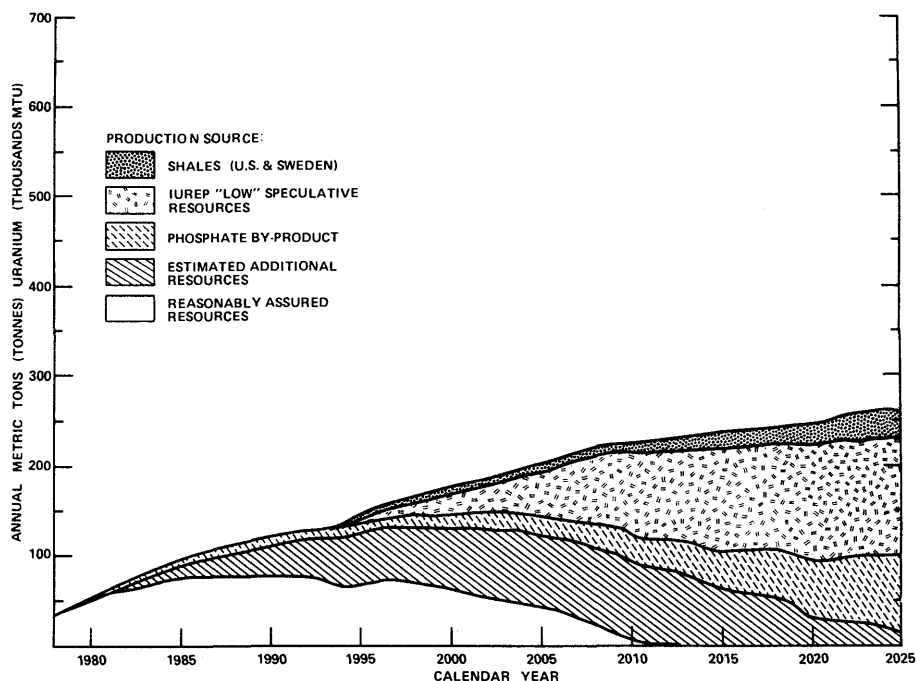


FIG. 8. INFCE study – total projected uranium production capability based on 'optimistic' development scenario and IUREP 'low' resources.

varied according to the geologic nature of the resources, with the state of development of uranium resources, and with the industrialization level in the countries containing the resources. The projections are shown in Figure 7 for IUREP High speculative resources and in Figure 8 for the Low speculative resources. The figures show production from reasonable assured resources peaking in the late 1980's and early 1990's and from estimated additional resources around the end of the century. Contributions from speculative resources could allow production rates to continue to grow to levels of around 400,000 tonnes per year from the IUREP High estimates and to around 250,000 tonnes with IUREP Low estimates by the year 2025. By-product production from phosphoric acid and copper leaching could add tens of thousands of tonnes per year toward the end of the period. Production from shales was considered to start around the end of the century when they could be of economic interest.

Comparison of these projections to the previous demand projections suggests that there is considerable present doubt that the high energy level projections when using once through reactor and fuel cycle systems can be assured of a fuel supply.



The factors influencing uranium availability from deposit to deposit and area to area are not yet well established and are changing with time. Considerable room for improvement exists in defining these factors and in improving the methodology for projecting production capability, particularly for the long term and from speculative type resources. The technique of performing such analyses is in its early stages. The fundamental importance of understanding possible future uranium availability as well as the levels of resources requires that we continue to make such projections and strive to improve them. Several of the key variables in understanding future supply will be examined in other parts in this symposium.

## 11. Resource Economics

An element of resource appraisal of importance, particularly for uranium is the economic availability of the resources estimated. This includes the cost of producing the estimated resources and the prices that may result. Uranium resources of economic concern in the long term future, the next 50 years, may have considerably higher production costs than those of current commercial interest. Therefore, confining our appraisal to the types of resources currently available in the market place or expected to be soon available would provide an inadequate measure of the resources available and of interest for nuclear planning. It is clear that in many locations and situations uranium at much higher costs than currently experienced would still be competitive with oil and coal, or with alternative energy types being developed, such as solar or geothermal power. On the other hand, if we worked on appraisals that considered only the high cost resources, we would have inadequate information on uranium availability and costs in the near term.

Varying criteria and evaluation concepts have been used in performing economic evaluations. Criteria used include market prices, total cost plus return on investment, forward costs, or using factors such as ore grade as a substitute for or indication of economics. Improvements in the definition of suitable economic criteria and the more general adoption of such criteria would materially improve our concepts of potential future costs and prices of uranium. An aid to improving economic evaluation would be more complete and detailed information on actual costs of exploration and production for various types of deposits in various geologic, geographic, and political settings. Additional data and generalized deposit character-cost relationships need to be developed and to be made more generally available to resource appraisers. A continuing effort is needed to develop and maintain good cost information. The technology of exploration and production has been evolving; environmental, health, and safety regulations have been increasing; new ore deposit types, in new areas, are being found. Inflation and changing exchange rates between currencies adds to the complexity of performing current studies and complicates future projections.

## 12. Conclusion

Although the future role of nuclear energy is uncertain and undergoing current review and analysis it is clear that there will be a continuing growth in the need for uranium. The uncertainties in growth are accompanied by uncertainties about fuel cycle practice and reactor types to be used. World resource appraisal has received much attention but there is still considerable uncertainty about resources since many areas are incompletely known. Appraisal of resources must be accompanied by analysis of production levels attainable and the economics and prices of such production. The longer term outlook especially needs more attention. This symposium can assist in displaying the state-of-the-art for many of the related technological concerns and point the way to development of improved future estimates and evaluations.

# REFLEXIONS SUR L'APPROVISIONNEMENT MONDIAL EN URANIUM A L'HORIZON 2000 ET PERSPECTIVES ULTERIEURES

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## Abstract-Résumé

### CONSIDERATIONS REGARDING THE WORLD SUPPLY OF URANIUM UP TO THE YEAR 2000 AND THE SUBSEQUENT OUTLOOK.

In assessments of the dependability of the western world's uranium supply between now and the year 2000, and also of the conditions under which the next century will have to be faced, the following questions may be raised: (1) Are the resources in the ground sufficient? Are the reserves equal to those of other mineral substances? In the light of present estimates of speculative resources, when will it become necessary to develop the breeder system? The study concludes that exploration will have to be intensified and preparations made now for the commercial introduction of breeders as from the beginning of the next century; (2) Is the mining industry able to meet the increasing demand? Despite the reductions in nuclear power programmes which have helped mask the difficulties of the mining industry, the growth in demand continues to be rapid; however, it should be possible to cope with this if the uncertainties of the market do not discourage investors in the mining industry; (3) Will the mining industry have to face the possibility of a sudden surge in demand? The possibility of an abrupt reduction in supply or a sudden increase in demand cannot be ruled out. A crisis scenario based on assumptions which the authors consider to be realistic would result in a 300 000 t increase in demand within 15 years; (4) Can uranium circulate freely between the producer and the consumer? This is the most disquieting problem, in view of the unequal distribution of resources between the world's consumer countries and producer countries (not including the United States of America) and of the embargo risks threatening international trade on such a strategic product. The average level of dependence of consumer countries (except the United States) for uranium supplies is around 80 per cent; this situation is likely to induce the less privileged countries to develop processes — such as the breeder system — which conserve uranium and thus reduce their dependence.

### REFLEXIONS SUR L'APPROVISIONNEMENT MONDIAL EN URANIUM A L'HORIZON 2000 ET PERSPECTIVES ULTERIEURES.

Pour apprécier le niveau de sécurité de l'approvisionnement en uranium du monde occidental d'ici l'an 2000 et les conditions dans lesquelles sera abordé le prochain siècle, on peut se poser

les questions suivantes: 1) Les ressources en terre sont-elles suffisantes? Les marges de réserves sont-elles équivalentes à celles des autres substances minérales? En fonction des estimations actuelles des ressources spéculatives, quand devra-t-on développer la filière des surrégénérateurs? L'étude conclut à la nécessité d'intensifier l'exploration et de se préparer dès maintenant à l'introduction commerciale des surrégénérateurs dès le début du prochain siècle. 2) L'industrie minière est-elle apte à suivre la croissance des besoins? Malgré les réductions de programme nucléo-électrique qui ont contribué à masquer les difficultés de l'industrie minière, la croissance des besoins reste rapide mais devrait être satisfaite si les aléas du marché ne découragent pas les investisseurs miniers. 3) L'industrie minière peut-elle avoir à faire face à une éventuelle accélération brutale des besoins? Une réduction brutale de l'offre ou une augmentation soudaine des besoins ne sont pas à écarter. Un scénario de crise bâti sur des hypothèses que les auteurs considèrent comme réalistes provoquerait une augmentation des besoins de 300 000 tonnes en 15 ans. 4) L'uranium pourra-t-il circuler librement du producteur au consommateur? Il s'agit là du problème le plus préoccupant compte tenu de l'inégalité de répartition des ressources entre pays consommateurs et pays producteurs dans le monde (Etats-Unis d'Amérique mis à part) et des risques d'embargo qui pèsent sur le commerce international d'un tel produit stratégique. Le niveau moyen de dépendance des pays consommateurs (hors Etats-Unis) pour l'approvisionnement en uranium se situe aux environs de 80%; cette situation est de nature à inciter les pays défavorisés à développer des procédés économisant l'uranium tels que la filière des surrégénérateurs, diminuant ainsi leur dépendance.

## INTRODUCTION

Le contexte de l'approvisionnement mondial en uranium évolue rapidement et la nature même des problèmes change. Notre but est ici d'en faire un tour d'horizon et de situer leur importance relative.

Pour cela, faute de données suffisantes, nous limiterons notre étude au monde hors pays à économie planifiée, et nous examinerons les 4 problèmes qui peuvent se poser quant à la satisfaction des besoins mondiaux en uranium d'ici l'an 2000 :

- 1) Les ressources en terre sont-elles suffisantes ?
- 2) L'industrie minière est-elle apte à suivre la croissance des besoins ?
- 3) Peut-on avoir à faire face à une perturbation imprévue du marché ?
- 4) Le fait que la capacité de production globale prévisible permette de satisfaire les besoins à moyen terme est-il suffisant pour garantir la sécurité d'approvisionnement ?

Il est apparu au cours de l'étude que l'on ne pouvait raisonner de façon globale au niveau mondial ; en particulier on a été souvent conduit à isoler le cas des Etats-Unis qui, par leur situation particulière d'important producteur, se distinguent des autres consommateurs.

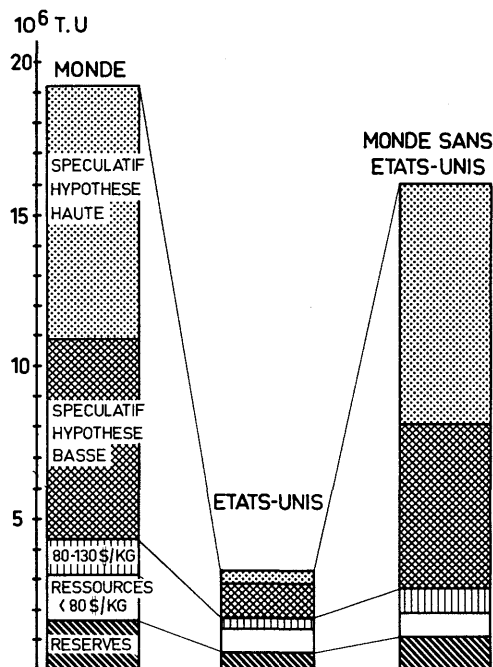


FIG.1. Ressources en uranium au 1<sup>er</sup> janvier 1977.

#### 1 - RESSOURCES EN TERRE ET BESOINS CUMULES

La figure 1 rappelle les chiffres des "ressources calculées" (ensemble des ressources raisonnablement assurées et des ressources supplémentaires estimées) établis conjointement par l'OCDE et l'AIEA (réf. 1), ainsi que ceux des "ressources spéculatives" évaluées par ces mêmes organismes dans le cadre de l'International Uranium Resources Evaluation Project (IUREP) (réf.2).

Il convient de noter que ces ressources spéculatives et leur répartition reposent uniquement sur des raisonnements géologiques et que les chiffres expriment les quantités actuellement envisageables de minerais de qualités économiques comparables à ceux des ressources calculées par l'OCDE/AIEA.

On observe que les ressources mondiales calculées (4,3 MtU) sont 2,6 fois plus élevées que les réserves proprement dites estimées par l'AEN/AIEA à 1,65 MtU. Les ressources spéculatives correspondent à 1,5 fois le total des ressources calculées dans l'hypothèse basse et 3,5 fois dans l'hypothèse haute.

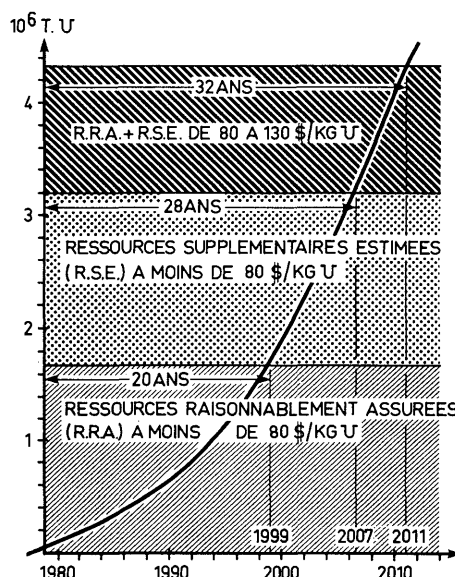


FIG.2. Comparaison des besoins mondiaux cumulés avec les ressources en uranium actuellement comptabilisées.

Un essai de répartition est fait entre Etats-Unis et Monde hors Etats-Unis, en assimilant les définitions américaines des ressources possibles et spéculatives (réf.3) à celles des ressources spéculatives basses et hautes adoptées par l'IUREP.

Il apparaît alors que si les ressources calculées des Etats-Unis représentent environ 40 % des ressources mondiales de la même catégorie, en revanche leur part dans le spéculatif n'est plus que de 10 %, ce qui traduit bien que cette région du monde est beaucoup mieux prospectée en moyenne que les autres.

La figure 2 permet de comparer ces ressources mondiales aux besoins cumulés, qui ont été évalués sur la base de nos dernières estimations du développement de la puissance électronucléaire.

Pour faciliter cette comparaison, il nous a paru préférable d'exprimer ces prévisions de besoins sous la forme de courbes simples plutôt que de zones de fluctuation possible entre scénarios extrêmes, en raison même de l'instabilité des hypothèses.<sup>1</sup>

<sup>1</sup> Ces évaluations ont été faites avec une teneur de rejet des usines d'enrichissement  $Nw = 0,2\%$  et les puissances électronucléaires installées suivantes:

1990	Monde	475 GWe (dont USA 175 GWe)
2000	Monde	1000 GWe (dont USA 300 GWe)
2010	Monde	2000 GWe (dont USA 550 GWe).

Nous avons pris en compte un recyclage limité et commençant en 1990. L'introduction des surrégénérateurs qui a été retenue (30 GW en 2000) ne modifie pas sensiblement la courbe des besoins cumulés avant 2010.

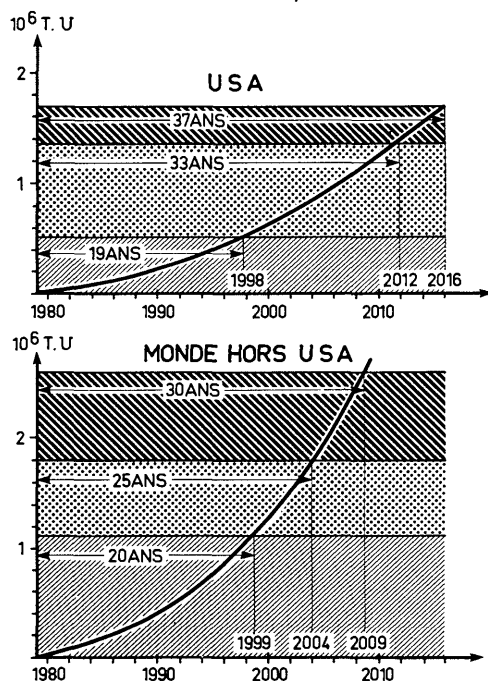


FIG.3. Besoins cumulés et ressources en uranium.

Ainsi, les ressources à moins de 80 \$ kg/U (réserves au sens AEN/AIEA) permettent de satisfaire les besoins mondiaux pendant 20 ans, ce chiffre étant porté à 28 ans si l'on y ajoute les ressources supplémentaires estimées dans la même catégorie de prix, et à 32 ans avec l'ensemble des ressources calculées.

Le côté rassurant de tels délais doit être néanmoins tempéré par la certitude qu'une fraction, non négligeable, de ces ressources ne pourra sans doute jamais être extraite en raison de contraintes dues à l'environnement (certains pays y seront plus sensibles que d'autres) ou encore de contraintes économiques. Il ne faut pas oublier que les coûts de référence des diverses tranches de ressources sont des "forward costs", qu'ils excluent par conséquent les dépenses déjà effectuées, les impôts sur les bénéfices, les charges financières et la rémunération du capital, et doivent être utilisés avec précaution lorsqu'il s'agit de prévoir l'arrivée de ces tranches sur le marché.<sup>2</sup>

Ces considérations globales masquent une caractéristique supplémentaire : celle d'une grande hétérogénéité géographique.

Si on se limite à la comparaison Etats-Unis/Monde hors Etats-Unis, on constate en effet sur la figure 3 que, d'ici l'an 2000, les situations

<sup>2</sup> En moyenne, on peut considérer que les réserves ne peuvent être exploitées que si le prix de vente est supérieur d'environ 50% à l'évaluation du "forward cost" (coût en aval).

**TABLEAU I. COMPARAISON DES RESERVES ET DES BESOINS D'ICI L'AN 2000 POUR LES PRINCIPAUX PRODUITS MINIER**

Eléments	Réserves en 10 <sup>6</sup> t de métal	Réserves/besoins cumulés de l'année d'évaluation à l'an 2000
V	9,7	7,5
Cr	523,2	5,7
Mn	1 814	4,9
Fe	90 500	4,5
Ti	300	4,4
Al	3 483	4,0
Ni	55,3	2,1
Mo	6,0	1,4
Cu	408,2	1,3
Sn	10,2	1,3
W	1,8	1,2
Pb	49	1,2
Zn	41	1,1
U	1,65	0,9

sont voisines. En revanche, de l'an 2000 à 2020, la masse des ressources américaines, associée à une croissance relative plus faible des besoins, conduira les Etats-Unis vers un avenir moins préoccupant. Il convient par conséquent aux autres pays de développer vigoureusement leur effort d'exploitation qui en principe devrait donner globalement de bons résultats, puisque la masse des ressources spéculatives est répartie essentiellement à l'extérieur des Etats-Unis d'après les évaluations de l'IUREP.

Si l'on compare par ailleurs le cas de l'uranium à celui d'autres substances minérales, on constate, comme le montre le tableau I (réf.1,4,5), que parmi un ensemble de 13 produits miniers, l'uranium occupe la dernière place si l'on compare ces réserves à la demande cumulée d'ici l'an 2000. Il en serait d'ailleurs de même si l'on prenait les ressources totales dans les catégories économiquement exploitables.

Pour maintenir 20 ans de réserve, le monde devrait dès maintenant trouver annuellement quelque 140 000 tonnes d'uranium.

Selon certaines opinions, la distribution naturelle de l'uranium serait homogène et la répartition des ressources calculées correspondrait à celle des dépenses de prospection effectuées. La part américaine dans les



TABLEAU II. DISPERSION DES RESERVES ET DE LA PRODUCTION POUR LES PRINCIPAUX PRODUITS MINIERES

Eléments	Pourcentage des <i>réserves</i> détenues par			Pourcentage de la <i>production</i> en 1977 venant de		
	1 pays	3 pays	5 pays	1 pays	3 pays	5 pays
Cr	73,9	96,5	98,2	28	61,3	78,4
Ti	65,9	93,0	98,2	22,1	53,1	71,1
Mn	45,0	90,5	97,7	34,3	65,1	82,5
V	74,7	94,8	97,3	40,2	87,9	95,2
Mo	49,5	79,1	96,5	58,4	87,3	99,1
U en 1977	31,7	67,7	87,5	37,8	81	93,8
en 1990				42,7	71,8	90,2
W	53,6	74,6	87,1	21,6	48,0	60,5
Ni	43,7	69,4	87,0	30,2	62,4	78,2
Al	26,0	67,6	78,1	30,8	57,1	70,8
Fe	31,1	59,4	76,0	29,4	52,7	66,6
Pb	35,6	58,0	71,9	17,3	44,5	58,2
Sn	23,6	50,8	69,0	26,1	51,7	71,3
Zn	22,8	55,0	68,5	20,0	53,5	56,5
Cu	18,4	44,7	58,0	17,0	43,9	61,8

ressources calculées mondiales pourrait venir à l'appui de ce point de vue, les Etats-Unis ayant fourni un effort de prospection particulièrement intense dont la mesure est également donnée par la valeur très faible du ratio ressources spéculatives/ressources totales déjà évoqué plus haut.

Cependant, si l'on considère la densité moyenne de minéralisation (c'est-à-dire la quantité globale de ressources en uranium - spéculatives comprises - par kilomètre carré de territoire) des Etats-Unis et celle du reste du Monde occidental, on trouve respectivement les valeurs 340 kgU/km<sup>2</sup> et 170 kgU/km<sup>2</sup> : c'est-à-dire que le territoire des Etats-Unis apparaît ainsi deux fois mieux minéralisé en moyenne que le reste du Monde occidental.

Le manque de coïncidence entre la répartition géographique des ressources calculées et celle des pays futurs consommateurs est bien connu. Ces derniers se souciant de prospector leurs propres territoires avant de porter leurs efforts ailleurs, il est évident que cette hétérogénéité de répartition de l'uranium n'est pas simplement le reflet de l'effort de prospection, mais traduit un caractère physique. Il ne s'agit pas du reste d'une particularité propre à l'uranium. Bien d'autres éléments la présentent à des



FIG.4. Répartition des ressources calculées (RRA + RSE de l'AEN/AIEA, 1977) du monde "occidental".

degrés divers : comme le montre le tableau II (réf.1,4,5), les trois quarts des ressources en vanadium, titane, chrome, manganèse, tungstène, molybdène, niobium, platine se situent dans trois pays seulement et pour près de vingt autres métaux, 65 % des ressources se localisent dans cinq pays.

L'examen du planisphère des ressources calculées (fig.4) montre que le nombre de provinces uranifères connues est assez limité et que chacune d'elles a des dimensions très réduites. Cette constatation entraîne que l'on ne peut se fier à aucune méthode d'estimation des ressources et des probabilités de découvertes qui s'appuierait sur des seuls raisonnements statistiques, et il est notoire que les chances de découvertes à l'intérieur d'une province connue sont plus élevées.

Cependant, les ressources de nombreuses parties du monde sont encore peu explorées et les chances de mettre à jour de nouvelles provinces uranifères sont loin d'être uniformes.

De l'examen de ces ressources, on peut donc retenir que, pour le programme de puissance installée actuellement prévisible, les ressources en terre connues paraissent suffisantes pour assurer la couverture des besoins d'ici l'an 2000 ; cependant, elles sont très inégalement réparties et le niveau d'exploration est également très différencié géographiquement sans que ceci suffise à expliquer cela. Les perspectives ne sont donc pas identiques pour tous et si les Etats-Unis (par exemple) ont un avenir à moyen terme relativement assuré, les autres pays doivent en revanche soutenir l'effort nécessaire pour découvrir l'uranium des ressources spéculatives, d'autant plus que le mode de répartition irrégulier rendra ces résultats très aléatoires ; ces difficultés vont encore augmenter dans le futur car une partie importante des ressources à trouver se situe probablement dans des conditions de gisement plus difficile à déceler (gisements aveugles, types de concentration encore inconnus ...).

En ce qui concerne les perspectives plus lointaines, il est évidemment plus difficile encore de faire des prévisions. Néanmoins, il est possible de se livrer à quelques réflexions à partir des chiffres de ressources spéculatives évaluées par l'OCDE et l'AIEA. Pour cet exercice, nous prendrons comme chiffre représentatif des ressources spéculatives la valeur médiane entre 6,6 et 14,8 soit 10,7 millions de tonnes et nous considérerons que la part qui pourrait être découverte et exploitée serait comprise entre 50 % (5,3 Mt) et 75 % (8 Mt).

En y ajoutant 4 millions de tonnes correspondant aux ressources actuellement recensées (déduction faite de la majeure partie des schistes noirs dont l'exploitation nous semble peu probable et d'autres pertes inévitables) on obtient une estimation raisonnable du potentiel uranifère.

En considérant qu'il faut une moyenne de 4500 tonnes d'uranium pour alimenter un réacteur de 1000 MWe fonctionnant pendant 30 ans, on peut calculer le parc total qui pourrait être alimenté par l'uranium disponible (tableau III).

La consommation retenue, 4500 tonnes/GWe pour 30 ans de durée de vie, est celle des réacteurs à eau ordinaire de la génération actuelle, fonctionnant en cycle ouvert. Pour tenir compte de l'hétérogénéité du parc, d'éventuelles améliorations des performances des réacteurs à eau ordinaire et des possibilités de recyclage de l'uranium il aurait fallu calculer une consommation spécifique moyenne. Toutefois, compte tenu du caractère illustratif de cette étude, l'hypothèse simplificatrice adoptée est suffisante pour évaluer la puissance électronucléaire totale dont l'alimentation pourrait être assurée à partir des ressources en uranium retenues.

**TABEAU III. POTENTIEL URANIFERE ET PUISSANCE  
ELECTRONUCLEAIRE CORRESPONDANTE**

	Nombre de GWe possible (30 ans de durée de vie) (4500 t/GWe)
- Ressources connues (RRA + RSE) (4 Mt)	890 GWe
- Ressources connues (RRA + RSE) + 50% ressources spéculatives (9,3 Mt)	2070 GWe
- Ressources connues (RRA + RSE) + 75% ressources spéculatives (12 Mt)	2670 GWe

On peut se demander maintenant quelle pourrait être l'évolution du rythme d'installation, le rythme de construction étant donné par les programmes électronucléaires connus pour les prochaines années.

On peut rendre compte des perspectives plus lointaines en s'appuyant sur le modèle de Fisher-Pry couramment utilisé pour simuler le développement d'une activité humaine ou la consommation d'un produit dont les ressources sont finies.<sup>3</sup>

Appliqué à la puissance installée cumulée, et ajusté pour passer par les points 150 GWe en 1980 et 1000 GWe en 2000, ce modèle permet de tracer la courbe de puissance totale installée en tenant compte d'une durée de vie de 30 ans (Fig.5) et la courbe de puissance installée chaque année (Fig.6).

On constate qu'avec les hypothèses qui nous semblent les plus réalistes (ressources connues + 50 à 75 % des ressources spéculatives), la puissance totale installée passe par un maximum de 1500 à 1900 GWe, à une date comprise entre 2015 et 2020 et la puissance installée chaque année atteint un maximum de 60 à 80 GWe au tout début du siècle prochain pour décroître ensuite. Ceci signifie que la somme des ressources calculées et spéculatives n'est pas suffisante pour soutenir un taux de croissance du nucléaire correspondant aux hypothèses retenue plus haut (voir chapitre 1, note 1).

On peut d'ailleurs remarquer que toute modification (retraitement et recyclage de l'uranium, augmentation du taux d'irradiation, diminution de la teneur de rejet à l'enrichissement) qui permettrait de diminuer les besoins en uranium pour la vie d'un réacteur aurait le même effet que l'augmentation du même pourcentage du plafond des ressources. Ainsi, un gain de 30 % sur les besoins d'un réacteur a le même effet que la prise en compte de 75 % des ressources spéculatives, plutôt que de 50 %, et ne repousse donc les échéances que de 5 ans.

Dans ces conditions, pour répondre à l'augmentation de la demande énergétique en continuant à accroître la puissance électronucléaire installée à

<sup>3</sup> Courbe logistique du type  $y = \frac{a}{1 + b \exp(-\lambda t)}$

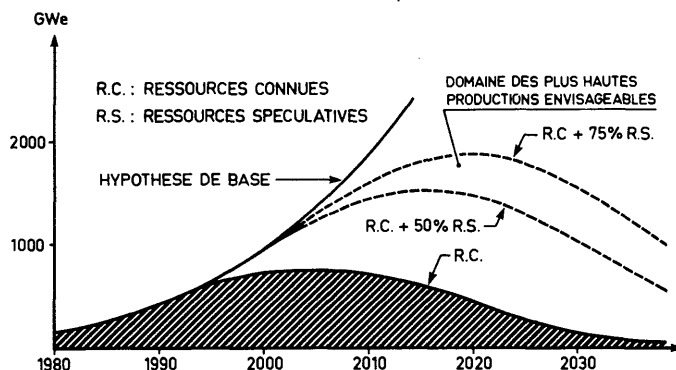


FIG.5. Puissance électronucléaire installée en LWR sans recyclage compatible avec les ressources en uranium.

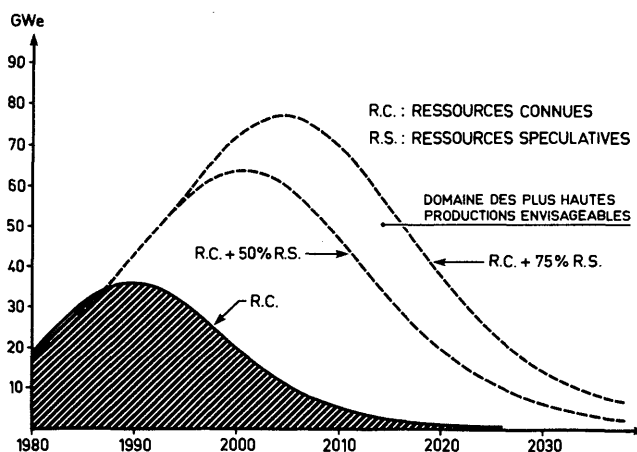


FIG.6. Démarrage annuel de réacteurs LWR compatible avec un niveau de ressources donné.

un rythme suffisant, il faudra développer une nouvelle filière dont la consommation en uranium soit d'un autre ordre de grandeur. La nécessité d'une introduction à grande échelle au niveau mondial des surrégénérateurs apparaît sur les courbes des fig. 5 et 6 dès l'an 2000, ce qui nécessite leur développement dans les pays industriels avancés dès les années 90.

## 2 - ADAPTATION DU DEVELOPPEMENT DE L'INDUSTRIE MINIERE A L'EVOLUTION DES BESOINS EN URANIUM

Pour l'industrie minière, les perspectives se sont profondément modifiées depuis 1973, comme en témoigne la figure 7. Ce graphique représente l'évolution des prévisions de besoins annuels mondiaux en uranium pour l'année

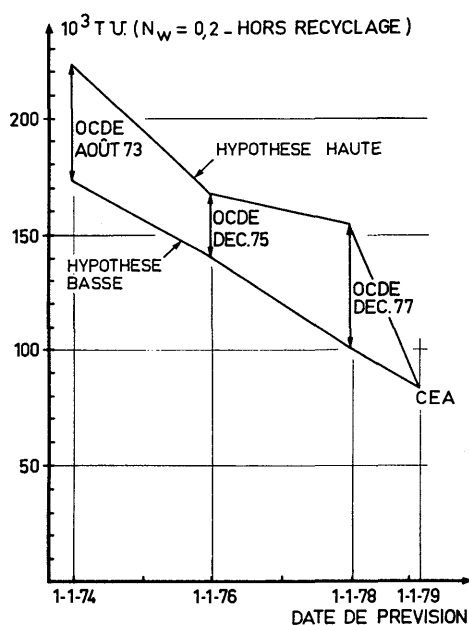


FIG. 7. Evolution des prévisions de besoins mondiaux annuels pour 1990.

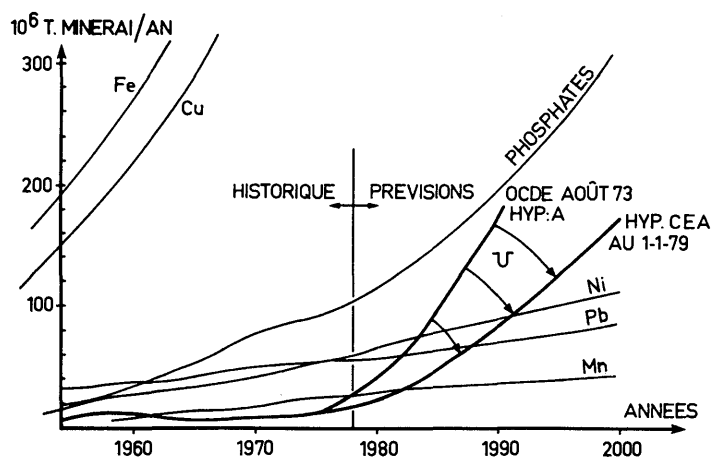


FIG. 8. Production des industries minières exprimée en quantité de minerai par an.

1990, en fonction de l'année au cours de laquelle la prévision a été faite (réf. 1, 6, 7). Depuis Août 1973, ces prévisions ont été réduites de plus de la moitié.

Il nous paraît cependant vraisemblable que les péripéties de la mise en place du nucléaire vont s'estomper devant les réalités de la faim énergétique, et que la tendance à la baisse ne se poursuivra pas.

Pour satisfaire la demande à l'horizon 1990, même dans le cadre des évaluations les plus basses, on peut voir sur la figure 8 que la croissance demandée à l'industrie minière de l'uranium demeure très forte (réf. 5, 8, 9) par rapport à celle des autres industries minérales, comme celles du plomb, du nickel ou du manganèse. Il apparaît en particulier que la quantité de minerai d'uranium à extraire par an, au début des années 90, sera équivalente à celle de minerai de phosphate extraite actuellement.

Si l'on tient compte du fait que les taux de découverte nécessaires pour exploiter l'Uranium en carrière sont plus importants que dans le cas des phosphates ou autres grandes substances minières, on constate que la date à laquelle l'industrie extractive de l'Uranium aura rejoint le niveau actuel de celle du phosphate (en termes de tonnage total déplacé) peut devenir encore plus proche.

Par ailleurs, les gisements d'uranium sont dans l'ensemble beaucoup plus petits que ceux des autres grandes substances minérales, ce qui constitue un handicap certain. Ainsi, sur 285 mines en service aux Etats-Unis en 1977, seulement 25 avaient une capacité supérieure à 150 000<sup>4</sup> tonnes de minerai au cours de cette année (réf. 10, 11). On sait que, pour l'ensemble des matières minérales, les mines de capacité supérieure à 150 000 t de minerai par an fournissent 90 % de la production de minerais ; or, dans le monde, parmi les 37 mines d'uranium d'une capacité inférieure à 150 000 t, seulement 3, c'est-à-dire 8 %, ont une capacité supérieure à 1 Mt/an, alors que dans le cas des phosphates on en trouve 41 sur 63, soit 65 %, et que pour l'ensemble des industries minérales, les chiffres sont de 476 sur 1052, soit 45,2 %. Cet obstacle à la mécanisation poussée des mines d'uranium est encore renforcé par le fait que le pourcentage des mines souterraines est beaucoup plus important que pour les phosphates, le cuivre, le fer, etc ... : plus de 60 % en nombre pour l'uranium contre moins de 25 % pour les phosphates dans la tranche des mines de capacités supérieures à 150 000 t de minerais/an.

Cette structure morcelée de l'industrie minière de l'uranium subsistera ; même si la mise en exploitation de quelques gros gisements en Australie, au Canada et au Niger permet d'ouvrir des mines supérieures au million de tonnes par an, leur pourcentage restera faible par rapport à l'ensemble de l'appareil productif, et on n'envisage pas dans un avenir proche l'ouverture de nouvelles exploitations du type de celle de Rössing, la seule qui soit comparable aux grandes mines de cuivre.

Compte tenu de ces caractéristiques et des décisions qui ont été prises jusqu'à présent, on peut se demander comment se présentent actuellement les perspectives de croissance de la production pour les dix prochaines années. C'est une entreprise très hasardeuse, car si les prévisions de besoins ont fortement baissé, il en est de même des prévisions de production. C'est pourquoi, simultanément, certains prévoient une pénurie en comparant les prévisions de besoins effectuées 1 ou 2 ans plus tôt avec les prévisions actuelles de production, alors que d'autres annoncent la surabondance en se basant sur d'anciennes prévisions de production et des prévisions de besoins actuelles.

<sup>4</sup> Soit une production d'environ 150 t d'uranium.

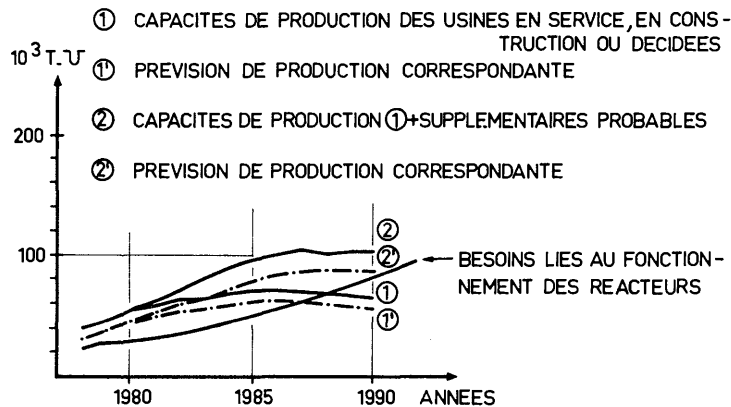


FIG. 9. Comparaison des besoins annuels et des prévisions de production: monde.

Le problème est de savoir si la baisse des prévisions de production correspond à une adaptation naturelle de l'industrie minière de l'uranium à la chute des prévisions de besoins, ou au contraire si elle correspond à des difficultés propres à la mise en production. Nous pensons que si le premier effet ne peut être entièrement écarté, il masque de façon notable les difficultés réelles de l'industrie minière à mettre en production de nouveaux gisements (main-d'oeuvre, incidents techniques et lenteur de montée en production, pressions politiques, autorisations et réglementations plus strictes, notamment sur la protection de l'environnement, etc ...). De ce fait, si un renversement de tendance, ou plus simplement une stabilisation des prévisions de besoins se produisait, les contraintes et le manque de souplesse de l'industrie minière pourraient, d'un seul coup, se faire fortement sentir.

La figure 9 permet de comparer les besoins annuels mondiaux aux capacités de production envisageables compte tenu des usines en service, en construction ou décidées et aux prévisions de production correspondantes en supposant un facteur de charge raisonnable.

L'excédent qui apparaît sur cette figure sera en fait largement réduit car :

- La courbe des besoins physiques de réacteurs a été construite hors contraintes liées aux contrats d'enrichissement et constitution des stocks de sécurité.

- S'il apparaissait des risques réels de surproduction, certains industriels réduiraient probablement leur production ou plutôt retarderaient au moins de nouvelles mises en exploitation.

Ces prévisions de production supposent de la part de l'ensemble des Etats une résolution politique qui écarte progressivement toutes les barrières ayant retardé plusieurs mises en route majeures ces dernières années. L'hypothèse paraît cependant justifiée par l'évolution qui se dessine dans des pays de poids déterminant, comme l'Australie et le Canada.



- ① CAPACITES DE PRODUCTION DES USINES EN SERVICE, EN CONSTRUCTION OU DECIDEES
- ① PREVISION DE PRODUCTION CORRESPONDANTE
- ② CAPACITES DE PRODUCTION ①+SUPPLEMENTAIRES ESTIMEES
- ② PREVISION DE PRODUCTION CORRESPONDANTE

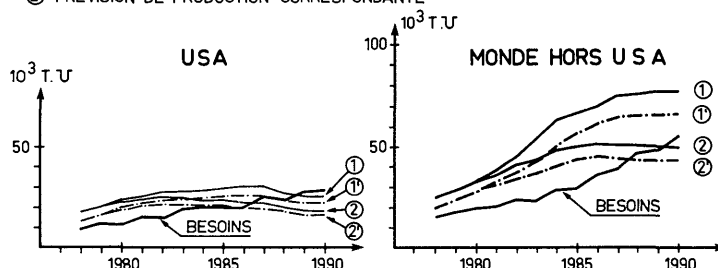


FIG.10. Comparaison des besoins annuels et des prévisions de production: Etats-Unis et reste du monde.

Sur la figure 10, le cas des Etats-Unis a été séparé de celui du reste du monde, ce qui met nettement en évidence que les possibilités d'excédent se situent essentiellement hors de ce pays. Les consommateurs américains auront donc probablement un intérêt économique à importer de l'uranium, ce qui accentuera davantage le déséquilibre constaté précédemment entre les cadences d'épuisement des ressources aux Etats-Unis et dans le reste du monde. On notera pour la deuxième moitié de la prochaine décennie le fléchissement des courbes de capacité des usines en service et commandées et la tendance au plafonnement des courbes qui tiennent compte des capacités probables. Il serait illusoire de considérer que le rythme des découvertes et des mises en service nouvelles, non comptabilisées dans ces graphiques, serait tel qu'il corrige notablement le tassement indiqué. En effet, parmi les très grands producteurs hors Etats-Unis, seuls l'Australie et le Canada semblent susceptibles de contribuer substantiellement et assez tôt à un tel redressement. A la suite de l'effort considérable de l'industrie en République Sud-Africaine, sa capacité atteindrait un maximum vers 85, avec peu de chances d'extension ultérieure, sauf bien entendu nouvelles découvertes majeures. De plus les politiques officielles de certains pays (stricte régulation du développement et des ventes, restrictions dans les domaines de l'utilisation des produits en aval du cycle) n'indiquent pas que ces Etats seraient prêts à laisser croître leur capacité de production jusqu'au maximum possible.

En résumé, il semble que l'industrie minière soit en mesure de soutenir le rythme d'expansion qui lui est demandé dans les dix prochaines années, mais que ses difficultés, aujourd'hui masquées par la baisse des prévisions de besoins, réapparaîtront brutalement en cas de stabilisation et surtout de remontée, et ce d'autant plus que des impressions fallacieuses de risques d'excédent auront fait baisser les cours et auront découragé les investisseurs miniers et surtout, bien entendu, ceux qui détiennent des intérêts dans des opérations à rentabilité modeste.<sup>5</sup>

<sup>5</sup> Les producteurs se protègent d'ailleurs par des contrats de vente à long terme conclus préalablement aux mises en production, et l'on voit même apparaître, compte tenu des accroissements des coûts et investissements miniers, des structures de facturations avec incorporation des prix de revient.

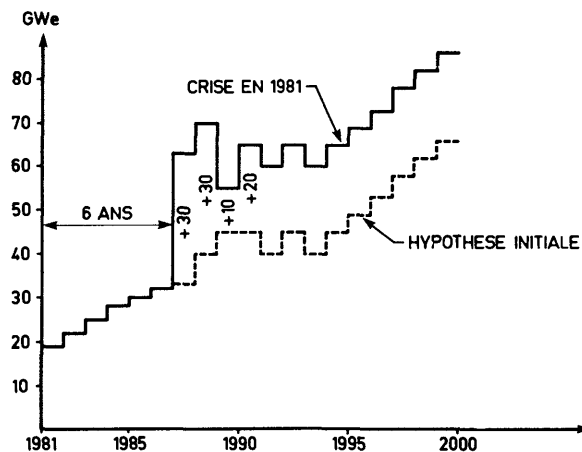


FIG.11. Hypothèse d'augmentation du rythme annuel de construction de centrales dans le monde en cas de crise en 1981.

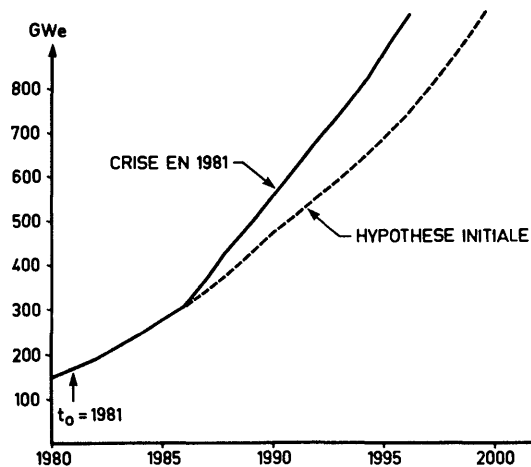


FIG.12. Puissance électronucléaire installée.

Cette perspective d'une éventuelle surcapacité à court et moyen termes a aussi pour effet de démobiliser le pré-financement des utilisateurs pratiqué communément jusqu'à ce jour (sous forme, notamment, d'avances sur livraisons), et de rendre plus difficiles de nouvelles mises en route.

### 3 - INCIDENCE SUR L'INDUSTRIE MINIERE D'UNE FLUCTUATION IMPREVUE DU MARCHE

L'actualité ne cesse de montrer, notamment dans le domaine énergétique, combien l'avenir est aléatoire, ce qui rend plus nécessaires encore les "marges de précaution".

Or, il n'est pas déraisonnable de penser que la production mondiale puisse être brusquement amputée d'une fraction non négligeable ou que n'intervienne une accélération rapide de la demande. Le premier type de crise pourrait correspondre au retrait partiel ou total d'un grand exportateur mondial. Ainsi, l'arrêt total d'une production équivalente à celle de l'Afrique du Sud entre 85 et 90, ou la réduction de moitié d'exportations équivalentes à celle du Canada ou de l'Australie entraîneraient au cours de cette même période un déficit d'environ 40 000 tonnes d'uranium. Mais un effet plus grave pourrait résulter d'un redémarrage brutal des commandes de centrales si l'industrie minière ne disposait pas des ressources et des moyens industriels nécessaires pour faire face à de telles fluctuations.

Pour illustrer ce point, nous avons imaginé une crise pétrolière grave intervenant en 81 ; il serait d'ailleurs plus exact de parler d'une prise de conscience soudaine du caractère inéluctable de la crise énergétique que, parmi d'autres, les experts pétroliers nous annoncent. Sur la figure 11 on a représenté une hypothèse de modification du rythme annuel de construction des centrales nucléaires dans le monde, à la suite de cette inflexion décidée brutalement en 1981. On a considéré que le raccourcissement des délais de construction, et la pleine utilisation des capacités disponibles de production de centrales permettaient de rajouter 60 GWe répartis sur 2 ans à  $t_0 + 6$  ans et  $t_0 + 7$  ans (fig.11). Puis un programme de nouveaux investissements lancé à  $t_0$  permettrait d'augmenter le rythme annuel de 10 GWe à  $t_0 + 8$  ans et de 20 GWe les années suivantes ; pour la simplicité du calcul, on a supposé qu'il s'agissait de réacteurs à eau ordinaire.

L'augmentation de la puissance électronucléaire totale installée correspondante est représentée sur la figure 12. Après l'accélération, le résultat sur la puissance installée est une augmentation supérieure à 200 GWe, soit près de 30 %.

Au niveau des besoins en uranium (fig. 13) l'effet est encore plus sensible du fait du poids des premières charges.<sup>6</sup> De plus, à cause des délais de cycle, l'augmentation des besoins s'amorce dès  $t_0 + 3$  ans et s'élève déjà à 6000 t à  $t_0 + 5$  ans, soit l'équivalent de la production de Rössing à cette date. Cet accroissement augmente d'année en année et, dix ans après la crise, atteint 14 000 t (production du Canada à cette date). L'accroissement de besoins cumulés s'élève à 300 000 tonnes (fig.14). 15 ans après la crise, ce qui représente environ dix fois la production mondiale de 1978.

Or cette période de 15 ans correspond aux délais de mise en production de nouvelles unités si les opérateurs ne disposent pas de gîtes suffisamment reconnus; on réalise les risques que pourrait présenter une situation où l'industrie minière n'aurait pas eu les moyens de se préparer, notamment par des explorations anticipées, à l'éventualité d'une augmentation soudaine des besoins.

#### 4 - TRANSFERT DE L'URANIUM DU PRODUCTEUR AU CONSOMMATEUR

En supposant que l'uranium en terre soit en quantité suffisante, que les capacités soient adaptées aux besoins,

<sup>6</sup> Dans un réacteur à eau ordinaire, une première charge nécessite 2,5 fois plus d'uranium qu'une recharge.

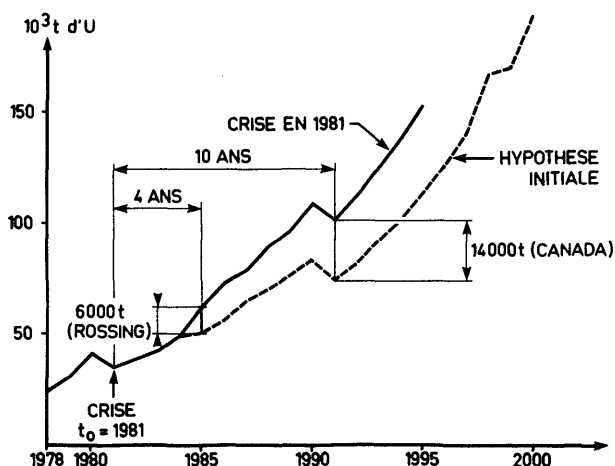


FIG.13. Besoins annuels en uranium naturel.

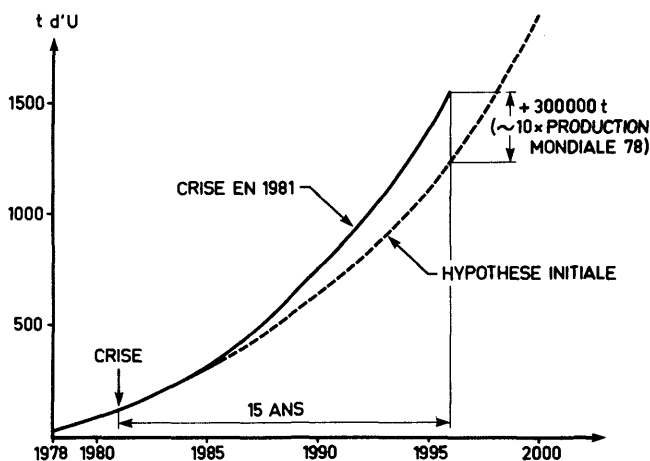


FIG.14. Besoins cumulés en uranium.

encore faut-il que le consommateur y ait un accès assuré. La figure 15 montre que d'ici l'an 2000, les principaux consommateurs resteront toujours les Etats-Unis, l'Europe et le Japon. Les autres parties du Monde (hors pays à économie planifiée) participeront pour 25% seulement à la consommation de l'an 2000. Or, la situation se présente de façon très différente pour les uns et pour les autres. La figure 16 représente la part des besoins en uranium qui pourrait être couverte par les productions nationales, autrement dit la

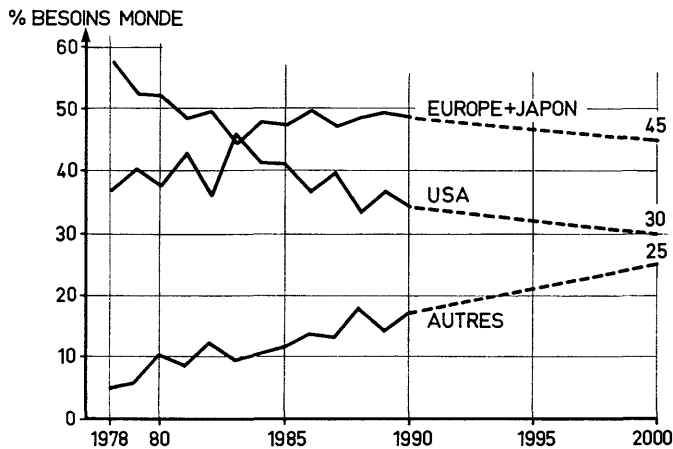


FIG.15. Répartition géographique des besoins mondiaux en uranium.

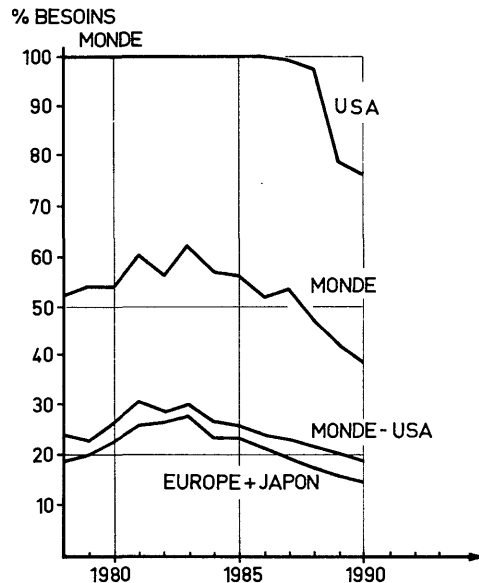


FIG.16. Evolution du taux de couverture des besoins en uranium par des productions nationales.

fraction qui n'aurait pas à franchir les frontières (en supprimant artificiellement les importations et les exportations non justifiées par des impératifs physiques). On constate alors que les échanges internationaux devront représenter près de 50 % du marché en 1980, et qu'ils dépasseront 60 % en 1990. Mais, si l'on exclut les Etats-Unis qui pourraient, en théorie, subvenir entièrement à leurs propres besoins, au moins jusqu'en 1987, le taux moyen de

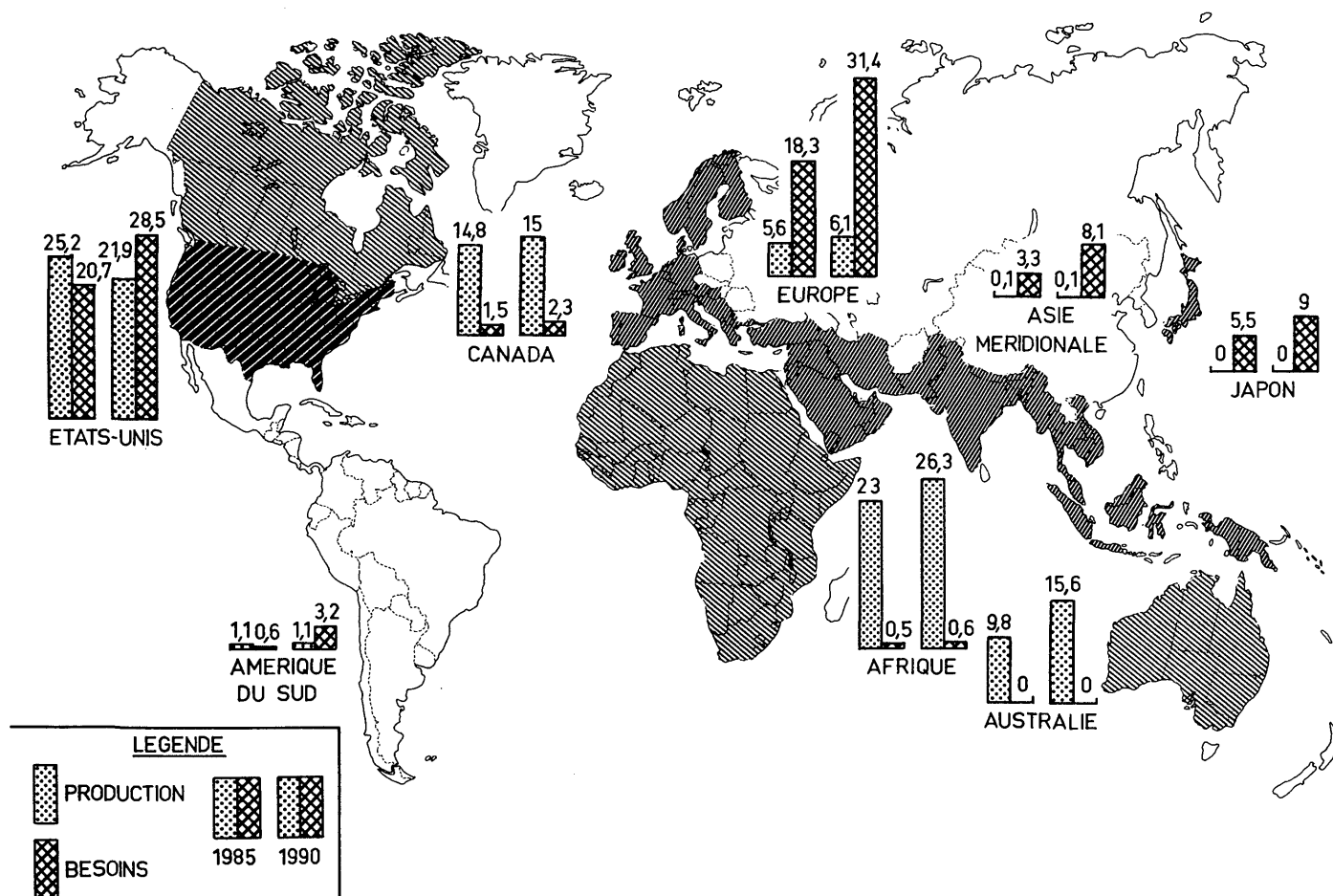


FIG.17. Comparaison des prévisions de production et de besoins annuels (en  $10^3$  t U) par zone géographique pour les années 1985 et 1990.

couverture des autres pays n'est plus que de 25 % en 1980 et de 20 % en 1990. En particulier, pour l'ensemble Europe-Japon, il faudra importer en moyenne 80 % des besoins en uranium, et cette dépendance augmentera encore en fin de période. Le planisphère de la figure 17 traduit également cette réalité. Pour chaque zone géographique, la comparaison des prévisions de production et de besoins en 1985 et en 1990 laisse apparaître qu'en dehors des Etats-Unis, ces grandes zones sont soit essentiellement productrices et donc exportatrices (Canada, Afrique, Australie), soit essentiellement consommatrices et donc importatrices (Europe, Asie Méridionale, Japon).

Cette constatation, du point de vue de la sécurité d'approvisionnement a une grande importance. En effet :

- A cause de la nature stratégique de l'uranium, les risques d'embargo sont plus importants que pour un produit banal. En particulier, le fait que les gouvernements des pays exportateurs contrôlent ces exportations et soumettent leur autorisation à certaines clauses parfois très contraignantes gêne la libre circulation de l'uranium.

- En dehors de tout problème politique, il est certain qu'en cas de pénurie les difficultés pour augmenter la production (environnement, disponibilité de main-d'oeuvre, lenteur administrative pour obtenir les permis) se résolvent beaucoup plus facilement lorsqu'il s'agit de faire face aux besoins vitaux de son propre pays que lorsqu'il s'agit d'exporter vers un pays éloigné.

- Pour se prémunir contre de tels aléas, la réaction normale des consommateurs est de stocker de l'uranium, ce qui est d'autant plus facile que le stockage de l'uranium est une opération aisée et peu coûteuse ramenée au coût du kilowattheure. Cette opération constitue une bonne protection pour le producteur d'électricité. Elle pourrait également en principe amortir les fluctuations du marché. En fait, on peut craindre que paradoxalement le résultat contraire soit obtenu, si les consommateurs essaient d'adapter leur niveau de stock en suivant de trop près l'idée qu'ils se font à un instant donné de la probabilité du risque de pénurie : si l'offre est supérieure à la demande, les cours ont tendance à baisser et parallèlement les détenteurs de stocks, jugeant leurs stocks de sécurité excessifs, s'en défont, ce qui augmente l'écart. Inversement, en cas de pénurie, les stocks sont jugés insuffisants, d'où une augmentation des besoins.

#### CONCLUSION :

En résumé, aux 4 questions que nous nous étions posées à propos de la sécurité de l'approvisionnement mondial en uranium d'ici l'an 2000, les réponses que l'on peut avancer sont les suivantes :

- Les réserves en terre paraissent actuellement suffisantes pour cette période, mais il est nécessaire de poursuivre un effort intense d'exploration pour maintenir le rapport actuel réserves/prévisions de production.

- Compte tenu des révisions en baisse des prévisions de besoins, l'industrie minière paraît en mesure de faire face à la demande actuellement

prévue, mais il faut bien prendre garde au fait que les difficultés rencontrées dans son développement ont conduit à des retards importants jusqu'ici masqués par des retards encore plus importants dans les programmes électro-nucléaires. Ceci pourrait changer en cas de stabilisation des besoins et surtout dans l'hypothèse d'une reprise de l'électronucléaire.

- L'adaptation de la production à une croissance brusque de la demande, consécutive par exemple à un redémarrage des programmes électro-nucléaires justifié par les tensions sur le marché des combustibles fossiles, poserait des problèmes insurmontables si l'industrie minière de l'uranium ne s'y préparait pas.

- Enfin, en dehors du cas particulier des Etats-Unis, l'uranium ne se trouve pas dans les zones mondiales de consommation. Ceci constitue un frein certain à l'adaptation de la production à la demande et d'autre part exige la libre circulation entre pays producteurs et pays consommateurs ; il s'agit certainement là du point le plus préoccupant en ce qui concerne la sécurité d'approvisionnement.

En conclusion, les recommandations que l'on peut faire pour améliorer la sécurité de l'approvisionnement des centrales nucléaires en combustible sont les suivantes :

- soutenir un effort d'exploration important, en particulier pour se donner les moyens d'accroître le rythme de production en cas de besoin;

- veiller à ne pas laisser les fluctuations du marché handicaper l'industrie minière ni l'empêcher de répondre aux objectifs de développement qui lui sont fixés pour le moyen et long terme;

- être conscient de la gravité du problème de la libre circulation de l'uranium entre pays producteurs et pays consommateurs;

- développer de meilleures techniques d'utilisation du contenu énergétique de l'uranium, permettant à moyen terme aux consommateurs de limiter leur taux de dépendance et à long terme de faire face à la raréfaction des gisements d'uranium à bas coûts d'exploitation.

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## DISCUSSION

B.S.I. MARENGWA: On what basis did you estimate the figures for speculative resources and world demand in future years given in your paper?

V. ZIEGLER: The estimates of speculative resources presented in the paper are those published in 1979 by NEA/IAEA in the IUREP (International Uranium Resources Evaluation Project) report.

No attempt is made in the paper to estimate actual production capacities on the basis of these speculative resources, but they are used, together with known resources calculated by means of the Fisher-Pry model, to give a rough idea of consumption, assuming that a maximum of 50–75% of the median value of these speculative resources will actually become available.

Demand is calculated on the basis of internal forecasts made by the French Commissariat à l'énergie atomique (CEA). It should be noted that demand is estimated at about half the high hypothesis of the 1977 NEA/IAEA Red Book and corresponds more or less with average INFCE (International Fuel Cycle Evaluation) forecasts.

A.E. BELLUCO: Were the IUREP speculative resources calculated by means of geological methods, or was some other system used?

V. ZIEGLER: The geology of the different countries involved was analysed in depth, in terms of the likelihood of finding uranium deposits. The principle of analogy was then applied, when conditions comparable with those located in known uraniferous provinces or districts were found, in order to estimate the quantity of uranium that the structure being studied could reasonably be expected to contain. This investigation was conducted on a country-by-country basis in the context of IUREP; the results were then put together and presented in terms of natural regions.

D.M. TAYLOR: I do not believe that it is correct to use an average value for speculative resources. The available values for these resources take the form of a range, and the actual amount of the resources could fall anywhere within this range. To take an average value as the most likely one may be misleading.

V. ZIEGLER: This question has been debated for more than two years in various international meetings without any solution having received unanimous approval.

D.M. TAYLOR: Also, when talking about known resources we must remember that there may be serious limitations on their availability. The 300 000 t of uranium in the Swedish shales, for example, may never be produced and, even if they were, the rate of production would be low (400 t/a).

V. ZIEGLER: It is true that part or all of certain resources included in the  $4.3 \times 10^6$  t of known resources by NEA/IAEA will never be available and that the figure of  $4 \times 10^6$  t used in our paper would be much too high if the extent to which these resources are likely to become available were taken into account. However, this was not the purpose of the paper.



## UTILITY INVOLVEMENT IN URANIUM EXPLORATION AND DEVELOPMENT – A GROWING TREND

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### Abstract

#### UTILITY INVOLVEMENT IN URANIUM EXPLORATION AND DEVELOPMENT – A GROWING TREND.

Nuclear power commitments by United States utilities at the beginning of 1979 represent 137 000 MW(e) of capacity from 147 nuclear units scheduled for operation by 1985 plus an additional 52 000 MW(e) from 48 units scheduled beyond 1985. Including the minority owners there are over 100 independent electric utilities in the United States of America with some financial commitment to these 195 nuclear power plants. United States uranium requirements to the year 2000 have been projected at 35 to 40% of the world requirements (exclusive of centrally planned economies). Also United States uranium resources represent a substantial fraction of present estimated world resources. Thus, decisions by US electric utilities regarding their financial involvement in uranium exploration and development can be expected to have a large impact on the development of the world-wide uranium industry. Unlike the situation in most countries with large commitments to nuclear power, the US government is not directly involved financially in uranium exploration and development except in a supportive role to the industry as a whole. Investment decisions by US utilities and US mining companies are based on their individual perceptions of the risks and benefits to be gained. Public attitudes towards nuclear power and public regulatory commission treatment of utility expenditures for resource development vary throughout the country. Thus, US utilities have shown a wide range of responses in formulating their uranium procurement strategies. About half the utilities with nuclear commitments are at present involved financially in uranium exploration and development. This paper traces the development of this trend and elaborates on the types of financial involvement and the factors that affect a utility's selection of its overall uranium procurement strategy. In the USA, for uranium at present under contract, the percentage coming from utility captive production rises from 4% in 1978 to 35% in 1985. Provided the present political uncertainty over the future of nuclear power in the USA is favourably resolved, the fraction of US requirements coming from utility captive production is expected to increase to over 50% before the year 2000.

## UNITED STATES NUCLEAR COMMITMENT

At the beginning of 1979 in the United States there were 69 nuclear power plants in commercial operation. These plants had a total net capacity of 50,970 MWe. Although this figure represents only 9% of the total electrical generating capacity of the USA, in 1978 nuclear power plants produced almost 13% of U.S. electricity. The difference in these percentages is indicative of the fact that nuclear power plants have lower operating costs than fossil plants and therefore are used primarily for base load power production.

NUS Corporation compiles and updates annually a listing<sup>[1]</sup> giving detailed information on commercial nuclear power plant commitments by U.S. utilities. A tabulation of the U.S. nuclear commitment based on this listing appeared in the Jan. 15, 1979 issue of Electrical World.<sup>[2]</sup> Table I from reference [2] shows that nuclear capacity is scheduled to increase to 147 units with 137,000 MWe by 1985 and that commitments exist for an additional 48 units. These 195 nuclear power plants have a total of 69 separate utilities as principal owner/operator. Including the minority owners of these plants, there are well over 100 independent U.S. electric utilities with some commitment to nuclear power.

## UNITED STATES URANIUM SUPPLY VS. DEMAND

Nuclear power developments within the United States can be expected to have a very large effect on the uranium supply and demand equation and on the development of the worldwide uranium supply industry. According to OECD-NEA estimates the U.S. was projected to have 48% of the world total (non-CPE<sup>1</sup>) installed nuclear capacity at the end of 1978. This percentage was projected to gradually decline but remained as large as 38% of the "present trend" projection at the end of 1990. Considering that after initial loading, reactors require a fairly constant supply of uranium over their 30 year lifetime, U.S. uranium requirement to the year 2000 have been projected at 35% to 40% of the non-CPE world requirements. U.S. uranium resources in the "reasonably assured" category producible at less than 80 U.S. \$/kg U have been estimated at 25-30% of the non-CPE world resources in this category. In the same cost category

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<sup>1</sup> CPE = Centrally Planned Economies

TABLE I. REACTOR COMMITMENTS BY UNITED STATES UTILITIES [2]

	<u>Units</u>	<u>MW(e)</u>
Operating Reactors	69	50,970
To operate in 1979	8	8,822
1980	11	11,195
1981	10	11,400
1982	14	15,138
1983	10	11,234
1984	18	20,292
1985	7	8,069
1986	11	12,962
1987	6	6,381
1988	7	7,852
1989	3	3,748
1990	2	2,050
1991	4	4,888
1992	--	----
1993	2	2,530
1994	--	----
Indefinite	11	11,553
Work Suspended	2	----
Total Committed	195	189,084

under "estimated additional" the U.S. is estimated to have as much as 50% of the non-CPE world resources. This high percentage is probably due, at least in part, to a greater relative effort within the U.S. to delineate resources in this category.

The shortfall between the projected U.S. uranium requirements and the reasonably assured domestic resources has been the subject of serious concern among U.S. utilities committed to nuclear power and among government energy planners.

## UNITED STATES URANIUM IMPORTS

NUS Corporation recently performed a detailed study<sup>[3]</sup> of the non-U.S.A. uranium supply situation under contract to the Electric Power Research Institute. Based on this work we<sup>[4]</sup> evaluated the role of foreign uranium in meeting domestic U.S. requirements. As of Jan. 1, 1978, foreign sales of uranium by U.S. producers represented only 5% of total domestic uranium sales. Purchases of non-U.S. origin uranium by domestic utilities represented about 11% of total domestic purchases. On average equal demand pressures on domestic and foreign resources would result if about 30% of U.S. requirements over the next 20 years were purchased from non-U.S. sources.

Considering the premium that all utilities must place on assurance of supply, and on the national interest in developing its indigenous energy resources, we believe it unlikely that U.S. imports will rise as high as 30% of domestic requirements. A more realistic estimate would be 20-25%. The presently identified U.S. uranium resources in the "reasonably assured" category available at less than \$130/kg U will satisfy the lifetime requirements of about 172,000 MWe of nuclear capacity. Import of 25% of our uranium requirements will allow very little growth in our presently committed nuclear capacity of 189,000 MWe without a substantial increase in domestic reserves. This can only be accomplished through continued expenditures for uranium exploration and development.

## UTILITIES NEED ASSURANCE OF LONG-TERM URANIUM SUPPLY

Among minerals being mined today uranium is unique in that it has only one end use - production of electrical energy by nuclear reactors. Thus, decisions by the uranium mining industry to develop new sources of supply are particularly sensitive to the perception of nuclear power growth. Exploration activities in the USA as evidenced by surface drilling statistics increased by 31% in 1977 from the level in 1976.<sup>[5]</sup> A further increase of 15% was estimated for 1978. However, utilities with firm nuclear commitments tend to discount such optimistic projections for several reasons: <sup>[6]</sup>

- Exploration and development drilling are very sensitive to market changes.
- Drilling efforts in the past have often been overestimated - frequently falling short due to constraints on financing, equipment, men and materials.

- Exploration and development decisions by the various sized oil and mining companies in the uranium business tend to be made in the narrower confines of their own individual interests and circumstances rather than just that of the industry demand.
- The industry is heavily involved in litigation.
- The prospect of new regulatory requirements on mill construction introduces considerable uncertainty in production costs.

A utility's commitment to a nuclear power plant can easily involve several billion dollars - in capital costs for construction and in enrichment and fuel fabrication contracts covering operating needs for several years in the future. Failure by a utility to deliver the uranium required under their enrichment and/or fabrication contracts would result in substantial cost penalties for alternate arrangements. Considering this level of financial commitment and that the useful life of a nuclear power plant is at least 30 years, utilities perceive their need for an assured supply of uranium over a time period that is long relative to the usual mining company projections of reserves for a given property (~7 years) and relative to the time required to produce uranium after beginning exploration (~10 years).

In an attempt to reduce the uncertainty involved in relying on the mining industry, by itself, to develop adequate uranium supplies, a few U.S. utilities with nuclear commitments, such as Southern California Edison Co., (in 1970) and TVA (in 1972) became involved in early uranium exploration activities. Since the mid 1970s providing assurance of supply for their long term needs has been a particularly strong factor in each utility's decision to finance uranium exploration and development. In recent years some public regulatory bodies have required the utilities to demonstrate that assured sources of fuel will be available for a minimum number of years of operation before necessary approvals for plant construction or rate increases will be granted.

#### DEVELOPING CAPTIVE PRODUCTION TO RESTRAIN PRICES

Notwithstanding their need for assurance of supply, most utilities did not begin to finance uranium exploration and development until after 1973 when assurance of a reasonable price for  $U_3O_8$  became their dominant motive. Prior to 1973 most utilities

were satisfied to take advantage of the depressed spot market for uranium and to pursue conventional short term and long term supply contracts.

Beginning in 1973 the realization by utilities and suppliers that near term uranium demand would outstrip supplies led to a dramatic change in procurement practices and to a rapid rise in uranium prices. This period was characterized by:

- rapidly rising spot market prices for uranium,
- reluctance by producers to enter into long term supply contracts,
- the necessity for renegotiating existing long term contracts,
- default on deliveries by one major supplier which is still under litigation.

The reaction to this situation by U.S. utilities has been to accelerate the trend to becoming increasingly involved financially in exploration, development, mining and milling - areas that traditionally have been the province of mining companies.

#### REGULATORY CONSIDERATIONS

Since U.S. utilities are protected by law from competition within their defined service areas, their operations and the prices they can charge their customers are subject to audit and approval by public regulatory commissions in each state. The state regulatory commissions are autonomous and independent and there is a wide diversity of opinion on such basic questions as the need for nuclear power. Utility service areas are not determined by state boundaries and quite often a single utility must answer to two or more regulatory commissions on decisions affecting its nuclear plants.

Public regulatory attitudes and rulings on how utility investments in uranium exploration and development should be treated in establishing prices for electricity vary throughout the country. In some areas investment in uranium supply ventures is recognized as a prudent activity for a utility with expenses allowed in the rate structure, with both the risks and the benefits accruing to the utility's customers. In other areas of the country regulatory commissions insist that such ventures be kept at arm's



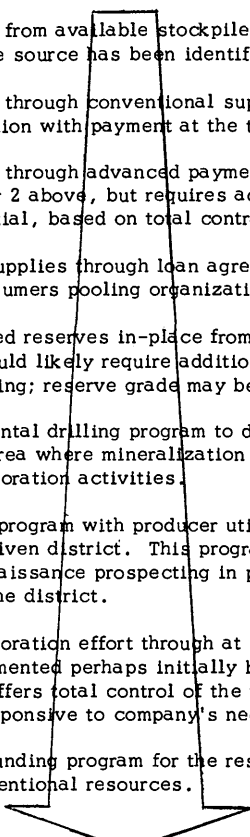
length from the utility so that the mining subsidiary would essentially have to compete with other suppliers to furnish the utilities uranium supply.

Two examples of the differences that exist in regulatory policies may be mentioned. Southern California Edison Company (SCE) made its decision to become involved in uranium exploration at a relatively early stage. In 1970 SCE formed Mono Power Company, a wholly owned subsidiary, and entered into a joint venture for uranium exploration with Rocky Mountain Energy Corp. Eventually this effort led to a decision to develop one of the joint venture's more favorable projects - a 900-t/d mine and mill at Bear Creek in Wyoming. The exploration program has continued to acquire rights to resources in the ground and to support development work on in-situ leaching projects. All this work has been funded by SCE's customers. In this case the utilities' customers take all the risks and will gain any benefits in the form of lower costs for electricity.

SCE is required to report annually to the California Public Utility Commission (PUC) on the results of its fuel procurement activities and to demonstrate the reasonableness of prices paid for fuel and energy. The California PUC's attitudes toward exploration and development may be summarized ☐ as follows:

- The adequacy of fuel and energy supplies available to electric utilities from traditional sources in the future is a matter of concern.
- Declining uranium supplies and escalating prices demonstrate the necessity for electric utilities to have some greater competitive leverage in dealing with traditional fuel supplies.
- Advance payments and interest-free loans utilized to secure a commitment of reserves or fuel supply are subject to approval through independent application proceedings.
- A substantial working interest position or the equivalent must be obtained in return for "up-front" money.
- Stockholder investment in exploration projects is discouraged because of the problems associated with establishing transfer prices.

1. Purchase uranium from available stockpiles, i.e., spot market. Minimal risk once source has been identified but high price.
2. Purchase uranium through conventional supply contracts; up to ten years in duration with payment at the time of delivery.
3. Purchase uranium through advanced payment supply contracts. Similar to Number 2 above, but requires advanced payments, possibly substantial, based on total contract quantity.
4. Secure uranium supplies through loan agreements or participate in a uranium consumers pooling organization.
5. Purchase identified reserves in-place from existing owners. This approach would likely require additional development or confirmation drilling; reserve grade may be lower.
6. Fund a developmental drilling program to define and evaluate resources in an area where mineralization has already been identified by exploration activities.
7. Fund exploration program with producer utilizing his past experience in a given district. This program may be broadened to include reconnaissance prospecting in potentially favorable areas bordering the district.
8. Develop own exploration effort through at least partial staffing in house, supplemented perhaps initially by outside consulting services. This offers total control of the program and ensures that it will be responsive to company's needs at any time.
9. Participate in a funding program for the research and development of non-conventional resources.



Increasing Risk but Increasing Potential  
for Obtaining Uranium at Lower Cost/  
Increasing Lead Time Required

*FIG.1. Methods of securing uranium supplies.*

In contrast to California, in New York State the Public Service Commission (PSC) questioned the wisdom of utilities entering the mining business specifically in connection with an application by Niagara Mohawk Power Corporation to guarantee \$18 million in promissory notes issued by its Texas subsidiary NM Uranium Inc. In this case the PSC held that the estimated costs of the uranium from the utilities project were so close to estimated future spot market prices that the attendant risks were not justified. An administrative law judge ruled that Niagara Mohawk decision in the uranium venture was, in fact, "not demonstrably irresponsible or imprudent based on the data

and forecasts available at the time the decision was made (mid 1975)." However, the judge recommended that the utility seek PUC approval before any further financing of its uranium subsidiary. In its final ruling in the case the PSC ruled that Niagara Mohawk could include in its rates only the lower of the costs of uranium from its Texas subsidiary or uranium purchased on the open market. This ruling, shielding the utilities' customers from risk, essentially requires the stockholders to assume all risks without a commensurate chance for profit. The ruling was alleged to be based on the fact that the utility had failed to seek prior PSC approval for its uranium venture and was not a commission policy decision on uranium ventures.

A policy decision may be forthcoming from extensive hearings before the same commission on the application of Consolidated Edison Company of New York (Con Ed) to invest \$15 million in uranium exploration through its subsidiary Fair Day Corp. Spokesmen have testified that Con Ed will not participate in such ventures unless it is allowed to pass the risk as well as the benefits of such ventures on to its customers. Con Ed has also testified that some mechanism must be found to allow utilities to negotiate and close uranium ventures without awaiting commission approval. Other New York State utilities have become intervenors in the case in support of Con Ed's position.

#### URANIUM PROCUREMENT METHODS AVAILABLE TO UTILITIES

Figure 1 identifies the various methods available to a utility for securing uranium supplies. In general, the methods are listed in order of increasing risk and increasing potential rewards in terms of obtaining uranium at lower cost:

##### 1. Spot Market Purchase

This method is normally used for small quantity purchases to adjust deliveries arranged several years before the existing requirements. As long as there is a significant demand there will usually be some uranium available for purchase on the spot market since this uranium usually results in the highest price for the producer. However, in recent years the quantities available for immediate delivery through the spot market have been relatively small and may be only a few hundred thousand pounds in any given month. Thus, a utility could not plan on obtaining a substantial portion of its requirement in the spot market unless it were also willing to maintain a high inventory level equal to at

least a two year supply. Carrying such a large inventory is expensive since carrying charges for most utilities are around 10% of the inventory value. At the present time many utilities especially those with no involvement in developing their own supplies have opted to pay the costs associated with carrying an inventory of 2 years or more in order to improve their assurance of supply.

## 2. Conventional Supply Contracts

This was the usual method for utilities to secure uranium prior to 1973. These contracts often covered delivery periods of 10 to 15 years, a time span comparable to the average life of a uranium mine. By selling the production beforehand, a producer can use such a contract to secure the necessary financing to complete his mine or mill facilities. Most long-term supply contracts negotiated since 1973 have contained provisions for delivery prices to be determined at periodic intervals throughout the contract duration; usually a year or so before delivery. To protect the suppliers against an unforeseen decline in demand such contracts usually specify a floor price consisting of a base price plus escalation. In recent years the concept of a negotiated or "World Market Price" has been used in some long-term contracts but there have been great difficulties in arriving at a suitable definition of such a price. These contracts usually provide for independent arbitration if there is no agreement on the "World Market Price" but the arbitrators have demonstrated a unique capacity for unpredictability that can easily leave one party to a contract in a state of shock unless the original contract is carefully worded to limit the arbitrators choices.

## 3. Advance Payment Contracts

In the strong seller market that existed in 1974-1976, competition among potential purchasers led to the introduction of various forms of advance payments. Security for such payments depend upon the reliability and reputation of the producer, his reserves and financial strength, and the time span between payment and deliveries. The two most common forms of security are prepayment bonds written by insurance companies, or a lien on, or security interest in, an ore body, reserve, stockpiled ore, or other yellowcake in the can.

#### 4. Loan Agreements and Pooling

Utilities with near term needs and longer term surpluses theoretically could arrange a loan either through a producer or with another utility who had an available stockpile. Such arrangements would help utilities minimize their uranium inventory carrying charges. Utilities' concerns about the borrower's ability to replace the borrowed uranium when needed have limited the use of this mechanism in the past. Recently announced plans for establishing fuel trusts to allow utilities to capitalize unneeded fuel inventories may eventually allow smaller total stockpiles to cover the utilities needs.

A pooling arrangement conceptually would allow several utilities to combine their uranium requirements and resources for procuring uranium to create a much larger demand and thus become a more significant entity in the uranium market. By pooling their resources the utilities would be able to share the cost of maintaining a stockpile and could share costs and risks in any exploration and development ventures. For the most part, U.S. utilities are fiercely independent and pooling arrangements, primarily for nuclear fuel procurement, have not been formed. The pooling arrangements that are operative have a much broader charter. A typical example is the Central Area Power Coordinating Group (CAPCO) consisting of the Cleveland Electric Illuminating Company, Duquesne Light Company, Ohio Edison Company and its subsidiary, Pennsylvania Power Company, and the Toledo Edison Company. CAPCO was formed as a power pool to share the costs of production and transmission of electricity from new large generating units. Coordination of nuclear fuel procurement for its members is only one of many CAPCO functions.

#### 5. Purchase of Identified Uranium Reserves In-Place

Under this option a utility could either buy the ownership rights to the reserves and assume the responsibility for production, or buy a production commitment by funding all or part of the operating costs of the owner/producer.

#### 6. Funding a Development Drilling Program

The normal arrangement involved here is the securing of an equity position or production commitment from the present property owners on the defined uranium reserves in return for funding a development drilling program to outline and develop reserves discovered through previous exploration efforts. The

TABLE II. SUMMARY OF UNITED STATES UTILITY INVOLVEMENT IN URANIUM EXPLORATION AND DEVELOPMENT<sup>a</sup>

Utility, Group or (Subsidiary)	Date of Entry or (Withdrawal)	Participant(s) <sup>b</sup> (Terminated)	Type of Activity <sup>c</sup>		U <sub>3</sub> O <sub>8</sub> Supply Began or To Begin <sup>d</sup>
			Initially	Now	
1. Arizona Public Service Co.	1976	Self Others	AED	AED	
2. Baltimore Gas & Electric Co.	1974	Intercontinental Energy Corp.	AE	AE	
3. Central and Southwest Fuels Inc.	1977 (1979)	Self Nuclear Dynamics Inc.	T	T	---
4. Central Power and Light Co.	See Central and Southwest Fuels Inc.				
5. Commonwealth Edison Co. (Edison Development Co.) (Cotter Corp.)	1974	Self Wold Nuclear Inc., Cleveland Cliffs, Inc.	AEDMM	AEDMM	1974
6. Consolidated Edison Co. of New York, Inc. (Fair Day Corp.)	1976 (1978)	Self	T	T	---
7. Consumers Power Co. (Plateau Resources, Ltd.)	1974	Self	AEDMO	AEDMO	1981
8. Detroit Edison Company (Saint Clair Energy Co.)	1977	Western Standard Uranium Co. & U.S. Energy	AD	AEDMM	1979
9. Duke Power Co. (Western Fuel Inc.)	1978	Ogle Petroleum	AEDS	AEDS	
10. El Paso Electric	See Arizona Public Service Co.				
11. Florida Power & Light Co. (Fuel Supply Services Inc.)	1976	Getty Oil Co. TXO Minerals	AE	AE	
12. GPU Service Corp. <sup>e</sup> (Cherry Hill Fuels Corp.)	1976	Self	AE	AE	
13. Gulf States Utilities (Varibus Corp.)	1976	Felmont Oil Co.	AE	AE	

Utility, Group or (Subsidiary)	Date of Entry or (Withdrawal)	Participant(s) <sup>b</sup> (Terminated)	<u>Type of Activity</u> <sup>c</sup>		U <sub>3</sub> O <sub>8</sub> Supply Began or To Begin <sup>d</sup>
			Initially	Now	
14. Houston Lighting and Power	1975	Self (Conoco)	AE	ADMM	1978
15. Long Island Lighting Co.	1976	Self Bokum Resources Co.	AD	DMM	1979
16. Madison Gas & Electric Co.	See Wisconsin Public Service Corp.				
17. Middle South Utilities Corp. <sup>f</sup> (Systems Fuels Inc.)	1978	Martin-Trost Assoc.	AE	AEDM	1987
18. Niagara Mohawk Power Corp. (NM Uranium Co.)	1975	U.S. Steel Co. New Mexico Uranium Inc.	AD	ADS	1978
19. Northern States Power Company (Cormorant Corp.)	1977 (1978)	Felmont Oil Co.	T	T	---
20. Pacific Power and Light Co. (Northern Energy Resources Co.)	1977	Teton Exploration Drilling Pioneer Nuclear Nuclear Dynamics Inc.	AE	AED	
21. Pennsylvania Power & Light Co. (Uranium Resources and Development Co.)	1975	Self	AE	AED	
22. PJM Utilities Group <sup>g</sup>	1978	Self	AE	AE	
23. Portland General Electric Co.	1976	Martin-Trost Assoc.	AE	AEDM	1980
24. Public Service Co. of New Mexico	See Arizona Public Service				
25. Public Service Co. of Oklahoma	1974	Combined with Central and Southwest Fuels Inc. - 1977			---
26. Public Service Electric and Gas of New Jersey	1975	Kerr-McGee Corp Homestake Mining Co.	AE	AEDM	1980

TABLE II (cont.)

	Utility, Group or (Subsidiary)	Date of Entry or (Withdrawal)	Participant(s) <sup>b</sup> (Termination)	Type of Activity <sup>c</sup>		U <sub>3</sub> O <sub>8</sub> Supply Began or To Begin <sup>d</sup>
				Initially	Now	
27.	Southern California Edison (Mono Power)	1970	Rocky Mountain Energy Co.	AE	AEDMM	1977
28.	Salt River Project	See Arizona Public Service Co.				
29.	San Diego Gas & Electric	See Southern California Edison Co.				
30.	Texas Utilities Generating Company (Texas Utilities Fuel Co.) (Chaco Energy Co.)	1974	Ranchers Exploration and Development Corp.	E	E	
31.	Tennessee Valley Authority	1972	Mobil Oil Corp. American Nickel & Copper Co. United Nuclear Corp. Federal American Partners American Nuclear Corp. International Nickel Co. Silver King Mines Co. Utah International	AEDM	AEDMM	1976
32.	Utah Power and Light Co.	1975	Self Mountain States Resources Co.	AED	AED	
33.	Virginia Electric Power Co.	(1977)	(Glory Mining Co.)	T	T	---
34.	Washington Public Power Supply System	1977	Fremont Energy Co.	AE	AE	1987
35.	Wisconsin Power & Light	See Wisconsin Public Service Corp.				
36.	Wisconsin Public Service Corp.	1975	Self Mineral Recovery Corp.	AD	AED	1976



- a. "Involvement" here means one or more of the activities listed in (c) below.  
It excludes conventional and unconventional funding activities which may be part of nuclear fuel acquisition but which do not include any of the activities listed in (c) below.
- b. "Self" means the utility acts in its own name, through a subsidiary who holds an equity interest or through an operator who does not hold an equity interest. The other participants listed hold some equity interest. All participants are not necessarily involved in all activities or ventures.
- c.
  - A Acquires equity interest in mineral rights
  - E Explores for uranium mineralization
  - D Develops uranium reserves
  - M Mines uranium ore
  - MM Mines and mills uranium ore
  - O Buys uranium ore for custom milling
  - S Obtains uranium by solution mining
  - T Activity has been terminated
- d. Future dates are estimates.
- e. General Public Utilities Service Corp. represents Jersey Central Power and Light Co. and Metropolitan Edison Co.
- f. Middle South Utilities Corp. represents Arkansas Power and Light Co., Louisiana Power and Light Co., and Mississippi Power and Light Co.
- g. PJM Utilities Group represents Baltimore Gas and Electric Co., Philadelphia Electric Co., Pennsylvania Power and Light Co., Public Service Electric and Gas Co., and Delmarva Power and Light Co.

investment required is considerably less than the cost of proven reserves but the risks of defining inadequate reserves are correspondingly greater.

#### 7. Funding An Exploration Program

This method could fund an experienced company in searching for new deposits in a uranium district with which they are familiar or a new company with sufficient corporate resources to justify a venture. The program could be broadened to include reconnaissance prospecting in potentially favorable, but as yet unestablished areas.

#### 8. In-House Uranium Exploration Effort

This method requires a dedicated effort on the part of the utility through at least partial staffing in-house, supplemented at least initially by consulting geologic/engineering services for most of the field work. The utility has total control of the financial and technical aspects of the program so that the program is more responsive to the utility's needs at any time.

#### 9. Research and Development of Non-Conventional Uranium Resources

Examples of prior successes in this area would be development of recovery processes for uranium as a byproduct of phosphoric acid production or from copper mining residues. Present areas of activity include in-situ solution mining. For the future perhaps research and development could lead to economic methods of recovering uranium from granite or sea water.

### UNITED STATES UTILITY INVOLVEMENT IN URANIUM EXPLORATION AND DEVELOPMENT

Table II summarizes publically available information on the involvement of over 40 U.S. utilities in uranium exploration and development. Through confidential sources we know of the involvement of several other utilities which are not included in Table II. At the present time about half of the U.S. utilities with nuclear power commitments have some form of financial involvement in uranium exploration and development. Table II includes only those utilities whose programs result in the utility gaining an equity interest in reserves either directly or through a subsidiary and does not include utilities which are merely involved in advanced payment contracts or advanced payments to

operators of ore buying stations. In some cases the utility may be involved in a number of joint ventures often with different partners in each venture. We have not attempted to sort these out in Table II, but instead have attempted to summarize each utility's overall program.

Where allowed by public regulatory bodies, utilities with large nuclear commitments have attempted to follow procurement strategies involving a prudent mix of the options listed in Figure 1. As shown in Table II, utility involvement in uranium exploration and development was an accelerating trend in the period 1973 through 1977. In the past year this trend has slowed primarily to await clarification of several important regulatory proceedings which will set precedents for utility investment in these activities. Utility attitudes toward such investments will also be affected by the direction the USA takes in its policy toward nuclear power.

As can be seen from Table II, utility involvement usually begins with funding exploration programs and acquiring equity interest in mineral rights. Relatively few utility programs have proceeded to the stage of mine and mill development. This is to be expected since the costs involved in land acquisition and exploration represent only about 10% of the final costs of producing uranium whereas capital and operating costs of mining and milling represent about 65% and 15% of the final costs respectively.<sup>[8]</sup> The real test for utilities will come as they have to make the decisions to bring their reserves into production. These decisions will require much greater commitment of utility funds, and will require larger in-house staffs with mining expertise.

## CONCLUSION

It is too early to judge whether utilities will be successful in producing less expensive uranium through captive production. However, the growing importance of this source can be seen from Figure 2 which gives uranium procurement by contract type for deliveries through 1985.<sup>[9]</sup> Most of the category labeled "other" is for uranium coming from captive production. In the USA uranium procurement from utility owned or captive production was a negligible factor prior to 1976. In 1976 it became possible for some utilities with captive production to develop firm production schedules. In that year a record amount of  $U_3O_8$  (75,800 t)

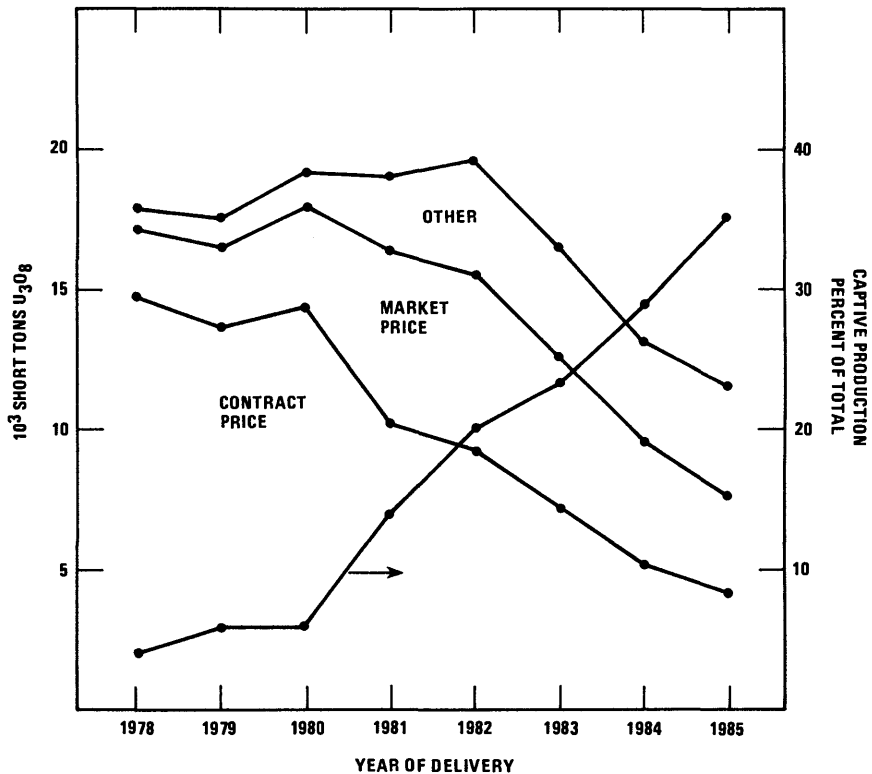


FIG.2. Procurement by contract type – United States purchasers from United States producers (1 July, 1978).

(NB. The left-hand scale is not in metric units).

was purchased by U.S. utilities and almost half of this amount was through captive production commitments.

This increasing trend is evident even if one corrects the percentages in Figure 2 for the fact that present commitments represent a decreasing fraction of total requirements for later years. Reference [9] indicates that as of July 1, 1978, 1985 commitments represented about 50% of 1985 domestic requirements. Even if all the unfulfilled requirements were to come from non-captive procurement, captive production would still increase to 17-18% of the 1985 totals.

Production schedules from newly developed captive production sources are likely to slip due to unforeseen physical, financial, labor, environmental and political factors. Although

this would tend to delay the impact of such production, this trend will be offset by the fact that some, possibly 50%, of the present unfulfilled requirements will also eventually come from captive production.

In the U.S. coal industry, captive production is becoming an increasingly important factor having risen from 5.5% of utility deliveries in 1965 to 11.2% of utility deliveries in 1975.<sup>[10]</sup> Government projections are that captive coal production will increase to 19% of production by 1985. The incentives for developing captive coal production are not nearly as great as for uranium since annual coal production is a much smaller fraction of U.S. reserves and a substantial fraction (~25%) of coal production is not used for producing electricity. Although utilities may be disappointed in the effect on uranium prices that they achieve through these efforts, once embarked on exploration and development ventures they have showed remarkable tenacity and staying power in pursuing their objectives. Provided that the political outlook for nuclear power in the USA becomes more favorable, with resulting more favorable regulatory rulings throughout the country, we foresee captive production of uranium by utilities becoming an increasingly important factor and reaching 50% of total U.S. production before the year 2000.

If present trends in uranium exploration and development continues, the USA will become more firmly entrenched as the world's marginal cost uranium producer. This trend will accelerate the move toward utility involvement in captive uranium production since this will be the only way to insure the necessary growth in the domestic uranium industry. Thus, captive uranium production could become the dominant development in the U.S. industry within the next 20 years. One effect of this trend will be to diminish the importance of the spot market and to make it more difficult to define a meaningful "market price" for uranium.

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### DISCUSSION

A. QUESADA: According to your paper the trend in uranium exploration is governed by movements in energy demand. However, in view of the Three-Mile Island accident and the increasing pressure of anti-nuclear groups in the United States of America, what do you think will happen in the future as far as the investment of capital by utility companies in uranium exploration is concerned?

R.P. SULLIVAN: The report of the Presidential Commission on Three-Mile Island will be presented later this month and will be of major importance for the near-term future of nuclear power in the United States. If this report is well-balanced and if it is recognized by the authors of the report that nuclear power can be produced with an adequate safety margin, with the necessary action being taken as a result of the lessons learned from the Three-Mile Island incident, it will be possible for nuclear power to move ahead again in the United States. As for the environmentalist opposition, what we need is for the Government to make it clear to the public that nuclear power is essential to the economic well-being of the country. In the long run I think the situation will become obvious one way or another.

M. ISLA: The experience of United States electrical companies in financial partnerships for uranium exploration, mainly joint ventures, is considered by some to have been unsatisfactory because of the lack of direct control that these companies have over the activities concerned. Do you think this might be another factor which could discourage companies from continuing to invest in uranium exploration?

R.P. SULLIVAN: I think that individual United States utilities will be reluctant to commit themselves to receiving a large proportion (greater than 25%) of their uranium requirements from foreign suppliers. However, if the United States uranium industry continues to be relied upon by the electrical companies as the main source of raw material, the companies will be obliged to continue with financial participation. In cases in which their past experience may not have been satisfactory they will probably have to become more deeply involved to ensure better results.

M.V. HANSEN: According to the "Conclusions" section of your paper, utility involvement in total United States uranium production will reach 50% by the year 2000. What is the extent of their present involvement?

R.P. SULLIVAN: Figure 2 of our paper shows captive production reaching 35% of uranium deliveries for the year 1985. This percentage will change as a result of two competing effects: slippage in production schedules will tend to lower the percentage, but some of the requirements for 1985 which are at present unfulfilled will be met by captive production. The latest Department of Energy market survey report (issued in August 1979) shows the curve to have reached a somewhat higher level than in our Fig.2, so the trend is still moving upwards. Provided that the problems affecting the future growth of nuclear power in the United States are satisfactorily resolved, I would expect captive production to fulfil 50% of requirements soon after 1990.

A.E. BELLUCO: Can you say how much money is being invested in uranium exploration in the United States?

R.P. SULLIVAN: A United States Government survey reported the figure of US \$314 300 000 for expenditure on uranium exploration and development for the year 1978. Data on the amount of money being invested by United States utilities are difficult to obtain and I am not able to tell you what fraction of the total expenditure is provided by them.





# THE UNITED NATIONS' ENDEAVOUR TO STANDARDIZE MINERAL RESOURCE CLASSIFICATION

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## Abstract

### THE UNITED NATIONS' ENDEAVOUR TO STANDARDIZE MINERAL RESOURCE CLASSIFICATION.

The United Nations' Economic and Social Council passed a resolution in July 1975 calling for the development of a mineral resources classification system to be used in reporting data to the United Nations. Following preparation of background papers and an agenda by the UN Centre for Natural Resources, Energy and Transport, a panel of experts recommended a classification system to the Council's Committee on Natural Resources. The Committee met in Turkey in June 1979 and has reported favourably to the Council on the proposed system. The classification system is designed to provide maximum capability for requesting and receiving data from the resources data systems already used internally by major mineral producing nations. In addition, the system provides for flexibility in adjusting to the particular needs of individual mineral commodities. The proposed system involves three basic categories of in-situ resources: R-1, reliable estimates of known deposits; R-2, preliminary estimates of the extensions of known deposits; and, R-3, tentative estimates of quantities to be found in undiscovered deposits. As an option for given countries and commodities, the R-1 category can be further sub-divided into: R-1-E, economic; R-1-M, marginal; and R-1-S, sub-economic. Finally, the classification scheme provides for all categories to have a parallel set of estimates of recoverable mineral quantities.

## INTRODUCTION

The United Nations has just completed the development of a standardized set of categories, terms, and definitions for use in obtaining, aggregating, and reporting data on mineral resources, including coal and uranium. The objective of this effort is to provide more uniform and reliable international information about resources inventories and expectations. This goal is shared by other international organizations, such as IAEA/NEA, who deal with the various resource commodities. Therefore, it is desirable for communication to be maintained among these various groups concerning the direction and progress of their independent efforts. This will

provide the mutual benefit of shared experience, eliminate unnecessary duplication, and enhance prospects for compatibility and understanding.

The author served as a consultant to the United Nations Centre for Natural Resources, Energy, and Transport (CNRET) in the preparation of background papers and a discussion agenda for an Expert Group meeting held in New York from 29 January to 2 February 1979. During the meeting the author acted as rapporteur for the eight experts drawn from a cross-section of nations, plus a small group of invited observers from interested national or international agencies.<sup>1</sup> As a result of this meeting a recommendation report was submitted to the UN Secretary General for consideration by the Committee of Natural Resources of the United Nations Economic and Social Council. The Natural Resources Committee initiated the effort through a resolution passed 25 July 1975. The Committee considered the experts' report at its recent meetings in Ankara, Turkey, from 5 to 15 June 1979. Based on their favourable reactions, the Economic and Social Council has encouraged the member nations of the UN to follow the recommendations of the Expert Group.

This paper summarizes some of the insights the working group gained from its experience and reports the resulting recommendations.

#### RESULTS OF THE BACKGROUND STUDY

The UN Economic and Social Council's interest in improving the quality of data about mineral resources was stimulated by a recommendation initiated by Canada. The final resolution called for seeking an agreement on terminology to be used in categorizing mineral resources. To this end, the Council charged CNRET to review present definitions and terminology for mineral reserves. After this, a group of experts would be assembled to prepare recommendations on a common set of definitions and terminology which would be suitable for reporting to the United Nations about mineral resources.

The background study conducted at CNRET revealed considerable diversity in the classification systems currently in use throughout the world. However, closer examination revealed that there are two characteristics in international data. One, most nations devote much of their

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<sup>1</sup> Participants are listed at the end of the paper.

attention to reserves found in known deposits. Two, despite separation of the world into many autonomous states, there are historical, linguistic, political, professional, and trade ties that have left an imprint on our resources terminology. As a consequence, despite superficial differences, most classification systems originated from a relatively limited number of mining or engineering cultures. These are primarily those of Eastern Europe and the Soviet Union, the mining regions of Western Europe, the coal and iron industries of the United Kingdom, and North America.

The classification schemes used in these areas as well as by countries who have been influenced by their mining traditions and training, provide a confusing array of diagrams, identifiers, and definitions. But the actual conceptual differences are not unmanageable. Two systems, in particular, have seemed to become recognized in recent years as the "core" efforts. One is that employed in the USSR and adopted in 1960 by the Council for Mutual Economic Assistance. The other is the terminology formally recognized by the United States Department of the Interior and a number of other Federal agencies. This evolved from the early work of geologists McKelvey, Blondel, and Lasky. The USDI system has been modified by Canada and is also used in several other countries.

Following the review of systems in current use, the next determination was to establish more firmly what the Expert Group could attempt to accomplish. Fundamentally this involved a choice between an effort aimed at achieving international uniformity in the measurement and classification of resources, or providing a system that offered an opportunity to draw information from existing national or individual commodity classification systems. The former goal may be an attractive objective, but requires the acceptance and adoption of one system by a large number of nations and individual mineral industries with diverse interests and capabilities. The latter approach would avoid these difficulties but would have to find common denominators for translating various national or commodity data into acceptable regional or world data sets. It was decided that the background papers and discussion agenda prepared for the Expert Group would be designed to follow the latter course. However ideal it might be, a universal classification system for all nations would be left to evolve over time. Perhaps the need for better assessments and international analysis will become so compelling that this could occur in the long-term.

When it assembled, the Expert Group recognized the need to agree upon the extent of the resource universe it wishes to mea-

sure, the conceptual design of a proposed classification system, and purposes for which the UN would request and assemble data under the proposed system. The strategy was then for the group to seek general agreement on fundamentals, then turn to the details of categories, identifiers, definitions, and various methodological problems. In particular there was a need for:

Recognition of the differences between engineering evaluation of a specific deposit or property and a geologist's regional resource assessment.

Identification and accommodation of the variations in divergencies found among national systems reflecting technology, economics, statistical objectives and the availability of raw data.

Accommodation of the differences encountered among the many nonfuel minerals and energy resources.

Determination of how many dimensions should be encompassed--geologic, economic, technical, institutional, and so forth.

#### THE SEARCH FOR COMMON DENOMINATORS AND CONCEPTS

At some point, all resource producing countries find a need to inventory their mineral stocks. This requires them to come to grips with the problem of what has actually been discovered and how it can be measured. The differences in the various national systems revolve about the details of the numbers of classes of discovered resources they need in their system, the required accuracy of measurement, and what are reasonable practices concerning the extrapolation of information beyond the limits of observations or mine development. Thus the problem of comparing international resource statistics is more a reflection of disparities in definitions and measurements rather than a major conflict in fundamentals.

One basic difficulty is drawing the line between what has been discovered and what has not. This is a problem not only between nations but a difficulty within any country's own system. It is symptomatic of having to deal with what in reality is not a sharp line of demarcation, but rather a transition zone between one that is surmised to exist beyond the limits of actual observation. There is an additional difficulty in that many mineral producing countries have limited interest, funds, work force, or experience in appraising undiscovered resources. None the less, an international classification system must be

able to accommodate undiscovered resource estimates. Even those countries that only deal in estimates of discovered, economically producible reserves data have implicitly recognized that there is a limit between discovered reserves and undiscovered resources.

Perhaps a more difficult task in international resources classification is how to deal with resources that are known to exist but are below a country's contemporary standards of usability. Regardless of economic or political systems, most countries are aware of mineral deposits to which the application of manpower, equipment, and transport is not justifiable under existing circumstances. In concept these quantities are not part of what the analyst can classify as currently known and producible resources. This distinction is quite clear, but the more important question is whether or not it has been reflected in the measurement and reporting of resources data. There may be good reasons why some countries do not distinguish between what is currently usable and what is not. However, this cannot obscure the need for this difference to be recognized in measuring the world's resources.

The recognition that known mineral occurrences fall into both producible and currently non-producible classes leads to consideration on whether geologic estimates of undiscovered resources should be similarly divided into usable and unusable categories. This introduces obvious difficulty of what costs and technology are relevant to determining whether or not an undiscovered deposit after discovery will be producible. There is a lack of unanimity on the answer to this dilemma. However, in the general case, the argument in favor of not extending economic considerations into the realm of undiscovered resources seems strong, even though there may be exceptions under specific circumstances.

In the initial structuring of mineral resource classification schemes there is a necessity to establish a lower limit to what is considered worth assessing. In essence this is a requirement which restricts both classification and the process of specific quantification to those mineral occurrences that are perceived to have some present or future usefulness and to separate them from mere "country rock." This requires, perhaps in a somewhat arbitrary manner, the setting of spatial, geologic, and mineralogic limits to what is being measured. Such limits, depending upon the mineral commodity in question, may be established on the basis of purely physical criteria rather than necessarily economic ones. Obviously such limits need to be established commodity-by-commodity and country-by-country.

One major issue that must be resolved in international classification is the question of the percentage of useful mineral that can be extracted by mining and recovered by processing. This is particularly critical in reporting on the resource quantities found in disseminated ores, extensive bedded deposits, and for most energy sources. The ultimate recovery from the total quantity found in place may be as little as ten percent. For many countries and for some commodities a convenient solution has been to consider resource estimates only in terms of in-place quantities. But this cannot negate the necessity of ultimately having to calculate the recoverable proportions of these mineral quantities if the user of the data is to have any concept of the true supply potential of the mineral resources that have been estimated. It seems apparent that classification systems should make provision for data both in terms of the quantity found in place as well as for that which is recoverable under specified conditions.

An international classification system must define precisely what is encompassed as well as what is not in each proposed category. This burden falls equally upon an international agency requesting resources data and on the respondent country when it provides data from its own internal system. It is also useful for international classification systems to be relatively simple in concept and general in definitions so that they can most readily accommodate information from many countries as well as for a variety of mineral commodities. Another hurdle is how to identify the various categories of resource data. While structure and definition are the critical issues, one should not minimize the problems of language, the profusion of terms used, and the affinity for any group for its own identifiers and interpretation of terms. As a consequence, identification of categories by letters and numbers, particularly combinations not used by any individual country, seems to have merit.

After its general deliberations about classification, the Expert Group affirmed its intent to recommend to the Secretary General a system that provided categories and definitions compatible with those in use by various countries and other international groups. It was considered important that the system reflect modern assessment techniques but still provide sufficient simplicity that its use would be within reach of all countries. The Group agreed in particular that the classification system should:

- (a) Facilitate the international exchange data, particularly by enhancing their comparability;

- (b) Ideally, be suitable for all mineral resources, or readily adaptable to the specific needs of particular mineral commodities;
- (c) Take account of measurement and collection procedures to the extent necessary to ensure that the system will be of practical value;
- (d) Provide for the inclusion of estimates concerning all mineral resources that are known or surmised to exist with varying degrees of assurance, as well as resources that are as yet undiscovered;
- (e) Make provision for both in situ and recoverable resource calculations;
- (f) Allow separate estimates of economic and sub-economic resources within those categories where such subdivision is feasible; and
- (g) Be primarily concerned with estimates of material quantities that are of economic interest over the foreseeable period of the next few decades. However, provision should be made for recognition of estimates or descriptions of mineral occurrences that fall outside of the major resource categories as defined.

#### THE CLASSIFICATION SYSTEM RECOMMENDED TO THE UNITED NATIONS

The Expert Group proposed that an international classification of mineral resources to be used by the United Nations have three basic categories, identified as R-1, R-2, and R-3. These three categories are differentiated according to the level of geological assurance that can be assigned to each category. They represent all of the in situ mineral resources that might be of economic interest over the foreseeable period of the next few decades. This time period would be expected to vary somewhat depending upon the mineral commodity being considered. The limit of economic interest would be established by economic or physical criteria suitable for that individual mineral commodity. Additional material whose lower economic potential causes it to fall outside the boundaries of "resources" as defined by the Group should be referred to as "Occurrences," and if considered at all should be done so separately with proper clarification of the derivation and meaning of estimates.

The three categories were defined as follows:

Category R-1 encompasses the in situ resources in deposits that have been examined in sufficient detail to establish their mode of occurrence, size and essential qualities within individual ore bodies. The major characteristics relevant to mining and processing, such as the distribution of ore grade, the physical properties that affect mining, the mineralogy and deleterious constituents, are known mainly by direct physical penetration and measurement of the ore body combined with limited extrapolation of geological, geophysical and geochemical data.

Quantities should have been estimated at a relatively high level of assurance, although in some deposits the estimation error may be as high as 50 percent. The primary relevance of such estimates is in the planning of mining activities.

Category R-2 provides for estimates of in situ resources that are directly associated with discovered mineral deposits but, unlike the resources included in Category R-1, the estimates are preliminary and based largely upon broad geological knowledge supported by measurements at some points. The mode of occurrence, size and shape are inferred by analogy with nearby deposits included in R-1, by general geological and structural considerations, and by analysis of direct or indirect indications of mineral deposition. Less reliance can be placed on estimates of quantities in this category than on those in R-1; estimation errors may be greater than 50 percent. The estimates in R-2 are relevant mostly for planning further exploration with an expectation of eventual reclassification to Category R-1.

Category R-3 resources are undiscovered but are thought to exist in discoverable deposits of generally recognized types. Estimates of in situ quantities are made mostly on the basis of geological extrapolation, geophysical or geochemical indications, or statistical analogy. The existence and size of any deposits in this category are necessarily speculative. They may or may not actually be discovered within the next few decades. Estimates for R-3 suggest the extent of exploration opportunities and the somewhat longer-range prospects for raw material supply. Their low degree of reliability should be reflected by reporting in ranges.

Each of the categories can be further subdivided as follows:

E - Those in situ resources that are considered to be exploitable in a particular country or region under the prevailing socio-economic conditions with available technology;



S - The balance of the in situ resources that is not considered of current interest but may become of interest as a result of foreseeable economic or technologic changes.

The subclassifications "E" and "S" are particularly useful for subdividing resource category R-1 and perhaps category R-2, but the Group does not expect that R-3 will generally be subdivided in practice.

Some countries may wish to further subdivide "S" to provide for an estimate of resources, "M", perhaps within a decade, that may become exploitable in the more immediate future as a result of normal or anticipated changes in economic or technical circumstances.

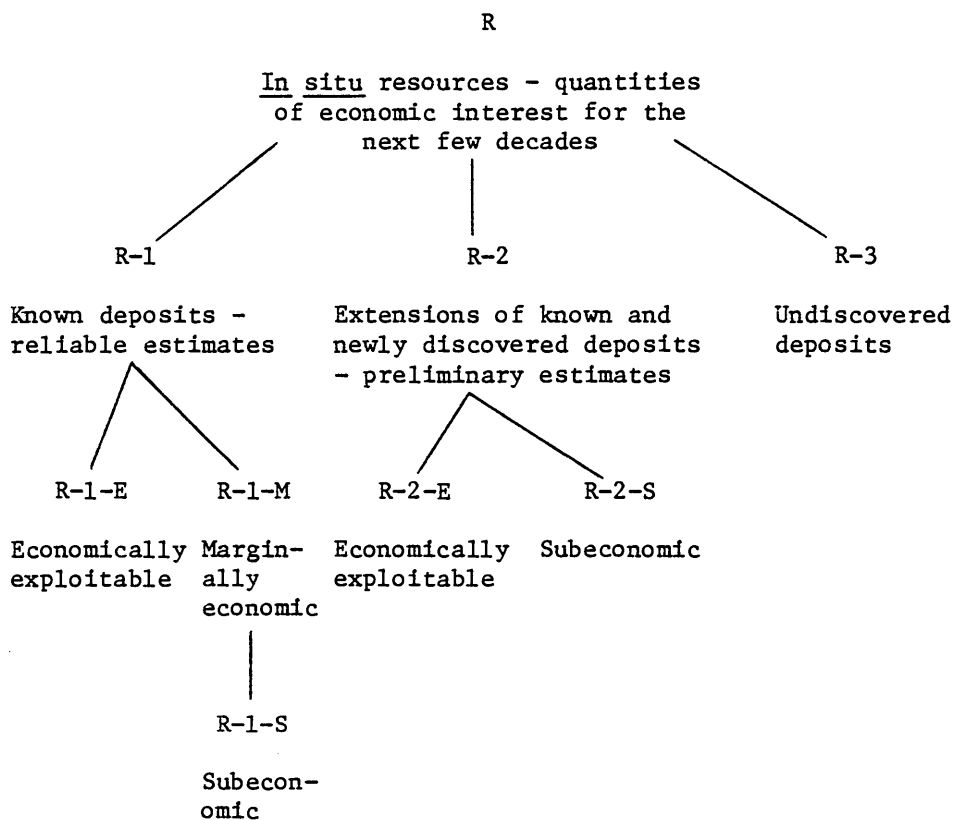
While directing its immediate attention to a classification system for in situ resources, the Group was fully aware that resource estimates for some minerals, such as oil and gas or uranium, are more commonly reported as estimates of recoverable metal or mineral. Since the recoverable content more closely approximates the quantity that may appear as mineral supply, the Group recommended the establishment of a set of categories and definitions for recoverable quantities. This would be in parallel to the in situ classifications, providing an opportunity to use either or both sets of estimates depending upon which is most suitable. It was noted that there can be no general definition of recoverability, or the point in the mining and processing at which it should be measured. These specifications must be established for each commodity.

The Expert Group could not arrive at a clear preference for a set of distinctive identifiers suitable for the recoverable categories. Several approaches were suggested. In its final report, it proposed the use of r-1, r-2, and r-3. However, there were proponents of several other methods using prefixes, suffixes, superscripts, or subscripts, such as pR-1, pR-2, and pR-3 or R<sub>r</sub>-1, R<sub>r</sub>-2, and R<sub>r</sub>-3. It was decided that this was not a matter of great consequence and that a final determination could be made as part of actual implementation by the United Nations.

The Expert Group closed its report with a cautionary statement, "If the classification system proposed here is placed into common use for international reporting of resource information, it will be only the first step towards general harmonization of resource classification. The collection, aggregation and dissemination of resource estimations on a world-wide scale are at present only carried out regularly by the International Atomic Energy Agency for uranium and the

World Energy Conference for other energy resources. If it is to be used for a reporting system, this classification system will have to be adapted for the specific requirements of individual commodities. For example, levels of assurance may have to be defined and recovery levels established. It is also recommended that both the definitions and the questionnaires to be used for individual commodities be tested carefully before actual use."

In my judgment, the Expert Group, in a very short interval of time, displayed not only a high level of professional competence and enthusiasm for a difficult task, but made a real contribution to the United Nations toward improving its work on mineral resources. Also, the report should serve the Group's wish that they were encouraging the expansion and improvement in resource estimation by individual countries, both in their internal data and for international purposes.

ANNEX 1: SCHEMATIC PRESENTATION OF CLASSIFICATION CATEGORIES<sup>2</sup>

<sup>2</sup>While the capital "R" denotes resources in situ, a lower-case "r" could identify the corresponding recoverable resources for each category and subcategory, such as r-1-E is the recoverable equivalent of R-1-E.

## ANNEX 2: PARTICIPANTS

(Expert Group Meeting, New York, 29 Jan. – 2 Feb. 1979).

**Members**

Alan A. Archer (Chairman),	Institute of Geological Science, London, United Kingdom;
Octavio Barbosa,	Companhia de Pesquisa de Recursos, Minerais, Rio de Janeiro, Brazil;
François Callot,	Bureau de Documentation Minière, Paris, France;
Günter B. Fettweis,	Institute of Mining, Montanuniversität, Leoben, Austria;
Kirill P. Kavun,	All-Union Institute of Economics of Mineral Reserves and Geological Prospecting, Moscow, Union of Soviet Socialist Republics;
Rabi N. Mishra,	Geological Survey of India, Bangalore, India;
John J. Schanz, Jr. (Rapporteur),	Resources for the Future, Washington, D.C., United States of America;
Jan Zwartendyk	Department of Energy, Mines and Resources, Ottawa, Canada

Observers

K.E. Koch	Federal Institute for Geosciences and Natural Resources, Hannover, Federal Republic of Germany
Vladimir Rogoznikov	Ministry of Higher and Secondary Specialized Education, Moscow, Union of Soviet Socialist Republics
Maurice V. Hansen	International Atomic Energy Agency
Noriko Iwase	Ocean Economics and Technology Office, International Economic and Social Affairs, United Nations Secretariat
Wolfgang Gluschke (Secretary)	Centre for Natural Resources, Energy and Transport, Technical Co-operation for Development, United Nations Secretariat

## DISCUSSION

B.S.I. MARENGWA(*Chairman*): It would appear that the expression "estimated mineral resources" as used in the paper does not have the usual meaning that it has in economic geology, where the terms "proven", "estimated" and "probable" mineral resources are used, but is being employed in a broader sense.

D.M. TAYLOR: I would like to comment on the use of the term "estimated resources" in the NEA/IAEA uranium resource classification (see Figure 1B of paper IAEA/SM-239/31, these proceedings, Session VII). It will be seen from this figure that our reserves (reasonably assured resources (RAR) which can be exploited at less than \$80/kg U) can be equated with Mr. Schanz's r-1-E, while the higher-cost RAR can be equated with r-1-S. Estimated additional resources (EAR) can be equated with Dr. Schanz' r-2 and our speculative resources with r-3. The "grey areas" between reserves and higher-cost RAR could be regarded as r-1-M.

J.J. SCHANZ: This is correct and reveals the intentions of the UN group. It is not a mere coincidence that there are three categories in the UN system, as in the NEA/IAEA scheme. But the intention is also to combine the sub-categories of the US system, such as "measured" and "indicated", or the a, b, c<sub>1</sub>, c<sub>2</sub>, d<sub>1</sub> and d<sub>2</sub> categories of the Soviet system into the three sets of data in the UN plan.

Although for most commodities classification has not advanced to the point where resources are estimated according to cost or price, as has been done for uranium, it is quite possible to interpret the sub-categories E, M and S in this way.

P.D. TOENS: I find it difficult to reconcile the NEA/IAEA classification, which is based on the cost of production, with that of the author of the paper, who suggests a system based on the concept of what is economically exploitable.

J.J. SCHANZ: While as a resource economist my natural inclination would be to incorporate prices or costs into the scheme, we have to recognize the difficulty of dealing with economic considerations in a multi-mineral and multinational framework. Thus, the terms "economic", "price" and "cost" are avoided in the UN report as much as possible. Instead we emphasized *usability* or *producibility*. The most important point is that, if a deposit actually is being or will be exploited, its output will be added to the materials and energy being used to meet the world's needs. In this context, price and cost are not relevant. However, the UN scheme does not exclude the use of price, cost and economic considerations when it is feasible to do so for particular commodities.

A.E. BELLUCO: For the category r-3, is there an evaluation methodology, a formula or other procedure for making the geological parameters used for performing these calculations compatible with each other?

J.J. SCHANZ: It must be recognized that a general classification scheme such as the one proposed cannot include a specific evaluation methodology. We expect, in the application of the general format to a specific mineral commodity,

that expert groups would have to devise detailed instructions in respect of the methodology and parameters suitable for including estimates in category r-3 for that particular mineral.

# A PROPOSED UNITED STATES RESOURCE CLASSIFICATION SYSTEM

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## Abstract

### A PROPOSED UNITED STATES RESOURCE CLASSIFICATION SYSTEM.

Energy is a world-wide problem calling for world-wide communication to resolve the many supply and distribution problems. Essential to a communication problem are a definition and comparability of elements being communicated. The US Geological Survey, with the co-operation of the US Bureau of Mines and the US Department of Energy, has devised a classification system for all mineral resources, the principles of which, it is felt, offer the possibility of world communication. At present several other systems, extant or under development (Potential Gas Committee of the USA, United Nations Resource Committee, and the American Society of Testing and Materials) are internally consistent and provide easy communication linkage. The system in use by the uranium community in the United States of America, however, ties resource quantities to forward-cost dollar values rendering them inconsistent with other classifications and therefore not comparable. This paper develops the rationale for the new USGS resource classification and notes its benefits relative to a forward-cost classification and its relationship specifically to other current classifications.

The author of paper IAEA-SM-239/3<sup>1</sup>, Mr. Schanz, and I have spent much time in recent years discussing the problems of communicating resource information between well-intentioned communicants. The problem is severe, even between English-speaking peoples educated in similar cultural and scientific situations. It gets progressively worse as we try to blend the interest and biases of government, industry, different nationalities, laymen, and political interests. But, since energy is a world-wide problem, we must learn to communicate our ideas about resource quantities in an understandable manner. That the United Nations group of resource experts was able to reach a consensus is remarkable. That that consensus can be correlated with certain United States resource classification systems, as well as with some other nations' classifications, is a great credit to the UN study group and to Mr. Jack Schanz.

This paper does not attempt to illustrate in detail exactly how the United Nations' system and a proposed US system can be integrated, but in discussing the US classification/nomenclature scheme, which we feel suits our national needs,

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<sup>1</sup> J. J. SCHANZ, these Proceedings.

## RESOURCES OF (commodity name)\*

AREA: (Mine, district, field, State, etc.) UNITS: (tons, bbls, ounces, etc.)

Cumulative Production	I D E N T I F I E D   R E S O U R C E S		U N D I S C O V E R E D   R E S O U R C E S	
	Demonstrated		Inferred	%                      Probability Range %
	Measured	Indicated		
			Hypothetical	Speculative
ECONOMIC	Reserves		Inferred Reserves	+
MARGINALLY ECONOMIC	Marginal Reserves		Inferred Marginal Reserves	
SUBECONOMIC	Identified Subeconomic		Resources	
Occurrences	Includes nonconventional and low-grade materials			

By: (author)

Date:

\* A portion of reserves or any resource category may be restricted from extraction by laws or regulations.

FIG.1. Classification of mineral resources - reserves.



I will refer to elements presented in Mr. Schanz's paper. Likewise, it would not be accurate for me to refer to it as *the* US system, because we do not have unanimity in US resource reporting. In particular, significant differences exist in uranium resource reporting between the Department of Energy, uranium subdivision, (DOE/u) and the Department of the Interior (DOI). The former reports tonnage numbers in terms of forward-cost categories, whereas the latter imposes only a generalized economic subdivision on the tonnage estimates. In addition, there are differences in definition that do not permit us to equate precisely the resource boundaries drawn on the basis of geological assurance of existence. Unfortunately, that is true also of the Nuclear Energy Agency/International Atomic Energy Agency (NEA/IAEA) classification, and I will mention where those differences lie.

This paper attempts to describe what we are trying to achieve generally for mineral resource classification in the United States, specifically for uranium. The specific classification/nomenclature system (Fig.1) is a modification of an earlier system reported in USGS Bulletin 1450-Y 1974, and one that has recently been agreed to by working groups from the US Geological Survey, US Bureau of Mines, Energy Information Administration of the Department of Energy, and the Securities and Exchange Commission. In this paper I refer to it as the proposed US resource classification system. Though we have not yet achieved final inter-agency approval for the system, we do have approval in principle and are at present writing the final drafts of the document. Most specialists are familiar with the broad outline of the proposed US resource classification system. The diagrammatic presentation shown here (Fig.1) is a variant of the earlier work but it adheres to the same basic principles, i.e. the classification is based on a matrix composed of economic subdivisions on the vertical axis and increments of geological certainty on the horizontal axis. The system allows for the use of all the subdivisions inherent in the matrix, but aggregations of appropriate subdivisions are permitted at the discretion of the estimator. Here we discuss only some of the principal aspects of this version, noting differences with the earlier version as appropriate. We now have a three-part subdivision of the vertical economic axis instead of a two part — dealt with later — and we have specifically allowed for the recognition of Occurrences of certain low-grade or remotely located materials about which we have no sense of any part of them possibly ever becoming economic as is required in the definition of a Resource. This distinction, therefore, calls for some specific cut-off concept to limit the bottom of the Total Resource diagram. This cut-off may be described in economic terms or in terms of grade, thickness, depth, or other physical resource parameters. An identical requirement was called for in the proposed UN classification that Schanz just discussed (IAEA-SM-239/3).

The question of a clear separation between Undiscovered and Identified Resources is tied up in the definition accorded the Inferred Reserve. We consider

the Inferred Reserve to represent the growth potential of the deposits recognized as making up the Demonstrated Reserve. Because the existence of these resources is confirmed marginal to the Indicated Reserve, we consider them to be Identified, but they are not yet satisfactorily delineated by the drill and, in a sense then, are not discovered. On the other hand, Undiscovered Hypothetical Resources bear no physical relation to Indicated Resources and are truly Undiscovered, no matter how good the prospect may be. The distinct classification of Inferred as representing growth potential of existing Reserves, which is a significant component part of the Total Resource, clearly separates the Undiscovered Resources from the Identified Resources. The UN classification, discussed by Schanz, recognizes these same boundaries; the UN's R-2 corresponds to Inferred and R-3 corresponds to Undiscovered. The DOE/u's category of Probable Potential and the NEA/IAEA's category of Estimated Additional Resources, however, include both the concept of Inferred and some part of Undiscovered Resources; such classifications require the aggregation of numbers with significantly different probabilities of occurrence, which pose serious statistical problems and do not serve to highlight an important and high probability, near-term potential Resource — the Inferred Reserve.

In the area of Undiscovered Resources, the proposed US resource classification allows for single-point-estimate reporting of Hypothetical and Speculative Resources, or it permits a range of values to be shown, reflecting probabilities of occurrence. I favour the reporting of a probabilistic range of estimates, both to give a visual portrayal of uncertainty as well as to leave room for a 5% or 1% probability assessment for those resources that might be there but are at the moment unconceived. This nicety helps make the classification system truly inclusive, which is very important, both theoretically and for purposes of long-term planning. Having said that, I am none the less well aware that most analysts will use the mean and ignore the extremes; but I still believe, as did the UN people, that we should encourage the expression of a range uncertainty in calculations of Undiscovered Resource potential.

As noted before, in the proposed US resource classification, the Undiscovered category is equivalent to the UN R-3 subdivision. The R-3 subdivision, however, does not allow for an estimate of the portion of that in-place Resource that might be economic, and hence recoverable. We believe that this is an important subdivision estimate to make so that the analyst can more readily compare present with future resource well-being and not be lulled into a false sense of security by a bias of definition to the high side, caused by reporting recoverable numbers on the one hand, and in-place numbers on the other. We would recommend that the UN consider the option of subdividing R-3 just as they have R-1 and R-2. The argument that we cannot know future economics, technology, and unknown geological conditions can be accommodated by remembering that we are only dealing with estimates.

The Measured and Indicated subdivisions of Identified Resources simply provide two levels of detail in Reserve calculations. We have offered the option of a combined reporting under the title Demonstrated, and that quantity would equate with the UN's R-1, with DOE's Reserves, and with NEA/IAEA's Reasonably Assured.

Now let us examine the economic subdivisions on the vertical scale. Previously, we recommended the subdivisions of Economic and Sub-economic but have since concluded that there is a genuine need for a gray area in the middle which we call Marginal. The UN also recognized this need and provided the suffix M for identification. In our definition of Marginal, we note that the economic conditions required for permitting recovery must be specified. This would permit us, for example, to identify a block of uranium resources as being Marginal Reserves at say US \$50 – 75/lb instead of having to present the remainder of the Total Resource beyond Reserves as being Sub-economic. We could also make such a division based on economically related parameters like ore grade.

Considering the three economic subdivisions in aggregate, we can report an in-place number (the Reserve Base as discussed later) which would be equivalent to the UN's R category. We recognize that there will be some materials included in the in-place tonnage estimates that may never be recovered, but we believe that it is important to retain them in the book-keeping because, certainly, on more than one occasion, entrepreneurs have re-entered abandoned mines in search of the unrecovered deposits as a result of changed economic or technological conditions. Our definition of Resource accommodates this total in-place inclusion by noting that "a Resource is an aggregate of valuable minerals from which an economic commodity may be withdrawn".

With respect to book-keeping, I should emphasize that, while the most common presentation of the DOI classification system has been in this format (Fig.1), this is not the only authorized format. Tabular formats with additional detail may be appropriate in some circumstances; certainly, for keeping track of abandoned deposits with potential for future extraction, one needs a detailed book-keeping system, all the components of which are not shown on this diagram.

Figure 2 shows a variant on the system that allows for the recognition of an in-place component of the Identified Resource without making an economic subdivision; we call this the Reserve Base. It is the in-place Resource perceived by the investigator to be worthy of detailed engineering study for purposes of determining an Economic Reserve. One may include all the in-place Demonstrated Resources or just that part analysed for whatever special purposes. As a result of the calculations, some part of that Reserve Base will be defined as an Economic Reserve; another part may be delineated as a Marginally Economic Reserve, given specific conditions; and, of course, a remainder will be rendered Sub-economic as a result of the mining methods or the specific mining plan adopted. With respect to the UN classification, the Demonstrated Reserve Base would equate to a part

RESOURCES OF (commodity name)\*

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Cumulative Production	IDENTIFIED RESOURCES			UNDISCOVERED RESOURCES	
	Demonstrated		Inferred	% _____ Probability Range % _____	
	Measured	Indicated		Hypothetical	Speculative
ECONOMIC	<div> <div>Demonstrated Reserve Base</div> <div>Inferred Reserve Base</div> </div>			+	
MARGINALLY ECONOMIC					
SUBECONOMIC					
	Demonstrated Subeconomic Resources	Inferred Subeconomic Resources			
Occurrences	Includes nonconventional and low-grade materials				

By: (author)

Date:

\* A portion of reserves or any resource category may be restricted from extraction by laws or regulations.

FIG.2. Classification of mineral resources - reserve base.

or all of R-1, and the Demonstrated Economic Reserve derived from the calculation would be R-1-E. For purposes of National Resource reporting, the boundaries of the Reserve Base will be explicitly defined.

Theoretically, I believe the proposed US resource classification is workable for all minerals and is, in fact, universal, i.e. all possible resources of a mineral are potentially included in the system. However, so far we have developed only a conceptual model for all minerals; the data for each mineral or class of minerals must be tailored to fit the framework. Operational classifications for coal and petroleum have been developed, and we have done some preliminary work on defining the boundaries for uranium. For uranium, and for purposes of national resource reporting, one might utilize a Total Resource minimum average grade of 0.01%  $U_3O_8$ . This is well below the average for currently mined Reserves and is richer than the 0.007%  $U_3O_8$  grade for the best of the Chattanooga shale, which possibly should not be considered at this time to be a Resource. Rather, in this system, with the above-defined limits, estimates of  $U_3O_8$  tonnage in the Chattanooga would be recorded in the block labelled Occurrences. Also, it is well to note that uranium from the Chattanooga is probably only economic as a by-product and should be recognized separately from other conventional Resources.

To provide an approximate understanding of the amount of an economically recoverable commodity in advance of detailed engineering analysis, it is useful to consider a tonnage associated with an approximate minimum average grade for the Demonstrated Reserve. In the USA, the average grade mined is about 0.13%  $U_3O_8$ , so possibly 0.1% might be considered as a geologically determinable, physical lower boundary for Economic Reserves. This same average grade could be projected into the Undiscovered category for an estimate of that part of the Undiscovered Resource that might be economic by today's standards. Likewise, an average grade suitable for defining the lower boundary of Marginal Reserves might be useful to subdivide the Reserve Base still further. One might also want to include concepts of thickness and depth with average grade in making geologically measurable judgements on the economic recoverability of resources. The point is, the system recognizes both geological and engineering processes in gaining a perception on economic recovery, requiring only that the parameters of the subdivision be stated.

The concept of Forward Cost provides one such economic subdivision, and it could be fitted into the proposed US resource classification system by determining which Forward-Cost categories represent the three economic subdivisions. The Forward Cost, however, used by itself obfuscates the discovery/consumption book-keeping, which is, after all, one of the principal reasons for keeping track of the Resources. For example, the tonnage estimate for US \$30/lb ore may change from year to year, but there is no way for an outside analyst to determine whether it is a function of change associated with costs, new

discoveries, or consumption. From my perspective, Forward-Cost subdivisions may be useful for some types of analysis, and the proposed US resource classification system recognizes their validity, but they do not stand in place of numbers reported in terms of their physical properties — tonnage and grade and also, perhaps, thickness and depth.

In summary, correlative Resource classification systems are required for good communication. The systems should provide for the inclusion of all Resources, be composed of clearly definable and statistically distinct subdivisions relative to geological certainty of occurrence, and be further subdivided economically, both by measurable physical properties that relate to economics as well as by engineering, cost, and price considerations. The proposed US Resource classification system, which is consistent with the proposed UN system, meets these criteria, and we urge the adoption of these or other correlative systems by all Resource reporting agencies. In so recommending appropriate changes in existing systems, we are well aware that changes in definition cause temporary disruption in annual resource report understanding. We believe strongly, however, that the times require clear communication of the best possible resource estimates properly classified.

## DISCUSSION

G. BIXBY ORDOÑEZ: Have uranium resources or mines been classified on a world basis in terms of tonnes of reserves or of production either into large, medium and small or into large and small only?

C. D. MASTERS: No, at least not by me.

B. S. I. MARENGWA (*Chairman*): What is economic in one area may not be economic elsewhere and so various factors, such as the nature of the infrastructure, have to be taken into consideration.

C. D. MASTERS: Yes I agree. For that reason I selected 0.1%  $U_3O_8$  as an *average* grade limit for the physical parameter defining economic reserves. No doubt some ore is being mined at a lower grade than 0.1% but, in the absence of a detailed engineering analysis of all identified resources, 0.1% is a useful marker for describing the approximate amount of materials richer than that lower limit.

B. BOYD: For the proposed treatment of Chattanooga shales, and presumably also of other unconventional sources of uranium, a time factor is applied. I wonder whether it is appropriate to apply this factor only to unconventional sources and not to conventional resources. Perhaps it would be better not to apply it in either case.

C. D. MASTERS: In fact, no time factor is applied differentially. By conventional resources we imply those which are more or less readily available, but clearly there are time factor variances. With regard to the unconventional Chattanooga uranium, we are simply saying that it is not a resource because, on

the basis of the value of the uranium alone, we cannot visualize economic recovery, so we state that it is not available now, nor will it be in the future. If you want to consider it to be available as a by-product, then this should be clearly stated, and information should be provided on the recovery rates for the principal product, so that the analyst can allow for whatever time factor variance he deems appropriate relative to the conventional ores also recognized.

R. N. CROCKETT: Could you comment on the effect that legal or technical restrictions of the exploitation of reserves would have within an internationally agreed system?

C. D. MASTERS: We think that the geological amount of the resource should be reported regardless of legal restrictions. But, if the figure reported includes known amounts restricted by law or otherwise, then the total resource quantity reported should be flagged and an explanation given in the text providing details of the restriction. In this way the geologist will have said what he knows and the public can decide for itself what to do about the restriction.





**PHYSICAL EXPLORATION AND ESTIMATION  
OF ORE RESERVES**

**(Session II, Part 2 and Session III, Part 1)**



## Invited Review Paper

# GEOSTATISTICAL ESTIMATION OF URANIUM ORE RESERVES

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## Abstract

### GEOSTATISTICAL ESTIMATION OF URANIUM ORE RESERVES

Since the early 1960s geostatistics have been applied for uranium ore reserves calculation, and, as in the case for other minerals, has been considerably developed. This is because of the ability of geostatistics to quantify clearly the main ore reserve questions, i.e. which are the geological (or in situ) ore reserves; what are the effects of a mining selection (recoverable reserves); and what is the precision of these estimates? These different concepts are presented in this paper as applied to uranium ore deposits. First, the specific problem of uranium is analysed, which is the importance of indirect measurements of grade by radiometry logging, which introduces imprecisions generally higher in the uranium grades and tonnages than those coming from the ore reserve calculation itself.

## INTRODUCTION

It is more than 15 years since the first paper [1] on the application of geostatistics to the estimation of uranium deposits was published by the late A. Carlier who was then in charge of the Reserves Section of the Commissariat à l'énergie atomique (France). Soon after this several other engineers from the C.E.A. (notably P. Formery, R. Coulomb and J.M. Marino) also contributed to the same subject [2]. After them, other contributions came from various countries [3–6]. After this initial burst of interest in uranium, almost nothing appeared for the next 10 years. The mining industry had turned its attention towards finding other minerals (bauxite, copper, iron, nickel) which required geostatistical studies of a different type (see Ref. [7] for an exhaustive bibliography of these studies).

The sudden renewal of interest in uranium following the rapid rise in oil prices in 1973 led to geostatistical studies to estimate the reserves of many uranium

deposits throughout the world. The organizations with whom the authors of this article are associated with have carried out more than a dozen studies on a wide range of types of uranium deposit, all of them under the strictest confidential conditions. This paper reviews the problems met in geostatistical studies of uranium ore bodies, based on the authors' experience in this field. Unfortunately, for confidential reasons, it is not possible to cite particular examples. It is this insistence on secrecy, not a lack of success, that has prevented the publication of case studies on uranium. In this paper, we hope to show that geostatistics provides a very suitable method for estimating the in-situ uranium reserves and the recoverable reserves.

## 1. THE BASIC PRINCIPLES OF GEOSTATISTICS

In this introductory section, we examine the basic principles of geostatistics before discussing its application to uranium. A more detailed account of these ideas can be found in Refs [8–10].

Geostatistics, as developed by Matheron [9], is simply the application of the theory of random functions – or more precisely, of regionalized variables – to the exploration and the estimation of natural phenomena. Probabilistic models are needed to describe many sorts of natural phenomena such as the grade of a mineral, because deterministic models do not give entirely satisfactory results.

The choice of a probabilistic model is also suggested by the spatial variability observed in these natural phenomena. This spatial variability has two apparently contradictory aspects to it:

(i) *A random (or erratic) aspect:* Rapid and unpredictable differences in grades are often observed from one sample point to the next;

(ii) *A structured aspect:* The grades usually show a gradual change from richer zones towards poorer zones, etc. This can also be seen in the idea of the zone of influence of a sample or of a drill-hole.

These two aspects, the structuration and the randomness are inherent in these spatially distributed variables. The theory of random functions allows us to quantify these two aspects and thus to estimate these variables. The three basic concepts in geostatistics are the *variogram*, the *estimation variance* and the *dispersion variance*.

### 1.1. The variogram

This function is used to quantify the spatial variability of the mineralization. For every pairs of points  $x$  and  $y$  in the space, the variogram is defined by

$$\gamma(x, y) = \frac{1}{2} E [Z(x) - Z(y)]^2$$

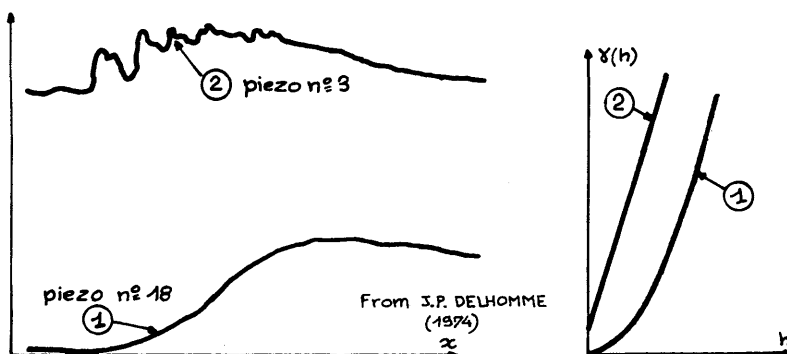


FIG.1. (a) Water pressure profiles; (b) Corresponding variograms.

To estimate this function from the sample data, we make the assumption that it depends only on the distance and the direction between the two points (i.e. on the vector  $\vec{h} = x - y$ ) and not on the location of the two points. This hypothesis is generally fairly reasonable, at least over sections of the deposit.

The variogram characterizes and quantifies the structured aspect of the mineralization as well as the random aspect.

#### 1.1.1. Continuity

The behaviour of the variogram at the origin allows us to distinguish fairly continuous phenomena from highly erratic ones. See Fig.1(a). The discontinuity at the origin, called the nugget effect, which can be seen in Fig.1(b), is characteristic of erratic phenomena and is often, though not always, met in uranium variograms. A few examples of very regular uranium deposits with little or no nugget effect have been found.

#### 1.1.2. Structure: Zone of influence and anisotropy

It is generally found that the variogram (in the direction  $\alpha$ ) reaches a maximum value and flattens out after a distance  $a_\alpha$ , called the range for that direction. The average difference between the grades at the two points no longer depends on the distance between the two points, i.e. the grades are no longer correlated. The range indicates the limit of the zone of influence of a sample. It need not be the same for all directions. For example, the correlation between points along the direction of a roll front is often much greater than the perpendicular to it. Note that a range of 100 m seems to be frequent in this type of ore-body, according to

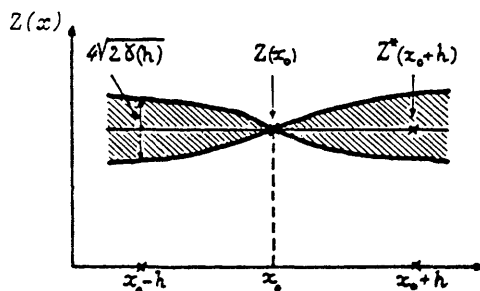


FIG.2. 95% confidence intervals estimating  $Z(x_0-h)$  or  $Z(x_0+h)$  by  $Z(x_0)$ .

Parker [11] and Sandefur and Grant [12], but the nugget effect can be of various magnitudes (not only 50% of the sill, as noted by these authors), which suggest that it may be too early to speak of typical variograms for the various types of uranium deposit.

### 1.2. Estimation variance

The estimation variance, which gives a measure of the accuracy obtained by estimating the true (but unknown) mean of a variable  $Z$  by an estimation  $Z^*$ , is defined as the variance of the error,  $Z - Z^*$ , that is

$$\sigma_e^2 = \text{var} (Z - Z^*)$$

From the mathematical aspect, this quantity is very easy to handle. It can be used to determine the appropriate confidence interval if the distribution of the values is assumed to be known. For example, when the distribution is assumed to be normal (i.e. Gaussian) the 95% confidence interval is then  $Z^* \pm 2\sigma_e$ .

From its definition  $\gamma(h)$  gives a variance of estimating  $Z(x+h)$ , or  $Z(x-h)$ , from the value of  $Z(x)$ , at least up to a multiplicative constant (see Fig.2).

From this, it is intuitively obvious that the variance in estimating the average grade  $Z_V$  over the volume  $V$  by another average grade  $Z_v$  over a different volume  $v$  can be deduced from the variogram. A typical example of this is when the average block grade is estimated by the average of several drill-hole grades. In this case,  $V$  is the block and  $v$  is the set of drill-holes.

In fact, the basic idea of geostatistics is that the accuracy of estimation is a function of the spatial distribution of the data relative to the zone to be estimated and also of the nature of the phenomenon studied. Since the variogram quantifies the continuity of the mineralization in space, it can be used to determine the accuracy of estimation.

It can be shown that the estimation variance associated with any linear combination can be calculated in the same way from the variogram. The weighting factors corresponding to the linear combination with the minimum variance can then be deduced from this. This process, which is called kriging, will be shown in the section on the calculation of in-situ reserves.

### 1.3. Dispersion variance and the idea of support

The variability of a grade within a given deposit depends on two factors: the size of the support  $v$  (i.e. the volume of the samples) and the size of the zone  $V$  being studied. It is well known that the grades obtained from drill-holes are much more variable than the average grades of blocks, which are in turn more variable than those of large panels. The dispersion variance  $D^2(v/V)$  of a volume  $v$  (the support of the samples) in a given other volume  $V$  (the field) is used to quantify this dispersion. It can be shown that  $D^2(v/V)$  can be calculated theoretically from the variogram equation and from the geometry of  $v$  and  $V$ . The selectivity of a mining method can be characterized broadly by the dispersion variance of the minimal unit of selection, a figure used in the calculation of the total recoverable reserves.

The most recent developments in geostatistics have been in the domain of estimating local recoverable reserves. This relies on the theory of recovery functions and of conditional simulations which are both discussed later in this paper.

### 1.4. In-situ reserves, recoverable reserves

The definition of in-situ reserves is, in principle, quite unambiguous: it is the total tonnage of ore or of metal actually contained in a certain volume in space (e.g. in a mining lease). No allowance is made for dilution due to the mining method or to the method of treatment. Consequently, the in-situ reserves are independent of the cut-off grade. However, the term 'in-situ reserves at a particular cut-off grade' is often applied in practice to ores not subject to a natural (or geological) cut-off grade. This is often the case for stratiform uranium deposits where a cut-off grade is used to differentiate the mineralized sections of drill-hole core from the waste. It is clear that no mining method could ever be selective enough to take only the ore defined in this way. However, the reserves thus defined can be considered as the in-situ reserves since this is the maximum amount of ore which could be recovered.

Defining the recoverable reserves is somewhat more difficult. The recovered reserves can obviously be determined from the quantity of ore actually recovered after the mining has been completed, but defining the recoverable reserves before mining starts requires that certain simplifying assumptions be made about the mining method to be used. In geostatistics the recoverable reserves are defined by

the minimal selection unit and by the cut-off grade. In the fifth part of this paper, the methods used in geostatistics for estimating the recoverable reserves are presented in detail.

### 1.5. The role of the computer

A general presentation about mining geostatistics would be incomplete if no mention was made of our basic tool, the computer. Without it geostatistics would never have developed to its present level. Some years ago, the computational time required for a geostatistics study was commonly regarded as prohibitive. These problems have now been overcome and today's kriging programs are as rapid as those based on polygons of influence: over recent years general packages of programs have been developed. They can handle data from very different types of deposit, incorporating not only grades but also pure geological and topographical data, using visual display units. In any case, the cost of the computer time is often negligible if compared with the whole cost of an ore reserve calculation. Nowadays the main computational problems consist in the increasing size of the basic data files, which commonly raise many hundred thousand records for uranium deposits, the management of which requires a computational organization which is fairly independent of the numerical processing of the data.

## 2. PROBLEMS SPECIFIC TO THE GEOSTATISTICS OF URANIUM

For a specialist in ore reserve estimation, uranium deposits pose two particular problems:

- (i) The mineralization varies enormously both in its geometric dispersion in space and in the numerical dispersion of the grades, which may vary from 0.1 to 100 kg U/t in a single deposit; and
- (ii) The grades are often calculated indirectly from the radioactivity readings rather than directly from chemical analyses.

### 2.1. Geometric variability

Geologists have listed several types of uranium deposit in which the mineralization occurs in extremely small geometric units (small lenses or small veins) often spread through a rock with a disseminated mineralization. It should be noted that this sort of deposit is not specific to uranium. Other minerals also show this type of structure. However, as the accuracy of the radiometric readings makes it possible to mine the uranium very accurately, geologists are more inclined to require a very fine estimation grid for uranium than for other metals such as copper or lead. This poses problems for the geostatistician charged with estimating the in-situ reserves



or the recoverable reserves for a given minimum size selection unit. To overcome these problems, the geometry of the deposit, which is rarely known accurately, should be simplified. The increased variability due to the imperfect knowledge of the geometry of the deposit can be taken into account later in calculating the estimation variance; this procedure also makes it easier to determine the spatial correlations. The reserves of uranium can then be estimated with no more difficulty than for other metalliferous ores. This remark is particularly applicable to low-grade disseminated ores and to large vein-type deposits. However, these problems still remain for deposits composed of many small veins.

## **2.2. The numerical variability**

The uranium grades found in small samples may range from 0.1 to 100 kg/t and their histogram has a characteristic log-normal shape. It is well known that linear estimation methods do not provide good results when used with variables such as gold and uranium, which have a highly skewed distribution like the log-normal in these cases. It is often best to work with the logarithms of the grades.

The amount of work in a study can be considerably reduced by keeping the following three points in mind:

(1) Since the blocks selected for mining will be around 1 to 15 m rather than several centimetres, it suffices to work with the regularized grades (i.e. the grades averaged over a length equal to that of the selection unit). This has two advantages: it reduces the number of data to be handled, and also reduces the number of grades with extremely high values.

(2) Because of the long tail of extreme values, distributions like the log-normal pose problems when variances and variograms have to be calculated. This can be overcome by calculating the variogram of logarithms of the grades and deducing the variogram of the grades themselves from it. The usual linear techniques such as polynomial regression kriging and least squares can then be used.

(3) Finally, if the regularized grades are still highly variable, log-normal kriging, which was specifically designed to handle these sorts of distribution, can be used (see Ref. [13]).

## **2.3. The grades obtained indirectly from gamma logging**

### **2.3.1. General remarks**

As uranium mining is highly profitable at present, most uranium deposits are fairly closely sampled. Unfortunately, the vast majority of sampling has been the less expensive variety of destructive drilling with gamma logging down the boreholes, which only provides an indirect measure of the uranium grade and which

must be recalibrated as often as possible from the correct chemically determined grades. So, despite the apparent wealth of data available, the results of all this sampling are somewhat unsatisfactory when it comes to reserve estimation. As early as 1964, Carlier [1] warned of the problems provoked by decreasing the ratio of chemical analyses to radiometric readings; he would certainly have been appalled by the low ratio prevalent today.

It is clear that, from the geologists' point of view, gamma logging is an exceedingly useful tool for determining the morphology of the deposit in detail. On the other hand, it is simply not accurate enough for estimating the quantity of metal contained on a deposit. Faced with this problem, the geostatistician can take one of two approaches:

- (1) He can transform the radiometric readings of the samples (eventually the regularized samples) or of the mineralized sections into uranium grades and then use this data to estimate the average uranium grade for each block and later for the deposit; or
- (2) He can estimate the average block grades directly from the radiometric readings and the chemical analyses found in the surrounding region.

### 2.3.2. *Transforming the radiometric readings into grades*

The theory shows that the integral of the gamma-log profile is proportional to the radium grade of the corresponding section of drill-hole core. The oldest method was to establish the constant of proportionality (either by comparison with the results obtained under laboratory conditions or by reading the appropriate numerical tables), to ask the geologist to estimate the ratio between the uranium and the radium and then to define a deterministic relationship between the gamma-log readings and the uranium grades. In practice, this method is suspect on several grounds, and so it is preferable to establish the relationship statistically from the experimental data.

There are two different procedures for doing this:

- (1) On studies where the gamma logs of the samples are to be converted directly into uranium grades, the relationship between the two can be established statistically by linear regression (or by regression on the logarithms) for the mineralized sections for holes where both the chemical analyses and the gamma-log readings are available (see the example of a stockwerk deposit in section 4.4.3). In the case of disequilibrium, the regression should be carried out on radium equivalents and not on the gamma-log readings themselves. One practical difficulty met with this method is that the lengths of the mineralized sections are not constant. So the mineralized sections have different variances which must be taken into account in the regression.
- (2) Calculating the uranium grades from the gamma logs is even more complicated in the case of three-dimensional deposits. Since the gamma-log reading

is proportional to the integral of the radioactivity in the 'zone of influence', it is necessary to 'deconvolute' the gamma-log readings, and then to calculate the regression between the radium equivalents obtained from the deconvolution and the chemical analyses.

In practice, the correlations observed experimentally are often found to be rather low; this happens for two reasons:

- (1) Uncertainty over the relative positions of the gamma logs and the chemical analyses; and
- (2) Instability in the deconvolution (it is very sensitive to the number of data in the neighbourhood).

Finally, the 'philosophy' of geostatistics places great emphasis on the influence of the size of the support of the sample; the individual data values are of little importance in themselves since the average grade in a particular volume in space is going to be calculated precisely by taking the average of the numerical data.

For these reasons, we often prefer to calculate the linear regression after averaging grades found by the two methods over lengths of 50 cm or 1 m or more. Taking the averages over these lengths (or regularizing as it is called) is very important when the relative vertical displacement between the gamma logs and the chemical analyses is unknown. It also often allows us to avoid the problems of deconvoluting the readings. The theory of co-kriging suggests that it would be better in practice to use a formula such as

$$Z_x^{*c} = m_v^{*c} + \beta(Z_x^r - m_v^{*r})$$

where

$m_v^{*c}$  is the average grade in a volume  $v$  around the drill-hole, estimated by kriging the available chemical assays

$m_v^{*r}$  is the average grade in a volume  $v$  around the drill-hole estimated by kriging the gamma logs from the same drill-holes.

This method appeals to geostatisticians because the coefficient  $\beta$  is the same for the whole deposit and is deduced from the variograms and the cross-variogram between the gamma logs and the chemical analyses. The parameter  $m_v^{*c}$  is estimated locally from the chemical data in the neighbourhood. So the procedure is a sort of local regression on the variables which can, for example, take into account local disequilibrium.

### 2.3.3. Co-kriging

When the ratio of chemically analysed holes to gamma-logged holes is high and when the chemical data are spread evenly throughout the deposit, it is worthwhile mixing the two types of data directly (without carrying out a preliminary regression) to estimate the average chemical grade of a block (co-kriging)

$$Z_v^e = \sum \lambda^\alpha Z_\alpha^c + \sum \mu^\beta Z_\beta^r.$$

A detailed example of this method is given by Guarascio [5].

The weighting factors  $\lambda^\alpha$  and  $\mu^\beta$  are found automatically by solving the system of linear equations for the co-kriging. The coefficients of these equations depend on

- (i) The relative positions of the block to be estimated V and the sample points, and
- (ii) The variogram and cross-variogram values.

The theory shows that it is worth while co-kriging the data when there are a fairly large number of holes which have been analysed chemically. If all the holes have been gamma-logged and analysed chemically, the gamma logging does not provide any additional quantitative information; it only acts as a double check and the co-kriging degenerates to a kriging using only the chemical data. On the other hand, if the number of chemically analysed holes is small, the co-kriging degenerates to a regression like the one described in the preceding paragraph.

#### *2.3.4. The grades obtained from the gamma log are less accurate*

The estimation variance obtained from the co-kriging equations automatically takes into account the relative accuracy of the two types of sample. By contrast, this is often forgotten or ignored when the estimation is based on chemical grades and those obtained by converting the gamma-log readings. It should be remembered that the accuracy with which the mineralized volume is known is considerably improved by using the gamma-log readings as well as the chemical analyses, but that the average grade of the deposit depends only on the results of the chemical analyses. For example, in a deposit containing 200 gamma-logged holes of which only 15 have been analysed chemically, the relative standard deviation on the tonnage is about 5% whereas the corresponding figure for the average grade is about 30%.

### 3. EXPLORATION OF THE ORE-BODY

Geostatistics allows us to determine the optimal drilling plan.

#### 3.1. Drilling plan

The use of systematic, regular drilling patterns is strongly recommended not only because it eliminates subjectivity in the choice of the drill-hole locations but also because it makes it easier to see the spatial correlations and to estimate them

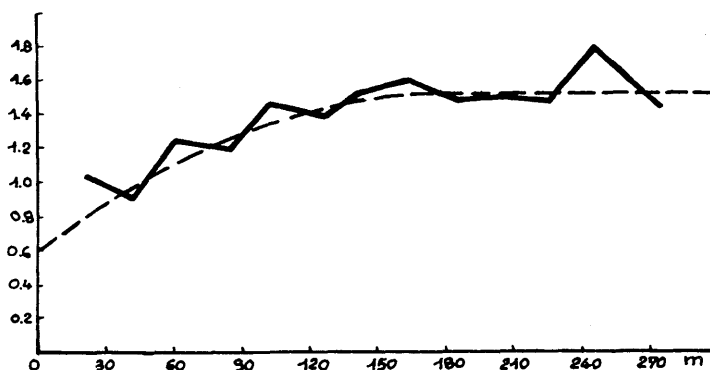


FIG.3. Example of horizontal variogram for  $U_3O_8$  accumulations (relative values).

more accurately. The drill-holes, which fall into a low-grade zone, are just as important as those which hit the ore.

A systematic sampling grid need not necessarily be a square one. It should be adapted to the anisotropy evident in the deposit (e.g. in a roll-front deposit). The variogram can provide a measure of the anisotropy and of its orientation, thus providing the orientation for subsequent drilling programmes.

It is important to determine the small-scale variability of the mineralization right from the outset of the drilling campaign. This is most easily done by drilling a number of holes on a much finer grid, usually in the form of a cross in a part of the deposit which is considered to be representative of the deposit. The advantages to be gained from a better understanding of the small-scale variability are two-fold: first, it helps in the interpretation of the morphology of the deposit (which is often difficult for uranium deposits where the grades vary enormously) and second, it is used in determining the value of the variogram for small distances. As we show in the following paragraph, we need to know this so as to be able to predict the characteristics of future drilling grids correctly.

### 3.2. The optimal size for a drilling grid

Once the variogram is known, geostatistics can be used to determine the accuracy of either a global or of a local estimate. It is then possible to optimize the grid size of any subsequent drilling campaign. In the case of uranium deposits, the usual problem is to optimize the size of future drilling grids after taking account of the existing holes rather than to specify the exact location of the holes to be drilled.

In the case of a tabular ore-body for instance, the problem will be to recommend a grid of vertical holes in order to achieve a given precision on a given

variable (which can be either the  $U_3O_8$  quantity of metal or the quantity of ore, in which case one will use the variogram either of the accumulation or of the mineralized width). The variogram of Fig.3 will allow computing the extension variance of each hole within its area of influence, thus allowing the computation of the total estimation variance of the ore-body (which depends, too, on the size of the ore-body). In practice, considering the total probable extension of the ore-body, one will compute the different precision associated with the successive doubling of the sampling density.

#### 4. CALCULATING THE IN-SITU RESERVES

##### 4.1. Review of the definitions

The three basic terms are geological reserves, in-situ reserves and recoverable reserves. The last two have already been defined so it only remains to define 'geological reserves'. This refers to the total potential reserves contained in the deposit and does not take into account any economic criteria or technical constraints. We have already seen that it is often convenient to use the term 'in-situ reserves at a particular out-off grade'. We shall continue this practice and so the term 'in-situ reserves' refers to the geological reserves whose point grade is above a certain minimum; that is, the term 'in-situ reserves' takes account of the effect of the cut-off grade and the term 'recoverable reserves' also takes account of the technical mining constraints imposed by the type of mining method adopted.

These parametrized in-situ reserves define implicitly a quantitative zoning of the deposit, and grade/tonnage curves for the whole deposit. It should be recalled once again that the grade/tonnage curve only relates to the spatial distribution of the point grades (or more correctly the grades of sections of core) throughout the deposit and therefore does not give any indication of the recoverable reserves since it would be impossible to select the blocks to be mined in drill-hole sized pieces.

The basic operation in estimating the in-situ reserves will be to estimate the average value of the variable over blocks the same size as the basic mining units. The traditional methods for calculating this average have been basically geometric (zone of influence, polygonal method, inverse distance methods), whereas the geostatisticians prefer to use kriging.

##### 4.2. Kriging

In this section a brief review of the basics of kriging is given. For more details see Refs [8,9]. The basic idea is to estimate the average block grade by a linear

combination of the samples in the vicinity of the block. To avoid having the estimate systematically biased by high grades or low grades, this neighbourhood must contain samples outside the block as well as those inside it.

Knowing the variogram allows us to evaluate the variance of any linear estimator by considering the relative positions of the block to be estimated and the samples in the neighbourhood. Kriging uses this property of the variogram and then selects the weighting factors which minimize the estimation variance of the block.

The weighting factors  $\lambda^\alpha$  are calculated from a system of linear equations whose coefficients are calculated from the variogram:

$$Z_V^* = \sum_{\alpha=1}^N \lambda^\alpha Z(x_\alpha)$$

with

$$\sum \lambda^\alpha = 1$$

$$\sum_{\beta=1}^N \lambda^\alpha \bar{\gamma}(x_\alpha, x_\beta) + \mu = \bar{\gamma}(\alpha, V)$$

where  $\bar{\gamma}(\alpha, V)$  is the average variogram value between any arbitrary point in the block  $V$  and the point  $x_\alpha$ .

The estimation variance is given by

$$\sigma_K^2 = -\bar{\gamma}(V, V) + \mu + \sum \lambda^\alpha (\alpha, V).$$

In practice, kriging plans which specify the size of the neighbourhood, the grouping of the samples, etc., are very varied and depend on the location of the data and on the variogram. The following example shows how important the data outside the block can be. The deposit considered is of the roll-front type and has an anisotropic variogram. Blocks  $a \times a$  ft are to be estimated from an  $a \times a$  ft square drilling grid using the central drill-hole and the eight surrounding values. Figure 4 shows the percentage weighting given to each of the nine samples used.

It should be noted that the central sample receives a weighting just slightly lower than a half and also that, because of the anisotropy, samples at equal distances from the centre do not have the same weighting factors.

• 3.	• 16.	• 3.
• 6.	• 44.	• 6.
• 3.	• 16.	• 3.

FIG.4. Two-dimensional kriging plan: weighting factors (in %) for a complete regular configuration.

This example shows what happened in one particular case. The weighting factors in other cases could be different, being dependent on the variogram and the layout of the data.

The methods to be used in dealing with uranium deposits depend largely on whether the deposit can be considered as basically two-dimensional or whether it is necessary to consider the third dimension as well.

#### 4.3. Two-dimensional calculation

These methods can be applied to two types of deposit:

- (i) Vein-type deposits where the data has been projected into the plane of the vein; and
- (ii) Horizontal (or inclined) sedimentary deposits with regularly spaced vertical drill-holes.

In both cases, the uranium accumulation expressed either as the total mineralized width  $P$  and as the total quantity of metal can be calculated along the direction of the drill-holes for each section of core. The problem is then reduced to the study of two regionalized variables  $P(x, y)$  and  $Q(x, y)$  whose variograms and cross-variogram can easily be calculated. The mineralized region will be defined as an area of the plane of projection. The variance in estimating this surface can be evaluated from the appropriate geostatistical formula. The average values of  $P$  and  $Q$  in this region can then be evaluated either by kriging or by taking a simple arithmetic mean (regular grid, no edge effect). This gives the estimated total tonnages of ore and of metal contained either in the whole deposit, or in a particular panel of it, as the case may be. It also provides the estimation variance and the confidence intervals, if required.

#### 4.4. Three-dimensional calculation

The method described in this section applies to deposits with a marked component in the third direction; either low-grade massive deposits; or thick sedimentary deposits.



In these two cases it is more convenient to calculate the in-situ reserves for panels whose height is equal to the height of a bench (in the case of open-cut mining) and whose horizontal dimensions are the same as for the drilling grid. The method to be used in estimating the reserves then depends on whether the minimum vertical selection is the same as the height of a bench or whether a finer vertical selection is possible.

#### *4.4.1. Deposits mined without selection in the vertical direction*

In this case, the estimation is exactly the same as for a metalliferous deposit (e.g. porphyry copper); the grades are regularized (averaged) over a height equal to that of the benches. By considering these grades and the geology of the deposit the panels belonging to the deposit are chosen and they are then estimated one by one. The tonnage of ore in-situ is found by multiplying the volume of all these panels by the density of the ore (if it is constant); the average grade is simply the average of the block estimates.

We strongly recommend this method of estimating in-situ reserves since it is always possible to decide on a minimal thickness (e.g. 1 m) and to consider panels of that thickness.

#### *4.4.2. When it is possible to make a vertical selection of the mineralized sections*

In tabular deposits, it is current practice to define the section mineralized as that part of core with a reading above a certain cut-off level, the rest being regarded as waste. Although this arbitrary cut-off does not correspond to any particular mining practice, it seems that this procedure is popular with the geologists and that this is why it is often used. It should be noted, however, that this method corresponds more to the calculation of recoverable reserves than to in-situ reserves (see later in text).

The limits of the mineralization for a particular cut-off grade are defined using these drill-hole readings for a series of benches of constant height (e.g. for an open-cut pit). This allows us to define two service variables, the tonnage (or percentage) of ore and the tonnage of metal, for each bench going down the drill-hole. These two variables can be defined for each cut-off grade; but, for a fixed cut-off they can be considered as three-dimensional regionalized variables, thus allowing us to do a complete geostatistical study – variograms, kriging, etc. The average value of the tonnage of ore and of metal can be estimated by kriging for each of the panels the size of the drilling grid. Summing these over the whole deposit gives the total in-situ reserves.

A practical way to reduce computing time in 3-D kriging, using only vertical drill-holes, is to use semi-random block kriging: it consists of assigning a single weighting factor to all the data in each block and assuming that each individual

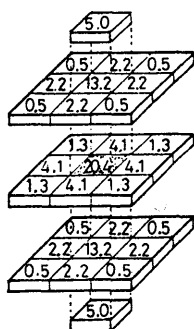


FIG.5. Three-dimensional kriging plan: weighting factors (in %) for the complete configuration.

sample is spread at random through the block. In the example of Fig.5, a group of 29 blocks is used, and the weighting factor corresponding to each block is displayed; the weighting factor of the central block is small (20%), which emphasizes in this case the influence of the outer samples. This is a frequent situation in uranium estimation where large nugget effect and short ranges are common.

## 5. RECOVERABLE RESERVES

### 5.1. General comments

The full extent of the recoverable reserves can only be defined unambiguously after the mining has been completed. The process of mining a deposit is extremely complex and varies with time and circumstances. A particular marginal block might well be taken at a time when more material is required for the crusher, or be rejected at another time when more metal is required. Consequently, the concept of recoverable reserves is strongly linked to some simplified model of the mining process. This simplification makes it possible to calculate the recoverable reserves.

The process of selecting or rejecting blocks for mining depends on two factors: a minimal volume of rock for which it is feasible to make a decision; and the information available at the time when the choice must be made.

Because of the possibility of radiometrically testing the block of uranium ore before the decision is taken, it is reasonable to assume that the average grade of a block is known perfectly at the time when the selection is made. So it remains to define the minimum size selection unit. This is not always easy to do since different size selection units are often used at different times during the mining. Two different cases can arise:

- (1) There is a consensus of opinion about the size of a block for selection. For example, in an open-cut mine where no selection is made within a level, the size of a block can be defined as a cube with the height of a bench. In a vertical vein the blocks can be defined as having the dimension of one individual blast area;
- (2) The mining method is very selective and the minimal block size varies depending on the condition at the time. The concept of a selection unit is no longer physically reasonable and its use in the calculations can only give an idea of the order of magnitude.

Those are two different methods of estimating the recoverable reserves which correspond to these two different situations. They are (1) to simulate the grades and thereby the recovery; and (2) to calculate the recovery directly using recovery functions.

## 5.2. Simulating the recovery

A mining engineer knows the mining method chosen for use in a particular ore-body. He knows the speed of production, the size and type of equipment being used, etc. and so, given a plan showing the grades of all the blocks, he can trace the sequence which the mining will follow. He can then calculate the tonnage of ore recovered and its average grade. Unfortunately, at the time when the recoverable reserves have to be estimated, the grades of all the blocks are not known exactly, so it is impossible to predict the exact sequence of mining.

In the case of stratabound deposits, the mining selection often mainly consists of the definition of the workable thickness. It is then possible to define service variables corresponding to this vertical selection: for each drill-hole the minable width and the corresponding uranium accumulation are calculated with a manual or automatic algorithm respecting the technical constraints of the mining process. Then, the calculation of the recoverable reserves is similar to the in-situ one, described in section 4.3. If a horizontal selection is also done (case of lenses . . .), this method is less accurate and it is necessary to know more precisely how the grades are distributed in the space.

A conditional simulation of the grades (or radiometric readings) is one way to obtain a plausible impression of how the grades are distributed, although this is difficult because the data available at the end of the exploration are still rather sparse [14].

Figure 6 shows an example of vertical cross-section in sedimentary uranium ore-body: it shows the real radiometry as measured when blasting. A simulated radiometric profile of the same deposit is given in Fig.7, and Fig.8 shows a vertical section in the same location that Fig.6 obtained from a conditional simulation of the deposit based on the information from exploration vertical drill-holes. The

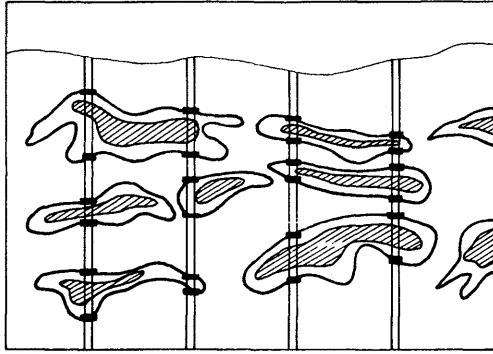


FIG. 6. Example of radiometric vertical cross-section as measured when mining.

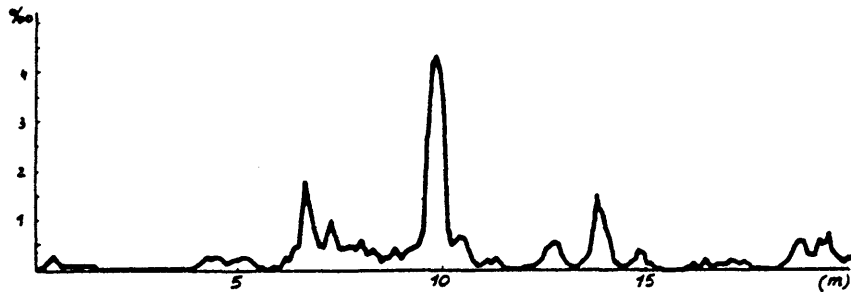


FIG. 7. Simulated radiometric profile.

close resemblance between these two figures is quite evident. Thus, it would be possible to use the results from the simulation to predict the recoverable reserves corresponding to a given mining method, for each of the cut-off grades. When the mining engineer has drawn the positions of the blocks to be mined on the plan showing the simulated grades, the contents are digitized and the computer provides the tonnages and average grades from the simulated block grades.

### 5.3. Direct estimation of the recovery

We have seen earlier that for low-grade massive deposits mined by open-cut methods, the selection process can be regarded as a choice between unit blocks of constant size whose grade is known. A local estimation of the recovery for each panel can then be made by using recovery functions. Detailed accounts showing how to use these functions for uranium deposits are given in Refs [15, 16]. So, in this article we simply say that the technique consists in estimating the local histogram of the grades of blocks to be selected for each panel of the deposit. To do this, we must know how to predict the histogram of block grades for the whole deposit from only the grades of the core sections (or from the radiometric readings).

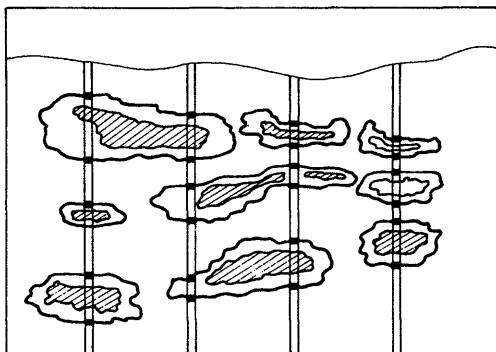


FIG.8. Example of conditional simulation of a radiometric vertical section (same location as Fig.6).

This problem is referred to as “the permanence of low distribution” and how to estimate a non-linear function of the grades so as to know the proportion of blocks whose grade is above a specified cut-off grade. This is the central problem in non-linear geostatistics.

The techniques developed from the work of Matheron [17, 18] and Maréchal [19] by the Geostatistics Centre now make it possible to resolve these problems routinely by using disjunctive kriging [15].

#### 5.4. Concluding remarks on the estimation of recoverable reserves

It should be emphasized once again that the recoverable reserves can only be estimated by making certain simplistic assumptions about the mining method to be used. This might shock mining engineers who always declare that the reality is always much more complicated. In fact the main reason for using these methods is that they make it possible to calculate the recoverable reserves. Secondly, experience has shown that these techniques provide results which are closer to the figures actually obtained from the mine than the estimates made by using an empirical ‘fudge factor’ to adjust the estimates of in-situ reserves.

## 6. CONCLUSION

A glance at the bibliography shows that geostatistics has been used for a long time for estimating uranium reserves. When the overall results of using geostatistics are examined, the following points can be made:

- (1) The systematic use of geostatistics would allow the standardization of such terms as in-situ reserves and recoverable reserves, and of the methods used for converting gamma-log readings to grades, etc.;

- (2) As with many other types of ore, geostatistics is much more useful when the data are abundant, as often happens with uranium, or when the ore has a clearly defined structure. From this point of view, much more spectacular improvements can be expected in the future from large low-grade deposits than from small complicated deposits;
- (3) Geostatistics can be expected to provide satisfactory answers to some of the problems specific to uranium choice of the ratio between radiometric readings and chemical grades, definition of a selection criterion using the radiometric readings taken on the ore before blasting, truck scanning, etc.

In conclusion, it can be said that geostatistics provides a good workable system for the routine estimation of uranium reserves, as well as for other metallic ore-bodies.

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## DISCUSSION

V. ZIEGLER: I would just like to mention that the method described by Mr. Marbeau is the one that has been used in France for calculating uranium resources since 1955, when it was introduced at the Commissariat à l'énergie atomique by Professor Georges Matheron.

J.P. GOMEZ JAEN: I agree in qualitative terms with the conclusions drawn by Mr. Marbeau about recoverable reserves in a deposit. However, it would have been more useful if he had given us some specific results. Secretiveness about such results, to my mind, detracts somewhat from the glamorous reputation currently enjoyed by geostatistics. In any case I believe that geostatistics has two very serious disadvantages which prevent it from being a universal panacea.

First, it does not give any idea of the position of local mineralization, as Mr. Marbeau mentioned, and a mine should be considered not so much in terms of the deposit as a whole as in terms of the ore mined day by day or month by month and, in the final analysis, in terms of the yearly balance.

Secondly, the cost of obtaining an appropriate and precise variogram of a very dense drilling grid, which includes the cost of a system of crosses, is prohibitive.

Finally it can be said that, on a large scale, polygonal or zone-of-influence methods have in any case proved reasonably satisfactory.

J.P. MARBEAU: To answer your general points first, I would say that geostatistics, without being a universal panacea, must be regarded the best method at present available for calculating reserves, subject to two conditions: (i) that there is close co-operation with geologists, especially in order to integrate the controlled limits of the deposit with the geostatistical model (examples of fruitful co-operation of this type are becoming more and more numerous, and the initial antagonism between the two disciplines has been surmounted); and (ii) that the geostatistical model is consistent in statistical terms (too many disappointing experiments have been the result of incorrect application of the theory).

In reply to the first of the specific points you raised, I should point out that the methods for calculating recoverable resources presented in the paper relate to feasibility studies of deposits; the aim is to evaluate the effects of the type of selection envisaged on relatively large units in order to be able to plan mining on a long-term basis. Your point relates more to the mining stage, when there is much more information available, and when very small units (of the size of the drilling grid) can easily be estimated by kriging.

Secondly, it is wrong to believe that the variogram requires an excessive number of drill-holes, since merely the fact of being able to master the variogram at an early stage makes it possible to quantify the effectiveness of any additional grid. Investment in a few additional drill-holes could thus enable savings to be made at a later date.

Thirdly, polygonal and zone-of-influence methods are useful as rapid and semi-quantitative indicators of tonnage and of the average in-situ content of a deposit at the mining stage. However, these methods are dangerous when used for a feasibility study, since they do not take into account the effect of selective mining and mine support and do not indicate the accuracy of results. Abundant examples of considerable errors are to be found in the literature both for uranium deposits and for other minerals.

R.G. WADLEY: I would like to endorse the views expressed by Mr. Marbeau concerning the application of geostatistics in the evaluation of uranium ore bodies. Our experience has also been that classical methods alone are inadequate to provide a sufficiently reliable estimate of ore reserves. The important point here, I believe, is that of scale: when a prospect is to be evaluated merely on the basis of the results of several tens of drill-holes, the application of geostatistics is inappropriate. However, in the case of a large drilling programme involving many hundreds or thousands of holes a geostatistical approach would seem to me to be not only desirable, but essential.



# CASE STUDIES IN GEOSTATISTICAL ORE RESERVE ESTIMATION OF URANIUM DEPOSITS

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## Abstract

### CASE STUDIES IN GEOSTATISTICAL ORE RESERVE ESTIMATION OF URANIUM DEPOSITS.

Geostatistical methods of ore reserve estimation for uranium deposits have been in use for 15 years but many practical problems of implementation of the technique have never been discussed or reviewed. It is the purpose of this review paper to sum up all what has been done in the field, especially concerning new developments for the case of North American deposits. The basis of the theory has already been extensively covered by other authors, and we intend to focus on the special problems which occur in different types of uranium deposit, taking real examples to illustrate each point. The paper starts with disequilibrium problems and new developments in their treatment, then discusses volume and tonnage estimation problems, optimum drilling plans, stressing the benefits to be obtained from relative or regressive variograms. The ore-waste distribution problem is reviewed and practical solutions proposed. Altogether, this paper is intended as the backbone for a handbook on geostatistical ore reserve estimation methods which could be used by governments or companies who wish to obtain the best possible assessment of their uranium reserves.

## 1. INTRODUCTION

The need to insist on the necessity to have accurate methods of ore reserve estimation in uranium is certainly superfluous. It is also well established now that theoretical methods like "geostatistics" and "kriging" which have been around for almost twenty years (Matheron [1]; Carlier, [2]) need considerable amounts of refinements and custom tailoring to provide a useful tool in defining the mineralized inventory of a uranium deposit. In this paper we intend to show by means of real case studies and examples which problems are to be encountered in the estimation of different types of uranium deposits and where to look for the solutions which are available so far. No formulae will be given and we intend

to point to geostatisticians uranium problems and to practitioners the few simple concepts which should help them. It is a fact that there is a considerable amount of on-going research in this area but practical tools are already available and have already been proven, although an essential difficulty is the secrecy involving mining operations and sometimes billion dollars lawsuits.

The different types of uranium deposits which we will consider in this paper, following Bailey and Childer's [3] classification, include strata-controlled deposits, like trend deposits of New Mexico, roll fronts or stack deposits of Wyoming, structure or fracture controlled deposits like Saskatchewan deposits as well as intrusive controlled deposits. Once again political and geological boundaries have little in common and the easiest known deposits may be of little interest to major producing organizations. It has to be taken as a fact however.

## 2. QUESTIONS TO BE ANSWERED

Solutions to be found for these different types of deposits will be different but the general questions are the same. One needs of course to establish with a good degree of confidence, the tonnage of mineable ore, its location and grade. A subsidiary question is of course how much more information is required to achieve a better precision. Overriding all these questions is the problem of the disequilibrium and correction factor to transfer  $\gamma$ -ray measurements into equivalent  $U_3O_8$  quantities. Using some already published examples and adding several new ones, we will thus insist on the applications of the concepts and show how one can establish with confidence available uranium reserves. Rather than describing methods we will refer the reader to published literature and concentrate on facts rather than theory, describing problems in the order given above.

### 2.1 Disequilibrium problems

This problem has been extensively reviewed recently by Dagbert and David [4]. One usually confuses under the same heading the problem of converting the gamma activity in a drill hole to equivalent  $U_3O_8$  which is the first step to take once readings are made. The method of Scott et al. [5] is usually well accepted. An additional problem well covered again by Scott [6], Czubek [7], Czubek and Zorski [8], Jonas [9], Conaway and Killeen [10] is due to the fact that the chemical

grade is from a piece of core and the  $\gamma$ -ray emission is from another volume. The physics of gamma emission is well known and this can be corrected. The real disequilibrium problem which is much more elusive and has a different importance in different parts of the world is of course due to the fact that the principal gamma source is  $^{214}\text{Bi}$  which might be there long after uranium which produced it is gone. The migration of  $^{222}\text{Rn}$  may generate another disequilibrium between  $^{214}\text{Bi}$  and  $^{238}\text{U}$  and systematic biases may appear. Again this is a well known problem and techniques have been designed to cope with it including  $\beta$ - $\gamma$  methods, closed-can, and D.F.N.<sup>1</sup> among others. Altogether however, it is usually virtually impossible to unravel all of the deterministic processes which create the differences between radiometric and chemical grades. For this reason statistical procedures have been designed to try and design "correction" factors to be used until a better understanding of all geological conditions can allow the design of better factors. The regression solution has been around for many years as can be seen in Figure 1, showing the mine of Urgeirica in Portugal. It may become totally inappropriate in cases like shown on Figure 2 (eolian origin). The regression is of course a lognormal regression technique which requires a lognormal hypothesis (Matheron, [11]; Carlier, [2]). This is met in many European deposits. It might be different in Wyoming and New Mexico.

Dagbert has designed a new distribution-free technique (Dagbert and David [4]) which allows the computation of a normalization of both distributions and expansion of the normalization functions in Hermite polynomials. The use of suitable numerical analysis techniques allows the definition of a correction equation and confidence intervals which are also expressed in polynomial expansions and coincide with the standard lognormal regression in the lognormal case. Hence for practical purpose there is a technique to "correct" radiometric samples as shown by line 1 in Figure 2A. It can be refined as geological understanding of the different zones of a deposit progresses as shown in Figures 2B and 2C which are subsets of 2A. An academic problem which remains but seems only academic at present is how to go from a correction factor on 6" or 5' samples<sup>2</sup> to a correction factor for 5-ton cars or 20-ton trucks?

Again it should be pointed out that when considering this problem on a worldwide basis, drilling habits are very different

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<sup>1</sup> D.F.N. = delayed fission neutron.

<sup>2</sup> Feet (1 ft(') = 30.48 cm) and inches (1 in.(") = 2.54 cm) are used throughout this paper

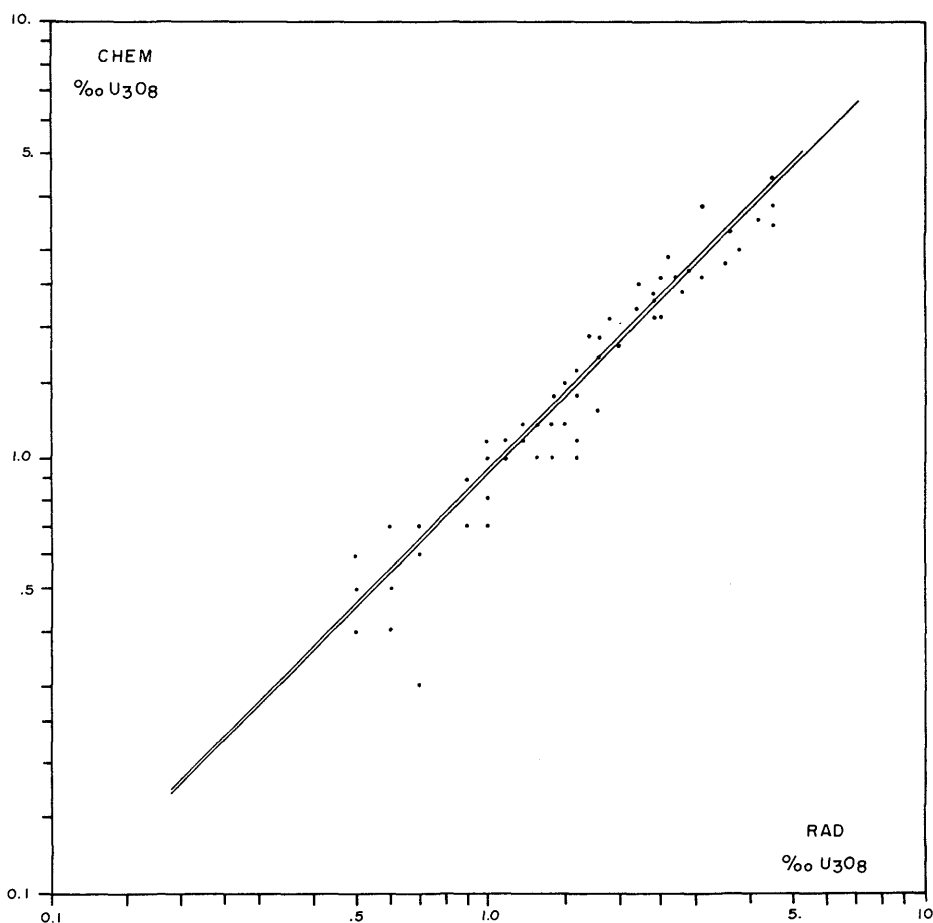


FIG.1. Correlation between radiometric and chemical values of the same drill-hole intervals at Urgeiriça. The upper line is the log-normal regression line.

in different parts of the world and one encounters mines with 95% of cored samples in Saskatchewan, 5% or less in Wyoming and an in-between proportion in Western Europe, hence its importance may vary from one mine to the other. In some cases, core recovery is poor and there is a preferential loss of mineralized particles. Then one should not rely on core values but rather on radiometric data. Once this first problem is settled, it is possible to have a data base build-up for each deposit. This data base may contain only a few hundred samples - sometimes less - or up to two millions. The time required to computerize data base is always grossly underestimated by people who are at their first experience. One should then be extremely

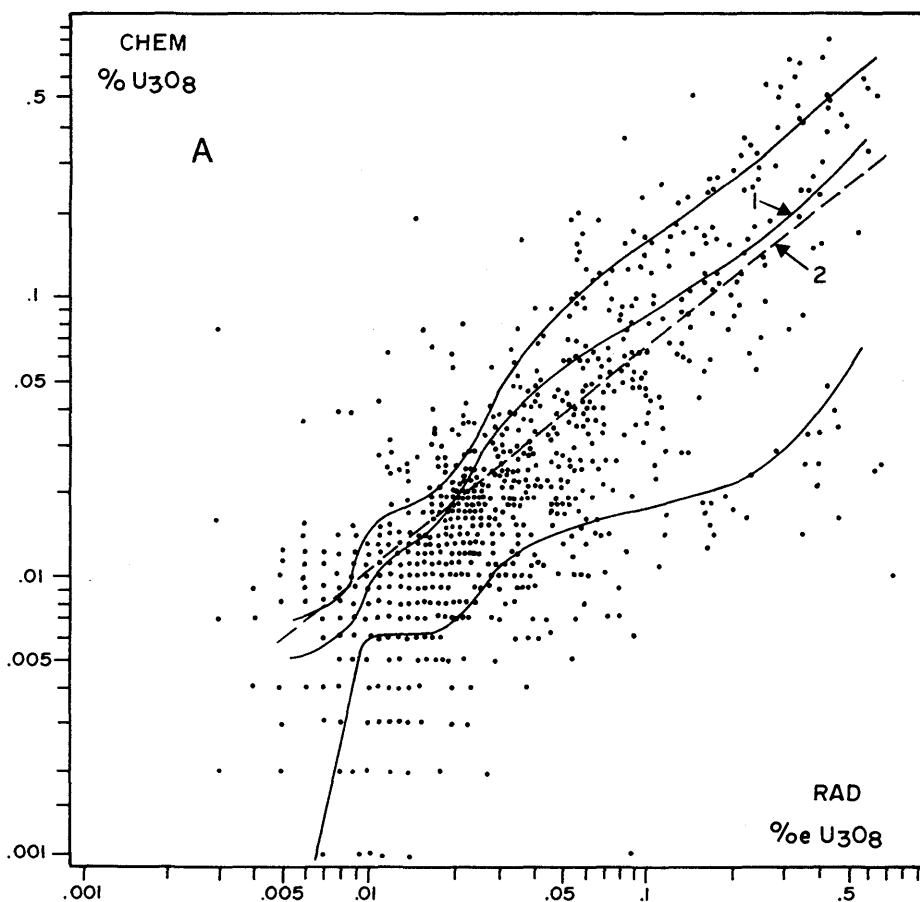


FIG.2. Correlation between radiometric and chemical values of the same drill-hole interval (1 ft) of an eolian deposit of Wyoming. In A, all the data available are put together. In B and C, only samples in two separate zones are shown. Line 1 is the regression line obtained by the distribution-free technique of Dagbert and David [4]. Line 2 is the log-normal regression line. Upper and lower lines are limits of a 84% frequency interval. (B and C on following pages).

careful in the organization of this data base, so that it does not cost \$500 computer time each time some information has to be extracted!

## 2.2 The definition of shape and tonnage of deposits

Early in the application of geostatistical techniques, the red pencil of a geologist outlining an ore zone has more than

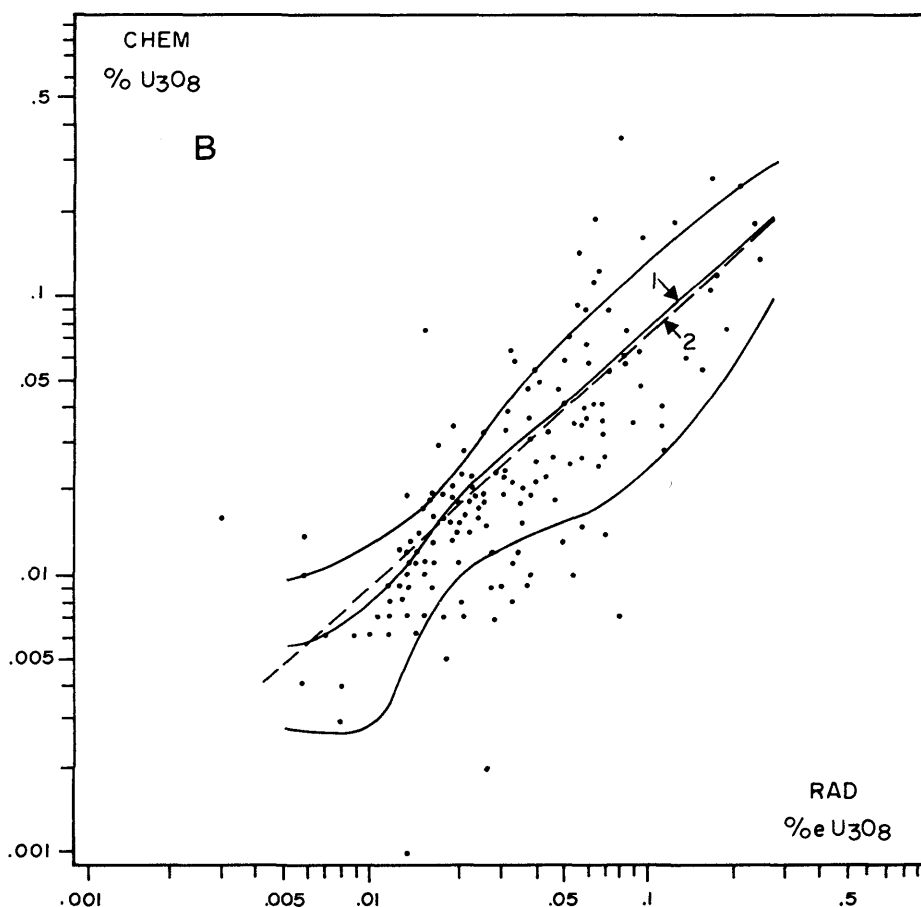


FIG. 2 B.

often been criticized. Statistics alone can certainly not replace geological experience and both geologist and statistician should accept to work together. Our point will be illustrated using the examples in both two and three dimensions. This problem was early called the geometric problem and formulae were given to roughly estimate the precision with which the area of a polygonal outline of a mineralization was known. We will successively review the case of trend deposits, vein-type deposits and finally 3D porphyry-type mineralization.

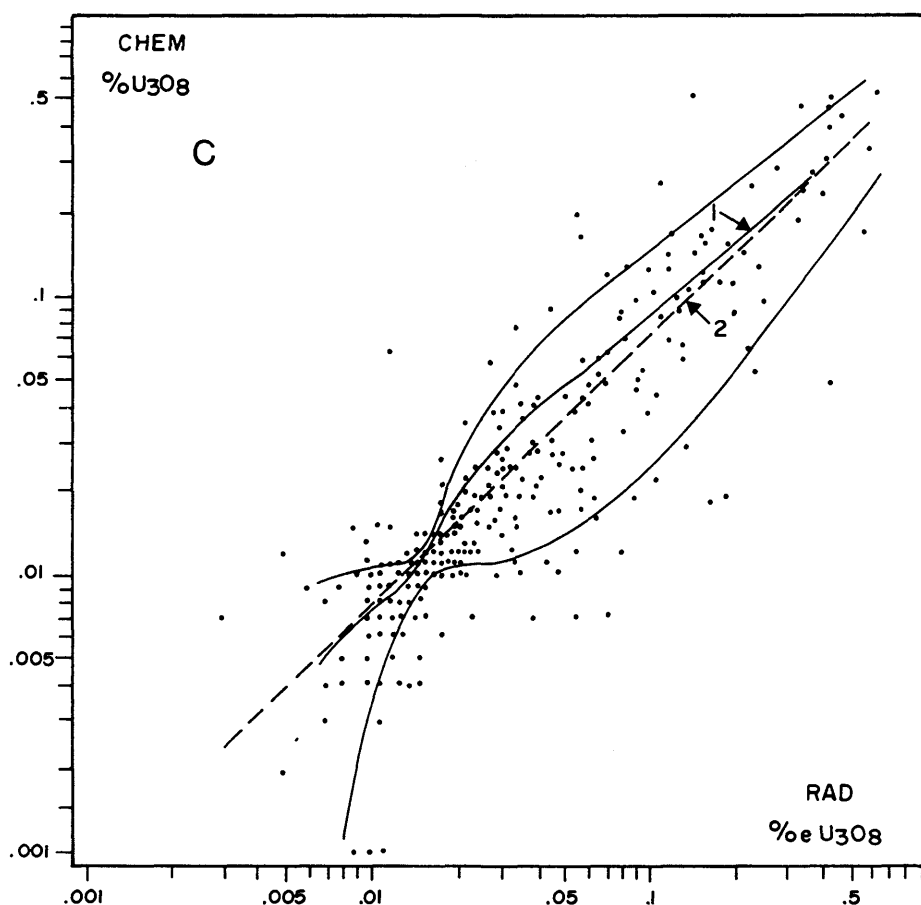


FIG.2 C.

### 2.2.1 Outlining a trend or roll-front deposit

It is well established that to obtain an estimation of the reserve of such a deposit, the total  $GT^3$  is the variable to work with, then one has to face a 2-D problem. This problem has already been well described by Matheron as early as 1962, by Sandefur and Grant [12] and more recently by Parker and Chavez [13]. To try and keep things simple the general problem is to put a contour around a mineralization where some intersections are defined as positive and others as

<sup>3</sup>  $GT$  = grade  $\times$  thickness.

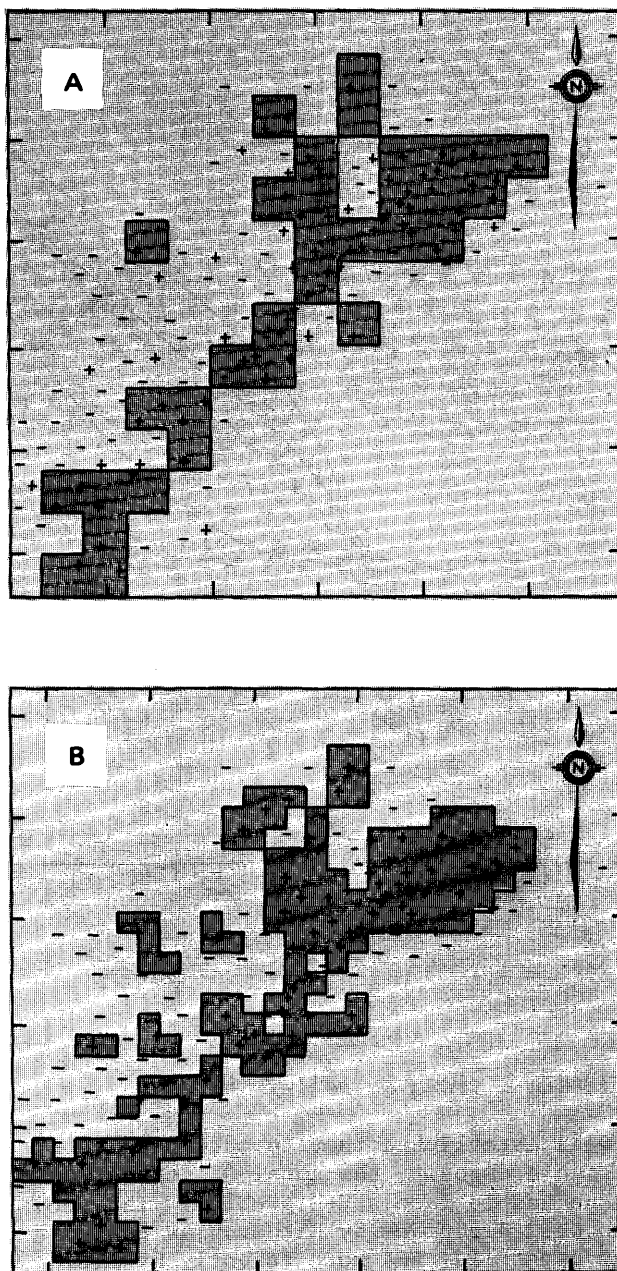


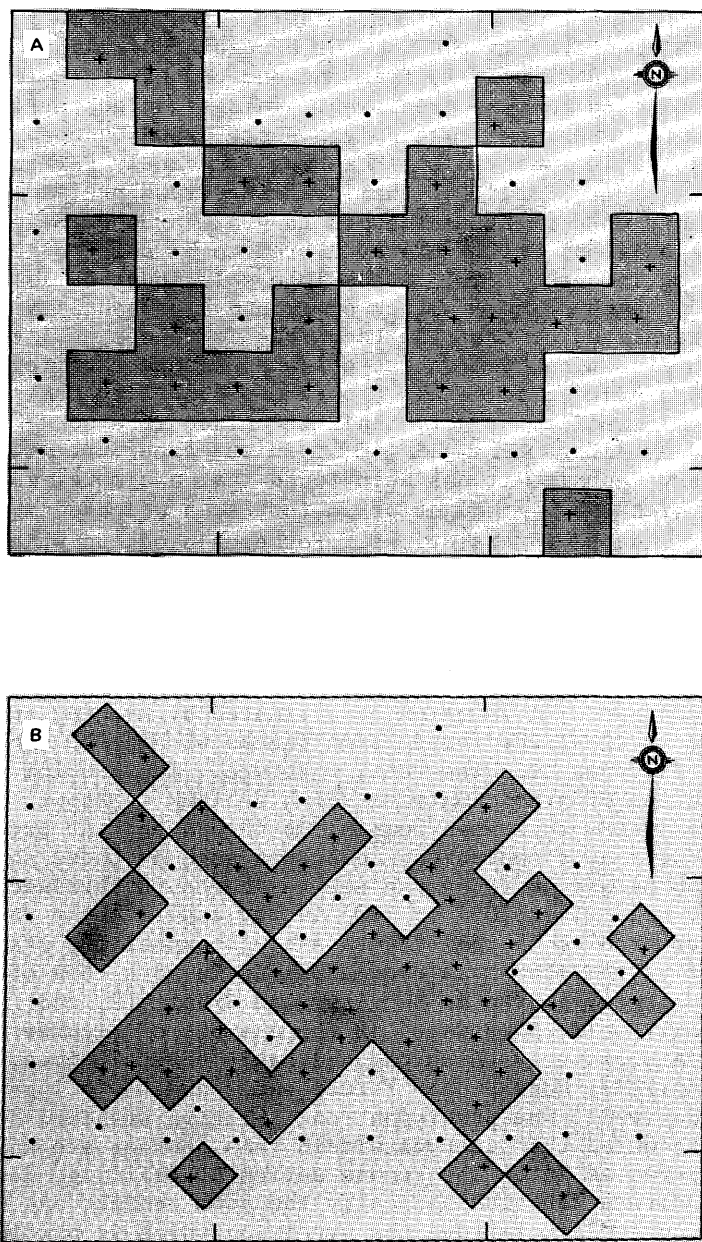
FIG.3. Estimation of the contour line of a flat-lying deposit with a 100-ft (A) and a 50-ft (B) block grid. + is a mineralized hole and - is a blank hole.



negative (not meeting a minimum grade for instance over a minimum thickness). In the case of a regular drilling grid, the polygonal rule is fine and one can use Matheron [14] formulae to compute precision. Things can go terribly wrong however when some huge polygons are drawn in open areas and are included in the reserves. Recently Parker and Chavez [13] pointed out that drilling crews prefer to leave plenty of room for antelopes to graze around the edge of the deposit and drill in proven area to be able to report interesting intersections! A possible solution then is to compute for each point or block of ground a probability that it is mineralized or not; the technique is to assign 1 to a "good" intersection, 0 to a bad one and compute a variogram and perform kriging to obtain a pseudo-probability that a block is mineralized or not. This technique is also described by Alfaro and Miguez [15]. In our experience with western sedimentary deposits it has been found that a simpler rule can be used which can be thought of in fact as a weighted average of "good" and bad intersections. For each block to which a status is to be applied, a window of influence is considered and the block is considered to be ore or waste depending on the number of positive and negative intersections encountered or the sign of the nearest intersection. The apparent arbitrariness of the process is in fact irrelevant as the defined contour will later on be estimated by kriging and according to the grade found, some blocks within the contour might be rejected from the total reserves. The problem hinges on the notion of precision with which a block is known. We shall re-discuss this when summing up the paper. At present let us accept as a convenience contours such as those shown on Figures 3A and 3B. One can see the contours obtained for different grid sizes, 50' or 100' in the case. About the particular problem of roll-fronts and more specially stacked ones, let us remember that over-drilling is more than often done, as already proven in Sandefur and Grant [12] and particularly well illustrated in the example of Figures 4A and 4B. On this property which changed hands several times, one can see drilling done by the former owner and the polygonal contour derived. The precision (one relative standard deviation) with which the reserves within this contour are known can be considered as 19%. The second series of drilling, doubling the expenditure revealed basically the same contour and  $U_3O_8$  content and did not improve the precision which is now 14%.

### *2.2.2 Working on vertical sections and recognizing the ore waste distribution problems*

Theoretically the problem is the same as above, except for a change in orientation. One should however be careful and make



*FIG.4. Polygonal estimation of the contour line of a flat-lying deposit with an initial 250 drilling grid (A), and after centring the grid (B). + is a mineralized hole and a dot (.) is a blank hole.*

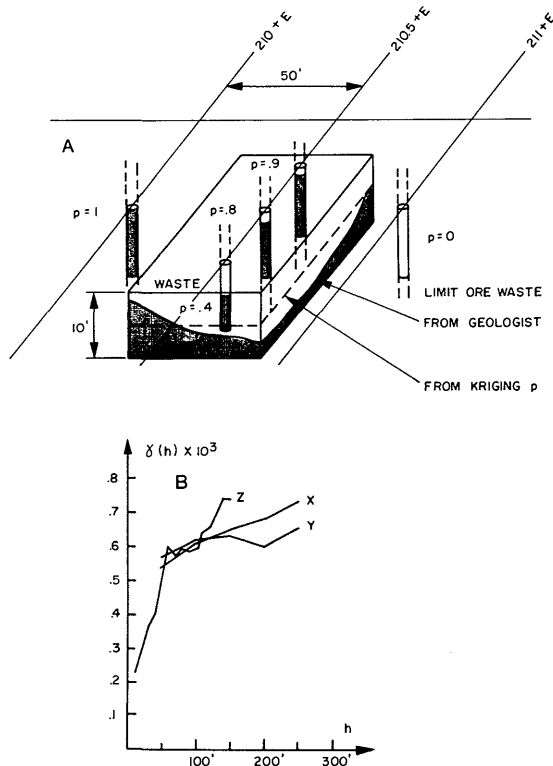


FIG.5. Estimation of the proportion of waste in a block by kriging the same proportion  $p$  in bench composites (A). The experimental variograms of  $p$  are shown on (B).

sure to consider several sections at a time to do the interpretation. Otherwise classical section methods and kriging can easily conflict. Careful examination normally shows kriging to be more realistic. Defining an indicator is also appropriate in this case, as well as in roll-front type deposits. It is a well-known fact that computer analysts trying to encompass ore deposits into regular boxes "matrices" often contradict a geologist's feelings and are said to dilute the ore, making ore lenses vanish on some occasions. The answer which was given earlier in porphyry type deposits (David *et al.* [16]) can be improved in sedimentary uranium deposits as the location of waste is usually fairly well known due to some continuity in the lateral extension of sand horizons. One can then keep the "box" model for the deposit, and for each - say 4' or 10' vertical intersection depending on the size of the mine, one can compute the proportion of ore and waste for each mining block. Examples are shown in Figures 5A and 5B, where one can

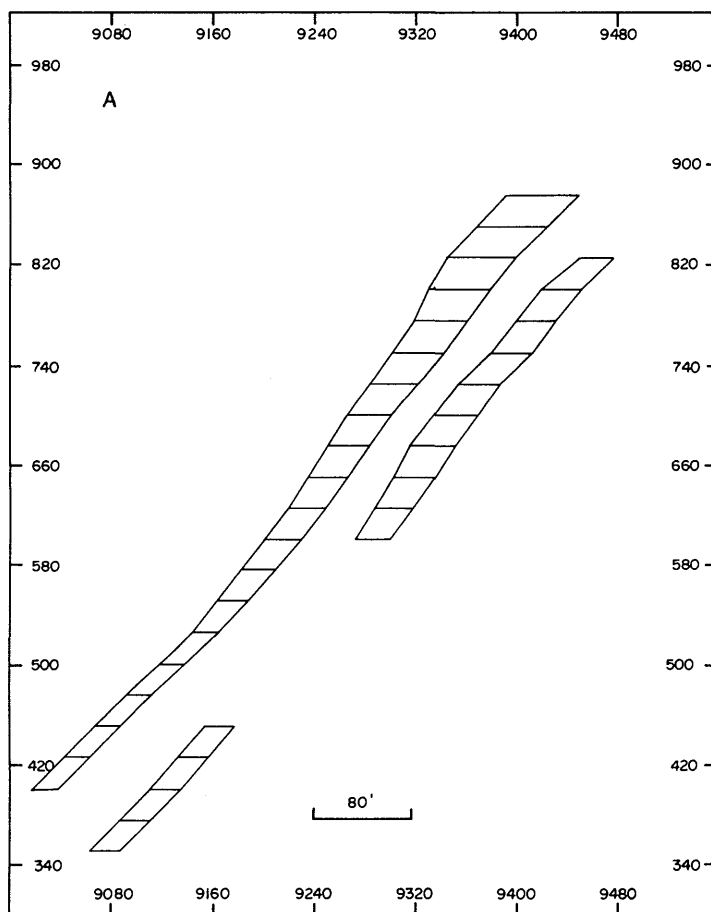


FIG. 6. Estimated contour of vein-type deposits. A is the estimated section of a gold deposit with the position of the hanging-walls and foot-walls derived by universal kriging. B is a longitudinal section of an uranium deposit showing isopach lines (m) also derived by universal kriging.

see a block overriding ore and waste, a variogram of proportion of ore and waste and as a feedback the best estimate of the proportion of ore and waste in the block.

In vein-type deposits, problems start with the definition of the data base; the number of drill holes is usually smaller but one has deviation problems, holes must be surveyed and one starts by re-computing the exact coordinates of each sample. One can choose to work with horizontal projection of intersections or simply record the point of entry and exit of the

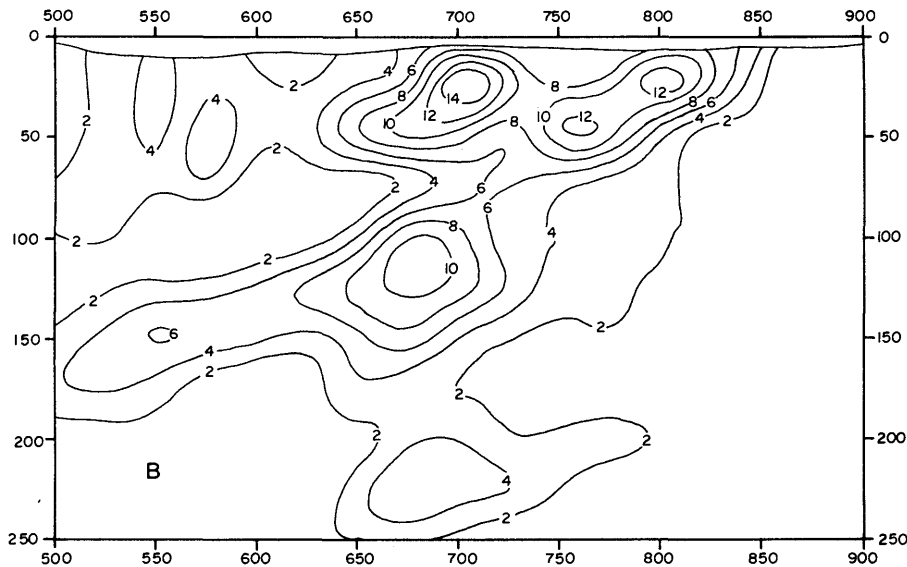


FIG. 6 B.

drill hole in the vein and produce by kriging the best estimate of the position of the footwall and hanging wall. This has been well described by Krige and Rendu [17] and they pointed out that kriging or universal kriging were ensuring that the footwall will not cross the hanging wall. Figures 6A and 6B show examples of cross sections obtained this way and isopach projected on longitudinal sections. A precision can of course be attached to the thickness of each block. One must also pay particular attention to the drilling diameter if holes are cored. Walls defined on EX or NQ core may not be the same! (David and Dagbert [18]).

### 2.2.3 Porphyry-type or bulky deposits

They present us with specific problems and all the techniques developed for porphyry copper deposits will apply. They have been covered for instance in David *et al.* [16]. Programs exist to help visualize the shape of the deposit in 3D; these should not be considered as simple gadgets as they are quickly produced and often as useful as a painfully constructed wooden or plastic model.

To conclude this section on geometrical problems, using the basic formulae which have been around for a long time, the remarks about the ore-waste distribution or so-called indicator

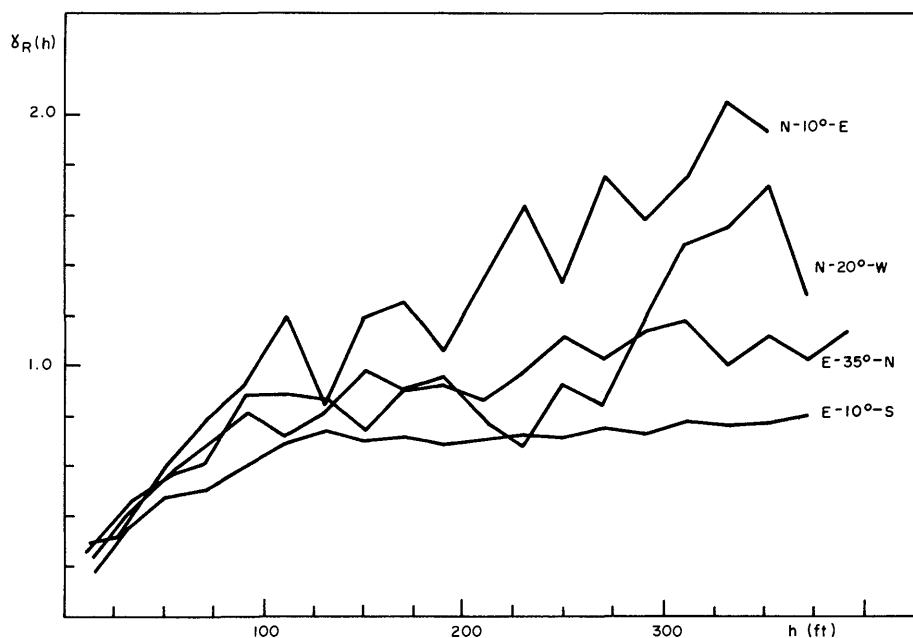


FIG.7. Experimental relative variograms of grade  $\times$  thickness (GT) along several horizontal directions in a Wyoming roll-front deposit.

function, computer graphics and remembering that contours are not definitive until grade estimations are made, it can be stated that an outline and confidence interval can be obtained for practically every uranium type of deposit. We have not covered the problem of the definition of a positive or negative intersection, it will be seen in the grade estimation section.

### 2.3 Grade estimation

Once again it is not our purpose to come back on the definition of kriging; among other documents where it has been well covered is the report of the "Advisory Group Meeting on Evaluation of Uranium Resources" of IAEA in Rome, 1976 and in Guarascio and Turchi [19]. We will simply recall that the basic principle of grade estimation is to obtain the best weighted average of all samples surrounding a particular block. The weighting is of course a function of the continuity of the mineralization.

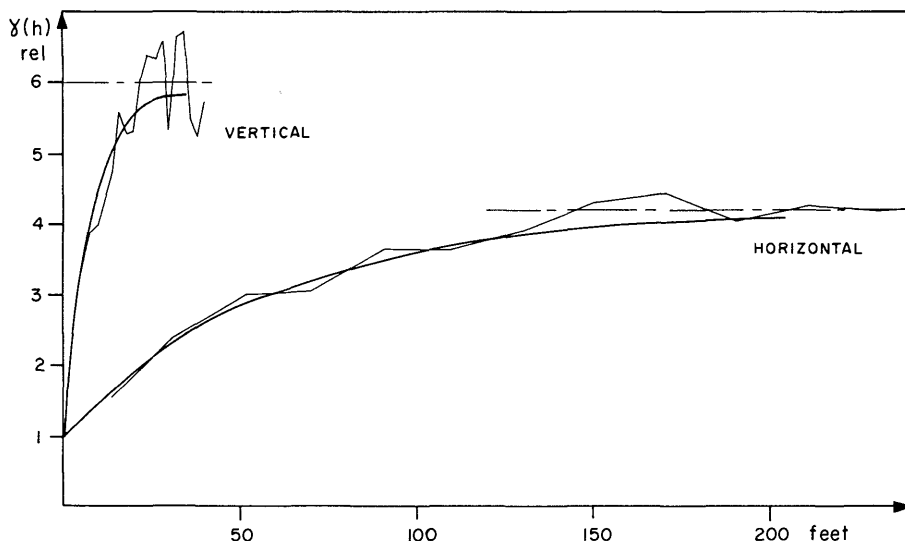


FIG.8. Experimental relative variograms of the  $eU_3O_8$  grade of 2-ft composites in a Wyoming roll-front deposit. The horizontal curve is the average of all the directional variograms in the horizontal plane. The smooth curves are spherical models fitted to the experimental curves.

### 2.3.1 Representing the continuity of the mineralization by means of a variogram

The way in which the continuity deteriorates with distance is well represented by a variogram (Guarascio [20]). We will use a series of figures to show how very different types of deposits do have variograms which do actually take geology into account. We will again consider 2 and 3 dimensional cases.

Figure 7 shows variograms obtained in a Wyoming roll-front deposit. GT variograms have often been shown (Sandefur and Grant, [12]; Knudsen and Kim, [21]). One can now state that they are almost always good which was a good surprise considering the intricacies of roll-front deposits. These variograms however only allow the estimation of 2-D blocks or total  $U_3O_8$  quantity, which makes sense due to the particular mining methods used in Wyoming or New Mexico's open pits. The total quantity of uranium in situ should eventually be recovered as mining is done virtually with a tea spoon! Techniques to obtain a good variogram have improved and the predictive sample re-use method recommended in Davis and David [22] or Knudsen and Kim [21] gives a good way to obtain "nice-looking variograms" although of course it is well-known that variograms are a very robust tool.

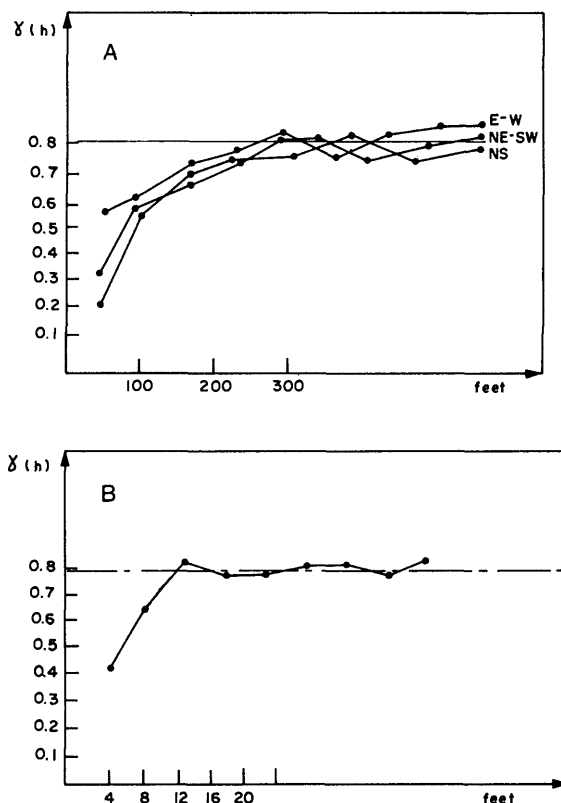


FIG.9. Experimental relative variograms of the  $U_3O_8$  grade of 4-ft composites in a New Mexico trend deposit along horizontal directions (A) and the vertical direction (B).

The previous variograms do not allow the estimation of regular blocks in three dimensions, which of course is a desirable feature. Obtaining a good variogram in three dimensions on a very flat deposit requires a lot of care and careful thinking rather than countless hours of computer time. Good and bad things may happen. Figure 8 shows the vertical and horizontal variograms of 2-ft composites in a Wyoming roll-front deposit, it is a textbook case! (Verly [23]).

Figure 9 shows equivalent variograms for 4-ft composites in a New Mexico trend deposit where one would have expected them to be nicer (Guertin [24]). One should be careful however to look at the scale on the vertical axis of Fig.8. The variograms shown are relative and sills of 6 mean that the accurate



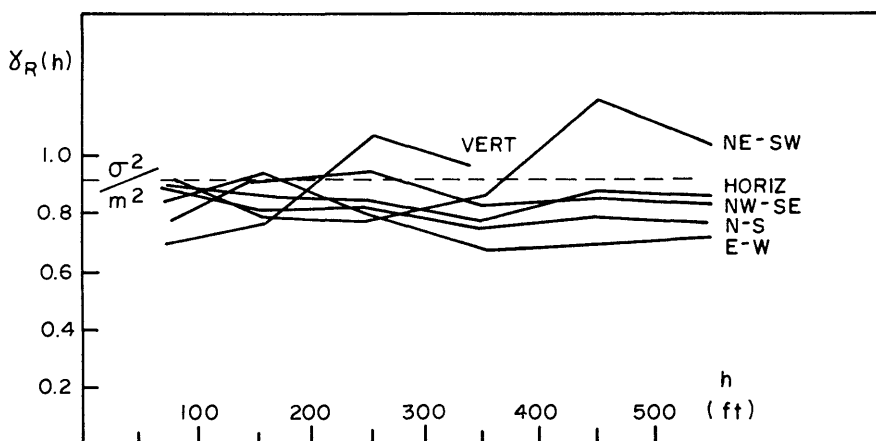


FIG.10. Experimental relative variograms of the  $eU_3O_8$  grade of 100-ft composites in another Wyoming deposit. *HORIZ* is the average horizontal curve.

estimation of the grade of a small block from surface drilling should be forgotten. Figure 10 is an honest account of a case where no horizontal structure could be found (Wyoming again). Deposits of origin other than sedimentary also exhibit good variograms as reported by Guarascio [25] in a volcanogenic deposit of Italy, once the proportional effect is taken into account. Figure 11 shows variograms from a vein-type deposit where continuity is not excellent but definitely sufficient to work out block estimates. Other deposits for which variograms have been reported include Imouraren in Niger (Parker *et al.* [26]), Ilimausaq in Greenland and Elliot Lake in Canada. Lucero-Michant [27] reported variograms in two argentinian deposits. Matheron [14] reported those of Mounana. Gagnon [28] computed those of Beaverlodge (Figure 12) and a rich Australian deposit showing a succession of rich and poor layers is certainly well described by the variogram of figure 13 from David [29].

As a conclusion to this part on continuity description one may say that one can always mathematically represent the continuity or lack of continuity of a uraniferous mineralization. It may sometimes require the usual precautions necessary in variogram computations, which are now normal procedures in modern geostatistical calculations. They are described for instance in David *et al.* [16] ("cleaning" a variogram) or in Dagbert and David [30] ("regressive" variograms). One should always test for the existence of a proportional effect

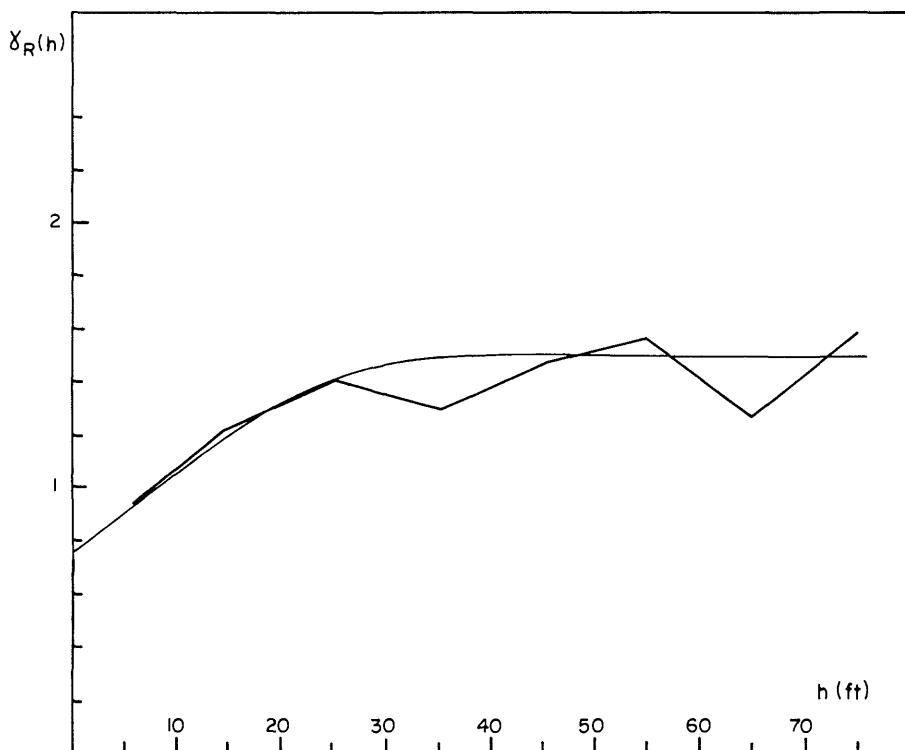


FIG. 11. Experimental relative variogram of GT along the horizontal direction in a subvertical vein-type deposit. The smooth curve is a spherical model fitted to the experimental curve.

which usually shows a perfect linear relationship between mean and standard deviation in different areas of the deposit (see Figure 14 after Verly [23]). The relationship may not be linear (Dagbert and David [30]).

### 2.3.2 Calculating block grades

Continuing in our review of problem solving now that a data base has been established, the disequilibrium problem has been dealt with and the geometry of the deposit has been outlined, grade estimates or estimates of uranium quantity have to be produced. Once again we will assume that our reader is familiar with the existence of the kriging technique which has been explained on numerous occasions (Guarascio [20]; David [29]; Journel and Huijbregts [31]) and we will concentrate on application problems when things do not happen the way they are expected to.

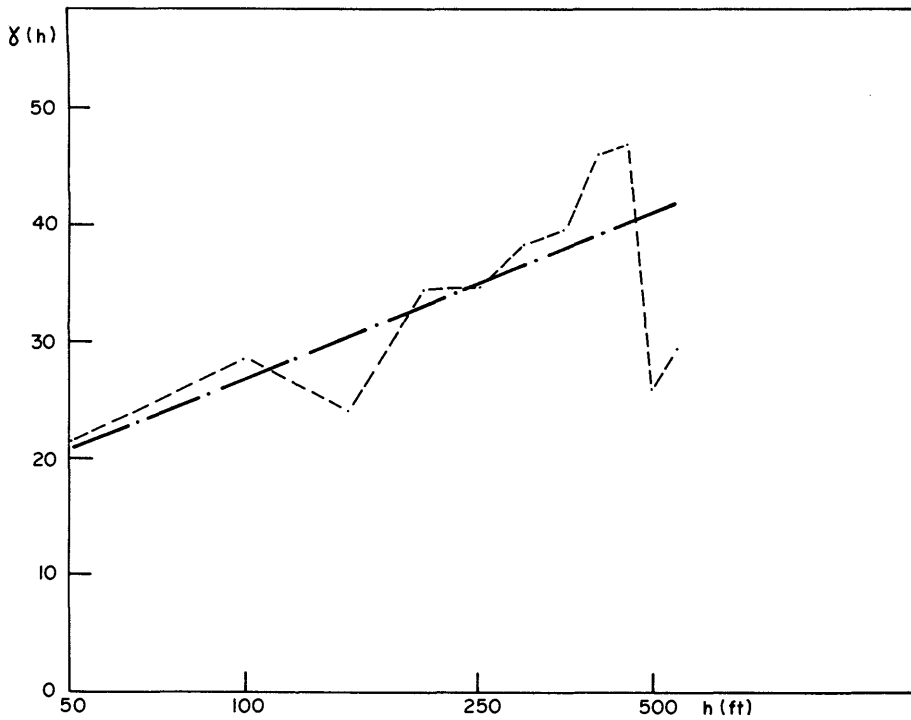


FIG.12. Experimental variogram of GT along the horizontal direction in a Beaverlodge deposit. The horizontal scale is logarithmic and the straight line is a de Wijsian model fitted to the experimental curve (From Gagnon [28]).

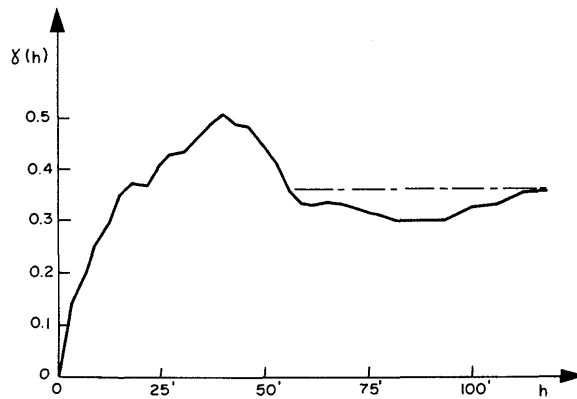


FIG.13. Experimental vertical variograms computed for increasing samples lengths in an Australian uranium deposit (From David [29]).

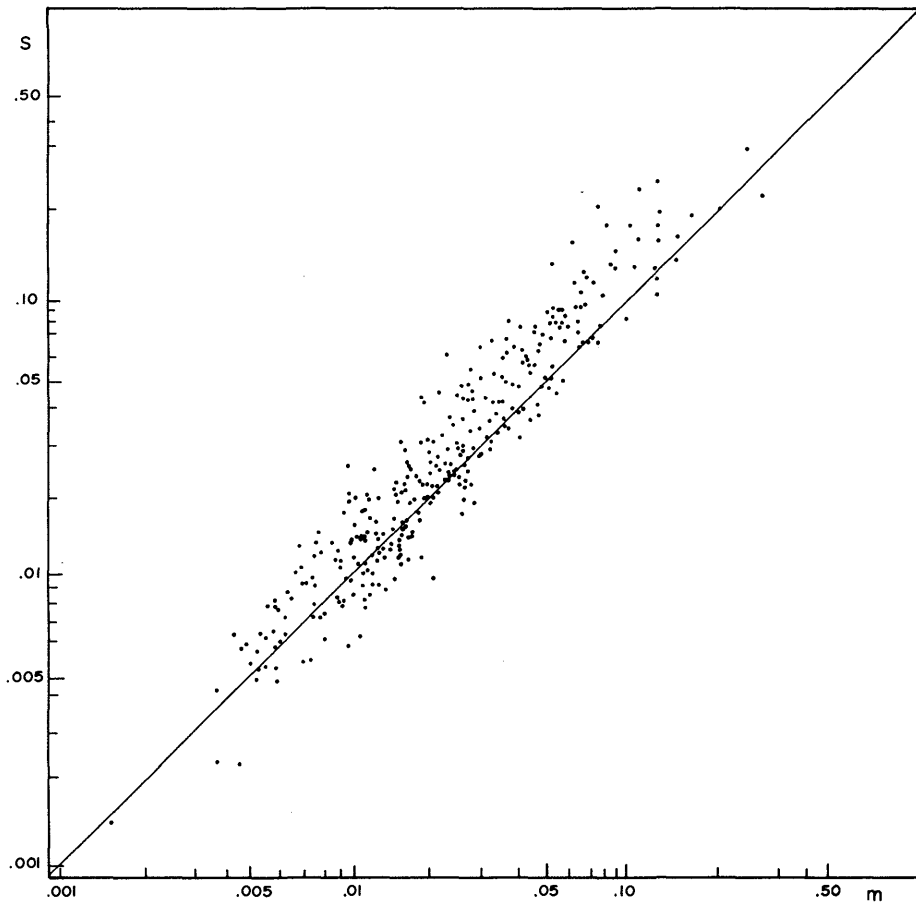


FIG.14. Correlation diagram of the mean ( $m$ ) and standard deviation ( $S$ ) of four successive 6-in. samples in 2-ft composites in a Wyoming roll-front deposit. Values are expressed in %  $eU_3O_8$ .

#### Working on GT values

When working in 2-D a problem is to define the "good" intersections. This is illustrated by Parker and Chavez [13], discussed by Sabourin [32] and Journel [33] or Metz [34] where a related problem occurs in phosphate. Most solutions which might be very different in theory give practically equivalent results and anyway one finds that company's geologists usually override any automatic program they are

proposed, to edit each intersection according to their own criterion. One should certainly accept the fact that they can see geological details which were not fed in the computer but one must also admit that there are frequent inconsistencies which do not help in the definition of variograms. Once this definition of intersections has been made then a variogram is easily computed whether samples are evenly distributed or not. Kriging is then done within the contour outlined in the previous section. Depending on the contrast between high grade and low grade samples, the seriousness of the unavoidable overestimation due to the exclusion of "non-ore" intersections from the data base may vary. This is always the same well known fact that rich samples are surrounded in average by blocks which are less rich. Hence one can be sure that blocks estimated with samples which are all above, say a 0.20%  $U_3O_8$  cut-off, will be overestimated. The problem is well defined but the remedy is not always simple if one wants to stay in two dimensions. One sure thing is that universal kriging, disjunctive kriging or lognormal kriging are not helping much despite their sophistication. The general idea is that some GT less than cut-off must be included in the data base for kriging the inside of the contour. This of course is what will cause some of the blocks within the contour to drop below the cut-off and also why we said earlier that this contour was rather a convenience than a rigid definition. A good discussion of this problem is given in Verly [23] who tested which samples should be included in the kriging data base for different parts of a roll front, using the predictive sample re-use method. In some cases, it is found that one can simply set the low GT values at zero, and let the intersection thickness at 6'. This however cannot be generalized to all deposits and should be tested for each one. This is one of the reasons why a minimum understanding of geostatistics must be acquired before using computer packages, however simple they may seem to use.

#### Working in three dimensions

This is theoretically the best method, it is not always feasible but one has seen in Figures 8 and 9 that good variograms can be obtained, thus allowing the complete 3-D estimation. Then again the definition of ore and waste block is made after the results of kriging, rather than by an *a priori* definition based on intersections only. There are still more questions to be answered which will keep changing the reserves, like the internal waste distribution problem and optimum definition of hanging wall and footwall or open pit.

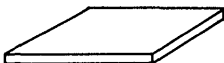
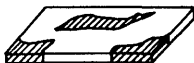
<u>GEOLOGY</u>				<u>MINING</u>		
Reserves are calculated on blocks 100' X 100' X 4'				Selection is done on smaller blocks 10' X 10' X 4'		
						
				internal waste is identified and deleted		
BEFORE MINING STARTS ONE CAN COMPUTE WITH IN EACH LARGE BLOCK THE PROPORTION OF ORE AND WASTE AND OBTAIN BETTER ESTIMATION OF RESERVES.						
Level	Tons	Grade	pounds U <sub>3</sub> O <sub>8</sub>	Tons	Grade	pounds U <sub>3</sub> O <sub>8</sub>
.	.	.	.	.	.	.
.	.	.	.	.	.	.
11	197,333.	0.135	533,194.	138,236.	0.182	504,287.
12	140,000.	0.139	390,000.	100,943	0.184	371,672.
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RESERVES BY KRIGING ONLY				RESERVES BY KRIGING PLUS O/W PROPORTION. K.GUERTIN, M.DAVID		

FIG.15. Computation of the internal waste in 100 ft X 100 ft X 4 ft blocks in a New Mexico trend deposit.

Using both radiometric and chemical data in block estimation

In deposits where both radiometric and chemical grade measurements are available in a similar proportion, Guarascio [25] suggested to use both of them simultaneously in the estimation of blocks. It means that the estimate is a linear combination of all values of both types available around the block. This method is called co-kriging. However comparative studies show that in most of the cases there is no need to use such a sophisticated method. Simpler substitutes like correction of radiometric data in places

where chemical values are lacking and kriging with uncertain data do equally well in terms of estimation variance and provide comparable estimates. A possible danger in blindly using co-kriging lies in the fact that if all chemical samples available are far from the block to estimate they may be assigned a significant weight (since the sum of these weights is forced to be equal to 1) whereas radiometric data closer to the block have less influence.

#### The internal waste problem

This is illustrated in Figure 15 and discussed at length with several solutions offered in Dagbert and David [35]. In uranium trend deposits the experience is often that in fact more tons of low grade material than expected are encountered. One should speak of internal ore in waste blocks! It should be accepted that one cannot at the same time predict how much ore there is and where it is. The several solutions offered include correction factors based on experience, available only if mining is already in progress, lognormal shortcut (David, [29]) may be not entirely theoretically correct, - it is a shortcut - but proven by experience like at Pima or Similkameen (Raymond, [36]), disjunctive kriging, expensive and based on hypotheses which are highly difficult to prove and which usually give results very close to lognormal shortcut, kriging of a limited number of small blocks within large blocks to establish a distribution of errors or actual distribution of small mining blocks within large blocks. Theoretical discussions about this problem will abound at the APCOM meeting in Tucson (1979) but no check against production of actual pounds of uranium has been offered yet. Available checks are either on copper, or if in uranium on re-predicting samples only. It seems however that on a monthly basis all methods should check and predictions match production.

### 3. ESTABLISHING RESERVES CATEGORIES AND DRILLING FOR MORE CERTAINTY

This is another problem which in theory has long been settled. We feel that in practice it is not because the concept of certainty is vague and varies from one individual or company to the other. Let us start with some positive points.

#### 3.1 Establishing the precision with which a total $U_3O_8$ content is known

This has been treated for instance in David [37] or David [29] and one can certainly state a confidence interval

given any drilling pattern. The technique usually involves kriging an entire contour at once and special numerical techniques have been developed to handle the huge matrices it implies. Question is, does this answer make sense? Is it sufficient for the operator? If as it often happens, large quantities of the dollar value of the deposit are concentrated within small blocks then one may want to better know these particular blocks, where the relative risk may be the same than everywhere else but where the dollar value of a 10% error can be tremendous.

### 3.2 Establishing the precision on small blocks and classifying reserves

This is also a simple problem. Kriging will return a precision, absolute or relative for each estimated block and then one may tally blocks which are known with a given precision or better. This way one can usually define a contour which matches a geologist's definition of proven or probable. One should however rely on the computer for consistency only rather than for real decision. One can also by tallying blocks known up to a given precision define "nested" deposits which are better and better known and recompute for each "interior" the precision with which the entire deposit is known. This is a lengthy iterative procedure which may not be needed. Again, before going into sophisticated calculations and using *a priori* ideas one should carefully review with a company what its particular needs and objectives are.

For a given deposit, the needs will vary depending on corporate policies or political situations! It may also be a function of the strong habits and requirements of a mine planning department which wants "precise" answer at any cost!

## 4. CONCLUSION

As a conclusion to this review paper, we will insist that the theories developed twenty years ago have now evolved into a workable tool for almost any uranium deposit situation, and the theory is flexible enough to accommodate any new type of mineralization which can be encountered. The literature on the subject, although sometimes difficult to gather and recent master's thesis by Guertin [24] and Verly [23] provide detailed step by step studies of western U.S. uranium deposits which can be considered as the present states of the art. Gathering all these information and the interesting examples regularly provided by Guarascio, Parker, Journal, or Sandefur



one may now not be far from having a handbook of uranium deposit estimation for the eighties. If one is sometimes frightened by the maths really involved in the background, let us at least remember the magnitude of the challenge.

### ACKNOWLEDGEMENTS

The authors are grateful to all the geologists and mining engineers who help develop geostatistical methods for uranium deposits estimation by their questions and criticisms. Special thanks are due to K. Guertin, G. Verly, M. Davis, D. François-Bongarçon and J.M. Belisle who contributed to finding some of the results presented in this paper.

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## DISCUSSION

B.S.I. MARENGWA (*Chairman*): Why speak of "geostatistics" when all the method involves is the application of statistics to geological work?

M.DAVID: Geostatistics is not merely statistics; the prefix "geo" is the most important part of the word. Serious mistakes may occur if the geological aspect of geostatistics is forgotten, as has sometimes occurred in the past.

V. ZIEGLER: In response to the Chairman's question I might point out that it is the regionalization of variables which distinguishes geostatistics from statistics in general. This was first demonstrated in South Africa (schools of Krige, de Wijs, Sichel etc.).

B.S.I. MARENGWA (*Chairman*): I would also like to ask Mr. David what he would do in case of a total lack of correlation between chemical analyses and radiometric surveys?

M. DAVID: Correction factors (regression lines) have been developed for any distribution of radiometric and chemical values. But of course if there is almost no correlation one should not expect one's predictions to be much good!

J.A. PATTERSON (*General Chairman*): Since the estimation of potential resources appears to present considerable problems, the development of applications of geostatistics seems to me to be desirable.



# **A GUIDE TO THE LOWER LIMIT – COMBINATION OF SIZE AND GRADE – OF DEPOSITS OF INTEREST FOR URANIUM RESOURCES**

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## **Abstract**

**A GUIDE TO THE LOWER LIMIT – COMBINATION OF SIZE AND GRADE –  
OF DEPOSITS OF INTEREST FOR URANIUM RESOURCES.**

An attempt has been made to establish a method of classifying in broad terms low-tonnage/low-grade uranium resources in terms of size, recovery grade and gross sales revenue at various price levels. To arrive at a basis for discussion, the following assumptions are made: (i) An overall average  $U_3O_8$  recovery grade of less than 50 g/t will never constitute a viable proposition; (ii) That no deposit, or cluster of deposits, containing less than 500 000 t ore will constitute a viable deposit; and (iii) Revenue requirements would vary according to the needs of the producer. It is, however, suggested that a gross sales revenue of US \$50 000 000 over the life of the mine (10 years) is regarded as an absolute minimum requirement for viability. From these data a family of curves has been constructed using a tonnage/grade combination which gives a gross revenue of US \$50 000 000 for various price categories in terms of \$/kg U. These curves may then be used to categorize deposits and to ascertain whether they are likely to constitute a viable proposition within the constraints listed above.

## **INTRODUCTION**

In attempting to arrive at a minimum grade/tonnage combination for what could constitute a viable proposition, many imponderables exist and it is unlikely that it will ever be possible to present a set of guidelines that will be universally acceptable or applicable. Nevertheless, in our efforts to assess national and global uranium resources, it is felt that an attempt should be made to arrive at a consensus regarding what, in very broad terms, constitutes the size/grade/gross sales revenue combination necessary to define the minimum requirements of a viable proposition, which could be put into a global inventory. The thoughts presented here are the first tentative steps in this direction and are presented with the hope that they may stimulate discussion which will ultimately lead to an internationally acceptable set of standards.

## 1. BASIC ASSUMPTIONS

In establishing the minimum requirements for a viable proposition, certain basic assumptions, which would apply up to the year 2025, have to be made. It is suggested that the following restraints could apply to deposits containing less than  $30 \times 10^6$  t ore with a  $U_3O_8$  grade less than 2 kg/t. They could apply to open-cast or shallow low-cost underground mining operations. However, as the establishment of the lower limit of the tonnage/ grade combination is being considered, it is suggested that a  $U_3O_8$  grade of less than 1 kg/t and a reserve of less than  $10 \times 10^6$  t should define the low-grade/low-tonnage package.

### (a) Recovery grade

It is assumed that an overall average  $U_3O_8$  recovery grade of less than 50 g/t will never constitute a viable proposition. Where potentially valuable by-products occur with the uranium, it is assumed that such a deposit will never constitute a viable proposition where the value of the total minerals recovered is less than the gross sales revenue on 50 g/t  $U_3O_8$ .

### (b) Minimum size

It is considered that a deposit containing less than 500 000 t ore is unlikely ever to constitute a viable proposition on its own. A number of smaller deposits may, however, constitute a viable unit when considered collectively. It is convenient, in the latter instance, to regard each small deposit as an increment to a total ore reserve which requires a corresponding increment in investment for the unit as a whole. Thus, the same set of parameters would apply.

### (c) Gross sales revenue

The minimum gross sales revenue demanded will vary according to a multiplicity of widely divergent requirements which will be determined by the available infrastructure, as well as by logistical, metallurgical, geological and political restraints, and by company or national policies. Because the question of profitability is not explicitly taken into account in this treatment, the distinction between cost and price is therefore not relevant.

To arrive at an acceptable norm, it is quite impossible to take all the above parameters into consideration. It is therefore considered that the best basis to work on would be a minimum gross revenue accruing over a given period of time. A figure for the gross revenue is naturally wide open to debate and could vary from say US \$50 000 000 to anything up to US \$1 000 000 000 over a life of 10 years, for example, in terms of 1978 US dollars. To make it possible for the

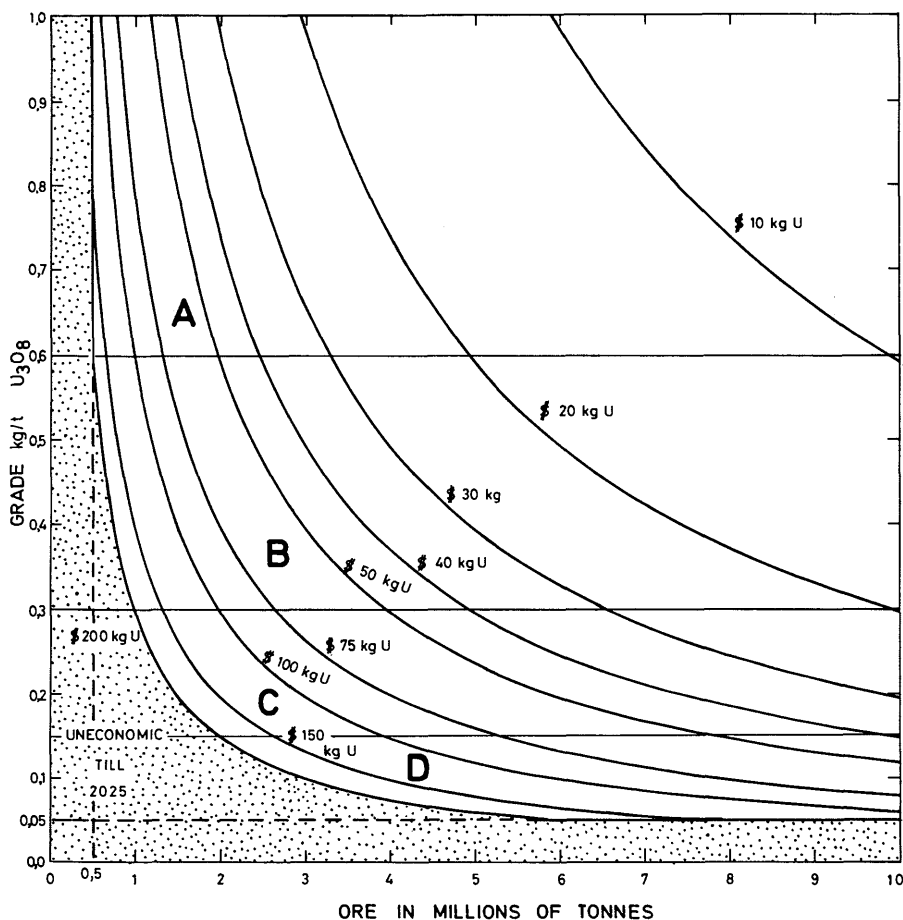


FIG.1. Family of curves showing gross revenue of US  $\$50 \times 10^6$  over a life-of-mine for deposits containing up to  $10 \times 10^6$  t ore and less than 1 kg/t  $U_3O_8$ .

smaller isolated deposits to be incorporated into an international inventory, it is suggested that a figure somewhere in the vicinity of the lower limit of US \$50 000 000 be adopted. This assumes that over a mine life of 10 years this figure would be the minimum requirement to bring a small operation into production and redeem the capital.

For example, at current prices this would mean that a proposition containing  $2 \times 10^6$  t ore at an average  $U_3O_8$  recovery grade of 0.4 kg/t would break even at the current price of uranium of about US \$75/kg U. This indicates that the basic assumption demanding at least US \$50 000 000 gross revenue is of the right order.

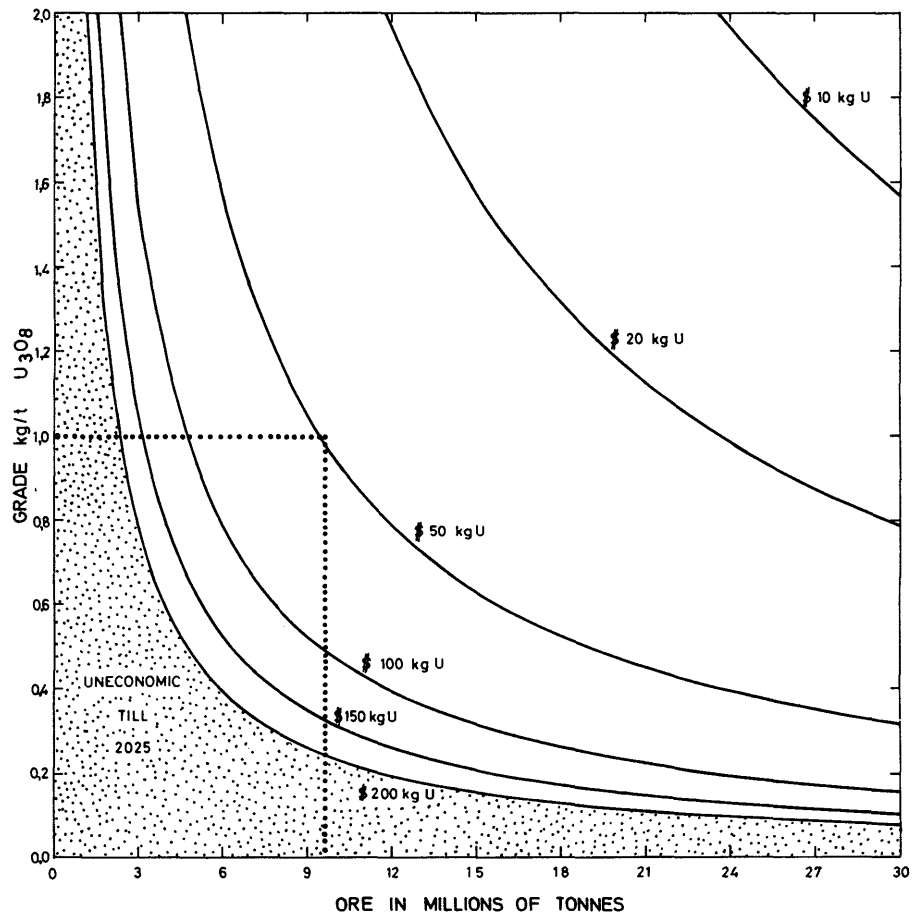


FIG.2. Family of curves showing gross revenue of US  $\$400 \times 10^6$  over a life-of-mine for deposits containing up to  $30 \times 10^6$  t ore and less than 2 kg/t U<sub>3</sub>O<sub>8</sub>.

#### (d) Price of uranium

It is assumed that up to the year 2025 the price of uranium will not exceed US \$200/kg.

## 2. INTERPRETATION OF CURVES

Referring therefore to Fig.1, it can be deduced that if the above-mentioned constraints are accepted and a US \$50 000 000 gross revenue regarded as the



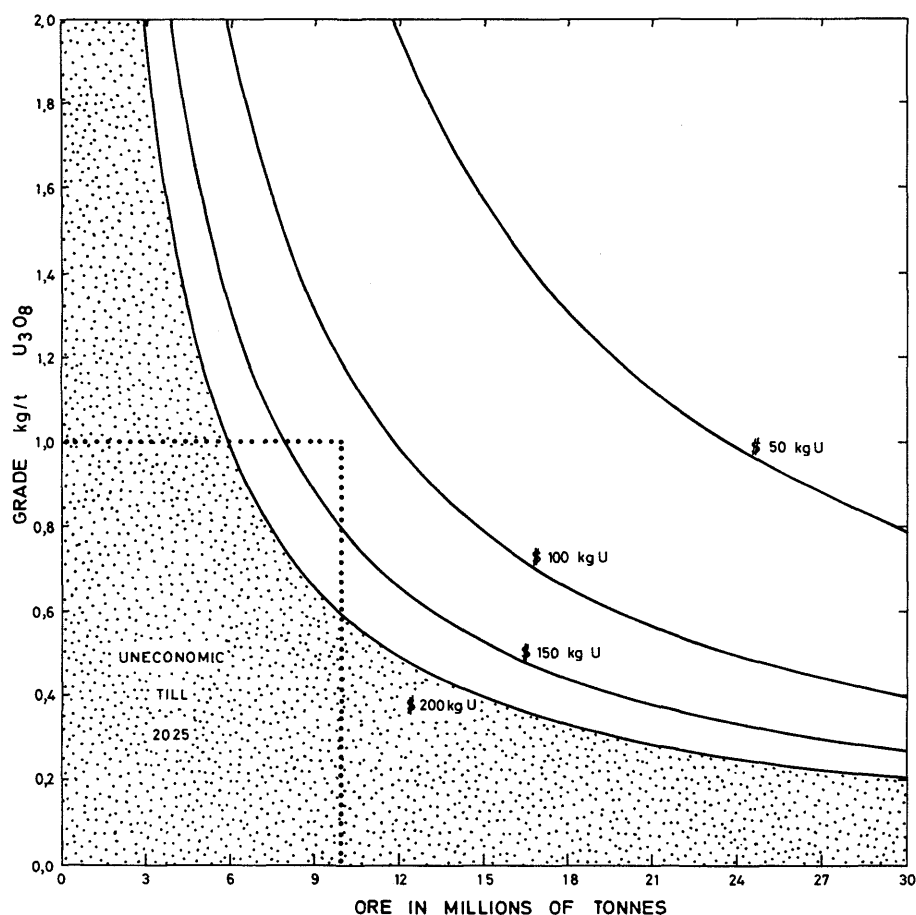


FIG.3. Family of curves showing gross revenue of US \$1000  $\times 10^6$  over a life-of-mine for deposits containing up to  $30 \times 10^6$  t ore and less than 2 kg/  $U_3O_8$ .

minimum figure, then it is possible to construct a family of curves, using tonnage/grade combination which would give this revenue at various price categories in terms of \$/kg U. At an upper limit of US \$200/kg U, for example, and with the constraints mentioned above, no deposit lying within the stippled area would be included in the international resource inventory up to the year 2025.

Figures 2 and 3 illustrate the effect of higher levels of the gross-revenue constraints of US \$400 000 000 and US \$1 000 000 000 respectively. From Fig.2 it will be seen that to meet a gross-revenue requirement of US \$400 000 000 within the 10 000 000 t reserve, and a  $U_3O_8$  grade up to 1 kg/t  $U_3O_8$ , the constraints are considerably higher. The lower  $U_3O_8$  grade limit is now about

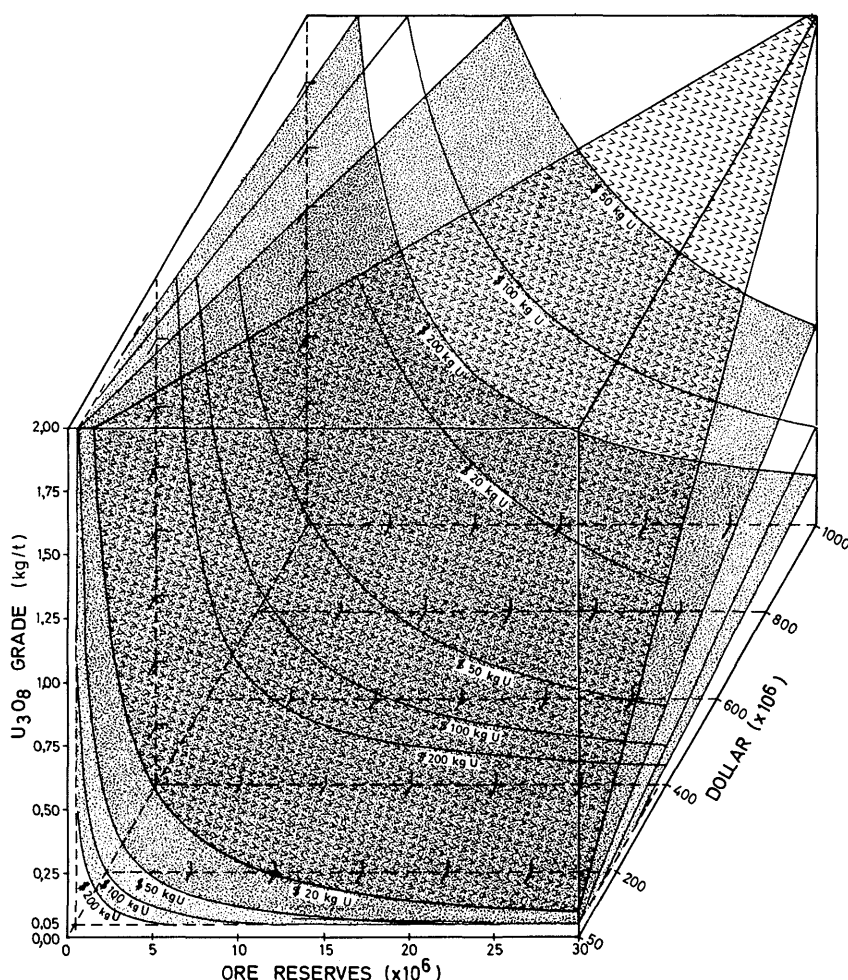


FIG.4. Family of curves showing gross revenue of US  $\$50 \times 10^6$  to  $1000 \times 10^6$  over a life-of-mine for deposits containing up to  $30 \times 10^6$  t ore and less than 2 kg/t  $U_3O_8$ .

0.2 kg/t, and the minimum tonnage requirement about  $3 \times 10^6$  t. Figure 3 illustrates that for a gross revenue of US \$1 000 000 000 and given a  $U_3O_8$  grade up to 1 kg/t  $U_3O_8$  and maximum tonnage of  $10 \times 10^6$  t, the constraints are still higher. The lower-grade limit is about 0.6 kg/t  $U_3O_8$  and minimum tonnage requirement is  $6 \times 10^6$  t. If higher grades and lower tonnages are considered, the constraints obviously change as illustrated on the graphs. Figure 4 is an onion-peel diagram which attempts to put together Figs 1–3 in a 3-dimensional representation of the various constraints and their combinations, and illustrates how reserve and grade demands increase with the escalation of gross-revenue requirements.

### 3 RECOMMENDATIONS

Should the concept presented above meet with the general approval of the Working Group, it is suggested that the proposals presented here may serve as a basis for establishing an internationally acceptable norm for categorizing uranium resources which contain less than 1 kg  $U_3O_8$ /t ore and with a reserve of up to  $10 \times 10^6$  t.

Assuming that a US \$50 000 000 gross-revenue requirement is acceptable, it is then suggested that the deposits could be categorized as illustrated in Fig. 1, and that within each category the price requirement could be stated. For example, a deposit containing  $2 \times 10^6$  t ore and with a  $U_3O_8$  grade of 0.5 kg/t would be classified as B/50/75.

### ACKNOWLEDGEMENTS

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### DISCUSSION

M. V. HANSEN: What sparked off the idea of performing this experiment?

P. D. TOENS: In my opinion it is becoming necessary to attempt to define, in broad terms, the low-grade/low-tonnage package for the Red Book and other studies in order to provide guidelines on what should and what should not be included in a global inventory. It is obvious to those of us involved in these studies that there are some deposits which are not, but should be, included in resource estimates; conversely, some which are included should not be.

M. V. HANSEN: Have you had any opportunity of testing this theory in a real case?

P. D. TOENS: Apart from carrying out a number of theoretical exercises it has not been possible to test this concept. It is not particularly important what figures are used to define the lower limits. What is important, however, is that these limits be defined so that uniformity may be achieved in global resource estimates when dealing with the low-grade/low-tonnage package, which collectively could assume considerable significance.

J. J. SCHANZ: Are your figures expressed in terms of current or constant dollars?

P. D. TOENS: We use 1978 US dollars.

J. J. SCHANZ: What price change limits do you assume for the future?

P. D. TOENS: We work on the assumption that the price of uranium will not exceed US \$200/kg U.

J. J. SCHANZ: Do you assume that there will be technological changes in the foreseeable future?

P. D. TOENS: Yes, I think the technology will have to improve if the lower limit of my model is to be realistic.

D. M. TAYLOR: Have you studied similar families of curves using cost instead of price, and gross expenditure instead of gross revenue?

P. D. TOENS: No, we have not. This is something which the Joint NEA/IAEA Steering Group could look into if it is decided to adopt a classification system. However, in my model cost of production and gross expenditure over a given period are very closely linked and could mean the same thing.

J.A. PATTERSON (*General Chairman*): What is the significance of the lettered categories A, B, C and D shown in Fig. 1?

P. D. TOENS: They are used for dividing the low-grade package into various arbitrary grade categories.

J.A. PATTERSON (*General Chairman*): The use of these grade groups implies that there would be no consideration of minimum grades – or costs – when classifying a deposit. The diagram thus represents only a preliminary or partial step towards the evaluation or classification of a deposit. It would have been more useful if it had shown the tonnage and average grade calculated after application of a minimum thickness and grade cut-off.

# ESTADISTICA Y GEOESTADISTICA. KRIGISMO Y UTILIZACION DE LAS FUNCIONES HEMIVARIOGRAMICAS EN LA INVESTIGACION ESTRUCTURAL DE LA YACIMIENTOLOGIA DEL URANIO

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## Abstract-Resumen

### STATISTICS AND GEOSTATISTICS: KRIGING AND USE OF HEMIVARIOGRAM FUNCTIONS IN THE STRUCTURAL INVESTIGATION OF URANIUM DEPOSITS.

After presenting some general conceptual considerations regarding the theory of regionalized variables, the paper deals with specific applications of the intrinsic dispersion law to the determination, description and quantification of structures. It then briefly describes two uranium deposits in Córdoba province, the study of which yielded the basic data and parameters for compiling the geostatistical results presented. Before taking up the matter of structural interpretations, it refers briefly to the mathematical relationship between the number of sampling points available and the number of directions that can be investigated by the variogram method and also emphasizes the need for quantifying regionalization concepts on the basis of a table of absolute dimensionalities. In the case of the "Rodolfo" deposit it presents and comments on the hemivariograms for concentrations, thicknesses and accumulations, drawing attention at the same time to the existence of significant nest-like phenomena (gigogne structures). In this connection there is also a discussion of the case of iterative lenticular mineralization on a natural and a simulated model. The "Schlagintweit" deposit is dealt with in the same way, with descriptions and evaluations of the subadjacent structures revealed by the hemivariographic analysis of grades, mineralization thicknesses and accumulations. This is followed by some considerations on the possibility of applying Krige and Matheron correctors in the moderation of anomalous mineralized thicknesses. In conclusion, the paper presents a "range ellipse" for grades; this is designed to supplement the grid of sampling points for the "Rodolfo" deposit by means of Matheronian kriging techniques.

### ESTADISTICA Y GEOESTADISTICA. KRIGISMO Y UTILIZACION DE LAS FUNCIONES HEMIVARIOGRAMICAS EN LA INVESTIGACION ESTRUCTURAL DE LA YACIMIENTOLOGIA DEL URANIO.

Después de abordar algunas generalidades conceptuales sobre la teoría de las variables regionalizadas, se tratan en este trabajo determinadas aplicaciones de la ley de dispersión intrínseca a la determinación, descripción y cuantificación de estructuras. Seguidamente se describen en forma somera dos yacimientos uraníferos de la provincia de Córdoba, cuyo estudio

proporcionó los datos y parámetros de base que permitieron la elaboración de las consecuencias geoestadísticas que se exponen. Antes de entrar en el capítulo de las interpretaciones estructurales, se hace una breve referencia a la relación matemática existente entre el número de sondeos de que se dispone y el de las direcciones posibles de investigar variográficamente, a la vez que se plantea la necesidad de cuantificar los conceptos de regionalización basándose en una tabla de dimensionalidades absolutas. Comenzando con el yacimiento “Rodolfo” se exponen y comentan los hemivariogramas de tenores, potencias y sus acumulaciones, señalándose en el mismo la existencia de interesantes fenómenos de folimorfismo interencajante (“structures gigognes”). Se trata también en este punto el caso de mineralizaciones lenticulares iterativas sobre un modelo natural y otro simulado. El yacimiento “Schlagintweit” es tratado en forma similar describiéndose y cifrándose las estructuras subyacentes reveladas por la investigación hemivariográfica de leyes, espesores mineralizados y acumulaciones. Se hacen después algunas consideraciones sobre la posibilidad de aplicar los correctores de Krige y de Matheron en la moderación de potencias mineralizadas anómalas, finalizándose con la determinación de una “elipse de alcance” para tenores, destinada a completar la malla de sondeos del yacimiento “Rodolfo” mediante las técnicas del krigismo matheroniano.

## 1. INTRODUCCION

Es interesante y fértil en consecuencias la tarea de delimitación y complementación de las nociones de “variable independiente” y “de carácter regionalizado”, que cubren el espectro a que alude la primera parte del epígrafe.

La utilización de la “función intrínseca” llevada a representaciones gráficas de base biaritmética, propicia la consecución de una herramienta de positivo empleo en la determinación, por una parte del esquema de distribución reinante y, por otra, del “alcance” o estereo-influencia de una regionalización ligada al fenómeno mineralizador, permitiendo a la vez la detección cifrada de la posible existencia del efecto “nugget” en el mismo; la posesión de estos parámetros de influencia permite la aplicación de los “correctores de leyes” en base a la participación de múltiples aureolas de datos enmarcantes.

Siendo, por definición, la Geoestadística la práctica de la teoría de las variables regionalizadas, su principal campo de acción se refiere al estudio de fenómenos de regionalización y caracterización de las estructuras que les sirven de soporte y a la cuantificación de los errores a los que quedan afectadas las estimaciones emergentes de su aplicación.

Para lograr sus objetivos, la teoría de las variables regionalizadas posee básicamente las siguientes metodologías de trabajo que pueden agruparse por una parte en métodos transitivos, de raíz geométrica, y por la otra en nociones derivadas de la teoría intrínseca que se expresan por la vía probabilística de las Funciones Aleatorias.

En el presente trabajo y en función de un grupo de datos de base, correspondientes a partes de dos yacimientos argentinos ubicados en la provincia

de Córdoba, se ha intentado extraer analíticamente — utilizando el hemivariograma que es una de las principales herramientas propuestas por la teoría intrínseca — la mayor cantidad posible de consecuencias de naturaleza estructural tendientes eventualmente a propiciar la aplicación del krigismo en la asignación local de tenores sujetos a errores minimizados, en un determinado sólido.

Para el logro de las aproximaciones de referencia — y ya en conocimiento de la existencia de algunos detalles estructurales revelados por las observaciones geológicas directas oportunamente realizadas — se ha analizado minuciosamente cada gráfico hemivariográfico experimental en su transición al modelo teórico capaz de brindar en forma directa y numéricamente expresadas, las pautas que se desea investigar.

El estudio de los fenómenos de transición traducidos en ligeras inflexiones de las curvas depuradas resultantes, permite detectar la existencia de estructuras que conforman regionalizaciones de diferentes rangos dentro de un mismo yacimiento, cuyo conjunto puede llegar a identificarse con el caso de verdaderas “estructuras folimórficas interencajantes”<sup>1)</sup> o “structures gigognes” de la literatura geoestadística francesa.

Los parámetros emergentes de estos análisis son de dos órdenes: vectoriales, ligados al valor del límite que separa el campo de una regionalización de aquel en el que rige la ley del azar puro; estos parámetros proporcionan las dimensiones métricas de la figura o sólido destinados a amparar un futuro tratamiento krigeano. Los segundos son parámetros escalares identificados con varianzas y covarianzas, poseyendo especial importancia entre las primeras, “ $C_0$ ” o “efecto pepítico” y “ $C$ ” ( $C_1$ ,  $C_2$ , etc.) magnitud que interviene en el cálculo de los “coeficientes de ponderación” “ $\lambda_i$ ” que minimizan la varianza de estimación y que se obtienen mediante el “Sistema Matheron”; por su complejidad, todos estos cálculos finales deben efectuarse por computación.

## 2. APLICACIONES DE LA LEY DE DISPERSION INTRINSECA EN LA CUANTIFICACION DE ESTRUCTURAS

Contándose con un buen número de datos de base, referidos a tenores, potencias mineralizadas y acumulaciones potencia-tenor de algunos yacimientos uraníferos ubicados en el ámbito de la Delegación Centro-CNEA, se procedió a efectuar una exhaustiva explotación de los mismos con miras a la extracción de conclusiones geoestadísticas sobre la vertebración espacial de las estructuras de soporte de sus mineralizaciones.

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<sup>1)</sup>  $\varphi\omega\lambda\epsilon\acute{o}\delta$  (G.C.),  $\varphi\omega\lambda\acute{\iota}\acute{\alpha}$  (G.M.) = nido.

## 2.1. Modelos naturales estudiados

Los ejemplos que se describirán en este trabajo corresponden a determinados sectores de los yacimientos “Rodolfo” (Valle de Punilla – Córdoba) de tipo estratiforme, y “Schlagintweit” (Sierra de Los Gigantes – Córdoba) de tipo “amas”.

El primero se aloja en estratos tércicos de la Formación Cosquín, que a su vez consta de tres miembros: el inferior, compuesto de conglomerados y areniscas feldespáticas de matriz arcillosa, el miembro medio, formado de limos y arcillas calcáreas y el más conspicuo portador del uranio, que en algunos casos llega a alojarse también en el miembro inferior, y el miembro superior que se compone de areniscas arcósicas y limos calcarenosos que tornándose progresivamente más calcáreos hacia los sectores cuspidales llegan a componer finalmente un horizonte netamente calcáreo y algo tufáceo que se comporta como una verdadera capa-guía a todo lo largo de la continuidad Norte-Sur de la formación; este banco que participa de elementos provenientes tanto de la Sierra Grande (rocas graníticas eopaleozoicas que dieron origen a la Formación Cosquín) como de la Sierra Chica (metamorfitas precámbricas que originaron la Formación Casa Grande, que descansa pseudo-concordantemente sobre la F. Cosquín) constituye un verdadero hito estratigráfico. En general, las sedimentitas portadoras de la mineralización uranífera se encuentran en posición homoclinal en un ángulo de unos 30° hacia el Este, bajo el bloque metamórfico de la Sierra Chica que las suprayace por efecto de una gran fractura compresiva submeridiana que buza en el mismo sentido con una intensidad de aproximadamente 60°; otro juego de fracturas subparalelas a aquella y de más reciente data hace infrayacer al Cuártico bajo los sedimentos tércicos, dando lugar a estructuras de imbricación. Determinados sectores del yacimiento presentan estructuras plegadas de envergadura hectométrica, probablemente ligadas a juegos tafro y/o esfenogenéticos operados en el seno del cristalino que les sirve de sustrato.

En detalle la mineralización se presenta en agrupaciones de forma elongada, de entre 3 y 20 mm de longitud, compuestas de cristales de aspecto micáceo (carnotita y tyuyamunita) de tamaños individuales comprendidos entre 0,5 y 6 mm; las mismas aparecen implantadas preferentemente en aquellas áreas donde el material pelítico de ganga se encuentra decolorado, o bien en geodas de contracción de arcilla con calcita secundaria, casos en que la carnotita ocupa los espacios porales residuales. Estas agrupaciones carecen de lineación definida y de todo control lineal o planar, dado que los sedimentos albergantes no poseen “estructuras sedimentarias notorias”. A su vez, la frecuencia local de estas agrupaciones da lugar a la formación de verdaderos lentes de limitaciones muy indefinidas que se disponen subparalelamente a la estratificación. La longitud de estos lentes varía entre 10 y 20 m y sus potencias, siempre proporcionales a la misma, oscilan entre 5 y 10 m. Un análisis variográfico posterior (Lucero M.,



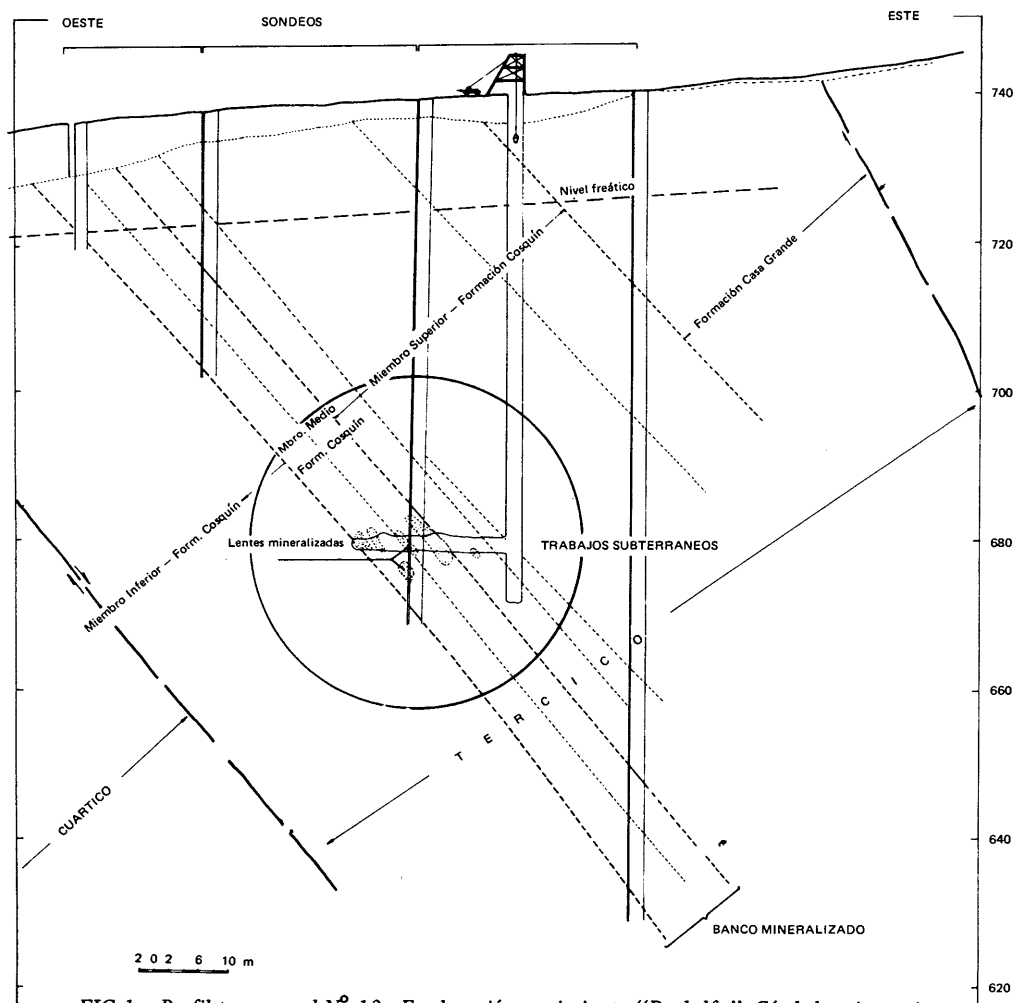
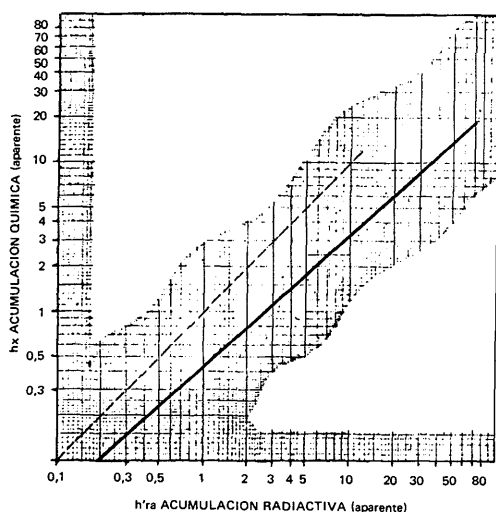


FIG.1. Perfil transversal N° 13. Exploración yacimiento "Rodolfo", Córdoba, Argentina.

1973) permitió interpretar que "la longitud de onda del fenómeno mineralizador" era del orden de los 15 m, que coincide con el valor del "alcance" de 7,5 m obtenido por otra vía, que se establece en este trabajo.

En general se trata de sucesiones de lentes que desaparecen después de cierto recorrido sobre banco, siendo inmediatamente reemplazadas por otras hacia abajo, arriba o a continuación de la anterior, ocurriendo todo ello mayormente dentro de los límites de un paquete que en el sector Sur posee un espesor medio de unos 9 m y en el Norte, hasta 11 y 13 m. En todo el yacimiento no se advierten cambios sensibles de estas potencias generales de banco hacia profundidad (Fig. 1).

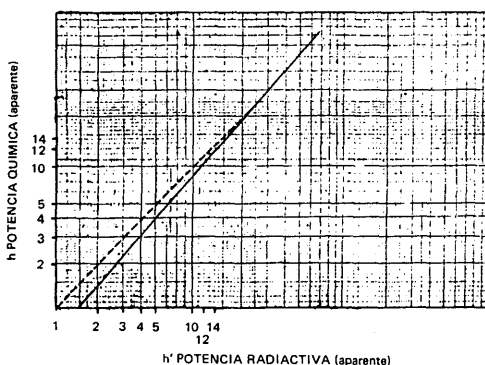
Recta  $h'ra/hx$ 

$$hx \begin{cases} \gamma = 2,38 \\ \sigma^2 = 0,856 \\ \sigma = 0,925 \end{cases} \quad h'ra \begin{cases} \gamma = 7,15 \\ \sigma^2 = 0,990 \\ \sigma = 0,995 \end{cases} \quad h'ra/hx \begin{cases} \sigma^2 = 0,0562 \\ N = 19 \end{cases}$$

$$r = \frac{\sigma^2 h'ra + \sigma^2 hx - \sigma^2 h'ra/hx}{2(\sigma h'ra \cdot \sigma hx)} = 0,9723; \quad p = r \frac{\sigma hx}{\sigma h'ra} = 0,9038$$

$$d = \frac{0,434}{2} (1-r^2) \sigma^2 hx = 0,01015 = 1 \text{ mm}$$

$$\text{Precisión} \begin{cases} 1 \text{ medida: } \sigma^2 w = \sigma^2 hx(1-r^2) = 467,7 \cdot 10^{-4} \\ \text{(a 95\%)} \quad 2\sigma w = 43,2\% \\ N \text{ medidas: } \sigma^2 w = \frac{\sigma^2 hx(1-r^2)}{N} + \frac{\sigma^4 hx(1-r^4)}{2N} = 45 \cdot 10^{-4} \\ \text{(a 95\%)} \quad 2\sigma w = 13,4\% \end{cases}$$

Recta  $h'/h$ 

$$h \begin{cases} \gamma = 5,29 \\ \sigma^2 = 0,295 \\ \sigma = 0,543 \end{cases} \quad h' \begin{cases} \gamma = 6,42 \\ \sigma^2 = 0,220 \\ \sigma = 0,469 \end{cases} \quad h'/h \begin{cases} \sigma^2 = 0,01265 \\ N = 19 \end{cases}$$

$$r = \frac{\sigma^2 h' + \sigma^2 h - \sigma^2 h'/h}{2(\sigma h' \cdot \sigma h)} = 0,986; \quad p = r \frac{\sigma h}{\sigma h'} = 1,1427$$

$$d = \frac{0,434}{2} (1-r^2) \sigma^2 h = 0,0018 = 0,2 \text{ mm}$$

$$\text{Precisión} \begin{cases} 1 \text{ medida: } \sigma^2 w = \sigma^2 h(1-r^2) = 82,6 \cdot 10^{-4} \\ \text{(a 95\%)} \quad 2\sigma w = 18,2\% \\ N \text{ medidas: } \sigma^2 w = \frac{\sigma^2 h(1-r^2)}{N} + \frac{\sigma^4 h(1-r^4)}{2N} = 5,6 \cdot 10^{-4} \\ \text{(a 95\%)} \quad 2\sigma w = 4,8\% \end{cases}$$

FIG. 2. Método Lea-estadístico.

El yacimiento “Rodolfo” ha sido explorado sobre una longitud meridiana de 2200 m mediante una malla de sondeos de 100 m por  $\approx 40$ , homotética del depósito, y sus reservas, estimadas a través de instrumentos evaluativos de raíz estadística fundados en correlaciones radiactividad/tenor, principalmente mediante el sesgo logestadístico de la combinación de dos rectas complementarias de correspondencia, una de ellas “potencia radiactividad/potencia real” y la otra correspondiente a las acumulaciones “potencia aparente X radiactividad/potencia real X tenor” (Fig. 2). A la vez y muy localmente se efectuó una exploración subterránea mediante galerías al nivel - 60 m (Fig. 1) muestreándose metro a metro en canaleta y en dirección Norte-Sur sobre un recorrido de  $\approx 40$  m; a este grupo de muestras, químicamente analizadas, se agregó a continuación otro

complementario correspondiente a parte de las chimeneas existentes, cuyos tenores y potencias fueron estimados radimétricamente y los que en base a la uniformidad de comportamiento estructural geológicamente comprobado por observaciones directas, se proyectaron sobre la horizontal a los efectos de poder comprobar mediante el concurso de un procedimiento de simulación, y sobre una longitud algo mayor, la conducta variográfica de toda la secuencia.

El segundo depósito se identifica con el yacimiento “Schlagintweit” correspondiente a una masa de tipo “amas” en ambiente de rocas intrusivas sobresaturadas, muy tectónicamente afectada y poseedora de una mineralización mayormente autunítica alojada en fracturas (diaclasas y fallas de muy diversa actitud espacial). Sobre la índole de la mineralización pueden mencionarse las siguientes circunstancias: tiene carácter monomineral, carece de ganga y es aparentemente supergénica (dos acompañantes ocasionales presentes como rareza, uranofano y fosfuranulita, son también supergenos). La mineralización está confinada entre la superficie y una determinada profundidad que no excede los 40–50 m; posee una ubicación preferencial con respecto a las grandes líneas de fracturación que pudieron actuar como trampas en la confluencia de algunos accidentes que forman estructuras diédricas en las que sus planos jugaron en un sentido el papel de avenidas y en otro el de pantallas de contención. Las evaluaciones se efectuaron en base a relaciones de correspondencia radiactividad/tenor (Lucero M., 1978). Este depósito, de unos 600 m de NO a SE por 200 de NE a SO, ha sido explorado según una malla regular de 20 X 20, salvo en un pequeño sector en el que se implantó una malla un tanto más densa.

## 2.2. Elección de direcciones a investigar

Un análisis variográfico completo requiere la investigación del comportamiento de los parámetros correspondientes (tenores, potencias mineralizadas o acumulaciones potencia-ley) en todas las direcciones posibles del campo mineralizado en estudio; sin embargo no tiene mucho sentido llevar esta tarea hasta el límite de las posibilidades materiales, dado el gran número de variables existentes. Teóricamente el número de direcciones posibles de investigar es muy grande y en general equivale a un 30% del total de puntos, en esquemas comprendidos entre los 50 y 500 puntos de base que figuran sobre abscisas en el gráfico de la Fig. 3 (materializados en un campo mineralizado mediante perforaciones o sus cuadrículas enmarcantes), observándose un ligero incremento de ese porcentaje al disminuir la cantidad de sondeos. Estas relaciones son ciertas para esquemas idealizados de puntos dispuestos en red cuadrada regular y encerrados dentro de una planta de traza circular a efectos de poder contabilizar su número con el mayor rigorismo posible. Estas circunstancias pueden llegar a reproducirse en la práctica, siempre que tales unidades (impactos de perforaciones o unidades de malla enmarcantes) se encuentren uniformemente distribuidas dentro de un

CUADRO I. RANGOS DE DIMENSIONALIDAD ABSOLUTA PARA 60 ORDENES TERRESTRES

Hiperescala Orden km		Megaescala Orden km		Macroescala Orden m		Microescala Orden mm		Nanoescala Orden $\mu$		Picoescala Orden Å	
1	6377.000	11	6227.539	21	6.082	31	5.939	41	5.800	51	56.600
2	3188.500	12	3113.770	22	3.041	32	2.970	42	2.900	52	28.300
3	1594.250	13	1556.885	23	1.520	33	1.485	43	1.450	53	14.150
4	797.125	14	779.442	24	0.760	34	0.742	44	0.725	54	7.075
5	398.562	15	389.221	25	0.380	35	0.371	45	0.362	55	3.538
6	199.281	16	194.611	26	0.190	36	0.186	46	0.181	56	1.769
7	99.641	17	97.305	27	0.095	37	0.093	47	0.090	57	0.884
8	49.820	18	48.653	28	0.048	38	0.046	48	0.045	58	0.442
9	24.910	19	24.326	29	0.024	39	0.023	49	0.023	59	0.221
10	12.455	20	12.163	30	0.012	40	0.012	50	0.012	60	0.110

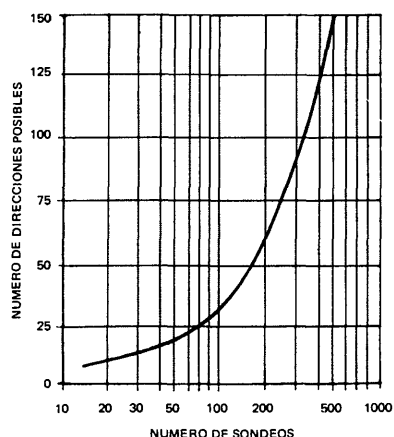


FIG. 3. Direcciones posibles de investigar variográficamente sobre reticulos de sondeos.

sistema bien regular, según red compacta y carente de espacios vacíos. El mencionado gráfico muestra las relaciones comprobadas, existentes entre los mencionados parámetros; en él, a partir de cierto número finito de impactos ordenados en una red de las características anotadas, puede obtenerse el número aproximado de direcciones posibles de investigar, según los diferentes rumbos que pueden pasar por los diversos alineamientos determinados por la trama regular del sistema. En los casos de cuerpos mineralizados de tipo amas, o por lo menos poseedores de espesores relativamente significativos, se hace necesario complementar la información con investigaciones del mismo carácter efectuadas según la vertical. En los casos presentados no ha sido posible llevar esta suerte de investigación sobre más de cuatro direcciones horizontales como máximo, y en solo uno de ellos extenderla en forma muy aleatoria a la dimensión vertical.

### 2.3. La interpretación estructural

Ante la evidente necesidad del empleo de una terminología de índole cuantitativa capaz de describir dimensionalmente las estructuras de diferentes rangos que se presentan en el estudio de las regionalizaciones, se propone la adopción de la “Tabla de escalas de dimensionalidades absolutas para 60 órdenes terrestres” (véanse Cuadros I y II).

Como puede verse, no es probable que se tropiece con una “nanoestructura” ni con una “hiperestructura” encontrándose casi todos los casos posibles de “regionalizaciones estructurales de los fenómenos mineralizadores” entre las categorías de “micro y megaestructura”; de esta manera, una regionalización del

CUADRO II. ALGUNAS APLICACIONES PRACTICAS DE LA DIMENSIONALIDAD ABSOLUTA

Ordenes	Ejemplos
1– 3	Continentes
4– 6	Islas, mares, etc.
7–10	Rasgos geológicos regionales
11–13	Ambientes regionales de depósitos
14–20	Depósitos minerales, campos petrolíferos, etc.
21–30	(Minería) potencias mineralizadas, etc.
31–....	Petrografía, mineralogía ...

orden de los 6 mm (detectable solo por observaciones directas de detalle y que en un variograma podría aparecer como un simple “efecto pepítico”) correspondería a una “microregionalización” de orden 31, una estructura de  $\approx 3$  m sería una “macroregionalización” de orden 22, y una estructura mayor, de por ejemplo 800 m, se identificaría con una “megaregionalización” de orden 14, etc. etc.

### 2.3.1. Yacimiento “Rodolfo”. Comportamiento de los tenores, potencias y acumulaciones

Comenzando por este depósito, se describirán los resultados obtenidos de los diferentes exámenes variográficos realizados. Contándose con un grupo moderadamente numeroso de sondeos implantados según una malla de 100 por 40 m, se efectuaron cuatro investigaciones de hemivarianzas en las siguientes direcciones: N–S, E–O, NE–SO, y NO–SE; naturalmente podrían haberse agregado otras direcciones más pero el volumen de información que hubieran aportado habría sido tan pequeño que sus resultados carecerían de la fiabilidad necesaria.

La Fig. 4 revela las siguientes circunstancias anomalíarías ligadas al andamiaje íntimo que constituye la infraestructura del espacio sujeto a este tipo de investigación.

En presencia de un “campo”, es decir el dominio dentro del cual juega la variable en estudio en función de un argumento “d” definido, y de un “soporte”, identificado con el volumen del tipo de muestra en base al cual haya sido explorado el depósito, se estará en condiciones de aplicar los formulismos conducentes al empleo de la función intrínseca, o hemivariograma, o sea, y por

definición, el error medio cuadrático que se comete al atribuir a la variable  $Y$  en el punto  $x$  el valor de  $Y$  en el punto  $x + d$ :  $\gamma(d) = 1/2 E [Y(x + d) - Y(x)]^2$ .

Recordando que el concepto de “tenor” independizado de su “soporte” carece de sentido, será necesario en primer lugar describir el tipo de muestra empleado en el reconocimiento del campo. En este caso se ha tratado de testigos de perforaciones (y radiotestigos) que en términos de equivalencia lineal poseerían una dimensión de  $\approx 2$  m, valor que comparado con el E.L. del total explorado ( $\approx 2250$  m) resulta tan insignificante que induce a considerar a este soporte como de carácter meramente puntual.

Analizando en primer lugar la dirección N–S que es la más rica en datos de base por corresponder con mucho a la mayor elongación del campo mineralizado objeto de una exploración efectuada en base a un distanciamiento de “ $d$ ” = 100 m, se observa la existencia de un efecto pepítico “ $C_0$ ” un tanto acentuado. Dado el relativamente pequeño volumen del “soporte” y la gran dimensión del argumento “ $d$ ” que se hace desplazar a lo largo de todo el campo, es muy probable que este efecto – evidenciado en la brusca discontinuidad de la función  $\gamma(d)$  que determina una tangente vertical en el origen – tenga sus verdaderas raíces en la existencia de una miniestructura que la escala explorativa es incapaz de poner en evidencia en el gráfico variográfico.

Dado que otra investigación más localizada de hemivarianzas, llevada a cabo a una escala 1/100 de la que se está tratando y que será objeto de posteriores consideraciones, tampoco pudo detectar el “alcance” de este posible fenómeno de transición, es probable que su magnitud pueda recaer en el campo dimensional de una microregionalización o bien en el de una macroregionalización de orden inferior (entre 25 y 30). Esta primera estructura, cuyos parámetros relativos a las magnitudes de “meseta” ( $C_x$ ) y “alcance” ( $a_x$ ) no es posible definir en base a varianzas sin el concurso de una investigación complementaria de mucho detalle, sería la de más pequeño orden existente en este yacimiento.

- Observando el gráfico, es posible detectar en el “modelo teórico” o “hemivariograma medio” emergente del experimental de base, una ligera pero notoria inflexión, determinante de un valor de “meseta”  $C_2 = 0,022$  y de un “alcance”  $a_2 = 175$  m; este último valor se identificaría en el caso de que el campo fuese isótropo, con el radio de la zona de influencia de una muestra y marca la presencia de otra estructuración de mayor rango que la anterior y correspondiente a una megaregionalización de orden 16–17 (Fig.4).
- Continuando con el análisis de la representación variográfica según la dirección N–S, surge la existencia de una tercera regionalización de gran envergadura que concurre a la reconstrucción estructural del espacio, siendo su parámetro escalar de valor de “meseta” o “palier”,  $C_3, 0,022$  y el “alcance” de la transición,  $a_3$ , de 870 m.

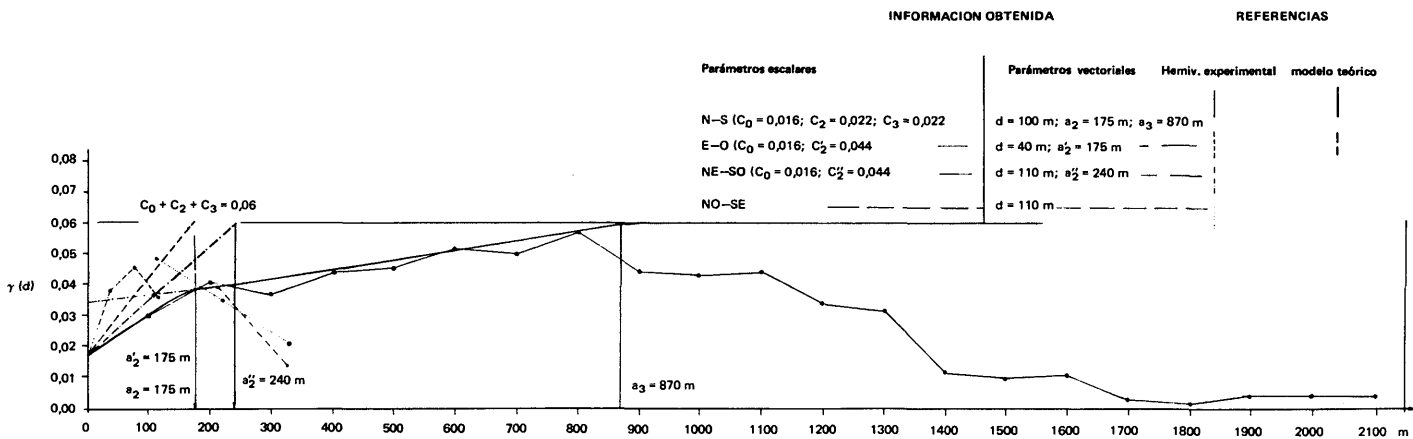


FIG.4. Hemivariograma de tenores, megaregionalizaciones. Yacimiento "Rodolfo", Córdoba, Argentina.



Hacia la derecha de la posición de " $a_3$ " se nota un paulatino decrecimiento de los valores de varianzas por falta progresiva de información; en el caso de que se hubiese contado con un mayor volumen absoluto de datos, sería de esperar:

- 1) una periodicidad del fenómeno iterativo megaregionalizante en un orden de 1,5 km como longitud de onda del hecho físico-químico mineralizador, y
- 2) que la siguiente cúspide del fenómeno de tipo deriva que entrañaría tal estructuración podría hallarse entre las hemivarianzas  $\gamma(d_{24})$  y  $\gamma(d_{30})$  que no es posible calcular dado que la exploración en base a " $d$ " = 100 m no supera actualmente los 2100 m de longitud meridiana.

Antes de tratar las restantes direcciones se incluirán los resultados de otro estudio basado en la extracción de muestras metro a metro en trabajos subterráneos de un pequeño sector del mismo yacimiento (Fig. 1); se trata de un muestreo "en canaleta" efectuado en forma un tanto selectiva sobre una continuidad de lentes mineralizadas separadas por breves tramos poseedores de leyes relativamente más bajas.

En la Fig. 5a puede verse el comportamiento de las hemivarianzas y su resultado final expresado en un esquema iterativo de tipo deriva, que poseedor de un efecto pepítico  $C_0 = 0,01$  señala la posible existencia de miniestructuras — probablemente las mismas a las que se aludió más arriba — que también escaparon al examen realizado a esta escala bastante detallada de trabajo; el valor de  $C_1$  es de 0,05. En el relativamente breve recorrido del argumento " $d$ " = 1 m a través de parte del campo en estudio, se advierte la presencia reiterada de una macroregionalización de orden 21 ( $a_1 = 7,5$  m) que indica la existencia de un pulso o ritmo del fenómeno mineralizador, de alrededor de unos 15 m.

Incorporando este fenómeno al del gráfico 4, se obtiene (Fig. 6) un panorama más completo del cúmulo de regionalizaciones de diferente rango que coexisten según la dirección en este caso investigada (N-S).

Resumiendo, se tropieza: a) en primer lugar, con una microregionalización, o macroregionalización de orden inferior, puesta en evidencia cualitativa por los efectos "nugget" acusados por los gráficos 4, 5a, y 6; b) en segundo, con una macroregionalización de orden 21, poseedora de un alcance  $a_1 = 7,5$  m; c) en tercero, con una megaregionalización de orden 16–17 ( $C_2 = 0,022$  y  $a_2 = 175$  m) y, finalmente, d) con una segunda megaregionalización de orden 14 ( $C_3 = 0,022$  y  $a_3 = 870$  m) que se identifica con la estructura de mayor rango existente en el campo según la dirección señalada.

Esta relación de regionalizaciones de diferentes rangos dentro de un mismo campo indica la existencia de un evidente "*folimorfismo*" de carácter *interencajante*, de sumo interés desde el punto de vista estructural de la mineralización del depósito.

El examen del comportamiento de los tenores en la dirección Este-Oeste, según la cual el módulo de desplazamiento del argumento " $d$ " es de solo 40 m, brinda muy pocos elementos de base, dado que siendo la dimensión transversal

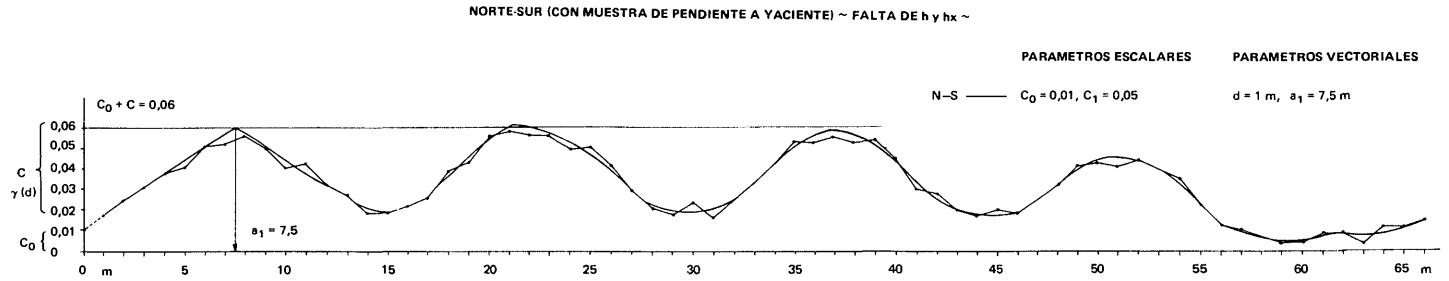


FIG.5a. Hemivariograma de tenores. Yacimiento "Rodolfo", Córdoba, Argentina.

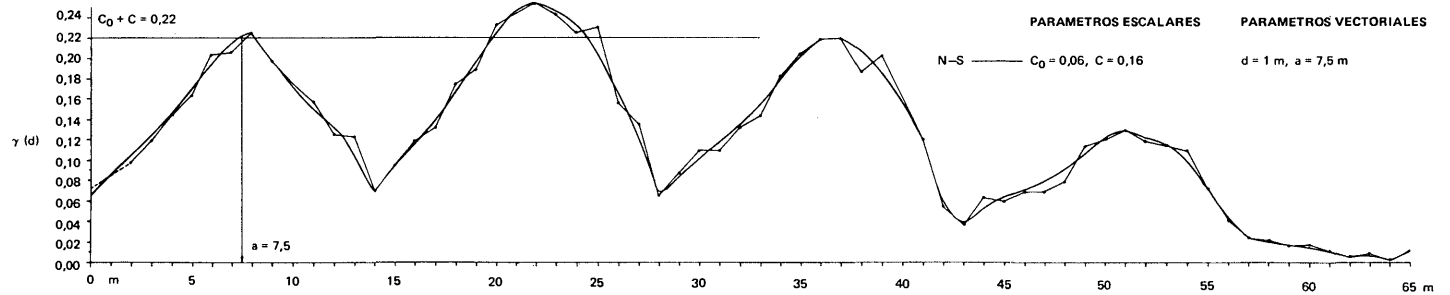


FIG.5b. Hemivariograma de acumulaciones potencia-tenor. Yacimiento "Rodolfo", Córdoba, Argentina.

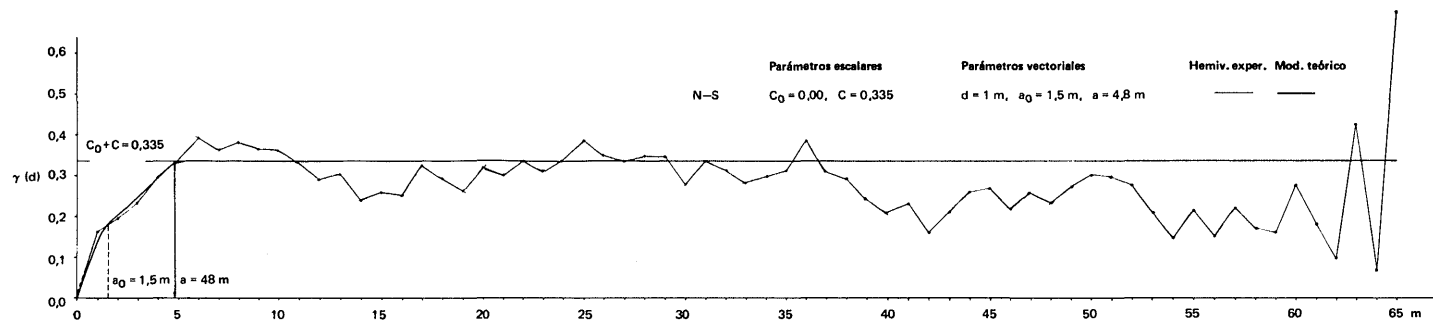


FIG.5c. Hemivariograma de potencias mineralizadas. Yacimiento "Rodolfo", Córdoba, Argentina.

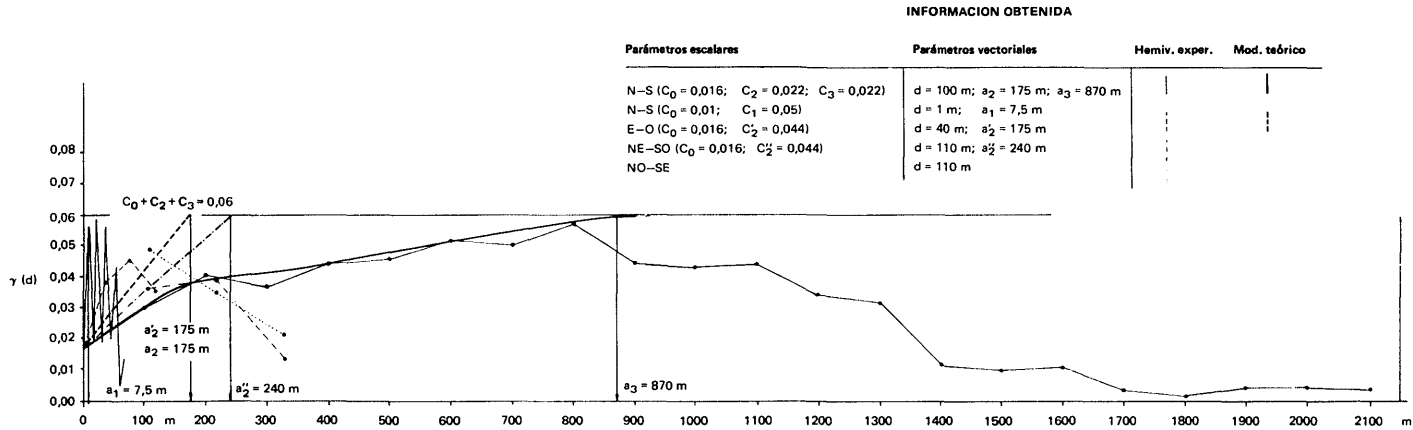


FIG.6. Hemivariograma de tenores. Yacimiento "Rodolfo", Córdoba, Argentina.

media del depósito de 140–160 m no fue posible obtener más de tres varianzas en total.

Siendo tan escaso el número de datos disponibles, las consecuencias emergentes del trazado del hemivariograma medio poseen un cariz un tanto especulativo que ha sido reforzado por algunas observaciones geológicas directas; es así como se ha aceptado la existencia de una estructura de carácter iterable y correspondiente a la dimensión megaescalar (orden 16–17), poseedora de un efecto de pepita  $C_0 = 0,016$  (que parece corresponder a una miniestructura), una meseta  $C_2' = 0,044$  y un alcance transicional de  $a_2' = 175$  m, coincidente con el de una importante estructura detectada sobre la dirección N–S y con la cual puede ser identificada.

En cuanto a la dirección NE–SO (que es en realidad N 22° E–S 22° O) con “d” = 110 m, por las mismas causas mencionadas para la anterior, posee solo tres valores de varianza, siendo el carácter de su alcance de transición también bastante aleatorio, aunque cuenta con el apoyo de observaciones geológicas previas; conservando el valor del efecto pepítico, el de la meseta  $C_2''$  sería de 0,044 como en el caso anterior, y el de  $a_2''$  de unos 240 m que aparentemente miden otra dimensión de la misma estructura de mesetas  $C_2$ ,  $C_2'$  y  $C_2''$ .

La restante dirección NO–SE no brindó ninguna información.

Tanto las conclusiones emergentes de las observaciones según la dirección E–O como de la NE–SO, parecen confirmar la existencia del folimorfismo que en forma tan evidente puso de manifiesto la N–S, no solamente en cuanto a la mineralización responsable del efecto pepítico (ya determinada por observación geológica directa) como a la correspondiente a los alcances  $a_2$ ,  $a_2'$  y  $a_2''$ .

La macroregionalización de rango intermedio entre las dos que se acaban de mencionar, vale decir la detectada por el muestreo metro a metro (radiométricamente comprobada en los trabajos subterráneos) se hace también perceptible en la dirección Este–Oeste, coincidente con la exploración por chimeneas a que ya se hiciera referencia.

En cuanto a la megaregionalización mayor, caracterizada por  $C_3 = 0,022$  y  $a_3 = 870$  m, solo es detectable en sentido N–S, dado que por su dimensión excedería la estrecha envergadura del depósito según la dirección E–O en la cual el yacimiento ha sido decapitado por la erosión hacia el Oeste y tectónicamente cercenado en profundidad hacia el Este.

El hemivariograma de tenores (Fig. 4), especialmente el que investiga la dirección N–S, parece ajustarse mejor a un esquema de transición matheroniano que por ejemplo a uno formeryano o wijsiano, entre los más frecuentes.

En efecto, la traza idealizada en base a la aplicación de la ecuación esquemática correspondiente así lo evidencia, siendo sus valores los siguientes:

	Hemivarianzas y sus corresp. covarianzas <sup>2</sup>	
$\gamma(d_1) = C_0 + C[3/2 d/\omega - (d/\omega)^3 \times 0,5]$	= 0,0235	0,0365
$\gamma(d_2) = C_0 + C[3/2 d/\omega - (d/\omega)^3 \times 0,5]$	= 0,0285	0,0315
$\gamma(d_3) = C_0 + C[3/2 d/\omega - (d/\omega)^3 \times 0,5]$	= 0,0378	0,0222
$\gamma(d_4) = C_0 + C[3/2 d/\omega - (d/\omega)^3 \times 0,5]$	= 0,0442	0,0158
$\gamma(d_5) = C_0 + C[3/2 d/\omega - (d/\omega)^3 \times 0,5]$	= 0,0498	0,0102
$\gamma(d_6) = C_0 + C[3/2 d/\omega - (d/\omega)^3 \times 0,5]$	= 0,0543	0,0057
$\gamma(d_7) = C_0 + C[3/2 d/\omega - (d/\omega)^3 \times 0,5]$	= 0,0577	0,0023
$\gamma(d_8) = C_0 + C[3/2 d/\omega - (d/\omega)^3 \times 0,5]$	= 0,0596	0,0004
$\gamma(d_{d \geq \omega}) = C_0 + C(1,5 - 0,5)$	= 0,06	0,00
$\gamma(d_9) = C_0 + C[3/2 d/\omega - (d/\omega)^3 \times 0,5]$	= 0,05999	0,00001
$\gamma(d_{10}) = C_0 + C[3/2 d/\omega - (d/\omega)^3 \times 0,5]$	= 0,0585	0,0015
$\gamma(d_{11}) = C_0 + C[3/2 d/\omega - (d/\omega)^3 \times 0,5]$	= 0,0550	0,0050
$\gamma(d_{12}) = C_0 + C[3/2 d/\omega - (d/\omega)^3 \times 0,5]$	= 0,0493	0,0107
$\gamma(d_{13}) = C_0 + C[3/2 d/\omega - (d/\omega)^3 \times 0,5]$	= 0,0412	0,0188
	etc.	etc.

Si bien los “alcances” de la primera megaregionalización poseen idénticos valores para las direcciones N–S y E–O (175 m) no sucede lo mismo con la tercera, que evidencia un “alcance” un tanto mayor (240 m); esta circunstancia, que da lugar a una representación en elipse, indica que se está en presencia de una “anisotropía geométrica” en el yacimiento, caracterizada por la diversidad de los “alcances” bajo conservación del valor de la “meseta”, y por el hecho de que una conveniente variación de direcciones puede restablecer el cuadro isotrópico realmente existente.

Con respecto a los restantes parámetros, potencias mineralizadas y acumulaciones potencia-tenor de este yacimiento, solo se ha contado con los datos emergentes de la exploración lineal subterránea mencionada, lográndose obtener los dos siguientes gráficos (Figs 5 b y 5 c).

En cuanto a las acumulaciones (Fig. 5 b) puede observarse que en forma muy semejante al hemivariograma de tenores, tiene lugar una sucesión reiterada de estructura que la intervención de las potencias no ha logrado borrar. Se observa la misma estructura, de alcance  $a = 7,5$  brindada por los tenores, siendo en el caso de las acumuladas  $C_0 = 0,06$  y  $C = 0,16$ . Como puede verse, persiste en la representación el efecto pepítico enmascarante de una estructura de rango menor que escapó por razones de escala al muestreo monométrico realizado.

El hemivariograma de potencias mineralizadas presenta otra conformación bastante diferente, observándose solo una pequeña estructura poseedora de un

<sup>2</sup> Siendo  $\gamma(d) = C_0 + \text{tg}\alpha \cdot d$   
y la covarianza  $= C_0 + C - \gamma(d) = C_0 + C - C_0 + \text{tg}\alpha \cdot d = C - \text{tg}\alpha \cdot d$ .

alcance transicional  $a = 4,8$  m (macroregionalización de orden 21–22) y a partir de este punto, una suerte de pequeñas variaciones que podrían interpretarse como las irregularidades de una formación mesetiforme de gran desarrollo longitudinal que hace pensar en la existencia de alguna estacionaridad de carácter local y en la carencia de signos de deriva.

Podría también señalarse una dudosa estructura de rango inferior, de 1,5 m de alcance que correspondería a una macroregionalización de orden 23. En general puede decirse que el tramo en el que se observa dependencia entre las funciones es bastante breve, alcanzándose rápidamente el dominio de la independencia propia de las funciones aleatorias. Se trata de una función de carácter continuo, carente de efecto pepítico por lo tanto y poseedora de un valor de  $C = 0,335$ . Se considera para ambos gráficos que el ajuste esquemático es de carácter sensiblemente matheroniano, como en el caso de los tenores.

Es interesante señalar que para el caso de estructuras lenticulares más o menos regularmente localizadas en el espacio basta efectuar un simple perfil de tenores en el que se trace la isopleta de la ley media para obtener una medida directa de la dimensión de las estructuras existentes.

El perfil de un modelo simulado (Fig. 7 b) muestra que la longitud de onda del fenómeno mineralizador, medido, por ejemplo, entre las progresivas 3,5 m y 13,5 m brinda un valor de 10 m para la estructura existente, cifra que coincide con la proporcionada por el hemivariograma del mismo modelo (Fig. 7 c); en efecto el “alcance” que mide el radio de la regionalización es de 5 m.

El modelo natural de “Rodolfo” proporciona un ejemplo real de lo dicho. Su perfil de tenores (Fig. 7 a) medido, por ejemplo entre las progresivas 37,5 y 53 m, brinda una estructura de 15,5 m, coincidente con la evidenciada por su hemivariograma (Fig. 5 a) donde el alcance “a” es igual a 7,5 m.

Esta metodología sería válida sólo para este tipo de estructuras y carece de la versatilidad y riqueza en consecuencias que caracteriza a las de raíz geoestadística elaboradas por la vía variográfica.

### 2.3.2. *Yacimiento Schlagintweit. Comportamiento de tenores, potencias y acumulaciones*

A los efectos de jugar estadísticamente con los tenores y potencias acumuladas, se ha hecho uso de un grupo de datos de base correspondientes a sondeos distanciados cada 10 m, malla cuadrada según la cual se ha explorado un determinado sector del depósito. Se han efectuado exámenes variográficos solamente sobre dos direcciones, la NO–SE correspondiente a la mayor elongación del amas, y la ortogonalmente relacionada con ésta; se ha agregado una tercera dirección identificada con la vertical, que por razones que más adelante se expondrán no suministró la información buscada.

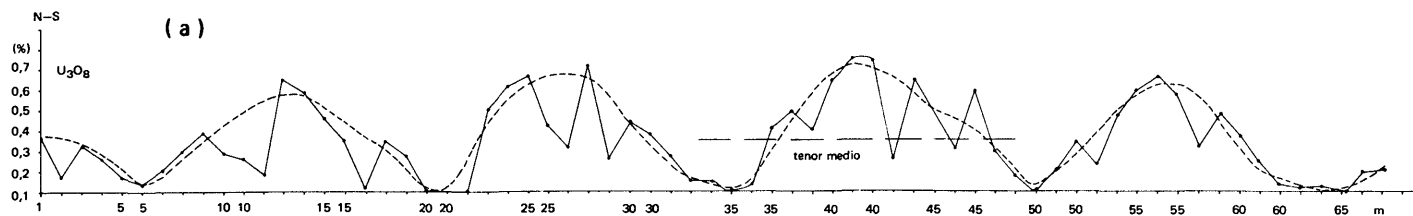


FIG. 7a. Perfil de tenores de un modelo natural. Yacimiento "Rodolfo", Córdoba, Argentina.

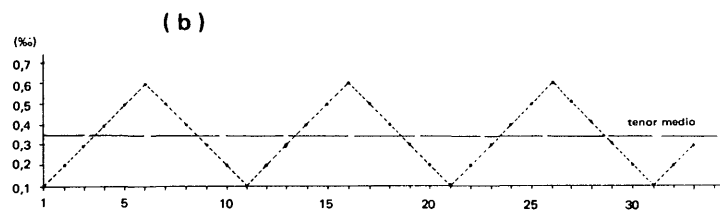


FIG. 7b. Perfil de tenores de un modelo simulado. Yacimiento "Rodolfo", Córdoba, Argentina.

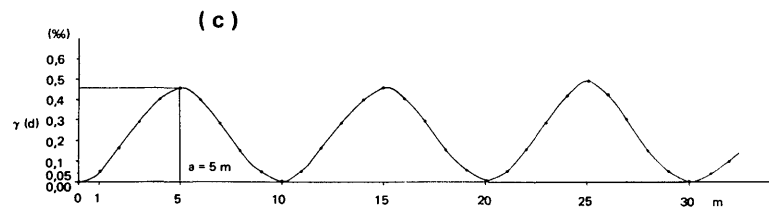


FIG. 7c. Hemivariograma del modelo simulado. Yacimiento "Rodolfo", Córdoba, Argentina.



En cuanto a la caracterización del “soporte” que brinda sentido a la noción de tenor, de acuerdo a la exploración realizada, puede también considerarse como de índole puntual.

Referente a las potencias mineralizadas, se ha tomado todo el desarrollo del yacimiento en sus dos dimensiones extremas coincidentemente con las direcciones más arriba mencionadas, pero respetando la vigencia de un argumento “d” igual a 20 m sobre ambas.

Comenzando por el hemivariograma de tenores que en base al amplio venero brindado por la representación biaritmética parece también ajustarse bastante bien a un esquema de Matheron, se tendría que una primera observación del gráfico de la Fig. 8 permite la cuantificación de un efecto pepítico  $C_0 = 0,023$  que probablemente — como en los casos anteriores — se encuentre delatando la existencia de alguna miniestructura que la escala de reconocimiento no ha logrado poner en evidencia mensurable.

Tomando en consideración la dirección NO—SE, más dotada en cuanto a datos de base, se observa la presencia de una primera estructura caracterizada por un valor de meseta de  $C = 0,03$  y uno de alcance de  $a = 18$  m, correspondiente a una megaregionalización de orden 19—20.

Una segunda estructura portadora de un valor de  $C_1 = 0,02$  y de  $a_1 = 84$  m se hace presente como la de mayor envergadura del depósito, correspondiendo a otra megaregionalización de orden 18—19; a partir de la posición de  $a_1$  en adelante, el valor de las varianzas comienza a decrecer rápidamente por insuficiencia de información.

La otra dirección estudiada, la NE—SO, posee menor número de elementos de base por lo cual sólo se pudieron conformar cinco valores de varianzas, encontrándose las dos últimas basadas en muy pocos elementos; por esa razón, los valores de alcance logrados no poseen la fiabilidad de aquellos brindados por la primera dirección. No obstante y con las reservas correspondientes al caso, se aceptaron los siguientes parámetros  $C' = 0,05$  y  $a' = 28$  m.

Con respecto a la dirección vertical puede decirse lo siguiente: de acuerdo a la profundidad media explorada ( $\approx 36$  m) y el intervalo decamétrico adoptado (por problemas de rendimiento precisional de la recta de correlación empleada) las probabilidades de enfrentamientos para el cálculo de varianzas son muy escasas y disminuyen muy rápidamente con los distanciamientos sucesivos. Por otra parte puede notarse que las últimas varianzas, especialmente la cuarta, es el resultado de la comparación de niveles poseedores de leyes muy bajas y muy similares entre sí, en razón de que la parte superior del depósito se encontraría empobrecida por probable arrastre mecánico del mineral, y la inferior lo es naturalmente por corresponder a sectores basales próximos a niveles de transición con el estéril franco.

Podría haberse logrado mejores resultados, en caso de haberse dispuesto de un numeral de leyes correspondientes a intervalos de influencia menores, por

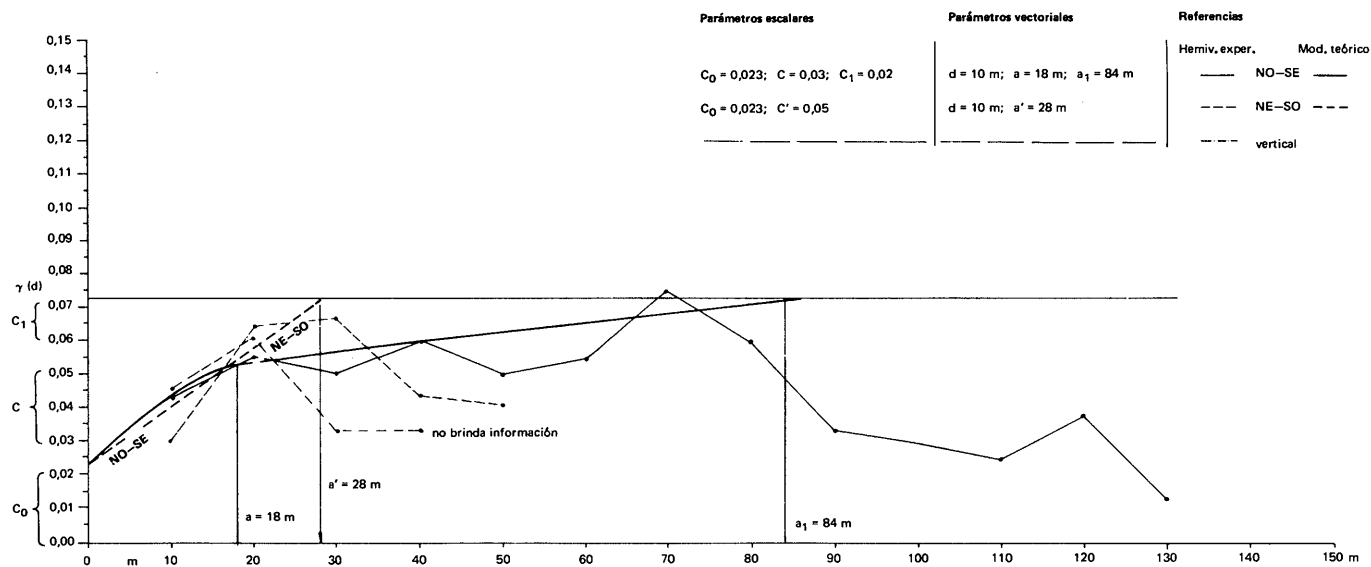


FIG.8. Megaregionalizaciones en el yacimiento "Schlagintweit", Córdoba, Argentina.

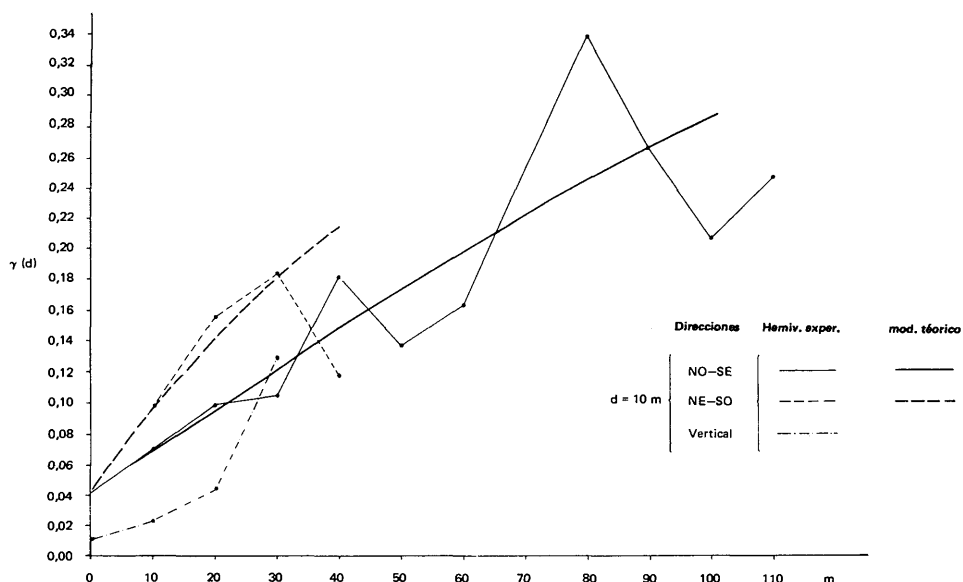


FIG.9. Hemivariograma de acumulaciones potencia-tenor. Yacimiento "Schlagintweit", Córdoba, Argentina.

ejemplo pentamétricos, pero con tenores químicamente obtenidos. Puede decirse que este examen no brindó ninguna información positiva en el aspecto hemivariográfico.

Recapitulando, se tiene que aparte de la posible miniestructura puesta de manifiesto en un efecto pepítico de  $C_0 = 0,023$ , un tanto acentuado, hay razones para suponer la existencia de otras dos estructuras que a diferencia de aquella son de carácter cuantificable: la primera de ellas, detectada por la investigación NO-SE con un valor de "alcance" de 18 m y por la NE-SO con uno de 28 m, y la segunda, señalada únicamente por la dirección NO-SE y poseedora de un "alcance" " $a_1$ " de 84 m, no confirmada según la dirección ortogonal por simple falta de información.

También en este caso aunque con menor peso informativo que en el del yacimiento "Rodolfo", podría señalarse la existencia de ciertas características "folimórficas" que una investigación de mayor detalle contribuiría probablemente a consolidar.

Es interesante señalar por otro lado que las dos medidas del "alcance" estructural de la primera regionalización mensurable mencionada, ya indican la existencia de una "anisotropía geométrica" traducible gráficamente en una elipse con elongación mayor de NE a SO.

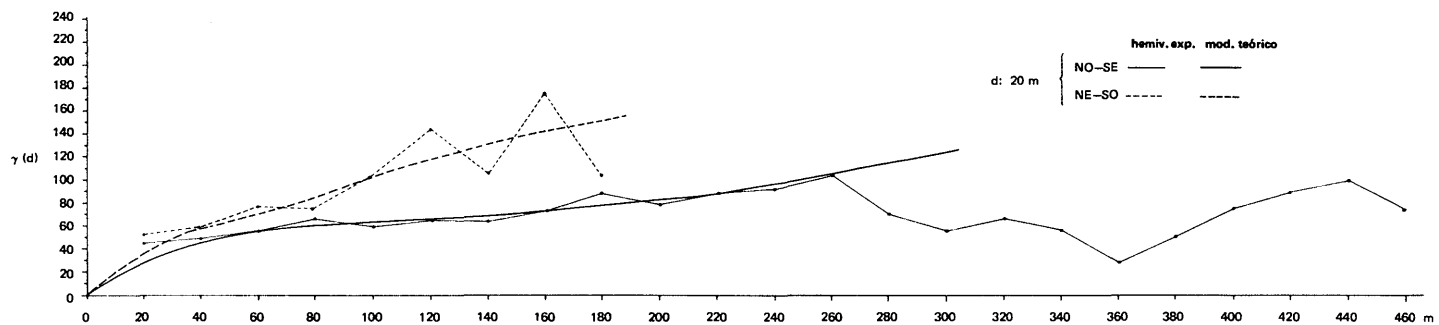


FIG.10. Hemivariograma de potencias mineralizadas. Yacimiento "Schlagintweit", Córdoba, Argentina.

Seguidamente se describirán las breves experiencias emergentes del análisis de la representación variográfica de las acumulaciones potencia-tenor (Fig. 9).

En su elaboración han intervenido los mismos sondeos utilizados en el hemivariograma de tenores, es decir los correspondientes a un breve sector del yacimiento que fuera explorado según malla de  $10 \times 10$  m.

Como puede observarse en la Fig. 9, los gráficos de acumulaciones hacen suponer, tanto según la dirección NO–SE como según la NE–SO, que la distribución parecería encontrarse en esquema “wijsiano” caracterizado por una varianza a priori al infinito y regido por la ecuación hemivariográfica: “ $\gamma(d) = 3 \alpha \ln d$ ”, de donde el único parámetro deducible es el del coeficiente de dispersión intrínseco.

La dirección vertical no brindó ninguna información aprovechable.

En lo relativo al comportamiento hemivariográfico de la dimensión “potencias mineralizadas” se aclarará que a los efectos de poder contar con el mayor número posible de datos, se utilizaron los sondeos equidistantes 20 m entre sí a lo largo y ancho de todo el yacimiento, eliminando con fines de regularización algunos impactos correspondientes al sector más densamente explorado.

Los resultados de la representación, que pueden verse en Fig. 10, indican que también en este caso el hemivariograma señala que ambas distribuciones parecen ajustarse sensiblemente a un esquema de De Wijs, por lo que caben para el caso las mismas consideraciones del punto anterior.

### 3. EMPLEO DE LOS PARAMETROS EMERGENTES DEL HEMIVARIOGRAMA EN EL KRIGISMO

Durante el proceso de una evaluación o como consecuencia de interrogantes surgidos con posterioridad a la misma, pueden plantearse determinados problemas tales como los siguientes, susceptibles de ser resueltos mediante la intervención del krigismo en sus diferentes formas y matices: a) puede suceder que en algún sector de un depósito explorado mediante perforaciones se dude de la realidad objetiva de un tenor “monstruo” (por defecto o por exceso) que desentone en forma muy acusada con respecto a los de los puntos vecinos o bien con la medianía del depósito; b) que el mismo problema se plantee con respecto a una “potencia mineralizada” sujeta a análogas incertidumbres; c) que falte un sondeo en el seno de una malla, y que se desee “integrarlo” atribuyéndole determinados valores de tenor y potencia; d) el siguiente caso que es una muy amplia extensión del último, consiste en que se considere necesario densificar artificialmente una malla de sondeos, intercalando datos deducidos de los parámetros de los realmente existentes, etc. etc.

El problema a) tiene solución mediante el “*Corrector de Krige*” (de fundamento estadístico) en base a varianzas generales válidas para la totalidad de un yacimiento, y a la certeza de encontrarse en lognormalidad y en esquema wijsiano, o bien con el “*Corrector de Matheron*” (de raíz geométrico-geográfica) en su empleo más corriente en base de las dos primeras aureolas de perforaciones que enmarcan al sondeo-problema; en este espacio no será tratado ninguno de estos dos métodos, que ya lo han sido en otros trabajos. También puede resolverse el caso mediante la aplicación del krigismo basado en los desarrollos de Matheron aplicables a un número “n” de sondeos vecinos, y ligado exclusivamente a la magnitud de las regionalizaciones existentes.

El caso b), a pesar de constituir un problema que se presenta con suma frecuencia en la práctica, resulta extraño comprobar que no parece haber sido nunca objeto de una atención muy especial, según se desprende de la bibliografía consultada.

De acuerdo a algunas experiencias comprobatorias realizadas, se ha podido establecer que si se conocen los principales hitos geoestadísticos relativos a los “espesores mineralizados” de un yacimiento, vale decir, “la varianza a priori de potencias”, “el coeficiente de dispersión intrínseco de potencias” y “los equivalentes lineales”, 1) del yacimiento, 2) del volumen de influencia de los sondeos, y 3) de la muestra, basta aplicar la conocida fórmula de Krige para moderar un espesor considerado como “anómalo” por exceso o por defecto, *con respecto a la potencia media de todo el depósito*.

De otra manera, empleando en la misma forma el “Corrector de Matheron” basado en la existencia de solo dos aureolas de sondeos circundantes ya citado, y previo cálculo de la *potencia media* de cada aureola (en este caso, la ponderación debe hacerse por los tenores) se consiguen análogos efectos moderadores, pero esta vez *con respecto a los espesores medios de los sondeos vecinos*. (Lucero M., 1978.)

Caso c) — En la resolución de este problema pueden aplicarse las dos metodologías de Matheron citadas en el caso a), es decir, la basada en las dos aureolas circundantes, y la del krigismo para un número de sondeos vecinos únicamente limitado por los valores de “alcance” variográficamente obtenidos.

Caso d) — Este problema, que puede presentarse ante la necesidad de tener que planificar una explotación, requiere para su solución, por una parte un minucioso trabajo de análisis geoestadístico de comportamientos hemivariográficos en base a los datos de la exploración realizada y por otra una voluminosa tarea complementaria de programación y computación sin cuyo concurso sería prácticamente insoluble.

La primera parte de este trabajo debe efectuarse bajo criterios eminentemente geológicos y capaces de limitar los dominios dentro de los cuales pueden aplicarse los valiosísimos aportes de las matemáticas aplicadas derivadas de la teoría de las variables regionalizadas. Un límite geológico, ya sea de orden tectónico,

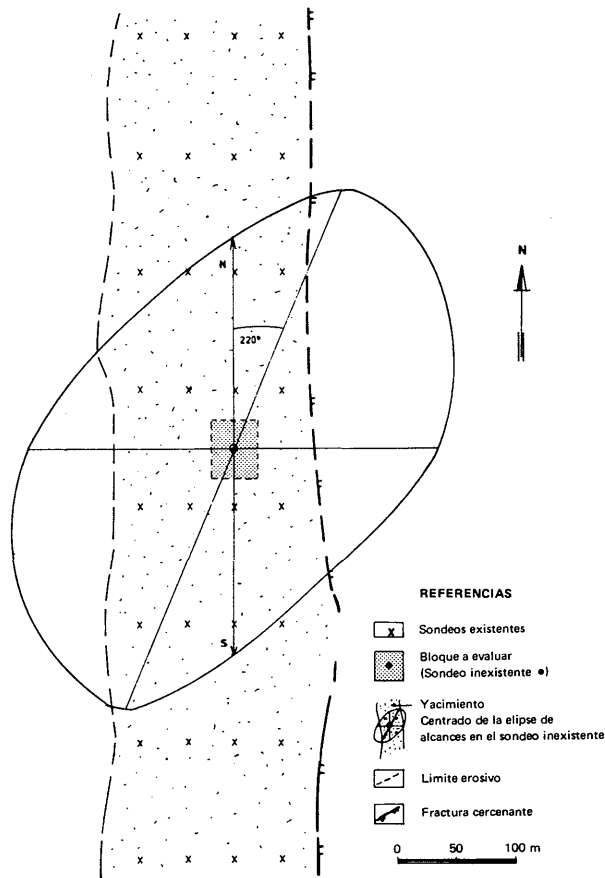


FIG.11. Ejemplo esquemático de evaluación de un bloque fuera de malla mediante krigismo con sistema Matheron. Yacimiento "Rodolfo", Córdoba, Argentina.

intrusivo, transicional etc., debe constituir una barrera infranqueable para especulaciones metodológicas de base matemática que pueden llegar a caer en el vacío por falta de continuidad de las geoestructuras que constituyen su soporte natural.

Los datos suministrados por los exámenes hemivariográficos son de dos órdenes, a saber: los adimensionados que consisten en varianzas (hemivarianzas en el caso) y sus covarianzas, y los de naturaleza vectorial, que miden en las diferentes direcciones investigadas las dimensiones de las estructuras presentes en el espacio.

Los primeros se refieren a la posición de las diversas "mesetas" o "paliers" que definen en ese campo particular las estructuras detectadas por el análisis;

puede visualizarse el diferente comportamiento de los parámetros “C” en relación a los respectivos “ $C_0$ ” — en caso de ser estos últimos diferentes entre una y otra dirección estudiadas — mediante la confección de gráficos orientados cardinalmente y contruidos a una escala cualquiera.

Los segundos cuantifican métricamente el desarrollo material de la regionalización (o regionalizaciones interencajantes en casos que como el presente se comportan folimórficamente) existente en el campo estudiado.

Los parámetros escalares que el geoestadístico extrae de los gráficos hemivariográficos constituyen una parte muy esencial de los datos en base a los cuales y por computación aplicada al “sistema Matheron” se obtendrán los coeficientes de ponderación “ $\lambda_i$ ” que finalmente proporcionarán la ley del sondeo-problema (en este caso, inexistente en el sentido material).

Los parámetros vectoriales identificados con los “alcances” que en las diferentes direcciones investigadas posee una determinada regionalización, posibilita la construcción de gráficos (circulares en casos isotrópicos o en elipse<sup>3</sup> en el de anisotropías geométricas); tales gráficos, confeccionados a la misma escala de representación del yacimiento y convenientemente centrados en el bloque cuya ley se desea estimar (Fig. 11) permiten determinar visualmente el número de sondeos capaces de ejercer una influencia fiable (zona de interdependencia de las funciones intervinientes) en la obtención de la ley buscada, de la que por otra parte se conocerá razonablemente el grado de precisión con que ha sido lograda.

En el presente caso y como puede verse en las Figs 11 y 12, el número de sondeos que intervienen en la estimación de la ley de un bloque hipotético extrareticular<sup>4</sup> es de 14, muy superior al de 8 que en el mejor de los casos puede ofrecer el empleo común del Corrector de Matheron basado en solo dos aureolas circundantes.

Una vez representados dentro de la elipse todos los sondeos intervinientes y el bloque a estimar, se une linealmente cada sondeo con el teóricamente centrado en ese bloque, y con cada uno de los restantes sondeos (Fig. 12)<sup>5</sup>.

<sup>3</sup> En el Centro de Computación (a cargo del Lic. en Física Nuclear J.P. Girardi) del Instituto de Investig. Mineras de la U.N. de S. Juan, dirigido por C. Rudolph, donde el suscripto tuvo la oportunidad de trabajar en equipo, se encuentran en etapas de investigación interesantes problemas de krigismo aplicados al yacimiento “Bajo de la Alumbra-Calamarca”, que serán objeto de una tesis doctoral.

<sup>4</sup> Se ha supuesto el caso de que en el yacimiento “Rodolfo”, explorado según una malla de 100 X 40 m se haya presentado la necesidad de densificar la información en base a su transformación en otra red de 50 X 40 m; se ha escogido para efectuar el krigismo, la regionalización más conspicua, registrada por tres diferentes direcciones de investigación y poseedora de una dimensionalidad megaescalar de orden 16.

<sup>5</sup> Procediendo con un rigorismo metodológico no justificado por el carácter meramente expositivo de estas aproximaciones, habría que rebatir la capa mineralizada a la horizontal, o corregir los “alcances” de posición no meridiana a los efectos de deformar convenientemente la elipse.



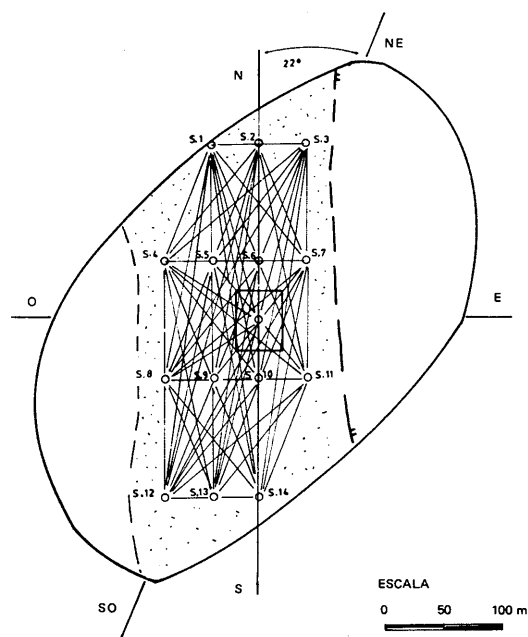


FIG.12. Trazado de las líneas sondeos-bloque y sondeo-sondeos. Yacimiento "Rodolfo", Córdoba, Argentina.

De esta manera, el sondeo 1 se une con el del bloque a estimar y con los sondeos 2, 3, 4 ... 14, calculándose luego mediante computación el valor de las covarianzas de las siguientes relaciones:

- 1) *muestra-muestra* (sondeo 1- sondeo 2, sondeo 1- sondeo 3, etc. etc.);
- 2) *muestra con respecto a sí misma* (sondeo 1- sondeo 2, sondeo 2- sondeo 2, etc), valor que por identificarse con el del parámetro "C" de "meseta" se mantendrá invariable para todo el conjunto;
- 3) *muestra-bloque* (sondeo 1-bloque, sondeo 2-bloque ... sondeo 14-bloque),

calculándose por último los "coeficientes ponderadores lambda", todo lo que conduce finalmente a la estimación de la ley del bloque-problema. Esta operación debe repetirse para cada cuadrícula (bloque) hasta completar las exigencias impuestas por la nueva malla de sondeos que reemplazará a la primitiva.

En los casos de cuerpos mineralizados de tipo "amas", en los que aparte de las direcciones horizontalmente investigadas se cuenta con un buen examen según la dimensión vertical, el problema del krigismo se torna particularmente

complejo; en este caso, en lugar de una elipse capaz de satisfacer un esquema geológico estratiforme, será necesario hacer jugar un “elipsoide” que desplazándose de arriba hacia abajo y en todas direcciones según la horizontal y conforme a la necesidad de ir siendo centrado en cada bloque-problema, dará lugar a un número excesivo de cálculos, circunstancia que solamente un buen apoyo de ordenador puede obviar.

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# HISTORY OF THE EVALUATION AND EXPLOITATION OF A GROUP OF SMALL URANIUM MINES IN PORTUGAL

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## Abstract

### HISTORY OF THE EVALUATION AND EXPLOITATION OF A GROUP OF SMALL URANIUM MINES IN PORTUGAL.

In the period 1945–1962 a multiple small-scale uranium mining project was successfully operated in Portugal by the Companhia Portuguesa de Radium Lda, the Urgeiriça Mine and some 28 smaller mines produced ore between 1951 and 1962 and provided uranium concentrates for export under an agreement between the Portugal, the United Kingdom and the United States Governments. The uranium deposits occur in near-vertical fractures in the granites of the Beiras mostly having a NE-SW strike trend. The veins are the siliceous-pyrite-galena type with red jasper and black and white quartz gangue and much wall rock alternation and shearing. The primary uranium mineral is pitchblende and the secondary minerals tobernite and autunite occur in the near-surface zones. Of the 63 concessions controlled in 1945, the Urgeiriça Mine, dating from the radium period, was by far the biggest, but by mid-1946, after a very rapid preliminary evaluation, it was concluded that a total "possible" ore reserve of 1080 tons  $U_3O_8$  might exist in the concessions. On this basis an agreement was reached with the Portuguese Government for the production and export of between 120 and 150 t  $U_3O_8$  per year for a twelve-year period up to the end of 1962, and for a maximum of 1325 t  $U_3O_8$ . Production commenced on a regular basis at the beginning of 1952 and the total production quota of 1325 t  $U_3O_8$  was produced by 31 March 1962. Since Urgeiriça provided 50–80% of the total ore throughout the period the main services were therefore located at Urgeiriça while all housing, power units, mechanical equipment etc., for the smaller mines had to be mobile. The Company innovated several techniques which are now widely used in uranium mining, e.g. radiometric borehole logging, tunnel-type radiometric selector systems for mine cars, heap leaching of uranium and in-situ leaching of uranium. Success in the whole project was due to reliable ore reserve estimates based on classical methods, tight management control, standardization of equipment and methods and also the flexibility and willingness of management to take full advantage of what were then innovations such as heap leaching, radiometric sorting etc. Good organization was the key to the success of the whole project.

## INTRODUCTION

This paper is based on work done in Portugal by the Companhia Portuguesa de Radium Lda, Minas da Urgeiriça, Canas de Senhorium, between 1945 and 1962.

The Company owned or controlled a group of uranium-radium mining concessions and claims in the north-central part of Portugal, and from among these concessions the Urgeiriça mine and a group of smaller mines were brought into production between 1951 and 1962 and provided uranium concentrates for export. The basis of operations was an agreement and contract between the Portuguese Government and the Company, signed in 1949, renewed in 1955 and finally expiring in 1962.

### Location

Although the work described took place between 1945 and 1962 the paper is presented here in the belief that many of the evaluation and mining techniques and problems are relevant to similar occurrences which may be exploited in the future.

The uranium concessions are located in an area of about 10 000 km<sup>2</sup> in the provinces of the Beira Alta and the Beira Baixa. The principal mine, Urgeiriça, is about 230 km north-north-east of Lisbon and 20 km from the district capital town of Viseu. It is located on the main railway line and on one of the highways to Spain and France. (Fig.1.)

Urgeiriça lies to the west of the dominant topographic feature of the region, the Serra da Estrela mountain range which reaches an altitude of 1991 m and has the uranium deposits located to the west, north and east of it.

### Geology [Ref.1]

The uranium deposits occur in near-vertical fractures in the granites of the Beiras, which are part of the granites of the Iberian Meseta. Most of the uranium-bearing fractures have a NE-SW strike trend, parallel to the axis of the Serra da Estrela mountain range. Four districts of greater concentration of uranium-bearing fractures exist: (1) the Urgeiriça district to the west, (2) Trancoso district to the north, (3) the Guarda district and (4) the Belmonte-Sabugal district to the east of the mountains (Fig.1). The fracture pattern of each district is complex and both shear fractures and tension fractures are found and may be uranium bearing.

The veins vary in extent from some tens of metres up to several kilometres in length. They are of the siliceous-pyrite-galena type of uranium vein and have red jasper, ferruginous white quartz and black and white banded quartz as the

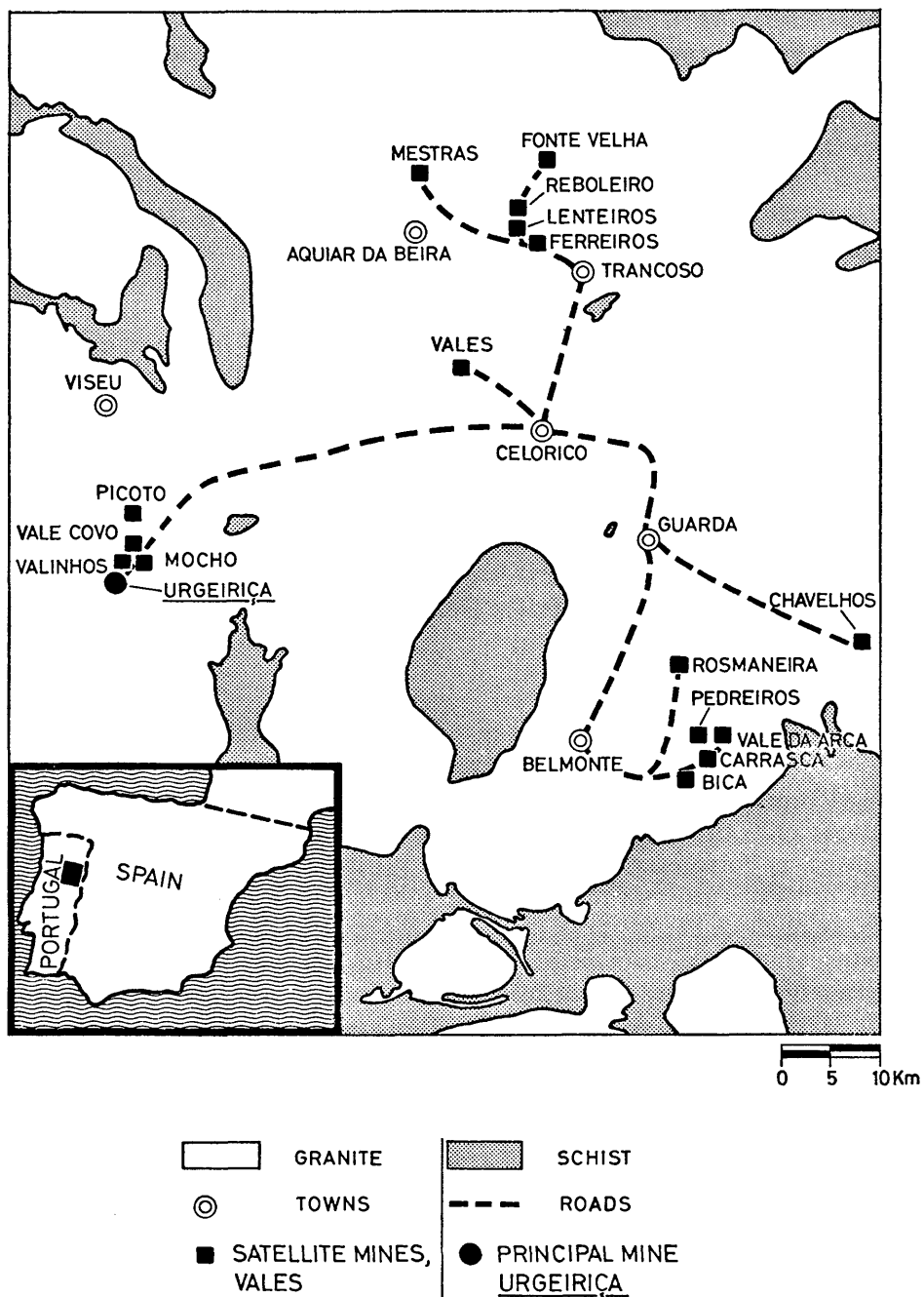


FIG.1. Uranium mining district in Portugal, 1951-1962.

main gangue rock. Considerable strike fault movement is shown by included altered silicified granite, fault clays and breccias. Wall rock alteration; sericitization, haematitization and kaolinization, together with strike faulting, produce many sections of weak ground, making mining difficult and at times dangerous. The primary uranium mineral is pitchblende, occurring principally in a microbotryoidal form, either as a dissemination in the red jasper or as a "sooty" form, loosely adhering to cracks and vughs in the veins. At the surface and in the near-surface zone, secondary uranium minerals, principally the calcium or copper hydrous uranium phosphates, are found. Other metallic minerals in the veins are haematite, iron pyrites, galena, sphalerite, chalcopyrite and small quantities of arsenopyrite.

The veins containing the ore-bodies may vary in width from 0.20 m up to multiple veins of 5 to 10 m, but the average is about 1 m. The angle of dip varies from 60° to vertical.

### History

Soon after the discovery of radium at the end of the nineteenth century, the Portuguese uranium-radium deposits were receiving attention and by 1907 a French mining group was working in the Sabugal area and another group was soon working in the Guarda area.

The date of the first uranium-radium concession, granted by the Portuguese Government Mines Department, was September 1909. The Urgeirica deposit was discovered in 1912 and the mining concession was granted in 1915.

The Urgeirica mine was active from 1912 up to 1926 and then for three years there was no activity in the uranium-radium mines of Portugal, but in August 1929, two British subjects privately bought Urgeirica and founded the Companhia Portuguesa de Radium Lda (CPR). Initially, interest was confined to Urgeirica but over the following years some other uranium concessions were acquired. From 1929 to 1938 the Company continued to operate Urgeirica under extreme difficulties, both technical and financial, and remained the only company attempting to produce radium in Portugal.

In 1942, the majority shareholder of the company was bought out by the United Kingdom Commercial Corporation Ltd., the United Kingdom Government war-time purchasing agency in Portugal. Under this control, the 14 mining concessions then owned were not operated but were maintained on a care and maintenance basis, but in late 1944 a more inspired interest was taken in CPR by the United Kingdom Government agency responsible for uranium supplies. A policy of buying out the minority shareholders of CPR and of acquiring other uranium concessions was pursued during the first half of 1945. This policy was completely successful and by August 1945 the Companhia Portuguesa de Radium Lda, was wholly owned by British Government interests and a total of 63 uranium concessions had been acquired.



## Basis of contract with the Portuguese Government

Of the 63 uranium mining concessions controlled by CPR in 1945, only three were then in operation and available for underground inspection. The Urgeiriça Mine, dating from the radium period, was 270 m deep with 13 levels developed and largely stoped out over a 300-m strike length. The solution of a fault problem revealed the possibility of extending the mine for a further 400 m. By 1946 it had been concluded that there might be about 800 t  $U_3O_8$  at Urgeiriça and a few hundred more tons in the other properties. It was on this basis of a total reserve of about 1080 t  $U_3O_8$  that negotiations were initiated with the Portuguese Government and plant construction started in 1949. The objective was to produce between 120 and 150 t  $U_3O_8$  annually over a 12-year period. The agreement signed was mandatory that no less than 120 t  $U_3O_8$  be produced each year and that a total of 1325 t  $U_3O_8$  be produced before 31.12.1962.

The contract also contained some clauses covering extremely complex arrangements in regard to which concessions could be in exploration or production at any one time. For example, only 10 concessions could be in 'production', i.e. stoping, at any one time, and before a worked-out mine could be substituted the Portuguese authorities had to be satisfied that no ore remained in that property. The complications which thus resulted in the overall technical policy are not dealt with in this paper.

## Construction and development policy

Because of the higher reserve tonnage and the existence of an infrastructure from the radium period there was no difficulty in deciding that Urgeiriça would be the centre of the organization. It was planned that the Urgeiriça ore reserve should be made to last throughout the contract period to 1962 and that development of the satellite mines should be as rapid as possible so that they should supply between 20 and 40% of the total ore milled in that period.

In planning mining production it was assumed that Urgeiriça would provide 80% of the total ore in the early years but that this would fall to 50% in the later years. With an assumed plant recovery of 90% and an average initial grade of 0.35%  $U_3O_8$  the plant was originally planned to have a capacity of 100–200 t ore per day but this was later expanded to 150 t ore per day.

The policy for the outlying mines was based on the assumption that their individual lives would be shorter than the contract period and therefore all housing, power units and mechanical equipment should be mobile. Sectional housing panels were designed to give houses of different sizes as required. Power units were standardized to facilitate maintenance and stocks of spare parts. Design of shafts, raises, winzes and galleries were standard and used in all the mines. Timber sets were made in Urgeiriça carpenters' shop and transported to the other mines.

The auxiliary services were centred at Urgeiriça and included the main mechanical and electrical workshops, carpenters' shop, store, office, mining surveyor and geological sections and were built on a scale sufficient for the requirements of the whole organization.

The chemical treatment plant was built at Urgeiriça and also the social services which included housing for 500 people, a medical post, hospital school, shop, social club and recreation facilities. As far as possible the social services provided facilities for the other mines.

The controlling board decided that the cost of product should be the equivalent of US \$6.66/lb<sup>1</sup> U<sub>3</sub>O<sub>8</sub> and the management was requested to ensure that this figure was not exceeded in the annual cost of production. The Company was also under very strong instructions to fulfil a minimum quota of production which was set at 120 t U<sub>3</sub>O<sub>8</sub> per year.

### Organization of the company

The local board of Directors, resident in Lisbon, comprised five members, three being British subjects and two Portuguese. The general manager of the Company from 1943 until the end of the contract in 1962 was resident at Urgeiriça mine and directly responsible to the board in Lisbon. The senior staff of the Company consisted of an assistant general manager, a chief mining engineer, the superintendent of the treatment plant, the chief geologist, the chief of the laboratory, the chief accountant and a chief mechanical and electrical engineer. The general manager was a Portuguese subject and the assistant general manager was British. The other senior posts were evenly divided between British and Portuguese subjects.

## GEOLOGICAL DEPARTMENT AND THE EVALUATION OF THE DEPOSITS

### General

The Geological Department planned, operated and assessed all surface exploration and diamond drilling, proposed mine exploration programmes, recorded geology and sample results in all mining work, advised on the geology of stopping mines and computed ore reserves for all properties. For convenience, the servicing of all the company's electronic equipment was also done by the geological department.

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<sup>1</sup> 1 lb = 0.453 kg. The lb unit is used frequently throughout this paper.

On all matters concerning ore reserves the chief geologist was directly responsible to the general manager and helped to formulate policy arising therefrom and related to the government contract.

On matters of mining geology and grade control the department provided information direct to the chief mining engineer.

### **Surface prospecting**

The early mapping (1945–1948) of the Company's concessions had been concerned with the more significant radioactive surface zones on each concession. After 1951 the whole surface area of all concessions was re-mapped in detail. Each concession, amounting to half a square kilometre in area, was surveyed by a geologist and surveyor and accompanied by a trenching team. Topography, geology and radiometric readings were recorded on a final 1:1000 scale plan and a written report on the geology and ore potentialities was prepared. Areas of greater interest were mapped on 1:500 or 1:200 scales. No work was allowed beyond the Company's concession or claim boundaries.

### **Diamond drilling**

Seven Craelius diamond drills were owned by the Company and directly operated by the Geological Department. Two of these were specifically for underground drilling and the other five for surface drilling. The geologists were responsible for hole planning, logging and assessments and under their supervision two junior engineers were responsible for the mechanical operation of the drills.

Over the period of eleven years, 1950 to 1960 (incl.), 62 055 m were drilled on 33 different properties. The annual meterage drilled reached a maximum in 1955 and thereafter decreased to the end of drilling in 1960. This pattern conformed to the policy needs of the company and allowed for mine development and stoping to be planned and completed before the contract terminal date in 1962.

Following surface mapping and trenching prospection, drilling was done on concessions where there was little or no earlier mining work. The ore-bodies at Valinhos, Vale da Arca, Môcho, Mestras and Pedreiros were principally discovered and defined by diamond drilling. On several other concessions, such as Bica, Carrasca and Reboleiro, surface drilling in conjunction with mining revealed the extent of the ore-bodies.

Development drilling was done on or below the extensions of many known ore-bodies, helping to define them and to estimate ore reserves so that the mining department could plan technical policy.

As the Company took over many old mines from the radium period for which no plans existed, the re-opening was fraught with risks and the drills were used on several of these small mines to locate old galleries, shafts and stopes before planning how to re-open the workings.

In hole assessment, sludge and core samples were taken, but the most important tool was the calibrated borehole Geiger counter for recording radio-activity. The 19-mm borehole probe used inside the Craelius drill rods was first developed in 1951 by AERE Harwell, England, at the suggestion of the CPR geologists. The calibrated Geiger counter results were checked on many later occasions as assessed ore shoots were mined out.

### **Geological mine plans**

The permanent mine plans for all properties were based on a block system with a unit of 200 × 120 m. Co-ordinates were normally a local system arranged near parallel to the average vein strike in each deposit. Nomenclature of the blocks was arranged with an alphabetical designation, generally NW—SE and a numerical designation NE—SW. Identification was completed by the level number. Three scales were in general use — 1:200 for the geological and assay plans which were separate but superimposable; 1:500 for the mine level plans used in planning work; and 1:1000 for general plans and ore reserve reports. Where necessary, groupings of blocks were made for the smaller-scale plans. Transverse sections along each 20-m interval main co-ordinates were also standard practice on the 1:500 scale.

### **Sampling and ore reserve estimation**

Sampling of mine development was done by teams of samplers, technically under the control of the chief geologist. All samples were channel samples, 20 cm wide, 3 cm deep and taken across the full width of the vein structure at 1-m intervals in all galleries, raises and winzes. Each sample was normally between 10 and 25 kg in weight and was prepared in the normal way by crushing, quartering, grinding and splitting. Assaying was done by a beta counter installation in the laboratory and a selection of samples were check-assayed by chemical methods for control purposes and to note any disequilibrium in the ores.

The normal ore reserve calculation methods were used. The ore cut-off was 0.15%  $U_3O_8$ , but in practice only zones with a minimum of five consecutive metres giving an average of over this grade were considered. Random high samples were cut to twice the average of the whole zone and the average re-calculated. A sampling factor reduction of calculated grade was determined and used for each individual mine. A dilution factor of 15% was used for all mines.

## Ore reserve classification system

### *Proven or measured ore*

This is ore exposed and sampled on four sides. The maximum vertical distance between levels, acceptable for this class, was 40 m and the horizontal distance between raises was 30 m.

### *Probable or indicated ore*

This is ore exposed and sampled on one, two or three sides, where there was reason to assume continuity from consideration of the nature of neighbouring ore blocks of similar type and comparable grade; and also ore defined by diamond drill holes and evaluated by samples of sands or cores or from calibrated Geiger counter borehole readings.

### *Possible or inferred ore*

This is ore for which quantitative estimates were based largely on a general knowledge of the geological character of the deposit in circumstances where no systematic sampling had been done and measurements might be approximate. Estimates in this category were regarded as no more than informed guesses indicating the degree to which the deposit merited exploration and development.

### *Ore reserves*

Ore reserve estimates were made at the end of each year for each property from 1946 onwards. At the end of 1946 the total ore reserves were stated as 1980 t  $U_3O_8$ , but only 80 t of this was in the proven/probable classes. It was on this basis that negotiations were started for the government contract, and when the agreement was signed in mid-1949, the total reserves were still at about the same figure although the proven/probable class had been increased to 375 t, mainly due to development work at Urgeiriça.

In late 1951 when production started, the total reserves were stated as 1700 t  $U_3O_8$  of which 580 t were then in the proven/probable class. The start of diamond drilling in 1950 and the increased mining exploration saw further increases to a maximum total figure of 2070 t  $U_3O_8$  in 1954. In the later 1950s, some re-assessment was necessary when drilled properties were opened up and mined, and as the poorer concessions were abandoned.

The ore reserves left in the mines at the end of the contract were estimated to be 540 t  $U_3O_8$ , 225 in the proven/probable class and 315 in the possible class. The total  $U_3O_8$  sent to the plant in direct ore mined amounted to 1360 t which,

TABLE I. PRODUCTION FROM ALL MINES, 1951–1962

Name of mine	Period of production	Distance from Urgeiriça (km)	Ore milled (t)	Av. grade (% $U_3O_8$ )	Total $U_3O_8$ produced (t)	Av. cost $U_3O_8$ (US \$/lb)
Urgeiriça	1951–62	—	241 060	0.305	766.9	4.65
Bica	1951–61	130	74 050	0.271	206.0	5.63
Vale da Arca	1956–61	136	32 130	0.265	80.1	5.04
Carrasca	1952–60	132	21 730	0.444	90.5	4.19
Valinhos	1955–60	2	13 000	0.243	41.8	8.64
Reboleiro	1951–55	80	12 920	0.257	26.6	13.65
Rosmaneira	1951–56	115	6 640	0.208	112.8	15.45
Ferreiros	1954–62	80	6 120	0.273		
Fonte Velha	1951–56	90	2 750	0.238		
Mestras	1955–58	75	2 650	0.294		
Picoto	1951–52	6	1 750	0.500		
Lenteiros	1951–55	80	1 520	0.380		
Mocho	1951–54	3	1 450	0.221		
Pedreiros	1958–60	130	1 270	0.281		
Vales	1957–60	40	810	0.185		
Vale Covo	1958–60	3	980	0.175		
Chavelos	1952–53	150	260	0.144		
Others <sup>a</sup>	1954–62	—	45 910	0.164		
Totals:			467 000	0.306	1324.7	5.70
Averages:			(44 475)	(0.306)	(126.2)	(5.70)

<sup>a</sup> Includes ore purchases, heap leaching, neutralization slimes, waters, sands, etc.

together with the 540 t left in situ, amounts to 1900 t. These figures do not, however, include the low-grade natural leaching ore nor the  $U_3O_8$  recovered from mine water, and other sources.

The total  $U_3O_8$  contained in the CPR concessions is therefore estimated to have been approximately 2000 t and thus the estimates at the beginning of the production period proved to have been sufficiently accurate to justify all the technical policies which were adopted by the management.

## PRODUCTION PERIOD 1951–1962

## Summary

The production period is best summarized by a tabulation of the final results as shown in Table I. Production commenced on a regular basis at the beginning of 1962 after a running-in period of several months in late 1951. The whole production operation was closed down on 31 March 1962 and thus the production period was effectively just under 10.5 years.

From Table I it can be seen that an average of 44.475 t of ore were milled each year with an average grade of 0.306%  $U_3O_8$  and an average of 126.2 t  $U_3O_8/a$  were produced. The average cost within the working period was US \$5.70/lb  $U_3O_8$ . The tonnage came from one small- to medium-sized mine (Urgeiriça) and 21 small mines – mainly very small. Favourable cost figures were held on only four of the mines, but these were all the larger tonnage mines and resulted in a satisfactory overall cost substantially below the company's target figure of US \$6.66/lb  $U_3O_8$ . It could have been argued that the smaller more costly mines should never have been developed, but this was a necessity as the annual production quota of not less than 120 t  $U_3O_8/a$  was an equally firm contract commitment. Grade appeared to be a much more important factor than distance from the treatment plant in reaching good cost figures, e.g. the Carrasca Mine, although 132 km from Urgeiriça, had the best cost per lb figure (US \$4.19) of the whole group because its average grade was 0.44%  $U_3O_8$  as compared with the overall average of 0.306%  $U_3O_8$ .

The contract figure of 1325 t  $U_3O_8$  was produced nine months before the limiting date of 31.12.1962.

A brief description of some of the more important of the mines follows:

(1) *Urgeiriça Mine (Urgeiriça and Ucha mining concessions) 1945–1962*

The Urgeiriça mineralized fissure zone consists of various strike faults and breccia zones with interconnecting tension fractures and branch veins. The strike is N 62° E and the dip is at a very steep angle to the south-east. The vein width was normally about one metre but widths of up to 10 m were reached locally and small stockwork zones existed. Except in the near-surface weathered zone, where secondary uranium mineralization was found, the ore consisted of finely disseminated pitchblende in red-black jasper veins or in veinlets in altered wall granite [2]. The high degree of brecciation, strong strike faulting and extensive kaolinization and sericitization of the wall-rock granite made this difficult ground to work and required close timbering in both development and stoping work. Three transverse faults divided the ore-body and several separate, near-vertical ore shoots were defined and worked.

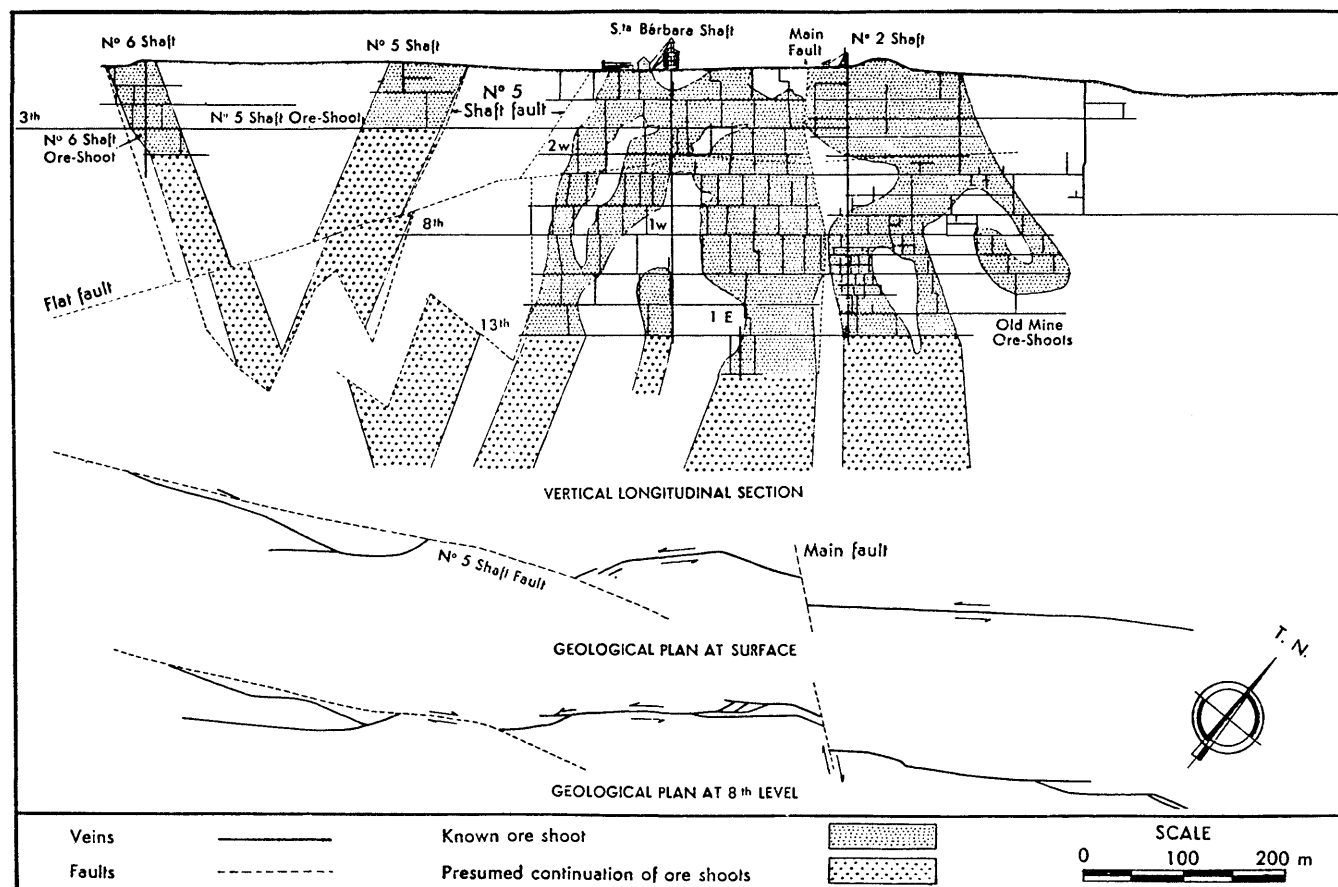


FIG.2. Urgeiriça mine. Longitudinal section and plans showing distribution of ore shoots and principal structural features.



A longitudinal section and plan of the mine in 1956 is given in Fig.2. The final explored length was 1200 m and the maximum depth was 380 m. The main fault divided the pre-1945 workings from the later workings.

Three vertical shafts served the main mine. The Sta. Barbara shaft, which was started in 1949 and sunk as required for development work, reached 380 m depth in 1959 and was the main haulage and personnel shaft.

In the Sta. Barbara shaft area, mineralization was more or less continuous over a distance of 270 m from the surface to the 4th level. Below the 4th level the ore-body split into a south-west and a north-east ore shoot about equidistant from the shaft (Fig.2). From development work and drilling it became evident that these ore shoots were narrowing and becoming impoverished below the 14th level.

Other areas which produced ore were the No. 6 shaft area, the No. 5 shaft, 3rd to 10th level zone, and the old mine. Drilling in the old mine showed that several richly mineralized veins, parallel to the main structure had been missed in the radium mining days and also that areas which were then uneconomic were now exploitable. Development of the old mine areas was slow because of the danger of re-forming the old levels and also because of the naturally weak ground.

Development of the 16th level was completed by mid-1961, after which no further development work was done because of the approaching end of the contract. Stopping by horizontal cut and fill started in 1951 and ceased in March 1962 at which time all the developed ore had been stoped. During the life of the mine from 1945 to March 1962, a total of 23 500 m of development works were done, 189 750 m<sup>3</sup> were stoped and 241 060 t ore were produced.

## *(2) Bica Mine: 1950-61*

The second biggest mine in the group, Bica, is situated east of the Serra da Estrela mountains about 130 km by road from Urgeiriça. For most of the period this mine served as the company's headquarters in the Belmonte-Sabugal area, supervised by a resident engineer.

The mine is located on a north-east trending red jasper vein, 0.75 to 2.00 m in width and dipping north-west at 70°. The mineralization was fine-grained pitchblende in the deeper zones and secondary uranium minerals near the surface.

Bica was one of the old radium mines and a shallow adit, shaft and old mined stopes had to be by-passed when mining exploration work started in 1950. In 1952 a vertical main haulage shaft was started after initial exploration and drilling had shown the probable continuation in depth of the ore-body. In mid-1955, horizontal cut and fill stoping was started above the second and third levels.

In March 1960 all development work was stopped because of the approaching end of the contract and at that time both shafts were 240 m deep and the 7th level

had been developed over a strike length of 230 m. Stopping between the 7th and 6th levels had been completed when the mine was closed down in November 1961.

During its working life for the Company, 5150 linear metres of mine development and 63 340 m<sup>3</sup> of stopping were done, and 74 050 t of ore were produced at Bica. A large tonnage of low-grade natural leaching ore was also produced.

(3) *Vale da Arca Mine: 1952–1953; 1956–1961*

Vale da Arca mine is situated six kilometres north-east of Bica and is located on the northeast continuation of the Bica vein. The vein zone, dipping at 60° to the north-west, consisted of three parallel veins with interconnecting veinlets giving a total mineralized width of ten metres.

A small amount of work had been done in the radium period, and the early mining exploration work by the Company in 1952–53 in opening up the old adit was not encouraging. It was only as a result of the diamond drilling programme started in early 1956 that the full extent of the ore-body was discovered. Mining development was re-started and three levels were developed over a strike length of 150 m to a depth of 80 m. Stopping started in late 1958 and the ore-body was finally worked out in June 1961. Stopes of 10–12 m width in very weak ground were worked.

(4) *Carrasca Mine: 1952–1960*

Carrasca is also located near Bica, being two kilometres north-west of it, but is not on the same vein system. A series of small ore-shoots were located on a group of narrow jasper and quartz veins. The mine was worked in the radium period and some old mined stopes had to be by-passed when mining exploration started in 1952. Stopping started in 1954 and continued until the mine was closed down in 1960. Although narrow, the veins contained sections with very rich values and routine samples of up to 13.0% U<sub>3</sub>O<sub>8</sub> were recorded. A maximum depth of 170 m and a strike length of 220 m on four different veins was developed. Drilling and mine exploration showed that no further ore-bodies could be expected and the mine was closed down in April 1960 and all equipment and installations removed.

(5) *Valinhos Mine: 1955–1960*

The Valinhos mining concession adjoins and lies north-east of the Urgeiriça concession. Although the mine is located on the Urgeiriça vein structure, the workings remained separate. The ore-body was unknown in the radium period

and was discovered in 1952 when systematic drilling along the north-east continuation of the Urgeirica vein found ore under an alluvial filled valley. The ore-body was completely outlined by drilling before mining development started in 1955. Two vertical shafts at the extremities of the ore-body were sunk to reach a depth of 130 m and four levels were developed over a strike length of 210 m. Stopping started in the spring of 1957 and continued until the end of 1960.

*(6) Reboleiro Mine: 1946–1955*

Reboleiro is located in the Trancoso area about 52 km north-east of Urgeirica and 80 km distant by road. The ore shoots were contained in a group of narrow, steeply dipping, black and white quartz veins forming part of a complex tension fracture pattern. Near-surface mineralization was secondary, but in depth the primary mineralization consisted of a loosely adhering sooty pitchblende in cracks and vughs in the quartz.

This was one of the two properties purchased by the company in 1944–45 and small-scale active mining exploration work was taken over in December 1945. Mining exploration and development were done on eight different veins in four separate groups of workings. The most extensive workings were at South Shaft where a depth of 110 m was reached and five veins were worked. Stopping started in late 1951 and continued until the exhaustion of all ore-bodies in mid-1955. Drilling had shown that no further ore could be expected and thus all equipment and installations were removed in late 1955 and the mine abandoned.

## General

The six mines which have been briefly described produced 84.5% of the ore tonnage and 91.5% of the  $U_3O_8$  during the whole protection period. In addition, smaller mines at Rosmaneira, Ferreiros, Fonte Velha, Mestras Picoto, Lenteiros, Mocho, Pedreiros, Vales, and Vale Covo produced ore from stopping and development operations (see Fig.1 and Table I).

Exploration and development work was also done on 13 other properties and a few of them, such as Chavelos, provided a certain amount of ore from development work but no mine stopping was done. The annual variations in the total work done are shown in Table II.

Development work on the 29 properties amounted to 52 190 m and stopping on 16 concessions totalled 364 860 m<sup>3</sup>. Stopping and development work on 20 concessions produced 421 100 t of ore.

The variations in annual totals (Table II) reflect both general policy and technical considerations. Mining development increased each year up to 1954,

TABLE II. ANNUAL FIGURES FOR DIAMOND DRILLING, MINE DEVELOPMENT, STOPING AND ORE PRODUCED FROM UNDERGROUND MINES

Year	Diamond drilling (m)	Mining development (m)	Stoping (m <sup>3</sup> )	Ore produced (t)
1945–1950	2 123	8 170	—	3 955
1951	3 583	3 412	2 890	4 723
1952	5 871	3 990	21 830	31 302
1953	6 398	5 020	30 240	38 515
1954	6 592	5 225	21 300	35 075
1955	8 663	2 953	35 350	35 135
1956	7 767	4 275	32 815	39 320
1957	6 698	5 980	36 430	44 140
1958	6 063	5 850	42 840	50 645
1959	5 941	4 905	50 800	51 535
1960	2 356	1 855	47 080	42 805
1961	—	522	37 975	38 540
1962	—	33	5 310	5 410
Totals:	62 055	52 190	364 860	421 100

but uncertainties regarding the renewal of the government contract in 1955 brought about a marked decrease in mining exploration and development in that year. After renewal was assured, increased development was done to secure the ore tonnage required for the second half of the contract. The progressive decrease in average grade also meant that more ground had to be developed in the later years to meet tonnage requirements. Both the cubic metres stoped and the tonnage of ore produced reached maximum figures in 1959. The close-down operation started in April 1960, two years before the end of the contract, and this was immediately reflected in a reduction of all mining work. From then to the end of the contract the only mining work done was that which was calculated to give the exact ore tonnage requirements. During this period the estimates were very complex; it was necessary to have sufficient ore to provide an exact tonnage figure to be achieved two years ahead, but no unnecessary development work was to be done. The decision not to develop the 17th level

TABLE III. ORE PROCESSING PLANT FIGURES

Year	Milling rate (dry t/d)	Feed grade (% U <sub>3</sub> O <sub>8</sub> )	Recovery (%)	Cost per dry ton milled (US \$)	Cost per kg U <sub>3</sub> O <sub>8</sub> (US \$)
1952	85.2	0.37	94.8	6.17	1.74
1953	112.0	0.35	88.5	4.78	1.48
1954	110.5	0.35	93.2	5.11	1.48
1955	104.5	0.34	92.3	4.77	1.45
1956	113.3	0.32	92.5	4.72	1.51
1957	133.8	0.27	89.2	4.31	1.68
1958	155.0	0.26	90.4	4.14	1.68
1959	154.5	0.27	90.1	4.18	1.62
1960	141.3	0.25	89.6	3.99	1.67
1961	142.1	0.23	90.6	4.51	1.99
3 months only	94.1	0.22	89.6	5.82	2.55
Averages:	125.2	0.306	91.0	4.60	1.64

at Urgeiriça was at times felt to be risky but in the end the stopes above the 16th level were exactly worked out on the last days of the tonnage requirement. At Bica the decision not to develop the 8th level was also taken nearly two years before the end and in that case the stopes above the 7th level were worked out in November 1961 so that from then onwards Urgeiriça and Ferreiros and some other sources had to be relied on for the terminal tonnage requirements.

### Ore grade

In the original planning in 1949–50, when only 375 t U<sub>3</sub>O<sub>8</sub> in proven and probable ore were in sight, the estimated average grade for ore to the plant was 0.35% U<sub>3</sub>O<sub>8</sub>, and at that time a provisional average cut-off grade of 0.20% U<sub>3</sub>O<sub>8</sub> was assumed. The average grade of ore from each mine is indicated in Table I and the annual average grade of ore entering the plant is given in Table III. As shown in Table III the estimated average grade was fairly well maintained up to 1956. The sharp fall occurred between 1956 and 1957 when the averages were respectively 0.323% and 0.274% U<sub>3</sub>O<sub>8</sub>. The grade then progressively decreased to 0.22% U<sub>3</sub>O<sub>8</sub> in the final year.

In the early years the main source of ore was Urgeiriça and Ucha but the high-grade ore from Carrasca was an important factor in keeping up the average grade. Development on the levels below the 12th in Urgeiriça had indicated that a fall in grade was to be expected from 1957 onwards and the exhaustion of the rich ore in Carrasca also contributed to the lower average grades. The average grade of all mine ore over the whole period was 0.31%  $U_3O_8$ .

The cut-off grade was changed to 0.15%  $U_3O_8$  at an early stage in the production period. Selection at shaft head was practised at all mines and a division into ore (plus 0.15%  $U_3O_8$ ), natural leaching ore (0.05% to 0.15%  $U_3O_8$ ), and waste (0.05%  $U_3O_8$ ) was practised. The low-grade, heap-leaching ore did not go to the plant but was treated in a separate process described below.

### Organization of mine services

#### *Mining organization*

The chief mining engineer and his assistant were resident at Urgeiriça but visited each of the mines once per week making full underground inspections in each case. The category and number of staff in resident control of each mine depended on the size and complexity of the operation but the larger mines were under the control of a resident engineer.

#### *Grade control*

Final ore grade for each producer was determined on chemical assay at the point of entry to the treatment plant, but up to that point all grade control was radiometric. All mines were issued with small portable GM counters calibrated to indicate waste, heap leach ore, and ore. At four of the mines tunnel-type radiometric car analysers were installed and classified all material extracted into the three classes. On all small mines, hand-picking assisted by GM counters was done to reduce waste transport to a minimum. Within limits it was always more economic to keep the plant operating at capacity and thus if any ore shortages developed the standard cut-off of 0.15%  $U_3O_8$  was temporarily reduced to ensure full capacity.

#### *Ventilation, safety and accident control, etc.*

As for the other services, the main control of these services was centred at Urgeiriça. A ventilation, dust control and radon inspector was responsible for the regular inspection and sampling of all mines and he reported to the chief mining engineer. An accident committee met once a month and training courses in rescue and first aid were held at all mines.

A hospital and main medical clinic operated at Urgeiriça and regular visits by medical staff were made to the other mines.

#### *Mechanical and electrical services*

The workshops at Urgeiriça were responsible for all mechanical and electrical maintenance on all mines. A foreman mechanic and electrician visited each mine once a week on maintenance inspections. The carpenters' shop was also centralized at Urgeiriça.

#### *Transport*

Transport of ore from the satellite mines was done on contract by a fleet of trucks of about 15 t capacity. Under the contract all materials from the central workshops at Urgeiriça were carried to the other mines on the return journey. The company maintained a fleet of Land Rovers and cars for staff transport.

#### *Laboratory*

The laboratory at Urgeiriça operated as a separate section to carry out routine analysis of samples from surface and mine exploration and development, stope control, ore stock control and treatment plant operation. Between 1000 and 1500 samples were analysed per month about 10% by chemical methods, mainly for checking equilibrium, and the others radiometric.

#### *General office, stores, etc.*

The chief of the general office at Urgeiriça was responsible for time-keeping, pay office, store, etc. on all mines. The chief storeman was the only purchasing agent of the Company and supplies to the other mines came through the Urgeiriça store.

#### *Mine labour*

Between 500 and 600 men were employed underground on all mines during the normal years, and, in addition, a further 300 to 400 surface workers were employed by the mining section on auxilliary mining services.

### ORE PROCESSING PLANT

The plant worked on a continuous shift basis employing 80 persons of whom 60 were on shift work and the others on maintenance. Over the whole

period the milling rate averaged 125.2 t/d and the cost per dry ton milled was approximately US \$4.60.

The figures for daily throughput, grade of feed, recovery and product grade for the years 1952 to 1962 are given in Table III. The year 1951 is not included as it was mainly a running-in period. During the period 1952 to 1956 the daily tonnage and grade were almost exactly those which the plant had originally been designed to treat, i.e. 100 t/d at 0.35%  $U_3O_8$ . No outstanding difficulties were met and the recovery was 91.2%.

During 1956 it was foreseen that the average ore grade was likely to fall to about 0.26  $U_3O_8$  and, in order to maintain  $U_3O_8$  output and overall cost per kilogram, it would be necessary to increase throughput by 40 to 50%. To meet this requirement various modifications were made to the plant in early 1957.

In 1958 the average daily throughput reached 155 t. From January 1957 to April 1960 the recovery averaged 89.8% but the concentrate grade fell. The final period, dating from May 1960 until the end of production in March 1962, coincided with the running down of the whole operation and further reduced the average feed grade to about 0.23%  $U_3O_8$ .

Several further alterations to the plant were made and by these measures, recovery was maintained at about 90% but product grade again fell.

The plant operating costs per dry ton milled and per kg  $U_3O_8$  produced are shown on Table III. Except for the last two years which were abnormal, owing to the approaching close-down, the cost per dry ton milled was progressively reduced during the operating period.

## OTHER SOURCES OF $U_3O_8$ AND SUBSIDIARY TREATMENT METHODS

Besides the ore produced by direct mining on the Company's concessions, followed by direct feed to the treatment plant, there were various other sources of  $U_3O_8$  and also other subsidiary treatment methods which contributed to the final total of  $U_3O_8$  produced.

### Ore purchases

The only ore purchases made by the Company were from the uranium prospecting and development section of the Portuguese Government, Junta da Energie Nuclear. The total ore purchased between 1957 and 1962 amounted to 4552 t with an average grade of 0.35%  $U_3O_8$ .



### Old surface dumps

Several properties contained uranium-bearing dumps dating from the radium period, either consisting of mine ore which had been below the cut-off grade of the radium period or of tailings sands or slimes rejected from the old radium plants. The total ore recovered from old dumps amounted to 7141 t with an average grade of 0.355%  $U_3O_8$ .

### Urgeirica treatment plant tailing's water neutralization slimes

The treatment plant tailing's disposal flow consisted of 600 to 650 t of water carrying 120 to 150 t of sands and slimes per day. Settlement of solids was effected in the tailings disposal dump and the clean water run-off from the top of the dump emerged in a sump at the base. About 250 t of water was returned daily for use in the plant but the remaining 350 to 400 t were sent to waste. The waste water had a pH of 3.5 and a  $U_3O_8$  content of 10 to 15 mg/litre, and was therefore passed through a neutralization system before joining a nearby stream. Precipitation took place in a series of pits and labyrinths. The resultant slimes were allowed to settle and dry in pits and were afterwards dug out and returned to the plant feed just before the rod mill. Over a ten-year period, 3930 dry tons of slimes with an average grade of 0.47%  $U_3O_8$  were produced from this source.

### Neutralization slimes from mine water

In some mines, pyrite in the ore, on being oxidized and attacked by the natural waters, produced a dilute sulphuric acid which, under oxidizing conditions, tended to leach small quantities of  $U_3O_8$  from ore in place or broken ore in stopes. A routine check was kept on the pH and  $U_3O_8$  content of waste mine waters. At Urgeirica the water was used direct by the treatment plant and on some other properties by the natural leaching system, but at Bica and Vale da Arca it was found to be worthwhile making a small neutralization system for excess waste mine waters. About 36 t of dry slimes with an average grade of 2.40%  $U_3O_8$  were produced in this way.

### Other ore sources

Small contributions were made by a few other sources, one of these was from waste mine timbers. Old timbers removed from the mine during re-forming work were found to be impregnated with uranium from mine waters. All old timbers were therefore used to fuel the furnaces of the sample preparation drying room and the resulting ash was recovered and treated. Over the period, about 14 t ash with an average grade of 2.22%  $U_3O_8$  were recovered.

### Water, liquors and concentrates

The above extraneous sources entered the treatment plant before the rod mill and were therefore subject to the recovery factor. The following sources entered after the rod mill and were considered to have 100% recovery.

#### Urgeiriça mine water

All the Urgeiriça-Ucha drainage mine water was used by the treatment plant and this water had a small but valuable  $U_3O_8$  content. The origin of the dissolved uranium was the same as described above. The leaching of  $U_3O_8$  within Urgeiriça mine was not uniform, and several studies were made of the process.

The volume of water pumped and the  $U_3O_8$  content varied seasonably. The grade of the water pumped expressed in mg  $U_3O_8$ /litre showed higher values in the spring and low values in the autumn. The monthly rainfall graph and the monthly total volume of water pumped corresponded very closely in form but were off-set by about five months indicating that drainage through the mine took that time. The total of 41 587 t  $U_3O_8$  from this source amounted to approximately 3.0% of the total production. This  $U_3O_8$  was considered to have been 100% recovered by the plant.

#### Stope leaching at Urgeiriça

An experimental attempt to speed up leaching in place was made in 1958 and 1959 – probably the first attempt at in-situ uranium leaching anywhere in the world. Backfill to stopes, originating partially from wall rock, often carried 0.02 to 0.03%  $U_3O_8$ . In an experimental stope between the 12th and 13th levels strongly acid liquors with a pH of 2.5 and a  $U_3O_8$  content of 2.13 g/litre were produced, and 121 kg  $U_3O_8$  were recovered in four months. The method gave results but it seemed probable that the  $U_3O_8$  would anyhow have been recovered by normal drainage over a longer period. This factor, combined with the inconvenience to normal mine operation, led to the discontinuation of the system.

#### The heap-leaching system

The Company pioneered and successfully developed a “natural” leaching or heap-leaching treatment process for low-grade ores. The range of grades treated by this process was approximately 0.05 to 0.15%  $U_3O_8$  and covered the low-grade ore selected at the surface hand-picking or Geiger counter selection installations. No ore was mined specifically for this treatment process.

The natural leaching possibilities of Urgeiriça ore were first noted in 1950–51 when it was found that an exposed dump of development ore had been very

substantially leached by rain-water over a period of six to twelve months. As it was realized that this phenomenon could be put to practical use, an early series of tests were carried out at Urgeiriça in 1951–1952, and it was found that ores from different deposits varied greatly in leaching efficiency if only water was added.

The natural leaching process is initiated by the aerial oxidation of pyrite in the presence of moisture and the subsequent attack on uranium minerals by the dilute sulphuric acid produced. The chemical reactions and many detailed experiments on Portuguese ores have since been described in a series of papers by several authors from the National Chemical Laboratory, Teddington. [3–5]

The methods employed at Urgeiriça and Valinhos gave recovery efficiencies of well over 80% and eventually reduced the turn-round period to eight or ten months. The biggest tonnage treated was from Bica mine and unfortunately this was also the most difficult ore to treat. Despite many efforts the overall leaching efficiency remained low at Bica. The principal reasons were that the pitchblende was finely disseminated and tightly locked up in the siliceous gangue and the calcite content of the gangue which consumed acid and also produced calcium sulphate which formed hard layers and inhibited drainage.

Over the ten-year period, 1952–62, a total of 83 600 t of low-grade ore was treated and 43 172 t  $U_3O_8$  were recovered. With an initial average grade of approximately 0.077%  $U_3O_8$  this gave an overall recovery of about 66%.

## PRODUCTION TOTALS

### Summary

The total production of  $U_3O_8$  based on the CPR laboratory assays was 1324.670 tons, but the final accepted figure was 1325.683 tons based on the assays from the purchasing agency and referees.

For the present purpose, discrimination by sources is based on the CPR assays. Table IV gives a summary of the main sources of  $U_3O_8$  making up the total.

As was to be expected, the Urgeiriça-Ucha mine was by far the largest producer, providing 49.42% of the  $U_3O_8$  which was directly mined and 57.89% of the total  $U_3O_8$  produced. Bica mining concession was the second largest producer and Carrasca and Vale da Arca, respectively third and fourth largest.

### Cost of product

An average cost of US \$5.70/lb  $U_3O_8$  was achieved, which can be broken down as shown in Table V.

TABLE IV. SUMMARY OF SOURCES OF  $U_3O_8$  PRODUCED

	$U_3O_8$ (t)	% of total $U_3O_8$ produced
<b>Ore from stoping mines:</b>		
Urgeirica and Ucha	654.580	49.42
Bica	188.109	14.20
Carrasca	90.020	6.80
Vale da Arca	77.790	5.87
Valinhos	34.509	2.61
Reboleiro	26.596	2.01
Nine other stoping mines	59.951	4.52
Sub-total	1131.555	85.43
Ore from development mines [3]	1.891	0.14
Ore from open-cast mines [6]	41.526	3.13
Ore purchases	14.180	1.07
Old surface dumps and various other ore sources	25.911	1.96
Slimes (Urgeirica neutralization etc.)	18.830	1.42
Water (Urgeirica & Ucha mine)	41.708	3.15
Liquors and concentrates (natural leaching, percolation leaching)	49.069	3.70
Total:	1324.670	100.00

TABLE V. COST OF PRODUCT

	Average cost (US \$/lb $U_3O_8$ )
Mining cost	2.89
Transport cost	0.29
Diamond drilling	0.20
Treatment plant	1.64
General overheads	0.51
Taxes, etc.	0.17
Total:	5.70

## **Final costs**

Amortization of capital expenditure had been calculated to be complete as at 31.12.1962, the limiting date of the contract, but in fact the contract tonnage figure was achieved on 31.3.1962 and therefore there remained a nine-month amortization of capital which is not included in the costs given in Table V. There was also a large item of closure expenditure incurred after 31.3.1962, which is not included in the cost of product figures given in this table. As shown, an overall figure of US \$5.70/lb  $U_3O_8$  was achieved under these conditions.

When the remaining nine-month amortization and the closure expenditure are included the final overall figure for the whole project is approximately US \$6.66/lb  $U_3O_8$ , or almost exactly the target cost.

## **STAFF, LABOUR AND SOCIAL SERVICES**

### **Staff**

The total number of staff which included all salaried personnel paid on a monthly basis ranged between 110 and 120 in normal years. Senior staff amounted to between 20 and 30 persons and the rest were junior staff. The total names of staff inscribed on the books of the Company during the production period was 255 and turnover was thus about 100% in the 11-year period.

### **Labour**

In the normal years the total labour inscribed on the books of the Company varied between 1000 and 1200. The total number of different names on the Company's books for the whole production period was 5800 and the labour turnover in the 11-year period was therefore about 500%. This high figure was caused by the many itinerant underground workers, who only worked seasonally on the bigger mines and also by the fact that on the small mines local labour was employed which did not move on to other mines when these short-term mines were closed down.

## **CLOSURE PERIOD: APRIL–JULY 1962**

Production of uranium concentrates ceased on 31 March 1962. There then followed a four-month period in which the affairs of the Company were finalized

and the assets arranged for hand-over to the Portuguese Government Agency, the Junta da Energia Nuclear (JEN), as required by the original agreement.

The total number of employees had already been gradually reduced from 1245 in early 1960 when reductions started, to 560 at the end of March 1962. In the following months staff and workers were reduced to about 150 and this number were finally taken over by the JEN at the end of July 1962.

Both staff and workers were given a final indemnity which was considerably above the minimum legal requirement for the dismissal of employees in Portugal. The planned gradual phasing of dismissals over nearly two and a half years, and the final indemnity made it possible to avoid, as far as possible, hardship and distress which might otherwise have been caused by the sudden stoppage of the activities of the Company.

In conformity with the terms of the agreement all fixed and movable assets were handed over in good condition to the JEN. Industrial buildings and housing were cleaned and made wind and weather tight, machinery and mining equipment was either taken to storage at Urgeiriça after being cleaned and serviced or, if left on the sites, was protected against the weather.

All shafts on abandoned mines were sealed with a reinforced concrete slab or were adequately protected by walls. Open pits which presented any possibility of danger were filled in or protected.

The Urgeiriça tailing's disposal dump, which was liable to give rise to dust in high winds was protected by turfing over all the walls and the top was covered with lime to neutralize acid waters. All natural leaching dumps and any others where there was any possibility of acid waters were also covered with lime which was well dug into the heaps.

Finally, at the end of July 1962, the Company's assets were handed over to the Junta da Energia Nuclear as representative of the Portuguese Government.

## CONCLUSION

In the 1950s a multiple small-scale uranium mining project was successfully operated in Portugal. It was based on one principal mine at which the main services and the ore treatment plant were located. Success was mainly due to a tight management control, standardization of equipment and methods and long-term planning involving the proving of ore reserves and the activating and closing down of the satellite mine on a strict time schedule. A contribution to success was also management flexibility and willingness to take full advantage of what were then innovations such as heap leaching, radiometric sorting, etc. Good organization was essential to the success of the whole project.

Another essential ingredient to success was to have reliable and sound estimates of ore reserve tonnages and their availability for all mines and at all

stages from the beginning to the end of the project. This was vitally necessary so that management could decide and control the necessarily complex technical policy. The evaluation of the ore reserves in these small deposits was based on classical ore reserve estimation methods but utilizing calibrated radiometric measuring methods appropriately checked by chemical analysis.

### ACKNOWLEDGEMENTS

In such a complex project, covering a period of over seventeen years, a very great number of persons were connected with, and made considerable and valuable contributions to, the fulfilment of the work and the author has called upon elements of all these contributions in preparing this history. The author therefore wishes to thank and acknowledge everyone who was in any way concerned with the CPR project.

The author wishes, above all to acknowledge the excellent management of the late Eng<sup>o</sup>. Joaquim de Sousa Byrne who was General Manager of CPR Lda at Urgeiriça from 1943 to the end of the contract in 1962 and who was principally responsible for the successful fulfilment of the project. Among his colleagues, the author wishes to acknowledge the very considerable contribution of Dr. J.A.E. Bennett, M.I.M.M., who shared the early assessment work with the author and was eventually Chief Geologist of the Company, and Mr. W.K. Brown, M.I.M.M., Assistant General Manager and Chief Mining Engineer from 1950 to 1956.

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## DISCUSSION

I. ASPELING: What is the magnitude of the sampling factor, and what is the justification (if any) for the arbitrary cutting-off of values?

M.V. HANSEN: The answers to both parts of your question are empirical. In 1946–47, prior to the production period, a trial stope some 20 metres in length known as the “Study Stope” was opened at Urgeiriça. Each one-metre vertical cut was channel-sampled at horizontal intervals of one metre extending vertically over 30 metres. All ore recovered was carefully bulk-sampled and checked against the 500–600 channel samples taken. As a result of these fairly extensive studies it was decided that the erratic high values would have to be cut. In addition, a reduction of 5% had to be made in the sample values to permit them to be equated with the bulk-sample values. Similar tests were performed at all the smaller mines. The sampling reduction factor never exceeded 10%.

M. DAVID: I would just like to point out that the excellent regression line which was shown in Fig. 1 of my paper<sup>1</sup> relates to one level of Urgeiriça. It indicates that radiometric measurements are appropriate in this case, which may not necessarily be true for many other deposits.

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<sup>1</sup> IAEA-SM-239/36, these Proceedings.



## GESTION DE DATOS PARA EVALUACION DE YACIMIENTOS EN NUCLEBRAS

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### Abstract-Resumen

#### DATA MANAGEMENT FOR EVALUATION OF DEPOSITS AT NUCLEBRAS.

The large volume of data generated in any deposit evaluation work based on information obtained from intensive drilling operations and produced by different types of equipment involved in prospecting calls for the establishment and use of standardized and unified systems of data collection. Thus, standardization of data collection and electronic data processing must be ensured if the management of such data is to be viable, i.e. effective from the technical and economic points of view. These two considerations form the basis of the Nuclebrás data management system, in which essential data are collected and processed by means of sub-systems called Catalogue of Samples, Drilling Report and Application Programmes, thus providing the basis for conventional statistical and geostatistical evaluations. It should also be noted that new data to be generated at the subsequent stages of planning and mining control will be added to this core, so that it has to be sufficiently flexible and generic to be used in the long term for a wide variety of purposes.

#### GESTION DE DATOS PARA EVALUACION DE YACIMIENTOS EN NUCLEBRAS.

El gran volumen de datos generado en cualquier trabajo de evaluación de yacimientos, resultante de sondeos intensivos y producido por los diversos equipos comprometidos en la prospección, obliga a la implantación de sistemas normalizados y unificados de colecta de informaciones así como a su utilización. Así, para que sea viable la gestión de esos datos, de manera que sea operacional desde el punto de vista técnico y de costos, se debe tener en cuenta la normalización de la colecta de informaciones y el proceso electrónico de datos. En estos dos tópicos está basada la sistemática de gestión de datos de Nuclebrás, que orienta la colecta de las informaciones indispensables así como su proceso posterior a través de sub-sistemas de Registro de muestras, Boletín de sondeos y Programas de aplicación, permitiendo de esta manera evaluaciones convencionales estadísticas y geoestadísticas. Debe destacarse además que la generación de nuevas informaciones en las etapas posteriores de planificación y control de explotación vendrán a añadirse a ese núcleo principal, obligándolo a ser lo suficientemente flexible y genérico para su utilización a largo plazo y con objetivos bastante dispares.

## 1. INTRODUCCION

Entre los factores que intervienen en un cálculo de reservas destaca sobremanera la gestión de datos, con la fijación de rutinas y normas de recopilación, lo que permite la utilización optimizada de todas las informaciones generadas durante la prospección y exploración, así como en el propio desarrollo de la explotación.

La experiencia de Nuclebrás indica como camino para esa gestión la centralización de la decisión, tanto para la implantación de normas de recopilación, como para la realización de los cálculos, por un grupo de especialistas que utilizan el proceso electrónico de datos.

Así se consigue la implantación de una metodología en la que se minimizan los caracteres subjetivos, permitiendo la utilización de criterios uniformes en los cálculos y, claro está, en la evaluación de las reservas.

De este modo se pretende dar una visión general de las rutinas implantadas para la utilización de los datos recopilados desde la fase inicial de prospección hasta la explotación de un yacimiento, mostrando al mismo tiempo la normalización de la recogida de informaciones y su gestión.

## 2. NORMALIZACION DE LA RECOGIDA DE INFORMACIONES

Durante la prospección y exploración realizada por Nuclebrás, dos fuentes generan informaciones con flujos que siguen caminos totalmente dispares. Por un lado están los equipos de campo que recopilan datos sobre los trabajos de prospección y, por otro, el laboratorio que analiza las muestras recogidas.

Así, cualquier rutina debe tomar en consideración esas dos fuentes normalizando los trabajos de campo y laboratorio, lo que se consigue a través de la:

- normalización de resultados analíticos,
- normalización de trabajos de evaluación.

### 2.1. Normalización de resultados analíticos

El esquema de normalización de muestras y resultados analíticos de Nuclebrás depende del objetivo para el que se recogan. Esto es, cada muestra es una entidad cuyas informaciones se deben archivar, sin importar que provengan de trabajos de geoquímica, levantamiento de mapas, etc. Es obvio que al iniciar una prospección se tiene apenas el sentimiento de que un determinado afloramiento pueda transformarse en un yacimiento, justificando por tanto la independencia en el objetivo de la recogida de informaciones (muestras), pues eventualmente éstas pueden ser utilizadas hasta la etapa final de exploración (cálculo de reservas).

Para conseguirlo es necesario que cada muestra tenga un identificador único que permita soportar el gran volumen de muestras originado por los distintos proyectos.

Con ese fin, se definió una numeración compuesta de ocho dígitos alfa-numéricos que permite millones de combinaciones.

Se ha creado un formulario de registro analítico donde la rutina implantada abarca la compilación por el solicitante del identificador único de cada muestra y los análisis a realizar, formulando de este modo una petición de análisis que se envía al laboratorio. Este último llena el formulario con los resultados analíticos y lo devuelve al solicitante.

## 2.2. Normalización de los trabajos de evaluación

Es importante focalizar dos aspectos básicos dentro de la organización de prospección y exploración de Nuclebrás. El primero, evidentemente, es la gran variedad de equipos comprometidos en la prospección con enfoques técnicos diferentes y, el segundo, la centralización de los cálculos de reservas por un grupo de especialistas.

La centralización permite la homogeneización y control de las informaciones recogidas a través de la implantación de normas en forma de formularios como el “Boletín de trabajos de evaluación”.

Así, los técnicos de campo encargados de los servicios sistemáticos de evaluación llenan esos formularios y los envían al grupo de cálculo, responsable de todos los trabajos posteriores.

Dichos formularios, bastante auto-explicativos, facilitan dos niveles de información:

- uno de ellos, totalmente objetivo, contiene el mínimo necesario para realizar una evaluación de yacimientos (localización, tamaño, recuperación y litología de las muestras colectadas);
- el segundo, hasta cierto punto subjetivo, se refiere a la descripción geológica del emplazamiento explorado (observaciones sobre el perfil).

Gánase así en operacionalidad y en homogeneidad, sin restringir excesivamente la necesaria creatividad e imaginación de los técnicos comprometidos en la prospección y exploración.

## 3. PROCESO DE DATOS

Existiendo la normalización de la recopilación de datos y establecidas las rutinas de flujo de las informaciones, es posible estructurar su gestión y respectivo tratamiento a través del proceso electrónico.

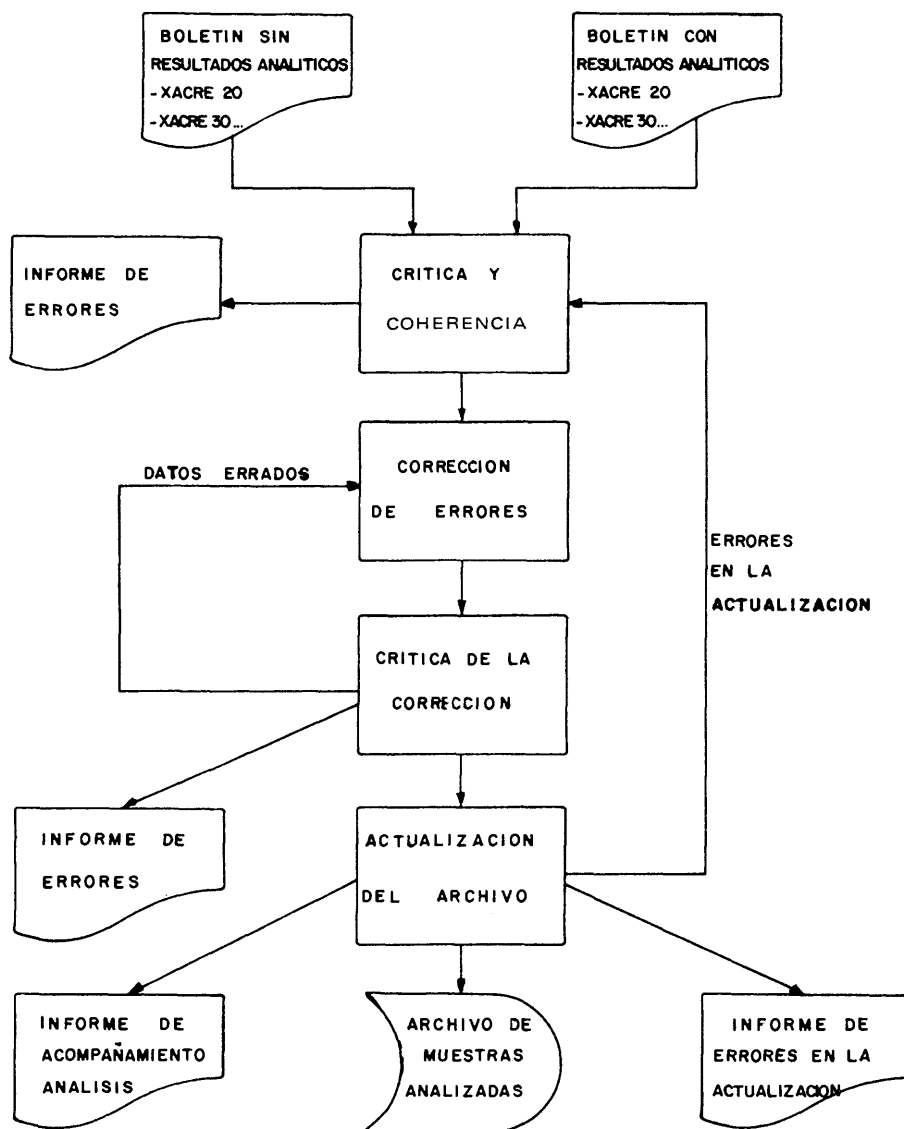


FIG.1. Flujo del subsistema registro de muestras.

De este modelo y con el fin de mantener la necesaria individualidad entre el campo, el laboratorio y el grupo de cálculo de reservas, se desarrollaron tres subsistemas interdependientes denominados:

- Subsistema de registro de muestras;
- Subsistema de informaciones para evaluación;
- Subsistema de aplicación.

### 3.1. Subsistema de registro de muestras

Este subsistema trata las informaciones procedentes del laboratorio sobre muestras recogidas por Nuclebrás y, como es independiente de la recopilación, lo comparten otros sistemas como por ejemplo Geoquímica, Geología regional, Control analítico, Petrografía, etc.

El flujo de la información y el respectivo proceso son bastante simples.

Una copia de la petición de análisis, o sea el Boletín analítico sin los resultados de los análisis es enviado por el recopilador al Centro de proceso de datos, donde es perforado y criticado, generando un archivo de muestras con análisis pendientes.

El laboratorio llena una copia de ese Boletín analítico con los resultados de análisis químicos y la envía al Centro de proceso de datos una vez completadas las informaciones generadas en la fase anterior.

La repetición de ese ciclo va creando el registro de muestras lo que, además de otros controles, permite que el gerente tenga una visión del tiempo empleado para los análisis y de las dificultades con métodos analíticos específicos; de este modo se puede actuar inmediatamente en cualquier punto del flujo en caso de anormalidades.

Por otro lado, se mejora la propia calidad del análisis a medida que se van automatizando y simplificando los tests de precisión y sensibilidad de distintos métodos y laboratorios.

La Fig.1 muestra el proceso del tratamiento del dato. Se puede ver que los Boletines analíticos completos y/o incompletos pasan por programas de crítica y coherencia generando un archivo con los registros errados, los cuales son corregidos vía terminal, siendo sometidos de nuevo al programa de crítica, donde los registros de datos ciertos van a actualizar el registro general, restando los errados que quedarán dentro del ciclo hasta que la actualización sea aprobada por los programas de crítica.

Las Figs 2, 3 y 4 muestran algunos de los informes generados por el subsistema.

RACRE12- NUCLEBRAS - EMPRESAS NUCLEARES BRASILEIRAS PAG 1067 | ARQUIVO DO ROSCOE  
 ELATORIO DE CRITICA DOS FORMULARIOS X A C R E 2 0 DATA 17/09/79 | ACRE0011  
 -----STATUS-----RUSCOE-----

ODIGO DO PROJETO : 101 LOTE : 333333 BOLETIM : 111111 DATA : 05.678 ALTERACAO : 2 |

RGAO REQUISITANTE : AAAAAAAAAA CENTRO DE CUSTO : 1234567891 ORGAO EXECUTOR : 888888888 |  
 DATA REMESSA REQUISITANTE : 12.378 RECEB. NO EXECUTOR : 050678 REMESSA DO EXECUTOR : 060778 |  
 035 108 201 301 | OK 0035/0036

ODIGO DO PROJETO : 101 LOTE : 333333 BOLETIM : 111111 DATA : 030578 INCLUSAO : 1 |

03 AAAAAAA L N 10,41 P 00,01 B 12,31 P T 00,10 6 11,01 | ERRO 0037/0038  
 \*\*\*\*\*  
 03 AAAAAAA L N 10,41 P 00,01 B 12,31 P T 00,10 6 11,01 | ERRO 0039/0040  
 \*\*\*\*\*

ODIGO DO PROJETO : 107 LOTE : 3333A3 BOLETIM : 111111 DATA : 050679 ALTERACAO : 2 |

25 \*\*\*\*\* L 0,1 W 12,41 I A ,001, L 0,01 N 1,231 K 0,1,0 | ERRO 0041/0042  
 \*\* \*\*\*\*\* \* \* \*\*\*\*\*

ODIGO DO PROJETO : 502 LOTE : 200001 BOLETIM : 222111 DATA : 050778 INCLUSAO : 1 |

01 MHHMM01 L 0,5 L 2,9 | ERRO 0043/0044  
 11 0000011 L 1,5 B | ERRO 0045/0046

ODIGO DO PROJETO : 502 LOTE : 200001 BOLETIM : 222111 DATA : 090778 ALTERACAO : 2 |

12 IIIIIII16 L 0,49 L 50 | ERRO 0047/0048

FIG.2. Crítica del boletín de análisis.

```

*** SISTEMA DE CADASTRAMENTO DE AMOSTRAS ***

- CORREÇÃO DE REGISTROS -
- DE ENTRADA -

HISTORICO DE CONTROLE DE LOTES PROCESSADOS
SELECIONE O LOTE PARA CORREÇÃO

0013  ACRE0013  G  010  79  04  24  13  50  000416  000208
0014  ACRE0014  G  020  79  04  24  14  40  000490  000245
0016  ACRE0016  G  002  79  04  10  16  57  000410  000202
0017  ACRE0017  G  022  79  04  11  10  35  000210  000071
0018  ACRE0018  G  011  79  04  21  16  53  000506  000159

```

AÇÃO : Digitar o nome do lote que será corrigido nesta sessão.  
Exemplo : ACRE0013

RESULTADO : Aparecerá a TELA 6

OBS : 1) Se o nome do lote não começar por ACRE aparecerá a TELA 15  
2) Se o lote escolhido não existir no Histórico aparecerá a TELA 16  
3) Se o lote existir no Histórico, mas já estiver corrigido, aparecerá a tela 17.

```

.... + .... 1 .... + .... 2                      + .... 7 .... + ....
XACRE20  I  000010  150179  106  600522  1  ERG01  281241001  0001
IPR      160878
XACRE20  I  000010  150179  106  600522  2  01  GCH363  0003
XACRE20  I  000010  150179  106  600522  2  02  GCH364  0004
XACRE20  I  000010  150179  106  600522  2  03  GCH365  0005

XACRE20  I  000010  150179  106  600522  2  22  GCH382  0022

```

AÇÃO : Posicionar o cursor (-) em baixo do dígito que se deseja corrigir e bater a tecla correta. Corrigir a tela toda posicionando o cursor. Quando terminar, apertar a tecla ENTER, que trará para a tela os seguintes 22 registros. Assim, corrigir este segmento todo, que irá sendo apresentado na tela de 22 em 22 registros. Quando terminar, apertar a tecla ENTER mais uma vez.

RESULTADO : Aparecerá novamente a TELA 7.

FIG.3. Corrección "on line" de boletines analíticos.

RACREC9

NUCLEBRAS - EMPRESAS NUCLEARES BRASILEIRAS

PAG 0001

RESULTADOS DE ANALISES NAO RECEBIDOS

DATA 15/09/79

ESCRITORIO :EBHO.PM

C.CUSTO : 6282016109

PROJETO : GANDARELA S.GEOL

C.CUSTO : 82212469

LOTE	BOLETIM	IDL	REQUIST.	DATA REMESSA	SITUACAO DO RESULTADO	EX E C U T O R ORGAO	RECEBIM.	REMESSA	ANALISES SOLICITADAS	NUM. DE AMUSTRAS	DETERMINACOES SOLICIT. RECEBID.	
000001	000008	201	EBHOPM	25/12/78	INCOMPLETO	CDTN	25/12/78	25/12/78	6	24	66	65
000001	000021	201	EBHOPM	25/12/78	INCOMPLETO	CDTN	25/12/78	25/12/78	5	24	47	43
000001	000023	201	EBHOPM	25/12/78	INCOMPLETO	CDTN	25/12/78	25/12/78	5	24	60	55
000001	000050	201	EBHOPM	25/12/78	INCOMPLETO	CDTN	25/12/78	25/12/78	5	24	74	60
000001	000060	201	EBHOPM	25/12/78	INCOMPLETO	CDTN	25/12/78	25/12/78	5	24	76	62
000001	000081	201	EBHOPM	25/12/78	INCOMPLETO	CDTN	25/12/78	25/12/78	3	24	72	58
000001	000082	201	EBHOPM	25/12/78	INCOMPLETO	CDTN	25/12/78	25/12/78	5	17	65	35
000001	000083	201	EBHOPM	25/12/78	INCOMPLETO	CDTN	25/12/78	25/12/78	5	21	60	51
TOTAL	PROJETO	GANDARELA S.GEOL	:		8				39	182	520	429
TOTAL	ESCRITORIO	EBHO.PM	:		8				39	182	520	429
TOTAL	RESULTADOS NAO RECEBIDOS	:			8				39	182	520	429

FIG.4. Informe de control del flujo analítico.



### 3.2. Subsistema de informaciones para evaluación

Este subsistema trata dos tipos de información:

- Informaciones sobre trabajos de evaluación presentadas por los formularios de “Informaciones generales” y “Ficha de colecta de muestras”.
- Informaciones sobre levantamiento de perfiles representado por los perfiles analógicos y/o registros digitales.

#### 3.2.1. Boletín de evaluación

Los datos de trabajos sistemáticos para evaluación provenientes de sondeos, pozos, trincheras, galerías, etc. se registran en los formularios normalizados ya citados.

El subsistema critica y somete a prueba la relación lógica entre los campos de esos formularios, creando el archivo de informaciones para evaluación.

Ese subsistema se encarga también de la fusión con el registro de muestras recogidas en los servicios de evaluación.

Esquemáticamente, el flujo de la Fig.5 indica las operaciones realizadas.

De este modo se critican los dos formularios: “Informaciones generales” (xacre 50) y “Ficha de colecta de muestras” (xacre 51), generándose un archivo de datos correctos y otro de datos incorrectos que después de las necesarias críticas y actualizaciones van a crear el archivo de informaciones de sondeos.

Mediante recuperación selectiva y fusión con el archivo de muestras analizadas y archivo de perfiles, se crea una matriz de trabajo que será utilizada por los programas de aplicación existentes.

En las Figs 6 y 7 se presentan algunos informes de ese subsistema.

#### 3.2.2. Informaciones sobre levantamiento de perfiles

Habitualmente, en Nuclebrás, el tratamiento de los levantamientos de perfiles constituye parte integrante del subsistema.

Dado que todavía es de uso intensivo el registro analógico para los perfiles Gama, SP y R, y que la utilización de la gravación discreta en cintas magnéticas o de papel se encuentra solo en sus inicios, aparece la necesidad de digitalizarlos.

Así, una vez realizados los perfiles, se envían las diagrfías al Centro de proceso de datos, donde son digitalizadas y representadas gráficamente, siendo criticadas visualmente por superposición con el perfil original.

Tras corrección de los errores que puedan existir, se genera un archivo en cinta magnética con los datos brutos.

En la fusión con el archivo de informaciones de sondeos, esas informaciones brutas de perfiles, principalmente las radiométricas, son corregidas por los factores de tiempo muerto, revestimientos, lama, etc., yendo a completar el Archivo general.

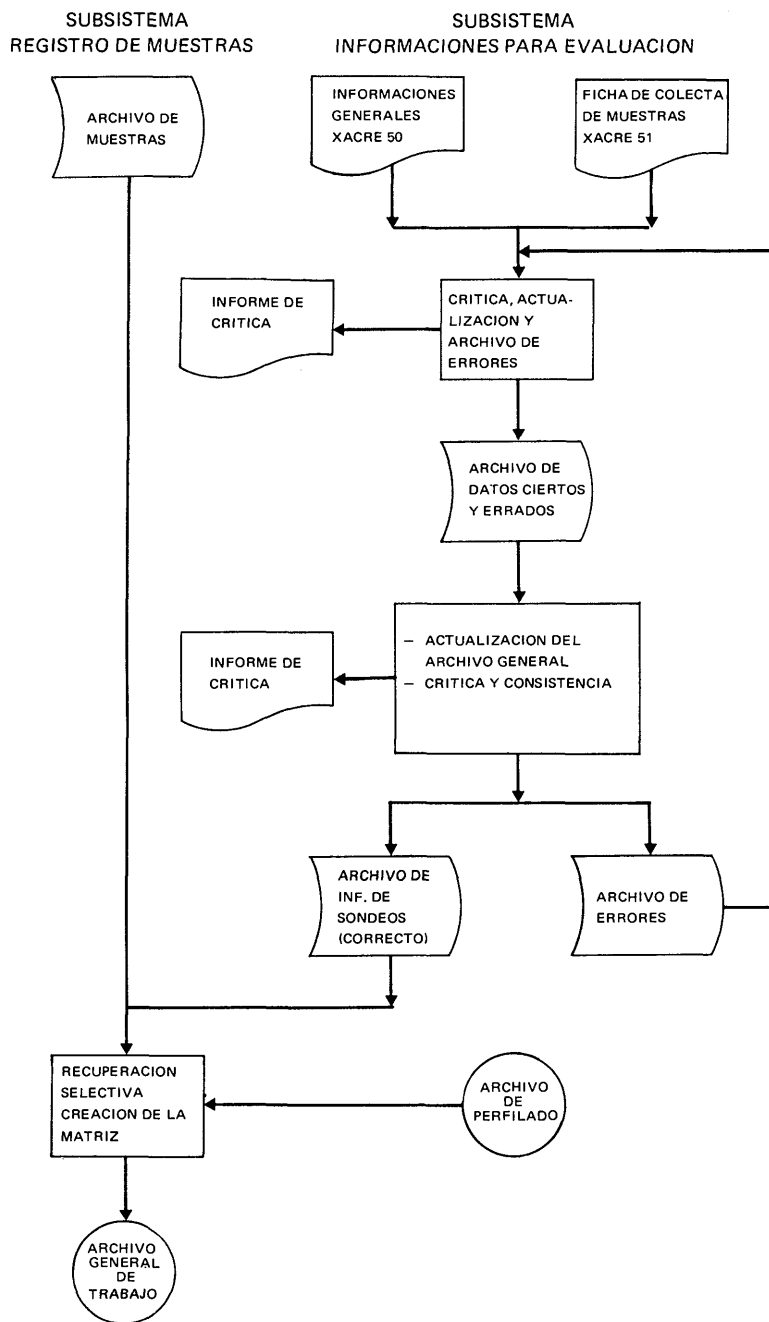


FIG.5. Flujo del subsistema informaciones para evaluación.

CDL IDENTIFICACAO	*****COORDENADAS*****			EXT. INCL	AZIM DO TRAB	*****NUMERO DE AMOSTRAS*****										**TEORES**		*****MINERALIZACAO*****			*TEOR MEDIO*	
	X	Y	DATA			<200	400	600	800	1000	1200	>1200	TOTAL	MIN.	MAX.	CORTE	TEOR PROF. INIC.	TEOR PROF. FINAL	ESP.	CORTE SEM	CORTE COM	
30 F#AF00101	413498	9496171	441.4	100	0	0	277	0	0	0	0	0	0	277	0	0	0	441.4	341.4	96.4	0	0
31							277	0	0	0	0	0	0	277	0	0	0	441.4	341.4	96.4	0	0
32							277	0	0	0	0	0	0	277	0	0	0	441.4	341.4	96.4	0	0
33							277	0	0	0	0	0	0	277	0	33	0	441.4	341.4	96.4	2	2
34							275	1	0	0	0	0	0	277	0	340	0	441.4	341.4	96.4	101	101
35							200	6	10	5	6	1	29	277	0	3700	0	441.4	341.4	96.4	190	190
36							277	0	0	0	0	0	0	277	0	0	0	441.4	341.4	96.4	0	0
30 F#AF00201	413504	9496188	438.4	100	0	0	344	0	0	0	0	0	0	344	0	0	0	438.4	338.2	96.1	0	0
31							344	0	0	0	0	0	0	344	0	0	0	438.4	338.2	96.1	0	0
32							344	0	0	0	0	0	0	344	0	0	0	438.4	338.2	96.1	0	0
33							344	0	0	0	0	0	0	344	0	23	0	438.4	338.2	96.1	2	2
34							344	0	0	0	0	0	0	344	0	130	0	438.4	338.2	96.1	97	97
35							253	17	19	12	9	10	20	344	0	2300	0	438.4	338.2	96.1	208	208
36							344	0	0	0	0	0	0	344	0	0	0	438.4	338.2	96.1	0	0
30 F#AF00301	413775	9496000	448.1	146	0	0	595	0	0	0	0	0	0	595	0	0	0	448.1	301.9145.0		0	0
31							595	0	0	0	0	0	0	595	0	0	0	448.1	301.9145.0		0	0
32							595	0	0	0	0	0	0	595	0	0	0	448.1	301.9145.0		0	0
33							595	0	0	0	0	0	0	595	0	29	0	448.1	301.9145.0		1	1
34							595	0	0	0	0	0	0	595	100	100	0	448.1	301.9145.0		100	100
35							535	20	11	12	9	5	3	595	0	1800	0	448.1	301.9145.0		79	79
36							595	0	0	0	0	0	0	595	0	0	0	448.1	301.9145.0		0	0
30 F#AF00401	413440	9496120	436.6	90	0	0	387	0	0	0	0	0	0	387	0	0	0	436.6	345.8	90.5	0	0
31							387	0	0	0	0	0	0	387	0	0	0	436.6	345.8	90.5	0	0
32							387	0	0	0	0	0	0	387	0	0	0	436.6	345.8	90.5	0	0
33							387	0	0	0	0	0	0	387	0	9	0	436.6	345.8	90.5	1	1
34							387	0	0	0	0	0	0	387	0	160	0	436.6	345.8	90.5	92	92
35							324	25	21	6	9	0	2	387	0	1310	0	436.6	345.8	90.5	114	114
36							328	23	17	7	3	2	2	387	8	1400	0	436.6	345.8	90.5	168	168

FIG.6. Ejemplo de resumen de las informaciones de sondeos.

NUCLEBROS PROJETO -	RESUMO DOS DADOS CADASTRADOS POR TRABALHO					25.09.79	FLA.	36
-----								
T O T A L I Z A C A O								
NATUREZA DO TRABALHO					:	FURU		
EXTENSAO TOTAL					:	41823		
NUMERO TOTAL DE TRABALHO REALIZADO					:	106		
NUMERO TOTAL COM ANALISE QUIMICA					:	52		
NUMERO TOTAL COM RADIOMETRIA					:	0		
NUMERO DE RESULTADOS QUALIFICADOS					:	14651		
NUM. TOTAL SEM QUIM. E SEM RADIOM.					:	54		
COORDENADAS LIMITES	:	X MIN.	X MAX.	Y MIN.	Y MAX.	Z MIN.	Z MAX.	
		644103	644103	7773101	7776626	1463	1519	
CARACTERISTICAS DE MINERALIZACAO: ELEMENTO	:	215		TEOR LIMITE	:	0	ESPESURA	:
		236				0		TeOR MEDIO :
		257				0		0
		263				0		0
		355				0	8.0	10.1
		401				0	8.4	10.1

FIG.7. Ejemplo de resumen general del proyecto.

### 3.3. Subsistema de aplicación

Esta es la parte más dinámica del sistema de cálculo de reservas. Se necesita un esfuerzo continuo en el seguimiento de las técnicas y desarrollo de los respectivos programas para mantenerse al par con la metodología actualizada de cálculo de reservas.

De manera general, los programas desarrollados se pueden agrupar en:

- Preparación de datos
- Cálculo por métodos convencionales
- Cálculo por métodos estadísticos
- Cálculo por métodos geoestadísticos
- Salidas de representaciones gráficas.

#### 3.3.1. Preparación de datos

En este grupo están comprendidos todos los programas utilitarios que preparan los datos para los cálculos subsiguientes.

Así, a partir del registro de muestras, las informaciones de sondeos y el archivo de perfiles, por recuperación selectiva, se organiza una matriz de trabajo donde las líneas son las muestras y las columnas las variables analizadas o representativas, como localización, recuperación, litología, leyes, etc.

Se crea esa matriz con las muestras normalizadas, o sea con un mismo tamaño y, por consiguiente, con las ponderaciones para que las diferentes leyes tengan la misma representatividad, sustituyéndose así la noción física de muestra por la de intervalos constantes de profundidad.

La normalización permite también que exista una relación biunívoca entre análisis químico y radiometría, creando la posibilidad de transformar los valores de cps en equivalentes químicos por regresión en los sondeos que contengan dos tipos de datos.

Para ello, después de las debidas correcciones por tiempo muerto, lama y revestimientos, es preciso ajustar el perfil de valores radiométricos con el perfil de valores químicos para cada sondeo, eventualmente con dislocaciones entre las dos observaciones, siendo el criterio empleado el de mayor covariancia.

Naturalmente, ese criterio no se usa a ciegas, pues los sondeos, en que las dislocaciones necesarias para obtener una buena correlación cruzada sean excesivas, se dejan para los cálculos subsiguientes.

El método utilizado es una extensión del indicado por Carlier (véase bibliografía), donde en lugar de utilizarse una regresión entre espesor químico/ espesor radiométrico y acumulación química/acumulación radiométrica, obteniéndose posteriormente la ley por simple división, se parte de acumulaciones químicas/acumulaciones radiométricas para intervalos constantes de espesores a partir del máximo radiométrico (Huijbregts y Guerra — 1977).

NUCLEBRAS	CALC	DE COVARIANCIAS CRUZADAS				FLA.	15			
PROJETO -		DADOS				MATRIZ DADOS QUIMICOS + RADIOMETRIA				
FURO - F01415										
VARIABEL - U308-NR		NUMERO DE DADOS -		43	MEDIA -	36.869	VARIANCIA -	3462.817	COVARIANCIA -	12455.505
VARIABEL - RADIOM.		NUMERO DE DADOS -		43	MEDIA -	155.154	VARIANCIA -	55049.993	CORRELACAO -	0.902
VALORES DAS COVARIANCIAS NAO CENTRADAS EM FUNCAO DA DISTANCIA MEDIDA NA DIRECAO "Z"										
VARIABEL 1 - U308-NR		VARIABEL 2 - RADIOM.								
0	Z1(X)Z1(X+H)	Z2(X)Z2(X+H)	Z1(X)Z2(X+H)	E(Z1(X))	E(Z2(X+H))	Z1(X+H)Z2(X)	E(Z1(X+H))	E(Z2(X))	N.PARES	
0.0	4822.156	79112.750	16175.916	36.869	155.154	18175.918	36.869	155.154	43	
0.40	3761.335	71203.937	16792.520	37.723	157.373	16068.633	37.223	158.327	42	
0.80	2723.976	54251.832	12624.934	38.448	159.928	12171.223	37.473	161.724	41	
1.20	2134.161	41039.797	9051.395	39.329	162.713	9752.094	38.072	165.121	40	
1.60	1917.665	34547.211	7633.875	40.209	165.625	8827.766	38.676	168.521	39	
2.00	1932.360	31615.767	6811.969	41.201	168.618	9092.566	39.299	172.242	38	
2.40	2046.400	29157.609	6268.238	42.220	171.314	9593.699	39.929	176.021	37	
2.80	2394.096	28880.797	6555.590	43.324	174.078	9955.664	40.649	180.026	36	
3.20	2210.307	31212.785	7706.492	44.461	177.300	8775.402	41.325	184.194	35	
3.60	1362.892	32011.891	8298.164	45.631	181.144	6081.709	42.099	188.492	34	
4.00	1256.604	27118.187	6688.574	46.967	185.176	4931.836	42.905	193.014	33	
4.40	1059.210	21854.902	5349.477	48.216	189.046	4428.117	43.543	197.721	32	
4.80	1055.642	19629.988	4774.027	49.610	192.713	4400.129	44.125	202.885	31	
5.20	991.709	19047.039	4587.176	50.747	194.978	4352.031	44.596	207.980	30	
5.60	969.483	18995.398	4610.844	52.186	195.756	3823.919	45.211	212.717	29	
6.00	939.597	18262.609	4636.848	53.282	198.370	3300.765	45.987	212.161	28	
6.40	918.840	17101.455	4765.301	54.941	201.227	3163.287	47.134	208.192	27	
6.80	915.676	16480.191	4906.816	51.538	200.898	3111.308	47.120	213.183	26	
7.20	1028.301	15756.824	4816.617	51.650	190.062	3169.013	46.345	219.564	25	
7.60	825.413	13653.992	4038.436	52.365	153.847	2735.798	36.610	224.596	24	
8.00	609.273	11435.691	3147.463	51.902	114.697	2267.517	27.114	226.358	23	
8.40	546.483	10041.914	2593.961	52.193	90.710	2045.985	22.665	226.929	22	
8.80	522.875	8946.270	2255.621	51.750	79.961	1859.361	20.816	222.667	21	
9.20	474.389	7592.332	1958.203	48.087	73.270	1667.244	19.581	201.680	20	
9.60	341.192	6012.687	1565.532	37.197	67.438	1237.098	17.296	156.805	19	
10.00	336.936	4716.691	1222.029	23.708	65.696	1059.999	16.340	106.671	18	
10.40	292.213	4337.020	1372.116	21.191	66.405	949.450	14.434	85.192	17	
10.80	129.393	4272.773	1386.087	19.547	65.653	436.493	6.374	77.406	16	
11.20	124.401	3214.493	983.100	19.850	48.742	435.990	6.232	74.485	15	
11.60	109.037	2349.174	692.806	19.589	35.917	338.834	5.141	71.050	14	

FIG.8. Ejemplo de análisis de desplazamiento entre informaciones químicas y radiométricas.

NUCLEBRAS

CALCULO DE BLOCOS

09.03.78

FLA. 49

PROJETO -

DADOS POR BLOCO AREA

ELEMENTO: A

NIVEL	RESERVA MEDIDA				RESERVA INDICADA				RESERVA INFERIDA				TOTAL BLOCOS
	N.BL.	TON. MINERIO	TON.	METAL	N.BL.	TON. MINERIO	TON.	METAL	N.BL.	TON. MINERIO	TON.	METAL	
1232.00	1	25070.43		9.71	0	0.0		0.0	0	0.0		0.0	1
1224.00	1	26495.96		4.31	0	0.0		0.0	0	0.0		0.0	1
1216.00	1	19871.98		3.97	0	0.0		0.0	0	0.0		0.0	1
1208.00	1	26495.96		3.64	0	0.0		0.0	0	0.0		0.0	1
1200.00	1	8049.54		0.80	0	0.0		0.0	0	0.0		0.0	1
1160.00	1	5198.44		4.07	0	0.0		0.0	0	0.0		0.0	1
1152.00	1	11361.54		11.17	0	0.0		0.0	0	0.0		0.0	1
1128.00	1	1986.45		0.94	0	0.0		0.0	0	0.0		0.0	1
1120.00	1	26495.96		26.01	0	0.0		0.0	0	0.0		0.0	1
1112.00	1	1425.55		2.85	0	0.0		0.0	0	0.0		0.0	1
TOTAL	10	152351.62		67.48	0	0.0		0.0	0	0.0		0.0	10
TEOR MEDIO				443.				0.				0.	

\*VARIANCIAS MEDIA:\*

ESPESSURA :0.31998E+01 ACCUMULACOES : 0.14595E+07 0.16836E+06 -0.65857E+09 0.52542E+07 0.0  
 ESP. RADIO :0.13886E+01 ACC. RADIO :0.30728E+07

FIG.9. Ejemplo de totalización del cálculo convencional de bloques.

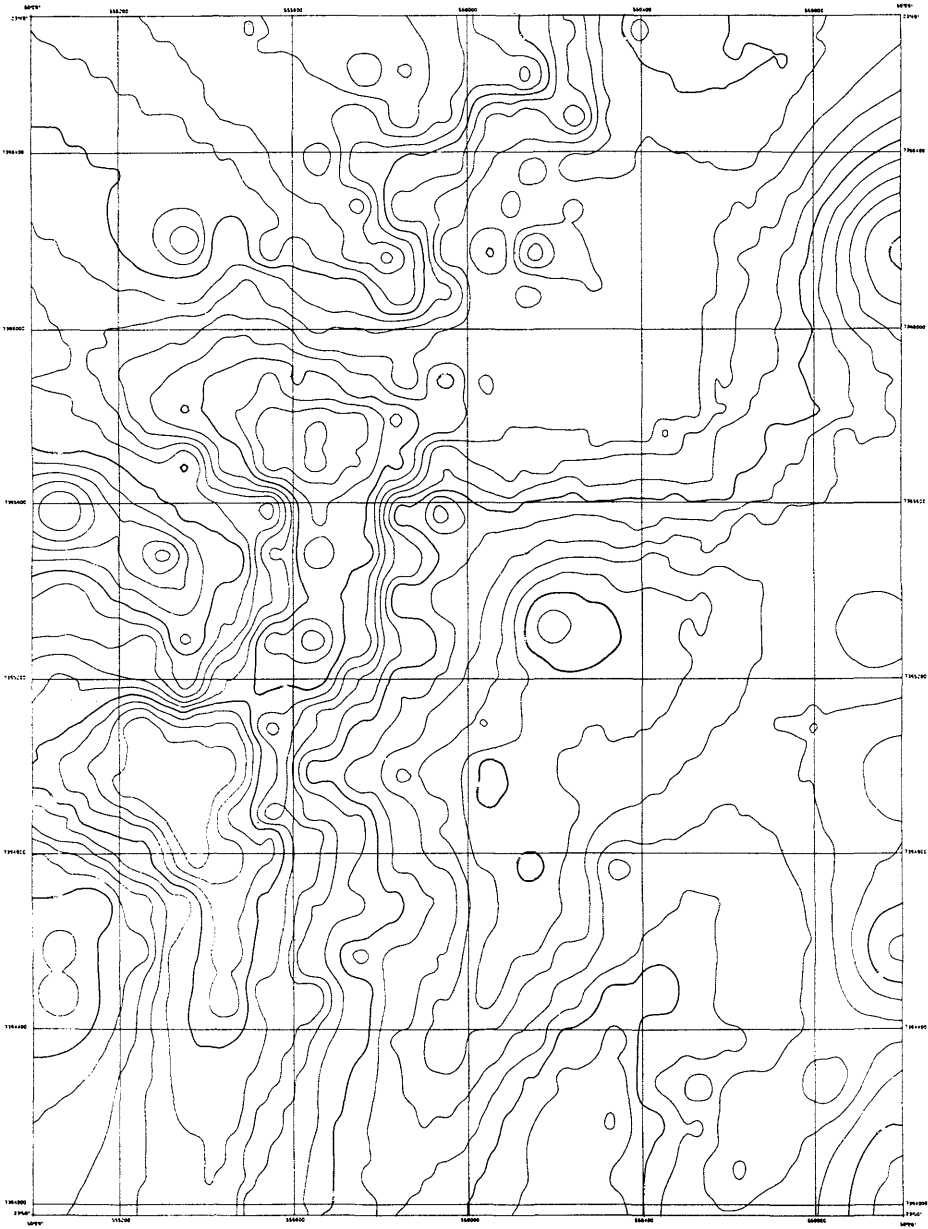


FIG.10. *Isolínea de la base de la mineralización.*



Así, disponiendo de las funciones de calibrado, todos los datos radiométricos se transforman en equivalentes químicos, completando la matriz de trabajo (Fig.8).

### *3.3.2. Cálculo por métodos convencionales*

Los programas de aplicación existentes en la Nuclebrás permiten la estimación de reservas por todas las técnicas convencionales, y se utilizan sistemáticamente en cualquier evaluación para obtener un modelo de comparación entre métodos.

A modo de ejemplo, en la Fig.9 se muestra un cálculo por bloques y en la Fig.10 un mapa de isólineas.

### *3.3.3. Programas de cálculo por métodos estadísticos*

Se refieren a programas que facilitan parámetros estadísticos uni y multivariados tales como estadísticas descriptivas, análisis de variancia, análisis de discriminación, análisis múltiple secuencial, etc., que se utilizan eventualmente para suprimir la redundancia de datos, definir y homogeneizar poblaciones, etc.

Esos programas no fueron desarrollados específicamente para el cálculo de reservas dado que el formato de la matriz de trabajo es modelo para cualquier sistema de Nuclebrás, existiendo una total interrelación con otros lotes como el sistema de estadística de muestreo geoquímico-SEAG, STATPAC-USGS y BMD-UCLA.

En la Fig.11 se da un ejemplo de test de distribución de frecuencia de una variable.

### *3.3.4. Programa de cálculo por métodos geoestadísticos*

Es un conjunto de programas desarrollado para permitir la evaluación de reservas por técnicas geoestadísticas, o sea análisis variográficos a través de variogramas directos y cruzados, con o sin efecto proporcional, o estimaciones por krigeage o cokrigeage, llegando hasta estimaciones locales con hipótesis de permanencia de distribución e incluso a la optimización de la cantidad de metal recuperable.

En la Fig.12 aparece un ejemplo.

### *3.3.5. Programa de salidas de representaciones gráficas*

Son programas que a través de la utilización del "plotter" e impresora del computador sirven de ayuda en cualquier etapa del cálculo de reservas, sea durante el montaje del archivo de datos facilitando la crítica de localización, valores aberrantes, etc., sea para interpretaciones de morfología y distribución de la mineralización.

S E A G

PROJETO -

CENTRO DE COSTO -

MATRIZ DADOS QUIMICOS+RADIOMETRICOS RGL A 1.

VARIABLE - A

NUM. AMOSTRAS - 2545

DISTRIBUICAO - LOGNORMAL

INTERVALOS - 6-STEPS

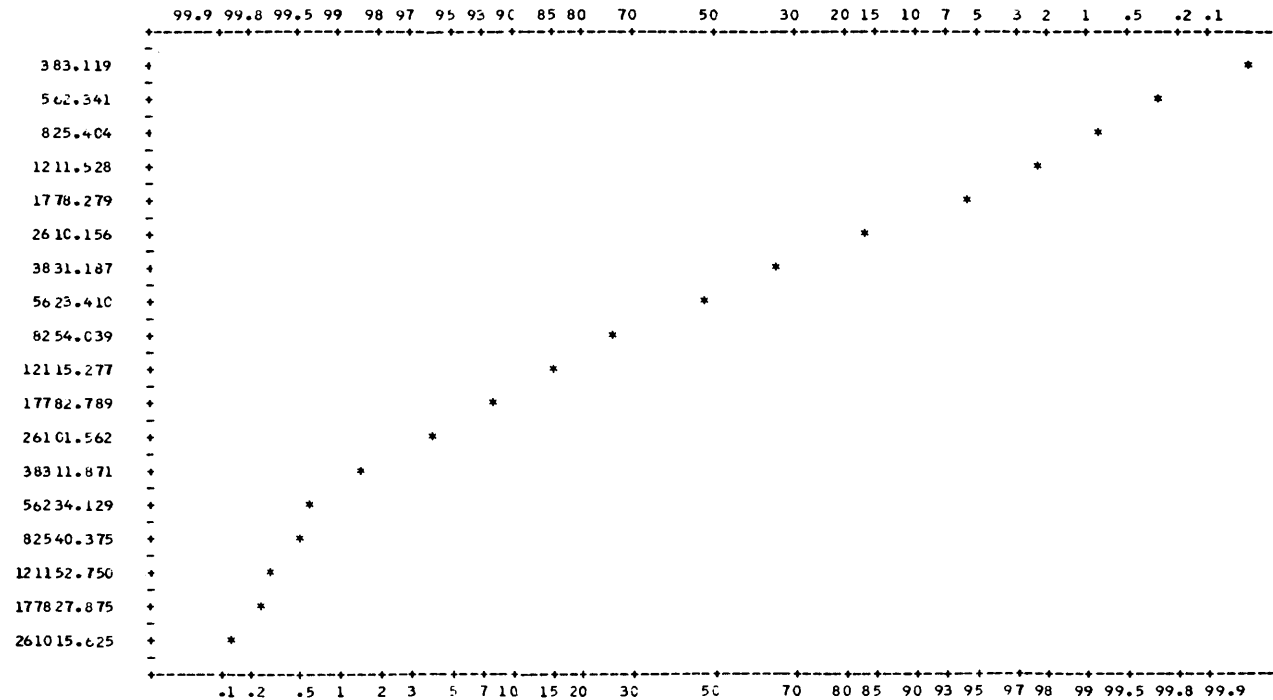


FIG.11. Ejemplo de test de distribución de frecuencia en una variable.

PROJETO -		DADOS - CALCULO DE RESERVAS POR COKRIGEAGE										TODAS AS AREAS
AREA :					-RESERVA TOTAL		--- (GLOBAL)					
NIVEL	COTA	PANEIS	V.FERTIL	VOL. MIN.	TON. MIN.	TON. A	TEORES: A	B	C	D	E	RESERVA
17	1288.	47	601600.00	601600.00	1431806.24	529.78	370.01	206.48	2401.14	1151.12	0.0	TOTAL
18	1280.	55	704000.00	704000.00	1675517.39	678.87	405.17	158.42	6548.94	1221.46	0.0	TOTAL
19	1272.	56	716800.00	716800.00	1705981.45	627.34	367.73	132.66	8907.07	967.19	0.0	TOTAL
20	1264.	55	704000.00	704000.00	1675517.51	725.03	432.72	177.59	7399.38	1040.05	0.0	TOTAL
21	1256.	49	627200.00	627200.00	1492733.62	515.18	345.13	224.10	4944.86	1039.84	0.0	TOTAL
22	1248.	48	614400.00	614400.00	1462269.32	525.25	359.20	178.15	3690.29	783.74	0.0	TOTAL
23	1240.	46	588800.00	588800.00	1401341.23	516.20	368.36	121.09	6341.83	960.18	400.00	TOTAL
24	1232.	48	614400.00	614400.00	1462269.05	792.38	541.88	182.70	15142.75	857.30	285.75	TOTAL
25	1224.	47	601600.00	601600.00	1431805.15	706.22	493.24	124.23	7669.24	1071.25	231.00	TOTAL
26	1216.	45	576000.00	576000.00	1370877.43	784.09	571.96	139.00	10124.98	992.08	125.00	TOTAL
27	1208.	40	512000.00	512000.00	1218557.73	794.89	652.32	218.20	6668.37	1437.56	127.11	TOTAL
28	1200.	39	499200.00	499200.00	1188093.55	636.87	536.04	165.18	5355.94	1123.59	0.0	TOTAL
29	1192.	34	435200.00	435200.00	1035773.86	805.37	777.55	180.41	6411.68	1273.98	0.0	TOTAL
30	1184.	23	294400.00	294400.00	700670.62	470.34	671.27	172.92	6883.12	1457.05	0.0	TOTAL
31	1176.	24	307200.00	307200.00	731134.38	500.57	684.65	174.66	8271.05	1480.85	0.0	TOTAL
32	1168.	24	307200.00	307200.00	731134.41	584.33	799.21	140.97	9743.84	1368.11	0.0	TOTAL
33	1160.	27	345600.00	345600.00	822526.50	545.65	663.39	147.01	8643.94	1199.25	920.00	TOTAL
34	1152.	20	256000.00	256000.00	609278.86	369.94	607.18	142.74	7035.90	1035.36	0.0	TOTAL
35	1144.	21	268800.00	268800.00	639742.78	273.97	428.25	151.81	6448.22	1185.28	0.0	TOTAL
36	1136.	19	243200.00	243200.00	578815.09	251.25	434.07	157.06	5776.19	1178.19	0.0	TOTAL
37	1128.	17	217600.00	217600.00	517887.43	212.83	410.95	159.64	6214.30	701.22	644.46	TOTAL
38	1120.	10	128000.00	128000.00	304639.85	74.68	245.14	132.23	7455.67	707.98	1302.88	TOTAL
39	1112.	7	89600.00	89600.00	213247.91	46.79	219.40	196.09	13699.99	934.87	0.0	TOTAL
40	1104.	3	38400.00	38400.00	91391.95	11.63	127.27	147.41	4170.03	1061.16	0.0	TOTAL
TOTAL		804	10291200.00	10291200.00	24493013.32	11979.44	489.10	164.48	7495.23	1106.87	504.53	TOTAL

FIG.12. Ejemplo de totalización del cálculo de bloques por cokrigeage.

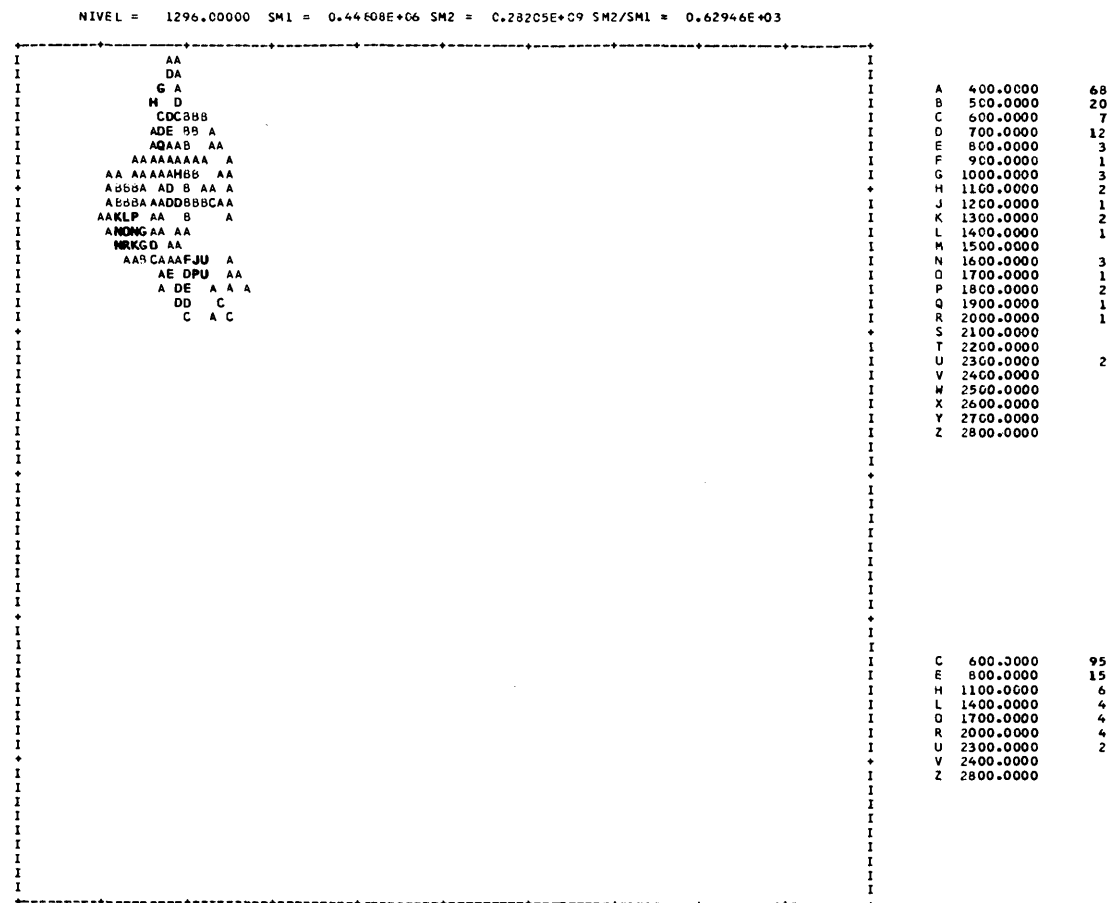


FIG.13. Ejemplo de distribución de bloques por clases de leyes.

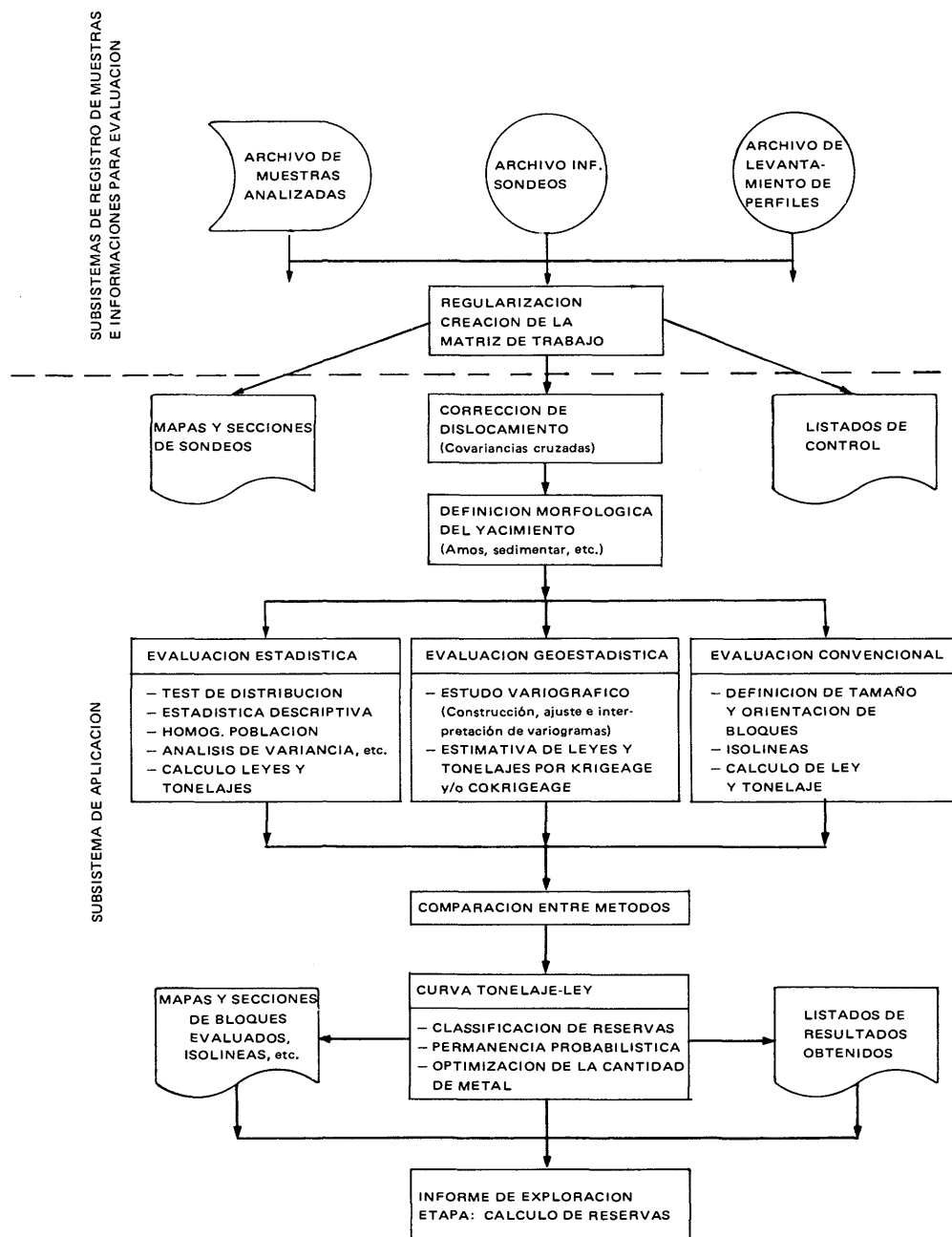


FIG.14. Flujo de la metodología para la evaluación de reservas en Nuclebrás.

De este modo se hacen mapas de localización de sondeos, secciones de sondeos, isolíneas, block-diagramas, secciones y mapas de bloques evaluados, etc.

En la Fig.13 se presenta un ejemplo.

#### 4. CONSIDERACIONES FINALES

Habiendo mostrado etapa por etapa la rutina de gestión de datos para el cálculo de reservas, con ilustraciones en las varias fases intermedias, en la Fig.14 se pretende dar una visión panorámica de toda esa metodología implantada hasta el momento de iniciarse los estudios de explotación.

Conviene resaltar que esta estructura no es rígida en modo alguno, sino que se presta a las adaptaciones necesarias según el caso.

#### AGRADECIMIENTOS

Los autores hacen constar su gratitud a todos aquellos que participaron en la realización de este sistema, haciendo patente que es un trabajo de equipo. Nada hubiera sido posible sin la contribución de los técnicos siguientes: Antonio Carlos Censi, Antonia Marli de Firmo, Adalea Dias, Carlos Alberto de Menezes, Charles Huijbregts, José Paulo Mansur Marques, José Augusto de Almeida Filho, Leila Sodero de Rezende, Maria Henriqueta H.B. Ornellas, Manoel Magarinos Torres y Nelson A. Xavier Filho. Desean expresar también su agradecimiento a la Dirección de Nuclebrás, a la Superintendencia de prospección y exploración mineral y a la Superintendencia de sistemas así como a sus órganos, por su autorización y facilidades, sin lo cual no habría sido posible presentar este trabajo.

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## DISCUSSION

A.E. BELLUCO: For your drilling evaluation do you use drillholes with core recovery or gamma logging?

C.A.G. DA VINHA: We use gamma logging only for defining the boundaries of mineralization. For evaluating the uranium content we use chemical analysis of cores recovered from diamond-drill holes. We do not accept recovery of less than 90% in the mineralized zones.





## **MINING AND OTHER URANIUM RECOVERY METHODS**

**(Session III, Part 2 and Sessions IV and V)**



# MINING METHOD AND GRADE CONTROL PRACTICES AT BAGHALCHUR, DERA GHAZI KHAN, PAKISTAN

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Dera Ghazi Khan, Pakistan

## Abstract

MINING METHOD AND GRADE CONTROL PRACTICES AT BAGHALCHUR, DERA GHAZI KHAN, PAKISTAN.

As a result of surface exploratory drilling in the Baghalchur area, the occurrence of significant uranium mineralization in two sandstone-filled paleo-stream channels has been proved. The ore in one of the channels is above the water table and is oxidized, while in the other, which is below the water table, it is unoxidized. The oxidized ore-body near the surface is being mined on an experimental basis by open-cast mining while the unoxidized deposit, being deeper, is to be mined by underground mining methods. It is planned to use the short-wall retreating mining method.

## INTRODUCTION

The Baghalchur site is located about 40 km north-west of Dera Ghazi Khan (Fig.1).<sup>1</sup> It is connected with D.G. Khan by 30 km of metalled road and 55 km of dirt road. The terrain is rather rugged and altitudes range from 500 to 1000 m above sea level. The climate is semi-arid desert type with an average annual rainfall of 12 cm. Temperatures are extreme both in summer and in winter. The area is thinly populated and the education level low.

Uranium was discovered in the Baghalchur area, D.G. Khan district, in 1963. Subsequently some trenching, pitting and aditing was done to:

- (i) Study the mode of occurrence of mineralization;
- (ii) Determine the subsurface extensions of the mineralization; and
- (iii) Obtain the samples for mineralogical studies and metallurgical testing.

Encouraged by the favourable results of these preliminary investigations a limited drilling programme was started in 1968 to determine the subsurface

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<sup>1</sup> For technical reasons it was not possible to reproduce the remaining five figures of Dr. Khan's paper. Readers interested in obtaining copies of these should contact Dr. Khan direct at the address given in the List of Participants at the end of these Proceedings.

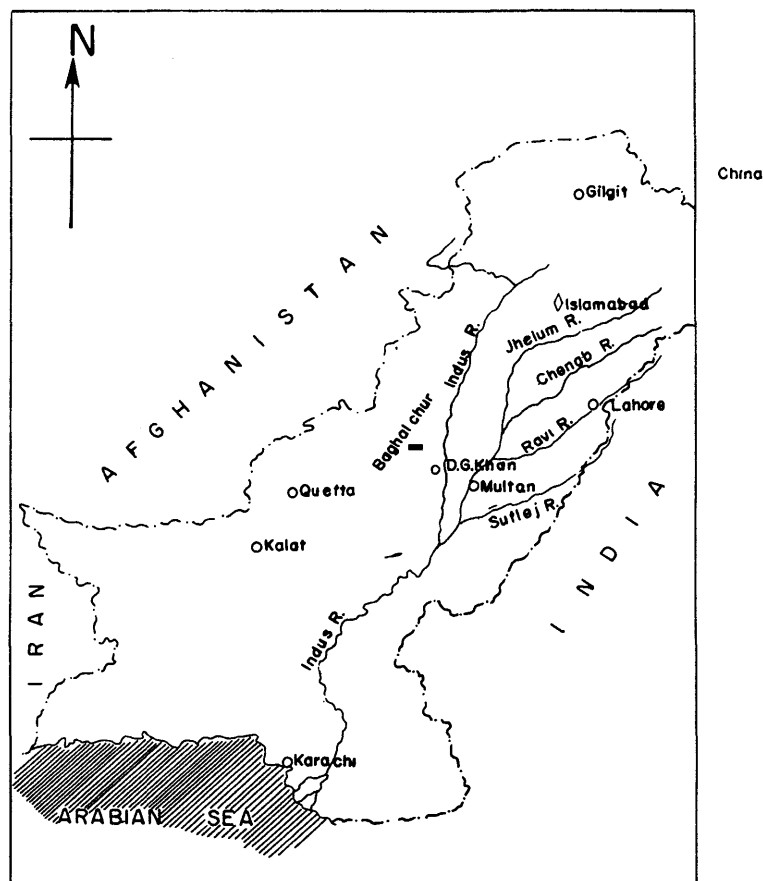


FIG.1. Map of Pakistan showing location of the mines.

extension(s) of uranium mineralization. Portable drills had to be used since all equipment had to be transported by camel, owing to the inaccessibility of the area. However, in 1971 UNDP/SF assistance was made available and a road was constructed for four-wheel traffic and truck- or skid-mounted rigs were introduced. The drilling operations for developing the deposit and for further exploration are still in progress.

As a result of exploratory drilling, two subsurface mineralized paleo-channels have been located. A small-scale surface mining operation has been started on the ore-body in one of the paleo-channels and plans to exploit the ore-body in the other paleo-channel are in hand.

## GEOLOGY

The rocks in the area belong to Siwalik System and range in age from Middle Miocene to Lower Pleistocene. The Siwalik System is divided into three groups – the Lower, Middle and Upper Siwaliks.

The Middle Siwalik group comprises grey-coloured medium- to coarse-grained, non-homogeneous arkosic sandstones, interbedded with grey and brown shales. These arkosic sandstones are believed to have been deposited as continental sediments.

The uranium mineralization is restricted to a sandstone unit locally known as Baghalchur Sandstone. This unit is about 60–75 m thick and is underlain by grey shale and overlain by brownish red shale.

Generally, at the bottom of the Baghalchur sandstone, there are lenses of conglomerate grit. These lenses are usually 0.6–1.0 m thick, but in places are up to 3 m thick. The conglomerate grit is composed of rounded to sub-rounded pebbles of quartzite and limestone in a sandy, silty or clayey matrix. The grit lenses may persist for hundreds of feet in the north-south direction but seem to pinch out in the east-west direction. They mark the base of paleo-channels.

Two paleo-channels have been discovered in the area. Channel No.1 is in the east while channel No. 2 is in the west, about 1–25 km from channel 1. The eastern channel is about 25 m stratigraphically higher than channel 2. These channels are braided and seem to have many small tributaries.

## EXPLORATION RESULTS

The exploratory drilling was initially done on a systematic grid pattern of 16 × 16 m spacing and later was modified to 32 × 16 m spacing. The following information has been revealed by drilling:

- (i) Uranium deposition is controlled by sedimentary structures, primarily sandstone-filled paleo-stream channels;
- (ii) The rocks dip about 9° SE, and the slope of the ore-body is about 6° southwards;
- (iii) The mineralization is spotty and irregular. The lenses vary in thickness from a few centimetres to 1 m in channel 1 and up to 2.0–2.5 m in channel 2;
- (iv) The thickness and grade of mineralization change abruptly;
- (v) The mineralization also occurs in different stratigraphic levels with intervening barren or low-grade sands;
- (vi) Oxidized uranium ore occurs above the water table in the form of tyuyamunite  $\text{Ca}(\text{UO}_2)_2(\text{VO}_2)_2 \cdot 5-8 \text{H}_2\text{O}$  whereas in the unoxidized zone below the water table it is found as uraninite and coffinite;

- (vii) The oxidized ore is in positive disequilibrium and thus the actual chemical grade is approximately 25% higher than the radiometric grade;
- (viii) The waste-to-ore ratio in channel 1 has been calculated to vary from 10:1 to 15:1 but is much more in the channel 2 area.

## OPEN-CAST MINING OPERATION

As the overburden in the area of paleo-channel No. 1 was very shallow, ranging from 0–13 m, a small-scale open-cast mining operation was started to exploit this ore-body. An account of the method and practices used in the mining operation is given below.

### Stripping

As mentioned above, the overburden thickness ranges from 0–13 m and consists of semi-consolidated sandstone with some lenses of hard sandstone 0–3 to 0–6 m thick. The conventional open-pit mining techniques are not very applicable to the major part of this deposit as the overburden is shallow and the area is dissected by natural drainage channels dividing the ore-body into a number of natural blocks. The stripping is being done in parts, and for this purpose the area between two drainage channels is taken as one unit. Hence, for the major part of the deposit the overburden is stripped as in a quarrying operation since most of it is higher than the bed level of the drainage channels.

Part of the overburden is being removed by a contractor and part by the PAEC. The stripping practice adopted by the contractor is as follows:

The area to be stripped is drilled with a wagon drill at  $2.25 \times 2.25$  m grid pattern using 62-mm-dia. bit. The depth of the holes is 3.50 m except in the last cut which is close to the mineralization. The holes are loaded with Wabonit, a TNT-sensitized ammonium nitrate powdery explosive, primed by Wabox 80%, a gelatinous N.G. high explosive. The firing is done with Wabocord (instantaneous detonating fuse). The blast rock is loaded and hauled by bulldozer-towed scrapers to the waste disposal area.

The PAEC practice adopted for the stripping operation is as follows:

The drilling is done with hand-held jack-hammers. The diameter of the holes to start with is 40 mm and after 1 m the

depth is reduced to 37 mm. The depth of the holes is 2.0 to 2.5 m. The drilling pattern adopted is  $1.50 \times 1.50$  m; and blasting is done similar to that explained above. The blast rock is loaded with  $2\text{-m}^3$  loaders into 18-t dump trucks for hauling to the waste dump.

At places the rocks have numerous cracks and blasting is not effective. Hence, at such places, the rock is removed simply by ripping with bulldozers equipped with rippers.

The overburden removal by the contractors and the PAEC is stopped at approximately 0–60 m from the top of the mineralized lenses.

### **Ore mining**

After the overburden is removed the area is again drilled at a  $1.50 \times 1.50$  m grid. The holes are then probed with Geiger-Müller type probes to recheck the mineralization level and to determine the thickness and configuration of the ore-body. The waste rock is carefully peeled off with bulldozers. After the ore is exposed the thick ore lenses are ripped with a bulldozer. But, should the ore lenses be small and not very thick, the ore is removed by pneumatic breakers. This practice is considered necessary in order to minimize dilution which is unavoidable if bulldozers are used at such places. The ore mined is loaded with the help of front-end loaders into 9-ton dump trucks. The grade of the load is checked and the ore hauled to a stock pile of relevant grade.

### **Grade control**

Great emphasis is placed on grade control during the mining operation since the deposit is low grade, and if the necessary care is not exercised there would be considerable dilution and loss of ore. To minimize dilution and to avoid loss of ore the following steps are taken for grade control:

- (i) As mentioned earlier the exploratory drilling had been done on  $16 \times 16$  m grid pattern. However, it was revealed that there were lenses smaller than 16 m, which had escaped detection during exploratory drilling. Therefore, all the blast holes drilled for overburden removal are probed by GM back-hole probes to locate any mineralized pods not detected during exploratory drilling.
- (ii) On the basis of the information available about the depth of the mineralization, a strict control on the last cut of the overburden is exercised by determining the levels frequently to avoid accidental exposure of the mineralization and its consequent mixing with overburden. After the removal of overburden

secondary drilling is done to penetrate the mineralization. These holes are also probed to determine the configuration of the ore lenses and the information so obtained is used to segregate the ore of different grades.

- (iii) The loaded trucks are probed at five places with T-probes and the average of the five readings is taken as the grade of the truck load.
- (iv) Small-capacity trucks are used exclusively for ore haulage to minimize the mixing of ores of various grades and also for obtaining relatively more accurate results with T-probes.

## UNDERGROUND MINING OPERATION

The unoxidized ore-body occurring in channel 2 is relatively deep-seated and the overburden ranges from 30 to 300 m. It is planned to exploit this ore by underground mining.

It is planned to exploit the ore-body in parts by approaching it through three inclines of  $2.7 \times 2.7$  m which would be connected through a cross-drift passing through the ore and following the same gradient as that of the paleo-channel. Two of these inclines would serve as a haulage-way cum air intake and the third incline would serve as an exhaust cum-escape route. It is planned to mine by the retreat short-wall mining method.

For support timber stulls shall be installed and the waste rock encountered during mining shall be packed in the excavated area. The ore and any excess waste rock (which cannot be accommodated underground) would be hauled from the mine face to the surface in 2-ton rubber-tyred buggies.

## CONCLUSION

This mining operation is the first of its kind in the country. It is a small-scale operation and is basically experimental in type. Apart from yielding raw material for the country's energy development programme, it would also result in manpower training which could be utilized for future national mining operations.

## ACKNOWLEDGEMENTS

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## DISCUSSION

A.E. BELLUCO: Am I right in thinking that the mineral is in disequilibrium? If so, are any corrections performed on the gamma-logs or other measurements used for evaluating the reserves in the deposit and for controlling production?

Haq Nawaz KHAN: We have a calibration drum containing ore of known chemical grade. The probes are calibrated almost every day in this drum and corrected.

R. WADLEY: Could you tell me please what extraction process you plan to employ at Baghalchur?

Haq Nawaz KHAN: The extraction process is to be acid leach with solvent extraction.

R.G. WADLEY: Can you give an indication of what you expect your mining, milling and extraction costs per tonne to be?

Haq Nawaz KHAN: The mining costs with open-pit operation based on an ore production of 100 t/d are around \$16.50/t.



## **ТЕХНИЧЕСКИЕ СРЕДСТВА И ТЕХНОЛОГИЯ КОНТРОЛЯ КАЧЕСТВА УРАНОВЫХ РУД В ПРОЦЕССЕ ИХ ДОБЫЧИ И ПЕРВИЧНОЙ ПЕРЕРАБОТКИ**

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### **Abstract – Аннотация**

#### **FACILITIES AND TECHNOLOGY FOR URANIUM ORE QUALITY CONTROL DURING EXTRACTION AND INITIAL PROCESSING.**

The paper surveys and generalizes various radiometric methods used to secure the highest possible uranium ore quality during extraction and initial processing. The radiometry equipment and other technical facilities used to enhance the quality of commercial-grade ores sent for chemical conversion are described. Actual data are quoted for ore impoverishment reduction, enhanced yields of unconditioned ore mass and enhancement of uranium concentrations in the commercial-grade ore. The effectiveness of each method is discussed.

#### **ТЕХНИЧЕСКИЕ СРЕДСТВА И ТЕХНОЛОГИЯ КОНТРОЛЯ КАЧЕСТВА УРАНОВЫХ РУД В ПРОЦЕССЕ ИХ ДОБЫЧИ И ПЕРВИЧНОЙ ПЕРЕРАБОТКИ.**

В докладе обобщена практика применения различных радиометрических методов с целью обеспечения максимально возможного повышения качества урановых руд в процессе их добычи и первичной переработки. Приведено описание радиометрической аппаратуры и других технических средств, используемых для повышения качества товарных руд, направляемых в химпереработку. Даны фактические данные по сокращению разубоживания руд, увеличению выходов некондиционной горнорудной массы и повышению содержания урана в товарной руде. Сделаны выводы об эффективности каждого из методов.

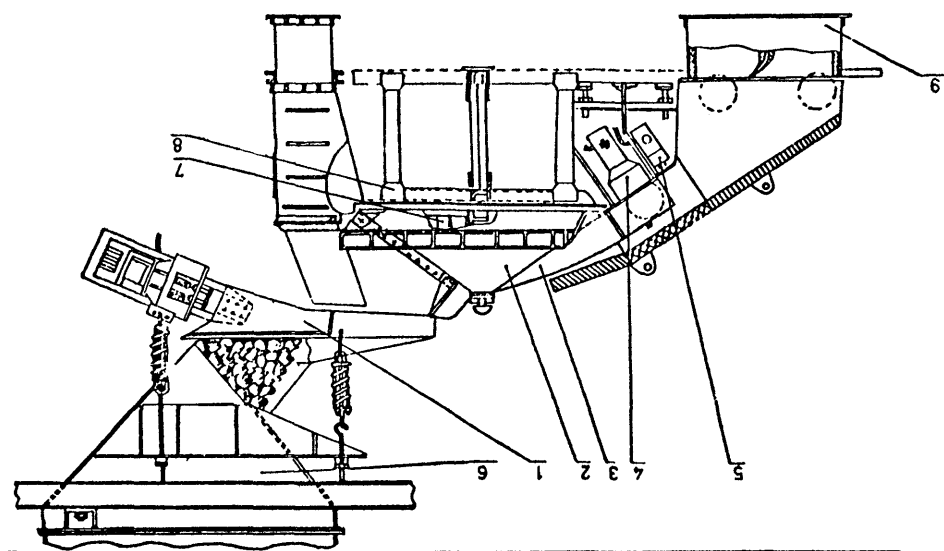


Рис. 1. Схема контроля качества урановых руд при их добыче и первичной обработке.

Развитие атомной энергетики в значительной мере связано с проблемой увеличения производства дешевого атомного горючего. Последнее же требует увеличения объемов добычи урановой руды, а следовательно и производительности рудников, что в свою очередь обуславливает применение систем отработки месторождений с массовыми способами отбойки руды и высоким разубоживанием. Гидрометаллургическая переработка таких руд характеризуется высокой стоимостью окиси-закиси урана.

Одним из направлений решения экономической проблемы атомной энергетики является снижение себестоимости химконцентрата за счет повышения качества урановых руд.

В докладе сделана попытка обобщить теорию и практику применения методов радиометрии с целью обеспечения максимально возможного повышения качества руд при их добыче и первичном обогащении.

На рис. 1 приведены схемы контроля качества урановых руд для систем с селективной (слева) и массовой (справа) отбойкой руды. Римскими цифрами показаны радиометрические методы, способствующие повышению качества руды, направляемой на гидрометаллургическую переработку. Остановимся более подробно на каждом из этих методов.

Известно, что по гамма-излучению урановых руд можно определять концентрацию урана радиометрическим методом непосредственно в естественном залегании.

Метод же определения содержания урана и мощности оруднения на обнаженной поверхности рудного тела условно назван радиометрическим гамма-опробованием, а те же определения в шпурах или скважинах — шпуровым гамма-опробованием, или гамма-каротажем скважин [1-3].

*Гамма-опробование забоев* применяется, в основном, при селективной отбойке руды и породы для выделения контуров рудных тел, в соответствии с которыми и обустраивается забой. Выделение гамма-излучения анализируемых участков при гамма-опробовании осуществляется или способом разностного эффекта с помощью специальных свинцовых экранов, установленных на детекторах переносных радиометров УР-4М, РПР-1, ПРК, ПРШ-4 и других, или радиометрами направленного приема типа РГН-2А, ПРН-4 и т.д. Радиометры направленного приема обладают резко анизотропной чувствительностью к гамма-излучению и двухканальной (основной и компенсационный) разностной измерительной системой, на выходе которой автоматически выдается разность показаний, фиксируемых каждым из детекторов.

*Шпуровое гамма-опробование* применяется для количественной оценки оруднения и определения границ рудных тел в шпурах, предназначенных для взрывных работ. Контроль шпуров осуществляется только для их правильной последующей зарядки взрывчатым веществом с целью селективной отбойки руды и породы.

Возможность получения результатов непосредственно на месте измерений, а также простота и дешевизна открыли путь для широкого применения этих методов опробования с целью повышения качества руды.

Привести конкретные данные по уменьшению разубоживания добываемых руд и повышению их качества (содержания урана) за счет использования радиометрических методов гамма-опробования на очистных работах, к сожалению, не представляется возможным, так как эти методы органически вошли в технологию ведения очистных работ с селективными способами отбойки руды с самого начала развития урановой промышленности.

*Гамма-каротаж скважин* применяется при разработке месторождений системами с массовой отбойкой руды, например, таких, как камерные системы с подэтажными штреками (ортами), или при открытой добыче урановых руд.

Взрывные скважины перед зарядкой взрывчатым веществом подвергаются гамма-каротажу с целью определения границ оруднения и выбора мест закладки взрывчатого вещества. Это позволяет произвести взрывные работы с минимально возможным разубоживанием и перемешиванием горнорудной массы.

Причем, на открытых горных работах взрывание рудной массы осуществляется в зажатой среде только на встряхивание, сохраняя естественный характер оруднения.

Для каротажа подземных скважин используются каротажные радиометры типа КРЛ-М ("Истра"), ПРКС ("Виток"), ПРШ-4 ("Дедал") с каротажным блоком детектирования [1]. Гамма-каротаж взрывных скважин на карьере осуществляется, как правило, каротажными станциями, оборудованными на автомашинах-вездеходах УАЗ-469.

В результате проведения гамма-каротажа скважин и селективной отпалки породы и руды на одном из месторождений по отдельным участкам камеры получено существенное (на 18 %) обогащение добытой руды по сравнению с валовой отпалкой на всю глубину скважин. При этом, выход хвостов (некондиционной горнорудной массы) увеличился в 1,38 раза.

*Радиометрическая сортировка в забое* отбитой горнорудной массы в ковшах экскаваторов или в емкостях самоходных погрузочно-доставочных машин применяется как способ первичного обогащения руд. Эта технология начала усиленно развиваться по мере использования на урановых рудниках и в карьерах большегрузных вагонеток, автосамосвалов и железнодорожных думпкаров. Дело в том, что с увеличением порций, по которым осуществляется сортировка руды, ее радиометрическая контрастность, или степень неравномерности распределения радиоактивного компонента, уменьшается. В то же время применение радиометрической сортировки отбитой горнорудной массы в забое, в свою очередь, позволило пересмотреть системы разработки месторождений или отдельные элементы этих систем, а в ряде случаев — отказаться от малопроизводительных систем горных работ с селективной отбойкой руд и перейти к высокопроизводительным системам с валовой и полуваловой отбойкой руды.

Радиометрическая сортировка отбитой горнорудной массы в ковшах экскаваторов осуществляется с помощью специальных приборов типа КР-2 и КР-4, состоящих из пульта, устанавливаемого в кабине машиниста экскаватора, и детектора (детекторов), смонтированного с применением специальных амортизирующих устройств на ковше одно-, трех- и четырехкубовых экскаваторов в свинцово-стальных защитных экранах.

Об эффективности ковшовой сортировки горнорудной массы можно судить по следующим данным. Средние коэффициенты разубоживания руды на открытых работах до внедрения ковшовой сортировки составляли 25,1-31,0 %, а по отдельным блокам величина разубоживания достигала 48 % при нормативном разубоживании, равном 20 %. Внедрение ковшовой радиометрической сортировки позволило их снизить до средней величины, равной 19,2 %, и максимальной величины, равной 22,8 %.

Как уже отмечалось выше, при неизменном граничном содержании разделения горнорудной массы с увеличением эквивалентного веса сортируемых порций руды уменьшается выход хвостов сортировки и снижается содержание урана в товарной руде. Так, практическая замена на одном из карьеров шестикубового ковша экскаватора на четырех- и трехкубовые привела к увеличению выхода хвостов сортировки на 3 % и 5-5,5 %, соответственно. При этом содержание урана в руде увеличилось на 3,3 % и 5,7-6,0 %, а разубоживание снизилось на 2,9 % и 5,2 %, соответственно.

Повышению эффективности радиометрической сортировки в ковшах экскаваторов в ряде случаев способствует уменьшение высоты уступа экскавации (например, разбуривание и взрывание при высоте уступа 15 м, а экскавация горнорудной массы в 2 или 3 подустапа).

ТАБЛИЦА I. ЭКСПЕРИМЕНТАЛЬНЫЕ ДАННЫЕ ПО СОРТИРОВКЕ ОДНОЙ И ТОЙ ЖЕ ГОРНОРУДНОЙ МАССЫ В ЕМКОСТЯХ ПОГРУЗОЧНО-ДОСТАВОЧНЫХ МАШИН ДО РУДОСПУСКА И В ВАГОНЕТКАХ ПОСЛЕ РУДОСПУСКА

Место- рождение	Система отработки	Выход хвостов сортировки (%)		Увеличение хвостов (%)	Примечание
		забойная сортировка	РКС		
А	Горизонтальные слои, сверху вниз	24,5	7,9	16,6	усредненные данные
	Подэтажное магазинирование	11,5	2,0	9,5	усредненные данные
Б	Подэтажное и слоевое (частично) обрушение	53,0	34,0	19,0	отдельные эксперименты
		4,2	1,7	2,5	
		27,0	0,0	27,0	
		17,3	10,8	6,5	
		25,4	11,6	13,8	усредненные данные

При подземных способах отработки урановых месторождений радиометрическая сортировка в забое имеет место только при использовании погрузочно-доставочных машин типа ЛБ-125/1000 и МПДН-1, которые оборудуются специальными радиометрами типа "Карат" или РПДМ-3. Определение качества горнорудной массы, загруженной в кузов машины, осуществляется в момент ее движения к рудоспускам и в зависимости от качества (руда или хвосты) машина разгружается в тот или другой рудоспуск. Забойная сортировка устраняет перемешивание горнорудной массы в рудоспуске, являясь практически первичным обогащением добытой руды.

В табл. I приведены сравнимые экспериментальные данные по забойной сортировке до рудоспуска и по гамма-анализу в вагонетках этой же горнорудной массы после рудоспуска.

Отсюда видно, что радиометрическая сортировка до рудоспуска увеличивает выход хвостов в среднем на 9,5-16,6 %, что приводит к увеличению содержания урана в руде на 8-15 %. Однако радиометрическая сортировка в забое применяется далеко не всегда, т.к. не везде используются погрузочно-доставочные машины, еще применяются самосвалы небольшой грузоподъемности и, наконец, бывают случаи, когда из-за характера оруднения забойная сортировка малозффективна.

ТАБЛИЦА II. РЕЗУЛЬТАТЫ КОНТРОЛЬНОГО ОПРОБОВАНИЯ РУД РАЗЛИЧНЫХ ТИПОВ

Тип руды	Число измеренных емкостей	Относительная ошибка (%)			
		всей партии руды	средняя единичного анализа	максимальная единичного анализа	
I	59	-0,7	± 31,0	+90,0	-36,0
II	50	+1,4	± 23,0	+72,0	-31,4
III	137	0,0	± 6,9	+22,5	-26,6
IV	85	-5,1	± 6,9	+43,3	-66,7
V	42	-6,0	± 22,7	+38,5	-62,3
VI	18	+5,8	± 22,4	+91,3	-36,9
VII	25	+1,2	± 24,0	+36,0	-43,8
VIII	7	-1,8	± 18,1	+56,2	-43,4

*Гамма-экспресс-анализ* горнорудной массы в транспортных емкостях (вагонетках, автосамосвалах и т.д.) давно стал неотъемлемой частью горно-эксплуатационного комплекса работ для быстрого опробования всей горнорудной массы. Высокая точность опробования руд в емкостях и возможность оперативного изменения значений бортового содержания в продуктах сортировки дают основание рассматривать экспресс-анализ руд, так же как и радиометрическую сортировку в забое, как первичный способ обогащения руд.

Радиометрические контрольные станции (РКС) для гамма-экспресс-анализа устанавливаются как под землей, так и на поверхности для сортировки горнорудной массы и позабойного учета добытого урана.

РКС оборудуются радиометрами типа РСР-3 ("Стрела") с блоком автоматической сортировки, РСР-6 ("Стенд"), РСР-7 ("Кварц") с батарейным питанием, которые имеют детекторы на газоразрядных счетчиках, и типа СГ-2М ("Старт"), СРК-2 ("Шифр") с цифropечатающим устройством, УРКС и другими, которые снабжены сцинтилляционными детекторами (от 2-х до 6-ти штук) [1, 4].

При необходимости РКС оборудуются весами, а в последние годы — приставками для записи данных анализа на магнитную ленту или перфораторами и теле-тайпными аппаратами для передачи данных на ЭВМ с целью обработки.

Погрешность гамма-анализа зависит от неравномерности распределения урана в руде, размера измеряемой порции, состояния радиоактивного равновесия, радоно-выделения из руд, метеорологических условий и других факторов и может достигать по отдельным анализам  $\pm (20-30) \%$ . Однако ошибки анализа партии руды не превышают  $\pm (5-6) \%$  и лежат в пределах технических условий для химических анализов урановых руд [4] (см. табл. II).



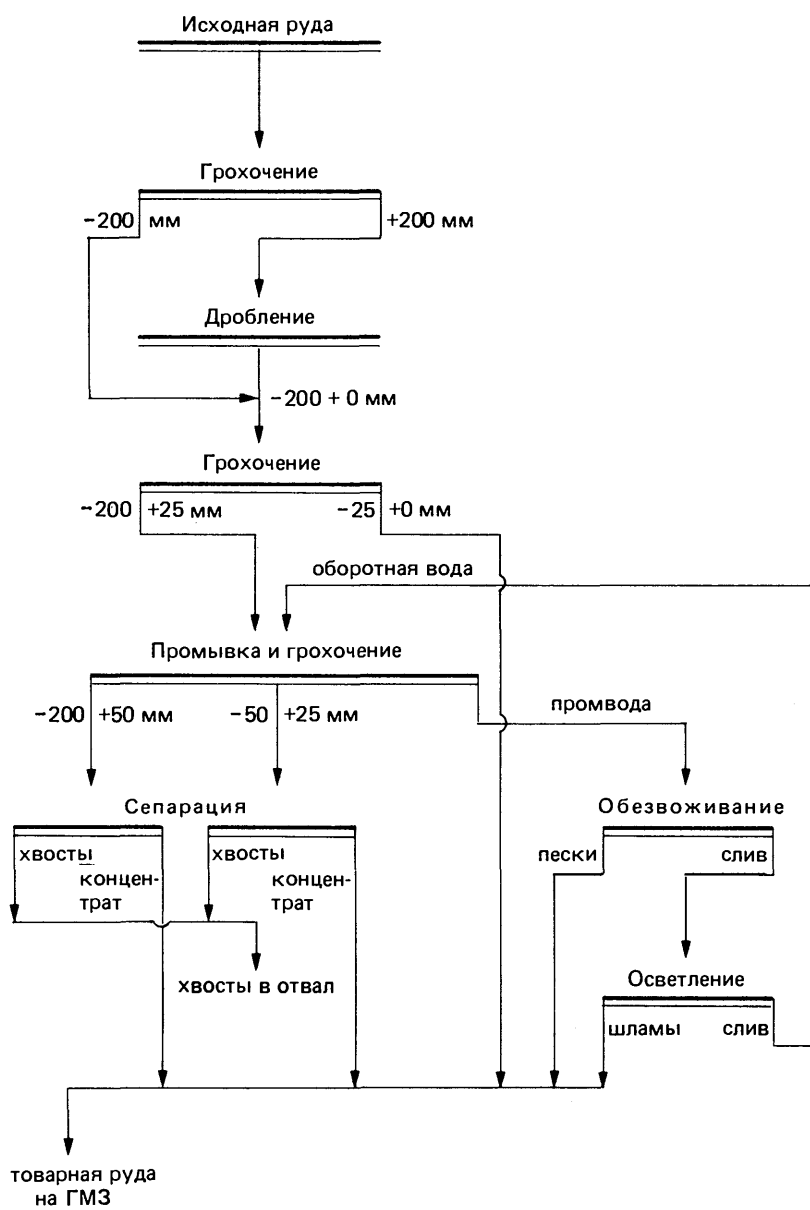
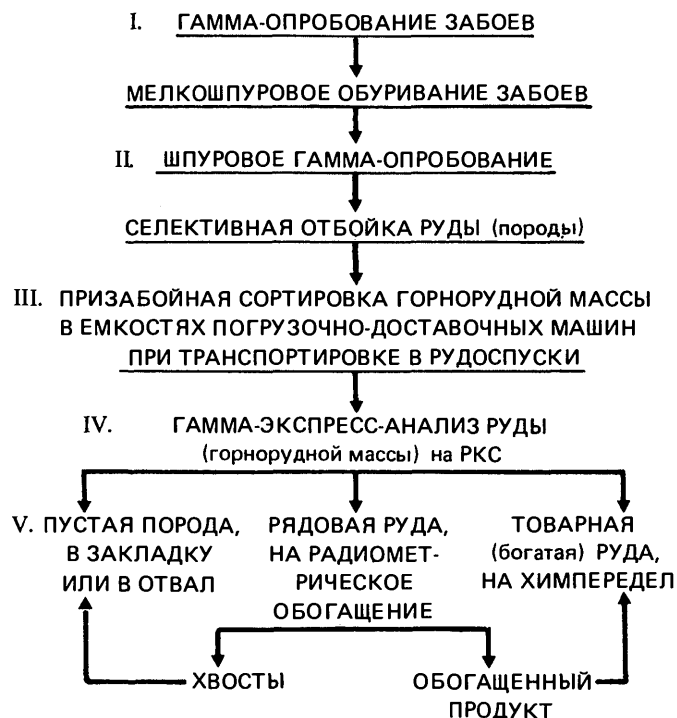


Рис. 2. Технологическая схема обогащения руды.

# А. СИСТЕМЫ ОТРАБОТКИ С СЕЛЕКТИВНОЙ ОТБОЙКОЙ РУДЫ



# Б. ОТКРЫТЫЕ И ПОДЗЕМНЫЕ СИСТЕМЫ ОТРАБОТКИ С МАССОВОЙ ОТБОЙКОЙ РУДЫ

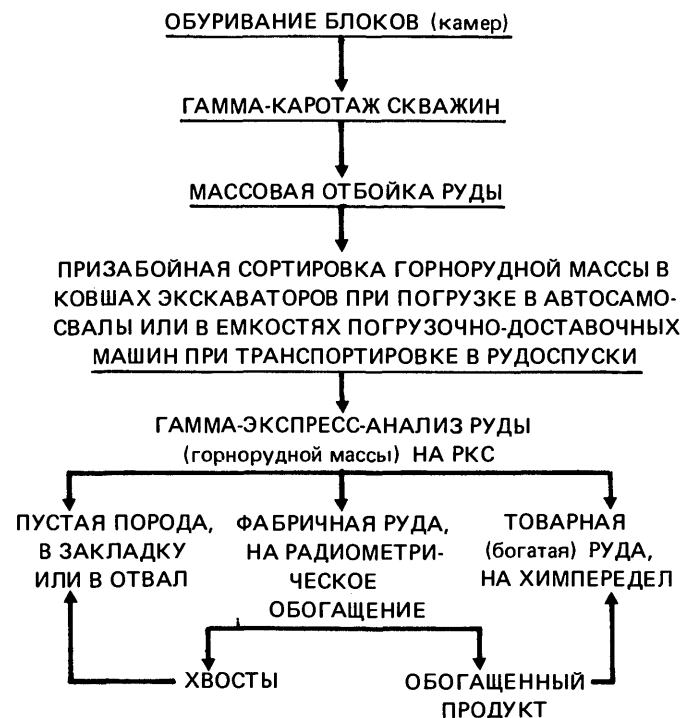


Рис. 3. Сепаратор для руды крупностью  $-200+50$  мм,

1 – виброуплотнитель, 2 – конусный раскладчик, 3 – неподвижная спираль конусного раскладчика, 4 – датчик радиометра, 5 – электропневмоклапан, 6 – бункер, 7 – привод раскладчика, 8 – рама сепаратора, 9 – сортировочная течка.

Как уже отмечалось выше, при отсутствии на руднике забойной сортировки ее функцию выполняет РКС, через которую, как правило, проходит вся горнорудная масса. Объем хвостов, которые отсортировываются в этом случае на РКС, составляет от 0 до 34 % (см. табл. I).

Таким образом, использование в технологическом цикле горно-эксплуатационных работ радиометрических методов и применение различных типов электронной аппаратуры обеспечивают существенное первичное обогащение урановых руд как в процессе ведения буровзрывных работ, так и при транспортировке и выдаче руды "на-гора".

Последняя, перед химперделом, и более тщательная сортировка или *радиометрическая сепарация (обогащение)* выделенной на РКС фабричной руды, осуществляется на радиометрических обогатительных фабриках (РОФ) [5].

Радиометрическая сепарация основана на использовании того же свойства урановых минералов, на котором базируются гамма-опробование и каротаж скважин, призабойная сортировка и экспресс-анализ руд в емкостях. Радиометрическая сепарация примерно для половины типов урановых руд является не только первичным методом их переработки, но и единственным процессом их механического обогащения.

Главное достоинство этого обогатительного процесса заключается в том, что он не требует реагентов, воды, топлива, а также измельчения руды, очень мало потребляет электроэнергию, практически не дает вредных выбросов в окружающую среду и поэтому обходится очень дешево.

Наиболее существенным недостатком радиометрической сепарации является ограниченная возможность обогащения мелких классов руды. Поэтому в настоящее время обогащаются, в основном, куски крупностью более 20 мм, хотя и есть случаи высокоэффективной сепарации класса крупности  $-20 + 15$  мм и даже  $-15 + 5$  мм. Вот почему при ведении буровзрывных работ очень важно не переизмельчать руду, а наоборот, добиваться максимально возможного выхода наиболее благоприятного для сепарации класса  $-150 + 30$  мм.

Технология радиометрического обогащения имеет свои особенности. На рис. 2 приведена технологическая схема РОФ, из которой видно, что она предусматривает нижеперечисленные основные операции [6].

*Дробление*, которому отведена вспомогательная роль. Его задача — обеспечить максимально допустимую для сепараторов крупность кусков, обычно не более 200-250 мм. Дроблению подвергаются только наиболее крупные куски  $+200$  мм.

*Грохочение* — одна из основных подготовительных операций для выделения несортируемой мелочи (например, класса  $-15$  мм) и разделения сортируемого материала на два-три "машинных" класса крупности с целью устранения резкого различия кусков по весу.

*Промывка* "машинных" классов с целью удаления с кусков радиоактивного шлама, который обволакивает кусковой материал, из-за чего куски пустой породы становятся радиоактивными и не выделяются в хвосты. Особенно эффективна

промывка для мелкого и среднего классов крупности. Иногда операции грохочения и промывки объединяются, т.е. осуществляются одним технологическим аппаратом.

*Сепарация* "машинных" классов, т.е. разделение их на концентрат (товарную руду) и отвальные хвосты, осуществляется на рудосортировочных машинах или радиометрических сепараторах. В настоящее время в промышленности используются следующие конструкции сепараторов: ленточные, у которых куски транспортируются и измеряются на ленточных конвейерах; вибрационные, которые аналогичны ленточным, только вместо конвейера применяется виброустройство; конусные, у которых куски подаются в зону измерения с помощью вращающегося конуса и направляющей спирали; и "сепараторы свободного падения", у которых подача кусков осуществляется комбинированным методом: с использованием вибрационного питателя и раскладчика, а также стабилизирующего транспортера, а их измерение осуществляется в свободном падении, когда они пролетают мимо детекторов радиометра. Несмотря на многообразие решений все сепараторы имеют основные узлы одного и того же назначения, а именно (см. рис.3):

- выходную часть бункера (6), обеспечивающую равномерный выход из бункера сортируемого материала;
- вибрационный питатель (1) с электромагнитным приводом;
- конусный раскладчик (2) с неподвижной спиралью (3) для равномерной раскладки и растяжки потока кусков перед зоной измерения;
- радиометр со сцинтилляционными детекторами (4) и счетно-решающим устройством для количественного определения содержания радиоактивного компонента в кусках с учетом их веса и выдачи команды;
- исполнительный механизм (5) (пневмоклапан или шиберное устройство) для механического разделения кусков на продукты сортировки;
- сортировочную течку (9) для сбора продуктов сортировки и передачи их на соответствующие конвейеры.

Принцип действия сепаратора ясен из приведенного рисунка.

Производительность сепараторов колеблется от 40 до 100 т/ч на рудах крупностью  $-200+50$  мм, 10-15 т/ч – на классе  $-50+25$  мм и 4-5 т/ч – на классе крупностью  $-25+15$  мм.

Действующие в настоящее время РОФ выводят в отвал от 25 до 43% хвостов от исходной фабричной руды, поступающей на РОФ, или от 50 до 80% от сортируемого класса крупности, выход которого по различным типам руд колеблется от 45,0 до 64,5%. Содержание урана в хвостах радиометрической сепарации колеблется от 0,010 до 0,015%. При этом коэффициент обогащения конечного продукта, направляемого на химпердел, составляет от 1,25 до 1,64.

Таким образом, применяемые технологические приемы повышения качества добываемых урановых руд за счет широкого использования радиометрических методов и приборов в процессах добычи и первичной переработки этих руд позволя-

ют повысить содержание урана в товарной руде, направляемой на химпереработку, в 1,5-2,0 раза, а иногда и более, и за счет этого значительно снизить стоимость химконцентрата — основного сырья для получения ядерного горючего.

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### DISCUSSION

A.E. BELLUCO: In Argentina we use a method very similar to that described by Mr. Temnikov in that we perform geological control of the mining and production of a uranium deposit in order to reduce dilution and losses of uranium to a minimum. In our experience such methods used for the Los Adobes (Patagonia), Tigre I and Tigre III (Sierra Pintada) deposits have given excellent results as regards estimation of the original reserve and final production. However, I would like to ask him whether such methods can be applied for all types of deposit. Are there not limitations on their use for ores which are not in equilibrium?

M.A. TEMNIKOV: I am glad that we have picked on the same method independently; this suggests that the system used in both our countries for controlling and increasing the quality of the ore mined must be a good one. I believe that radiometric methods of quality control for mined ore should be used on all uranium deposits, irrespective of the extent to which the ores are in radioactive equilibrium. Shifts in radioactive equilibrium need to be compensated by the introduction of appropriate correction factors; we do in fact do this.

V. ZIEGLER: I would like to ask you about radiometric sorting of uranium ores in a preconcentration plant after extraction from the mine or the quarry. Is this technique used on all deposits in the Soviet Union or only on those with particular characteristics, such as highly contrasted contents? I ask because the use of this process on two deposits in France over a number of years proved to be uneconomic.

M.A. TEMNIKOV: Radiometric sorting can of course be used only for those ores for which it is suitable. Radiometric sorting plants are designed only after thorough studies of the extent to which ores can be enriched have been performed and after the economic viability of the method has been evaluated.

V. ZIEGLER: But radioactive disequilibrium upsets the sorting process: if the radioactivity is too high in relation to the content, it will be uneconomic to treat the ore sent to the processing plant; if, on the other hand, it is too low, ore that would have been economic to process will end up as tails.

M.A. TEMNIKOV: You are right that disequilibrium between uranium and radium in ores is a disadvantage, but it is not of overriding importance. The radiometric devices in the separators are calibrated on the basis of the tails, that is, on the basis of the actual disequilibrium. If the coefficient of disequilibrium varies, it is necessary to check the radiometric calibration more often.

P.W.J. VAN RENSBURG: Down to what grade can your sorter separate values in ore in the + 50 mm material?

M.A. TEMNIKOV: The size class  $-200 + 50$  mm can be sorted with an effectiveness of 85–95%, depending on the contrast in the ore. The uranium content in tails is as a rule less than 0.01%.

P.W.J. VAN RENSBURG: Have you got a machine that can sort  $-25$  mm material?

M.A. TEMNIKOV: Yes, we have. It uses a relay system of measurement, transmission and storage of data on gamma activity. The piece being sorted passes a series of up to nine detectors, which gradually increase the amount of information on radioactivity. In this way the sensitivity of the ore-sorting machine can be increased.

P.D. TOENS: There is little doubt that it is necessary to have calibration facilities for both borehole and surface radiometric sampling. In South Africa we do have such facilities. When these are used, provided the ore is in equilibrium, we are able to estimate the uranium content to within 10% of its chemical value.

M.A. TEMNIKOV: Certain mining organizations in the Soviet Union also use standard boreholes; when these are made, ore with a known uranium content from the deposit being investigated is used. The degree of accuracy you mention is perfectly normal and sufficient for the purpose; our borehole logging equipment is equally accurate.

D.M. TAYLOR: "Standard" boreholes are now used in several countries, but care should be taken when using them for calibrating logging probes. NEA/IAEA are sponsoring an international project on borehole logging, and the initial results of a comparative study of "standard" boreholes in various countries have shown considerable differences in the data obtained, depending on the logging equipment and methods used. Important factors are the grain size of the material used for the borehole and the water content of the material — I was interested to note that Mr. Temnikov also corrects radiometric data for humidity.

## LA MINERIA EN LAS PIZARRAS URANIFERAS DE Saelices el Chico (SALAMANCA)

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### Abstract-Resumen

#### MINING OF THE URANIFEROUS SHALES OF Saelices el Chico (SALAMANCA).

In July 1974 ENUSA (the National Uranium Company) started mining operations in the FE-1 deposit in the municipal district of Saelices el Chico, Salamanca province. This is a deposit of the Stockwork type and is embedded in Cambrian shales of a distinct degree of metamorphism. The primary mineralization is found filling fissures and fractures of only slight longitudinal development. The secondary minerals were deposited in the zones affected by the hydrostatic level. The investigation of the deposit was carried out by the Spanish Nuclear Energy Board (JEN). In its final phases it consisted of a control drilling campaign and a wagon-drill campaign. The deposit was evaluated by the zone-of-influence method in accordance with the control drillings and the wagon drillings. The overall rate of production was 1 200 000 t per annum, of which 200 000 t were ore with a  $U_2O_8$  content of 0.08%. The output of the processing plant was 130 t  $U_3O_8$ . In 1976 ENUSA undertook a mining project of much greater scope in consequence of the decision to expand the ore processing plant (objective of the Quercus project). The annual output of the plant in question will be 600 t  $U_3O_8$ . For this the production of ore will be increased to 825 000 t/a (average grade 0.075%) and the total amount removed will be 9 million tonnes per annum. Account is taken of the whole of the FE deposit, which includes FE-1 and FE-3 and their respective extensions. The evaluation of the deposit was made both on the basis of the results of the control- and wagon-drillings and by the Kriging statistical method.

#### LA MINERIA EN LAS PIZARRAS URANIFERAS DE Saelices el Chico (SALAMANCA).

En el mes de julio de 1974 la Empresa Nacional del Uranio, S.A. (ENUSA) inició la explotación del yacimiento FE-1 situado en el término municipal de Saelices el Chico de la provincia de Salamanca. El yacimiento es de tipo Stockwork y está enclavado en las pizarras cámbricas de distinto grado de metamorfismo. La mineralización primaria se encuentra rellenando las fisuras y fracturas de pequeño desarrollo longitudinal. Los minerales secundarios se han depositado en las zonas afectadas por el nivel hidrostático. La investigación del yacimiento fué llevada a cabo por la Junta de Energía Nuclear española (JEN). En sus fases finales constó de sendas campañas de sondeos de testigo y de "wagon-drill". La evaluación del yacimiento se realizó por el método de las zonas de influencia tanto según los sondeos de testigo como los de "wagon-drill". El ritmo de producción era de 1 200 000 t anuales de todo uno de las que 200 000 t eran de mineral de ley 0,08% en  $U_3O_8$ . La producción de la planta de tratamiento era de 130 t de  $U_3O_8$ . En el año 1976 ENUSA acometió un proyecto minero de mucha mayor importancia como consecuencia de la decisión de ampliar la planta

de tratamiento de minerales, objeto del proyecto Quercus. La producción anual de la citada planta ampliada será de 600 t de  $U_3O_8$ . Para ello, la producción de mineral pasará a ser de 825 000 t/a de ley media 0,075% y el arranque anual de todo uno de 9 000 000 toneladas. Se ha considerado globalmente el yacimiento FE que engloba los de FE-1 y FE-3 y sus respectivas ampliaciones. La evaluación del yacimiento se ha realizado tanto para los resultados de los sondeos de testigo y "wagon-drill" como por el método estadístico de krigage.

## 1. INTRODUCCION

El incremento de los precios del concentrado de uranio, el aumento de su demanda mundial, la utilización de nuevas técnicas en la exploración así como de modernos aparatos radiométricos y por fin la aplicación de nuevos métodos de valoración, de técnicas mineras y de control de explotaciones, han llevado recientemente a la puesta en explotación de yacimientos de uranio con leyes medias inferiores al 0,1% de  $U_3O_8$  y con reservas de tipo bajo y medio.

Todo ello condujo a la iniciación en 1974 de la explotación minera en Saelices el Chico (Salamanca). La situación de esta explotación minera ha evolucionado muy rápidamente debido precisamente a las nuevas técnicas de valoración aplicadas, a los resultados obtenidos y al desarrollo de diferentes procesos de tratamiento. En el presente trabajo se exponen las diferentes técnicas aplicadas al laboreo del yacimiento, su evolución en el último cuatrienio y las previsiones de futuro desarrollo.

## 2. EL COMPLEJO URANIFERO

Los yacimientos están enclavados en los metasedimentos paleozoicos de distinto grado de metamorfismo de la provincia de Salamanca. La presencia de mineralizaciones, dentro de las formaciones de pizarras, no puede unirse ni a niveles estratigráficos determinados ni a particularidades petrológicas. Sin embargo, sí parece ir ligada a espacios tectonizados con mayor grado de relajamiento y por tanto de permeabilidad ya que aparecen rellenando las brechas de una entrecruzada red de fracturas de pequeño desarrollo longitudinal. La posterior alteración supergénica difumina esta red y se han redepositado productos secundarios a favor de la esquistosidad y las diaclasas.

El mineral primario característico de estos yacimientos es la pechblenda y los minerales secundarios resultantes de la alteración supergénica son óxidos negros, coracitas, uranotilo, autunita, torbernita y otros.

Los yacimientos se han agrupado en tres complejos: el primero y más importante, denominado complejo A, está situado en los términos de Saelices el Chico, Alameda de Gardón y Villar de la Yegua, con unas reservas seguras de 13 236 t de  $U_3O_8$  de las que descontando las 808 t de  $U_3O_8$  explotadas hasta



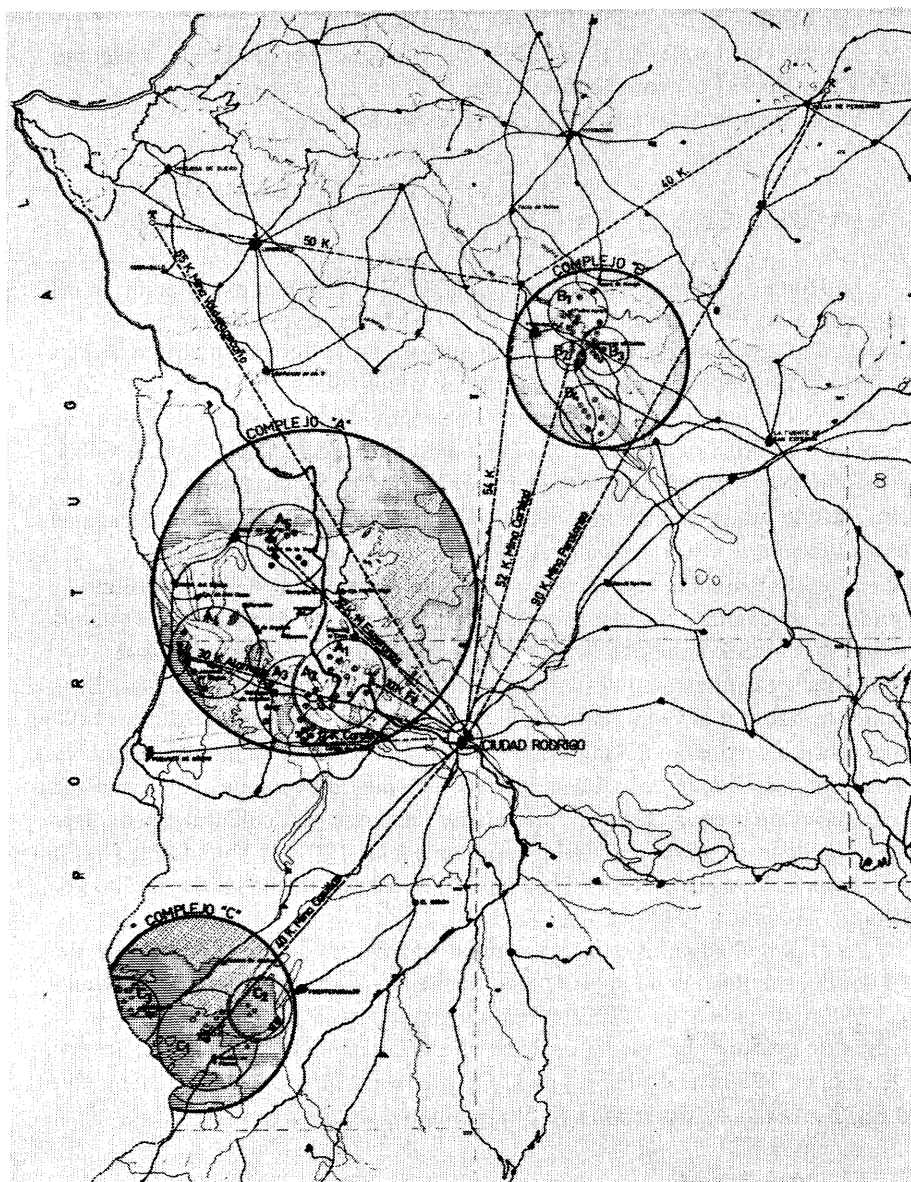


FIG.1. Situación de los yacimientos.

finales de 1978, quedan sin explotar 12 428 t. Las reservas probables del complejo A son de 506 t de  $U_3O_8$ .

Los complejos B y C son de menos importancia que el A.

Las reservas totales de la provincia de Salamanca son de 12 961 t seguras de  $U_3O_8$  y de 931 t probables de  $U_3O_8$ .

En la Fig.1 se detalla la situación de los yacimientos.

### 3. LA OPERACION ELEFANTE

La Junta de Energía Nuclear española (JEN) decidió la explotación uranífera del yacimiento FE-1 debido a que en 1972 existía la posibilidad de que el reculaje de un embalse en el río Agueda anegase parte del yacimiento e incluso comprometiese, por razones de seguridad, la explotación del resto. Ese mismo año se produjo la transferencia de los yacimientos de la provincia de Salamanca a la recién constituida Empresa Nacional del Uranio, S.A. (ENUSA). ENUSA recogió la idea y en colaboración con la JEN construyó una planta de concentrados de uranio con tecnología de JEN que tiene actualmente una capacidad de producción de 130 t de  $U_3O_8$  al año.

El yacimiento FE-1 se encuentra a orillas del río Agueda en el término municipal de Saelices el Chico, provincia de Salamanca. Se trata de un yacimiento de tipo Stockwork en el que las diferentes mineralizaciones uraníferas se encuentran anárquicamente distribuidas en grietas, fracturas y diaclasas. En sentido horizontal el yacimiento tiene unas dimensiones de unos 750 por 600 m. En el sentido vertical se distinguen tres zonas: la zona oxidada, superficial, con un espesor máximo de unos 20 m, la zona intermedia en la que aparecen pizarras no oxidadas de menor espesor y finalmente una zona reducida que puede llegar en algunos puntos a profundidades superiores a los 100 m. Abundan las limonitas en la zona oxidada que desaparecen en la reducida en la que existe abundante pirita.

La JEN realizó la investigación inicial de FE-1 mediante la realización de 247 sondeos de testigo de profundidad media de 75 m con una malla cuadrada de 35,35 m de lado y de 1729 sondeos de «wagon-drill» sin recuperación de testigos de profundidad media de 25 m en malla cuadrada de 10 m. La reserva calculada en 1973 fue de 1330 t de  $U_3O_8$  mediante métodos de zona de influencia. La Fig.2 muestra el histograma de bloques de sondeos de testigo.

#### 3.1. Diseño de la corta

Para el diseño de la corta se definieron tres parámetros básicos: relación estéril/mineral, talud final de la corta y ley de corte del mineral. De acuerdo con los precios de venta de 1975 que se situaban alrededor de dólares 15/libra de  $U_3O_8$  se obtiene una relación estéril/mineral máxima permisible de 12,77/1.

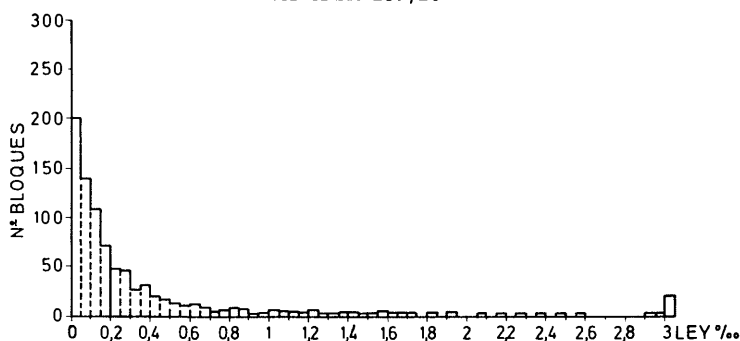


FIG.2. FE-1. Histograma sondeos de testigo.

La ley de corte establecida para el proyecto fue de 0,2%  $U_3O_8$ . El talud final se tomó de 45° para un banco de altura 3 m.

Asimismo para el diseño de la corta se realizaron una serie de zoneografías. Cada una de ellas es un corte horizontal del yacimiento por el nivel de base de cada banco de explotación. Se realizaron cada 3 metros desde el nivel 588 hasta el 666. En ellas se representaron las mallas de influencia de cada sondeo, la ley química diluída en los 3 metros de altura de banco, para los sondeos de testigo, y la radiometría diluída correspondiente a los sondeos de wagon-drill.

Para comprobar el contorno exterior de la corta se realizaron 29 secciones verticales del yacimiento.

Los arranques previstos para la corta total varían con el tipo de sondeo con que se hagan las previsiones, siendo más optimista, en el contenido metal, la previsión según wagon-drill.

En el Cuadro I figuran los arranques de la corta final que se obtuvieron como suma de los arranques de cada banco.

La relación estéril/mineral según los sondeos de wagon-drill es de 2,73, con ley media de 0,996% de  $U_3O_8$ , y según sondeos de testigo es de 7,58 con ley media de 1,02% de  $U_3O_8$ .

### 3.2. Métodos de explotación

Comenzó la explotación de la corta FE-1 en julio de 1974 para crear el stock de mineral necesario para la puesta en marcha de la planta de concentrados de uranio que se estaba construyendo en aquellos momentos.

#### 3.2.1. Perforación

La perforación se hacía con malla de 2 m, al tresbolillo, con una perforación por barreno de 3,40 m. La inclinación de los barrenos era de 53° con la horizontal. El diámetro de perforación era de 38 mm.

CUADRO I. FE-1. ARRANQUES PREVISTOS

	Sondeos wagon-drill			Sondeos de testigo		
	Toneladas	Ley (‰)	t U <sub>3</sub> O <sub>8</sub>	Toneladas	Ley (‰)	t U <sub>3</sub> O <sub>8</sub>
Mineral rico	2 030 174	1,37	2 781	442 884	2,58	1 142,6
Mineral marginal	1 437 225	0,47	675,5	1 066 527	0,38	405,3
Total mineral	3 467 399	0,996	3 456,5	1 509 411	1,02	1 547,9
Estéril	9 476 995	—	—	11 434 983	—	—
Todo uno	12 944 394	0,267	3 456,5	12 944 394	0,12	1 547,9

El equipo de perforación estaba compuesto por seis carros ligeros con martillos neumáticos de 27 kg y seis compresores que suministran un caudal de  $7 \text{ m}^3/\text{min}$  a  $7 \text{ kg}/\text{cm}^2$ .

### 3.2.2. Voladura

El explosivo utilizado es goma 2 EC y su consumo de 70 g/t volada.

El explosivo se cargaba manualmente y se iniciaba por un detonador eléctrico en el fondo de cada barreno.

### 3.2.3. Carga y transporte

La carga y transporte estaba contratada. El equipo utilizado por la contrata era de 2 palas de orugas de  $2,2 \text{ m}^3$  de cazo y 3 camiones de tres ejes de 25 t. La capacidad del equipo era de 600 000 t por relevo y año.

### 3.2.4. Radiometría

Cada barreno era testificado radiométricamente, cada 10 cm con una sonda conectada a un gammámetro. La media de esas medidas radiométricas se considera representativa de un bloque de 30 toneladas y mediante la curva de equivalencia radiometría-ley se obtiene la ley química del citado bloque.

Después de la voladura se controla radiométricamente cada cazo de pala cargado al camión. El equipo inicial de radiometría era de 4 aparatos de gamma-metría y cuatro sondas.

## 3.3. Producciones obtenidas

Las producciones obtenidas en la corta FE-1 desde julio de 1974 hasta el 31-12-78 han sido de 1 013 584 t de mineral de 0,772‰ de  $\text{U}_3\text{O}_8$ . Las toneladas de  $\text{U}_3\text{O}_8$  arrancadas han sido 782 y el tonelaje de todo uno 3 851 807. La relación estéril/mineral ha sido 2,8/1.

## 4. OPERACION QUERCUS

### 4.1. Introducción

El proyecto Quercus tiene como objetivo la ampliación de la planta de concentrados de uranio para obtener una producción anual de 600 t  $\text{U}_3\text{O}_8$ . Para ello es necesario tratar del orden de 825 000 t de mineral y arrancar unas 9 000 000 t de todo uno cada año. Para tener el mineral necesario a la vista en

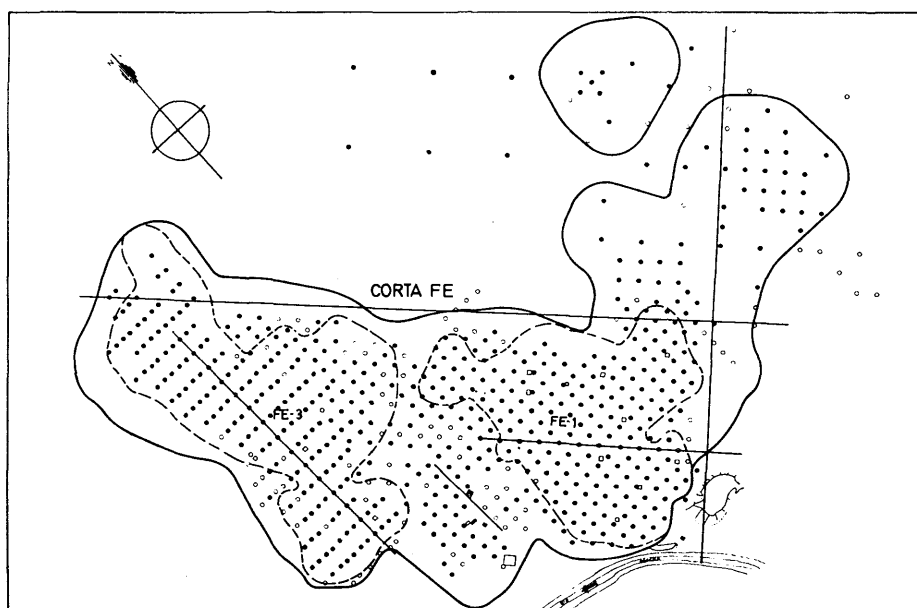


FIG.3. Contorno exterior de las cortas FE, FE-1 y FE-3.

el momento de la puesta en marcha de la ampliación de la planta de tratamiento se inició el arranque de la montera de recubrimiento del yacimiento FE-3 en el mes de marzo de 1977.

Las nuevas campañas de investigación de los bordes de los yacimientos llevadas a cabo por ENUSA en los años 1976, 77 y 78 han dado resultados satisfactorios y ello ha hecho posible la unión y ampliación de los yacimientos FE-1 y FE-3.

Considerando globalmente el yacimiento FE se ha acometido la realización de un nuevo proyecto de explotación unificando y englobando los anteriores de FE-1 y FE-3 en una sola corta (Fig.3).

## 4.2. Evaluación del yacimiento FE

### 4.2.1. Introducción

En la Operación Elefante, la evaluación de los yacimientos se efectuó exclusivamente por el método de las zonas de influencia.

En la Operación Quercus, aparte del método de las zonas de influencia tanto para sondeos de testigo como para sondeos de wagon-drill, se ha utilizado el método geoestadístico denominado krigage.

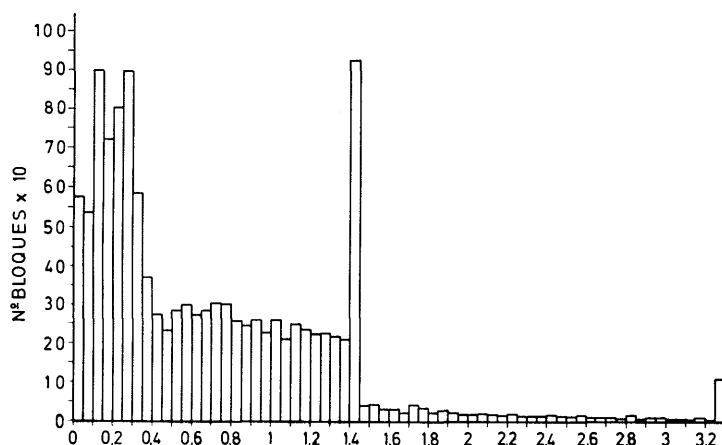


FIG.4. Yacimiento FE. Histograma sondeos wagon drill.

#### 4.2.2. Valoración según sondeos de wagon-drill

Los 3145 sondeos de wagon-drill se efectuaron según una malla cuadrada de 10 m de lado. El área investigada con este tipo de sondeos sólo abarca las zonas denominadas FE-1 y FE-3.

La Fig.4 representa el histograma de los sondeos de wagon-drill del total del yacimiento.

Como se puede apreciar existe un máximo de bloques para la ley de corte de 1,4‰ en U<sub>3</sub>O<sub>8</sub>. Esto es debido a que en FE-1 no se conocen las equivalencias radiométría-ley para las leyes superiores a 1,4‰. Por lo que todos los bloques con leyes superiores a 1,4‰ se consideran con una ley de 1,4‰.

Las hipótesis tenidas en cuenta para la evaluación del yacimiento son las mismas que se aplicaron para evaluar el yacimiento FE-1.

Para una ley de corte de 0,2‰ en U<sub>3</sub>O<sub>8</sub>, los resultados a obtener son los siguientes: ley media 0,868‰, t de mineral 7 005 000, y 6080 t de U<sub>3</sub>O<sub>8</sub>.

#### 4.2.3. Evaluación según sondeos de testigo

La evaluación del yacimiento FE según los 518 sondeos de testigo se ha realizado igual que para el yacimiento FE-1.

El histograma de frecuencia de bloques se representa en la Fig.5.

Igual que anteriormente para los sondeos de wagon-drill, cortando con una ley de 0,2‰ en U<sub>3</sub>O<sub>8</sub>, los resultados a obtener son los siguientes: Ley media 0,741‰, t mineral 7 462 500, y 5520 t de U<sub>3</sub>O<sub>8</sub>.

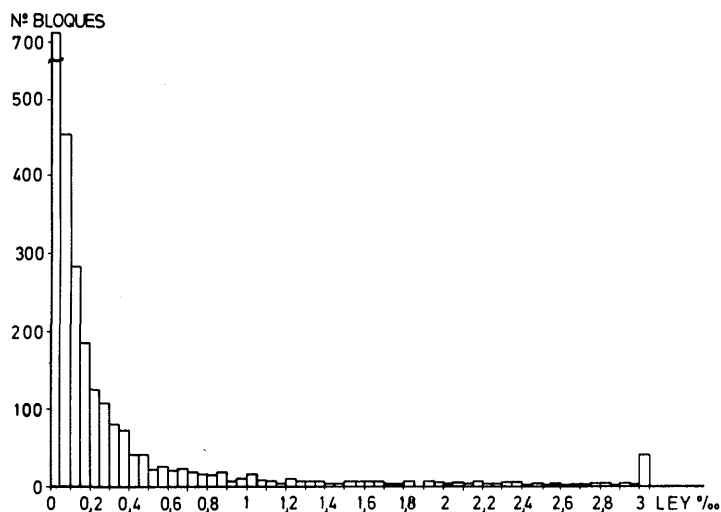


FIG.5. Yacimiento FE. Histograma sondes de testigo.

#### 4.2.4. Evaluación geoestadística por el método de krigeage

La valoración geológica del yacimiento según el método de krigeage se ha efectuado teniendo en cuenta las siguientes hipótesis:

- Solamente se han tenido en cuenta los sondes de testigo.
- El krigeage se aplica a bloques de  $35,35 \times 35,35 \times 0,25 \text{ m}^3$ .
- Las leyes, obtenidas por krigeage, de los bloques de 0,25 m de altura, se diluyen en tramos de 3 en 3 m, de acuerdo con la cota de los bancos de explotación. De esta forma cada ley diluida es representativa de un bloque de  $35,35 \times 35,35 \times 3$ , con densidad de roca de  $2,5 \text{ t/m}^3$ .

El histograma de bloques krigeados se representa en la Fig.6.

De acuerdo con este histograma para una ley de corte de  $0,2\text{‰}$  en  $\text{U}_3\text{O}_8$  se obtienen los siguientes resultados: ley media  $0,653\text{‰}$  en  $\text{U}_3\text{O}_8$ , t de mineral 15 196 875, y 9927 t de  $\text{U}_3\text{O}_8$ .

#### 4.2.5. Comparación de las distintas evaluaciones

Comparando los resultados obtenidos, para una ley de corte de  $0,2\text{‰}$  en  $\text{U}_3\text{O}_8$ , entre estas tres formas de valoración, se puede apreciar que las toneladas de mineral y las de  $\text{U}_3\text{O}_8$  previstas según krigeage son muy superiores a las previstas según los otros dos métodos de valoración; no ocurre lo mismo con la



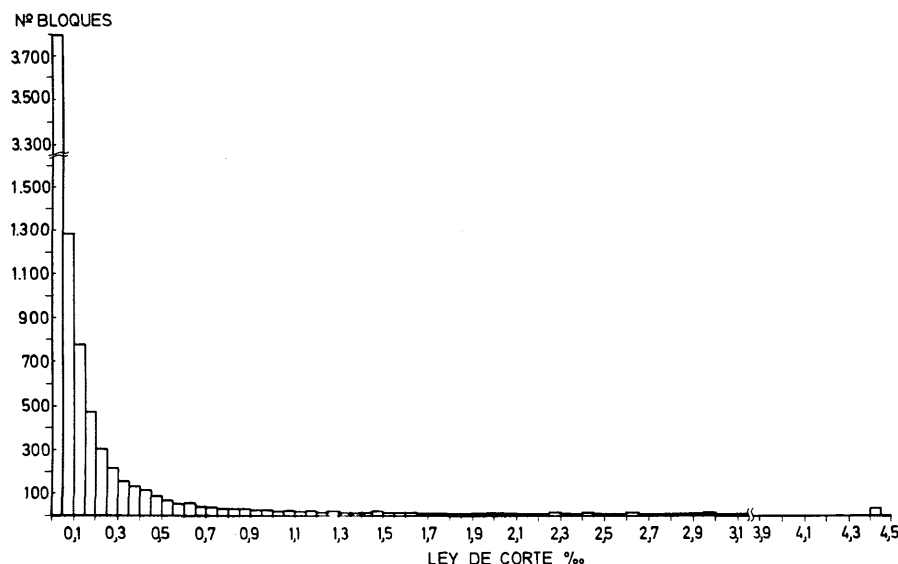


FIG. 6. Yacimiento FE. Histograma bloques krigeados.

ley media, ya que la ley prevista según krigeage es inferior a la prevista según wagon-drill y según testigo.

La evaluación según el método geoestadístico de krigeage aumenta las reservas de mineral pero disminuye la ley.

Se ha de tener en cuenta que la superficie investigada por sondeos de wagon-drill no abarca la totalidad del yacimiento ni en extensión ni en profundidad.

Ello induce a pensar, basándose también en el comportamiento práctico del yacimiento, que una evaluación geológica según wagon-drill, en la totalidad del yacimiento FE, será similar o acaso superior a la evaluación geológica por el método de krigeage sobre los sondeos de testigo.

### 4.3. Criterios de proyecto

La situación del precio de venta del concentrado de  $U_3O_8$  tiene una gran influencia en la determinación de la relación estéril/mineral máxima admisible y de la ley de corte del mineral.

#### 4.3.1. Relación estéril/mineral

Se ha calculado la relación estéril/mineral máxima admisible para precios de venta del concentrado de 35, 41, 88, 43 y 45 dólares/lb de  $U_3O_8$ .

CUADRO II. YACIMIENTO FE. VARIACION DE LA RELACION ESTERIL/MINERAL

Ley media (‰)  Precio de venta	0,75	0,7	0,6	0,5	0,4	0,3	0,2
35 dólares/lb de $U_3O_8$ ó 8 942 Pts/kg	23,96	20,76	16,54	12,14	8,21	4,78	1,83
41,88 dólares/lb de $U_3O_8$ ó 10 700 Pts/kg	37,24	32,87	26,70	20,34	14,46	9,07	4,17
43 dólares/lb de $U_3O_8$ ó 10 986 Pts/kg	38,17	34,84	28,35	21,67	15,48	9,77	4,55
45 dólares/lb de $U_3O_8$ ó 11 497 Pts/kg	41,98	38,36	31,30	24,06	17,30	11,02	5,24

En el Cuadro II se representa la variación de la relación estéril/mineral máxima admisible con el precio de venta del concentrado para distintas leyes medias del mineral.

#### 4.3.2. *Altura de banco*

Las especiales características de distribución del mineral en el yacimiento FE determinan la altura del banco de explotación. Se ha realizado el estudio de evolución de reservas en función de la altura de banco. De acuerdo con la valoración geoestadística se han comparado los tonelajes de mineral y de metal así como la ley media, en ‰ de  $U_3O_8$ , que se obtendrían si el yacimiento Fe se explotase en bancos de 0,25, 0,50, 1, 2, 3, 4, 5 y 6 m de altura respectivamente.

En la Fig.7 se representan las curvas de variación de la ley media y de los tonelajes de mineral y metal con la altura de banco.

La curva de distribución del tonelaje metal tiene un máximo para, aproximadamente, 2 m de altura de banco.

El tonelaje de  $U_3O_8$  recuperable para bancos de 2 m de altura de banco es 162 t superior al de 3 m. Esta diferencia no sería compensada por el mayor coste de explotación que se obtendría en caso de cambiar la altura de banco actual de 3 m a la de 2 m.

Por ello, se ha elegido la altura de banco de explotación de 3 m para la realización del proyecto de explotación.

#### 4.3.3. *Ley de corte*

Se ha calculado la ley de corte con las hipótesis económicas establecidas anteriormente para el año 1981.

En el Cuadro III se ha calculado la ley de corte para distintos precios de venta del concentrado de  $U_3O_8$ .

Para un previo de 41,88 dólares/lb de  $U_3O_8$  la ley de corte es de 0,105‰ de  $U_3O_8$ . Con esa ley de corte se obtendrían, de acuerdo con las tres valoraciones del yacimiento, leyes medias más bajas de lo previsto en el diseño de la ampliación de la planta de tratamiento.

La ley de corte se ha fijado en 0,2‰ de  $U_3O_8$  con lo que la ley media del yacimiento es de 0,653‰ según la valoración geoestadística, de 0,868‰ según la de sondeos de wagon-drill y de 0,741‰ en  $U_3O_8$  según los sondeos de testigo.

#### 4.3.4. *Talud final*

Se ha fijado el talud final de corta en 45°, de acuerdo con el estudio de estabilidad de taludes que se ha realizado.

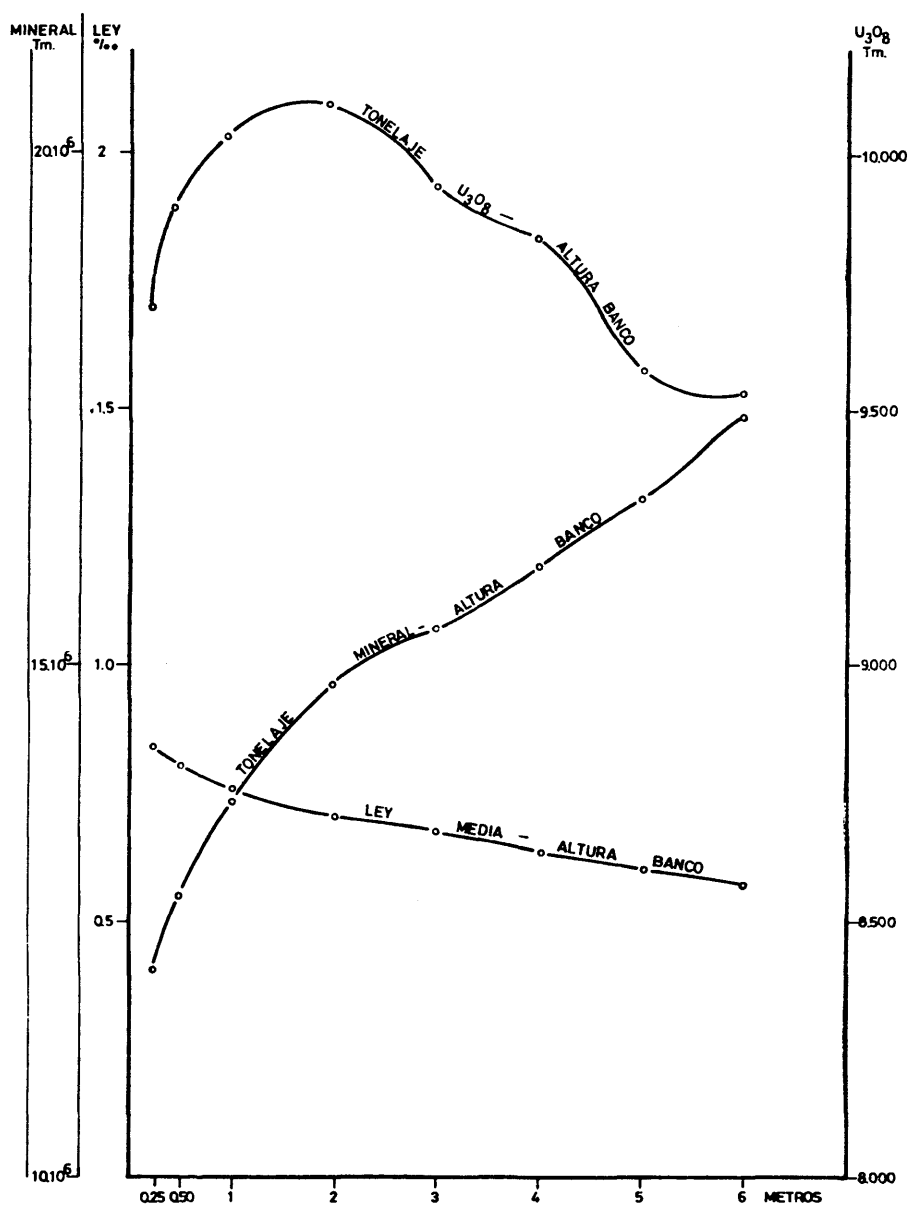


FIG. 7. Variación de reservas con la altura de banco.

**CUADRO III. YACIMIENTO FE.  
LEY DE CORTE DEL MINERAL**

Precio de venta del concentrado	Ley de corte
35 dólares./lb $U_3O_8$ ó 8 942 Pts/kg	0,125
41,88 dólares./lb $U_3O_8$ ó 10 700 Pts/kg	0,105
43 dólares./lb $U_3O_8$ ó 10 986 Pts/kg	0,103
45 dólares./lb $U_3O_8$ ó 11 497 Pts/kg	0,101

**CUADRO IV. PREVISIONES DE EXPLOTACION DE LA CORTA FE**

	Mineral (toneladas)	Ley ‰ $U_3O_8$	$U_3O_8$ (toneladas)	Todo uno (toneladas)	E/M
Wagon-drill	6 643 624	0,959	6 373	135 494 274	19,39
Testigo	6 759 378	0,789	5 333	135 494 274	19,04
Krigeage	14 160 939	0,657	9 312	135 494 274	8,56

#### 4.4. Diseño de la corta

Se han realizado las zoneografías del área a explotar análogamente a como se había realizado para el yacimiento FE-1. Se han considerado 42 zoneografías de bancos de 3 m de altura, desde el nivel 528 al 654 y 7 zoneografías de bancos de 4 m de altura, niveles 654 a 682.

En cada zoneografía se representan las leyes de los sondeos de testigo y de wagon-drill y las leyes de los bloques obtenidas por la valoración geoestadística.

#### 4.5. Arranques previstos

En el Cuadro IV se detallan los arranques que se obtendrán de la explotación de la corta FE según los tres tipos de valoración citados.

Hasta finales de 1978 se han explotado 1 073 135 t de mineral de ley media 0,753‰ de  $U_3O_8$ . Para ello, se han arrancado 7 778 167 t de todo uno, con lo que la relación estéril/mineral ha sido de 6,24/1. Las toneladas de  $U_3O_8$  extraídas de la corta han sido 808.

#### 4.6. Operación minera

##### 4.6.1. Perforación y voladura

El arranque es realizado por medio de perforación y voladura.

La perforación y la voladura presentan unas características especiales debido a la pequeña altura de banco (3 m) necesaria para evitar diluciones. Estas son:

- Necesidad de llevar diámetros de perforación pequeños
- Esquema de tiro de dimensiones reducidas
- Utilización de explosivos de cierta sensibilidad y potencia
- Sistema de encendido con cordón detonante.

El incremento de los costos de energía y personal y el deseo de aumentar el rendimiento del arranque, ha hecho que se vaya a diámetro y mallas de perforación de mayor tamaño.

En la actualidad se perfora a un diámetro de 64 mm con bocas de botones, utilizando 6 carros de perforación neumáticos.

Los barrenos se perforan según una malla al tresbolillo de  $2,8 \times 3,5$  m con una inclinación aproximada de  $62^\circ$  y una longitud media de 4,10 m.

La perforación se lleva adelantada respecto a la voladura con objeto de tener realizado el reconocimiento en detalle de las zonas a explotar con suficiente anterioridad.

Dicho reconocimiento se lleva a cabo con sondas conectadas a gammámetros manejados por un equipo de dos hombres (radiometrista y peón).

Con los datos obtenidos en este control se obtiene una ley media prevista para cada voladura, así como la situación de las zonas mineralizadas.

La carga de las voladuras se hace manualmente con goma 2 EC.

Los barrenos se cargan con 100 g/t como media de dicho explosivo.

Cada barreno está dotado de su correspondiente cordón detonante que se une a la línea general.

La línea general se tiende en zig-zag pasando por cada fila de barrenos (paralela al frente). Sobre ella, y a intervalos regulares, se sitúan los detonadores (del mismo número).

En un principio, este sistema de encendido presentó problemas debido al corte de la línea general al explotar los primeros barrenos, no transmitiéndose el encendido a los siguientes. Esto se solucionó utilizando detonadores de

## CUADRO V. DISTRIBUCION DE PERSONAL

Dirección	2
Otros técnicos titulados	11
Corta	33
Planta	31
Trituración	6
Taller y mantenimiento	25
Administración	6
Laboratorio	5
Oficina técnica	6
Exploración	33
Vigilancia instalaciones	13
Generales	7
Amplificación planta (Quercus)	8
Total	186

microrretardo situados al final de la línea general si solo existe un detonador de encendido, o de las líneas parciales si existen más, de tal forma que si un tramo es seccionado por una proyección u otra razón, el encendido es asegurado por el detonador en cola.

El microrretardo de estos detonadores debe de ser tal que no se inicien antes de que llegue a ellos (si no se diese corte de la línea) el frente de detonación transmitido por el cordón detonante.

#### 4.6.2. Carga, transporte y radiometría

En la actualidad la carga y el transporte se encuentran contratados, siendo también deber del contratista todo lo referente a mantenimiento de las pistas.

La carga, debido a las bajas leyes explotadas, reviste condiciones especiales, ya que debe realizarse un estricto control de leyes con objeto de evitar diluciones.

La necesidad de que el ensuciamiento sea pequeño limita el tamaño de cazo, por lo que en la actualidad se utilizan para la carga de mineral 4 palas de orugas de 2,2 m<sup>3</sup> de capacidad de cuchara.

El estéril es cargado con 2 palas de orugas de mayor tamaño de 4,5 m<sup>3</sup> de cazo, ya que no es necesario un control de leyes tan exhaustivo si se realiza debidamente el control de leyes en barreno.

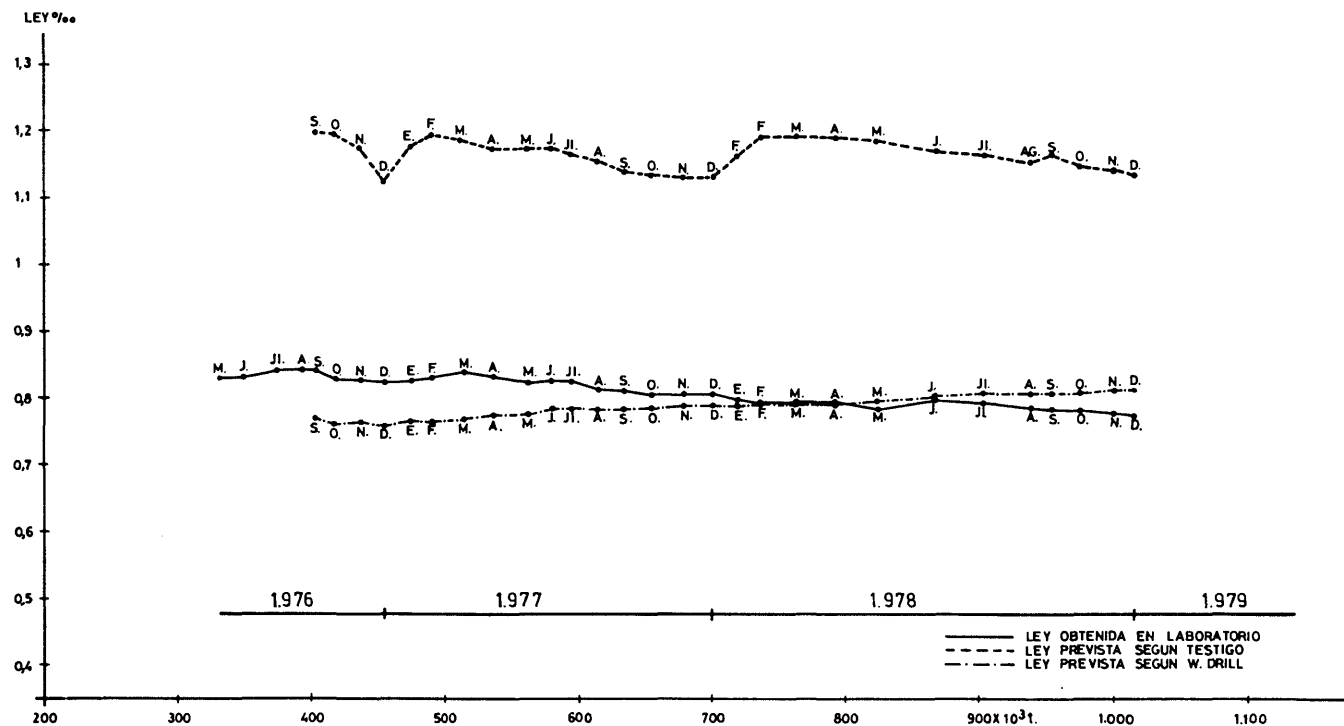


FIG.8. Ley media al origen en función de las toneladas de mineral obtenidas.



El control radiométrico de cazo está constituido por tres medidas, 2 en los extremos del cazo y 1 en el centro, de las que se hace la media y se anota el valor resultante para posteriormente calcular la media de cada día y de cada voladura.

El mineral es transportado, mediante 4 Dumpers de 32 t a la estación de trituración, donde por medio de una trituradora de impacto se reduce hasta un tamaño máximo de 10 cm aproximadamente, siendo desmustrada la salida de trituración para posterior análisis químico de las muestras.

El mineral triturado es transportado mediante un volquete de 300 CV hasta las eras de lixiviación estática.

El estéril es transportado en 3 Dumpers de 68 t a las dos escombreras existentes, dependiendo de la zona del yacimiento el que se envíe a una u otra.

#### 4.7. Personal

La plantilla de trabajadores de las explotaciones mineras de ENUSA en Saelices el Chico, incluida la planta de tratamiento, la formaban 186 personas a finales del año 1978.

La distribución de estas 186 personas por puestos de trabajo se relaciona en el Cuadro V.

### 5. RESULTADOS OBTENIDOS

#### 5.1. Leyes de $U_3O_8$

La Fig.8 muestra la evolución de las leyes medias obtenidas en la explotación del yacimiento desde el origen hasta el fin de 1978. Se observa que la ley media prevista de acuerdo con los sondeos de testigo, 0,113% de  $U_3O_8$ , es muy diferente de la realmente obtenida, 0,077% de  $U_3O_8$ . Sin embargo la previsión hecha por medio de los sondeos de wagon-drill 0,081% de  $U_3O_8$  se ajusta mejor a la producción real; la diferencia puede considerarse como indicativo del ensuciamiento (5,2%).

La Fig.9 muestra la evolución de leyes durante 1978. En esta figura se han incluido las leyes medias de radiometría de barrenos y las de radiometría de cazo. Se observa que durante 1978 el ensuciamiento respecto de la previsión de leyes por wagon-drill ha sido del 25% y del 10% respecto de la radiometría de barreno.

#### 5.2. Producciones y contenido metal

El Cuadro VI muestra la comparación entre la producción real obtenida y las previsiones según sondeos de testigo y de wagon-drill para el período

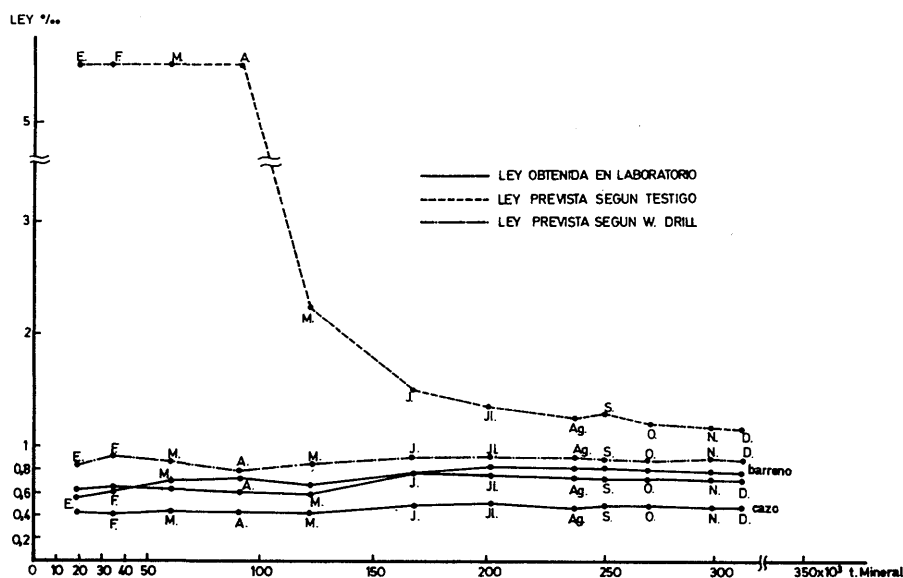


FIG.9. Ley media año 1978 en función de las toneladas de mineral obtenidas.

CUADRO VI. COMPARACION ENTRE LA PRODUCCION REAL Y LAS PREVISIONES SEGUN SONDEOS DE TESTIGO Y DE WAGON-DRILL (Período comprendido entre julio de 1974 y el 31 de mayo de 1978)

	t Mineral	Ley (‰)	kg $U_3O_8$	t Zafra	$Z^a = E/M$
Producción real	823 550	0,782	643 979	3 361 761	3,082
Previsión según sondeos de testigo	292 789	1,182	346 023	3 180 290	9,862
Desviación respecto a la previsión %	181,28	-33,84	86,11	5,71	-68,75
Previsión según sondeos wagon-drill	676 950	0,792	536 377	3 180 290	3,698
Desviación respecto a la previsión %	21,66	-1,26	20,06	5,71	-16,66

$$^a Z = \frac{t \text{ de estéril}}{t \text{ de mineral}}$$

# CUADRO VII. COMPARACION ENTRE LA PRODUCCION REAL Y LA PREVISION SEGUN LA VALORACION POR KRIGEAGE

(Período comprendido entre el 1.1.75 y el 31.5.78)

	t Mineral	Ley (‰)	kg U <sub>3</sub> O <sub>8</sub>	t Zafra	Z <sup>a</sup> = E/M
Producción real	758 435	0,787	597 059	3 102 381	3,09
Previsión según krigage					
Ley de corte 0,1‰	873 387	0,459	401 450	2 991 320	2,42
Desviación respecto a la previsión (%)	-13,16	71,46	48,73	3,71	27,69

$$^a Z = \frac{t \text{ de estéril}}{t \text{ de mineral}}$$

comprendido entre julio de 1974 y el 31 de mayo de 1978. Se observa que el contenido de U<sub>3</sub>O<sub>8</sub> real supera las dos previsiones. Se encuentra más próximo a la prevision wagon-drill en un 20% por encima.

El Cuadro VII compara la producción real y la previsión realizada por krigage para el período comprendido entre el 1-1-1975 y el 31-5-1978. La cantidad de U<sub>3</sub>O<sub>8</sub> contenida en el mineral extraído es 48,7% superior a lo previsto.

## 6. CONCLUSIONES

Uno de los problemas fundamentales de una explotación minera del tipo descrito es el de la previsión de las producciones que se van a obtener. Desde el punto de vista minero aunque los estudios geológicos y de investigación del yacimiento resultan insustituibles y constituyen una primera aproximación para la toma de decisiones de producción, no dan como se ha visto una idea exacta del yacimiento. Esta sólo puede obtenerse incorporando en las previsiones la propia experiencia del explotador.

Para las previsiones de producción se prevén actualmente los dos estimadores fundamentales, es decir, la ley de U<sub>3</sub>O<sub>8</sub> y el tonelaje de mineral a partir de datos de investigación poco correlacionados.

El diseño general de la corta se realiza basándose en los sondeos de testigo que son los que en la actualidad cubren un volumen mayor. Las leyes medias

a obtener en la planificación anual se deducen de la investigación realizada por sondeos de wagon-drill y las leyes medias mensuales se prevén a su vez mediante los resultados obtenidos en la perforación de barrenos de voladura.

El tonelaje de mineral se prevé de acuerdo con el estudio de krigeage que, para este estimador aunque da cifras inferiores a las reales, es el que más se aproxima a lo obtenido.

De esta forma y aunque con una metodología ciertamente empírica en alguna medida se consigue adecuar suficientemente las previsiones a la producción real.

Con ello no se resuelve totalmente el problema del explotador ya que sigue pendiente el tema de la localización precisa de las mineralizaciones. Este problema se mitiga con el aumento de producciones en la medida en que al crecer el tamaño de las voladuras y en consecuencia el bloque arrancado diariamente, aumentan las probabilidades de coincidencia de la previsión y de la producción para períodos de tiempo más cortos que el mensual. Uno de los objetivos actuales es tratar de que los cierres de balances de minerales realizados semanalmente sean lo más ajustados posible.

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## RECUPERATION COMPLEMENTAIRE D'URANIUM PAR LIXIVIATION EN PLACE SUR LE SITE DES BRUGEAUDS (LIMOUSIN, FRANCE)

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### Abstract-Résumé

#### SUPPLEMENTARY RECOVERY OF URANIUM BY IN-SITU LEACHING AT THE BRUGEAUD DEPOSIT (LIMOUSIN, FRANCE).

The actual mining operations at the Brugeaud Deposit (West Brugeaud and East Brugeaud) were followed by supplementary recoveries of uranium by means of in-situ leaching. There were a number of factors which favoured consideration of these operations: the amounts of uranium present at the edge of the stoped areas; the underground mining infrastructure, which did not require supplementary operations for the recovery of solutions; the nature of the rock, which presented a dense network of fractures and micro-fractures conducive to impregnation by the acid solutions; and the immediate proximity of a concentration plant. The amount of uranium recovered by in-situ leaching is close to 200 t. This production is approximately nine per cent of all the uranium extracted from the deposit. The cost of the metal obtained in this way was always less than FF 100 (FF of 1978) per kilogram of uranium.

#### RECUPERATION COMPLEMENTAIRE D'URANIUM PAR LIXIVIATION EN PLACE SUR LE SITE DES BRUGEAUDS (LIMOUSIN, FRANCE).

A la suite de l'exploitation minière proprement dite du gisement des Brugeauds (Brugeaud Ouest et Brugeaud Est), on a procédé à des récupérations complémentaires d'uranium par lixiviation en place. Plusieurs éléments favorables ont permis d'envisager ces opérations: les quantités d'uranium contenues en bordure des zones abattues; l'infrastructure minière souterraine, ne nécessitant pas de travaux complémentaires pour la récupération des liqueurs; la nature de la roche, qui offre un réseau dense de fractures et microfractures favorisant l'imprégnation par la solution acide; la proximité immédiate d'une usine de concentration. La quantité d'uranium récupérée par lixiviation en place est voisine de 200 tonnes. Cette production représente environ 9% de l'uranium total extrait du gisement. Le prix de revient du métal obtenu dans ces conditions est toujours inférieur à 100 FF par kilogramme d'uranium (FF de 1978).

## 1. GENERALITES SUR LE GISEMENT

### 1.1. Situation et structure (figures 1 et 2)

Le gisement des Brugeauds se situe dans le massif granulitique de la Haute-Vienne, à 30 km au nord de Limoges, dans le Limousin. Les premiers indices furent trouvés le 24 juin 1949 au cours d'une prospection de reconnaissance. Un vaste relevé radiométrique fut entrepris mettant en évidence de larges anomalies orientées NS avec un très grand nombre de points actifs. Ceux-ci se groupaient en deux zones distinctes: Brugeaud Ouest et Brugeaud Est, qui ont fait l'objet d'exploitations minières différentes.

La structure du gisement est extrêmement complexe dans le détail, mais son ossature est nette; les dépôts se sont effectués dans un réseau de fractures de direction globale NW. Ces accidents de direction NW sont des failles secondaires et surtout des fractures de réouverture de la mylonite<sup>1</sup> primaire; ils comportent deux types extrêmes entre lesquels existent tous les degrés de transition.

On trouve d'une part *des accidents simples continus horizontalement* sur plusieurs dizaines, voire centaines de mètres, et verticalement sur 30 ou même 50 cm; les puissances normales sont de 1 à 2 mètres. La minéralisation a une allure filonienne qui les colmate.

D'autre part, on observe *un réseau très dense de fractures de tous ordres*; les éléments les plus importants ont des puissances de l'ordre du décimètre; leur continuité n'excède jamais une douzaine de mètres; des accidents localement majeurs se perdent vite dans un réseau inextricable de multiples fractures secondaires.

En résumé tout se passe comme si le système de cisaillement complexe NS auquel appartient la grande faille avait provoqué la rupture et l'éclatement de la zone «mylonitique».

### 1.2. Minéralogie de l'uranium: formes de concentration et espèces minéralogiques

Les dépôts se sont effectués dans les ouvertures des systèmes fracturés que nous venons de décrire. Deux catégories sont à distinguer:

— Les *dépôts filoniens*, ne constituant qu'une faible partie du gisement (15% au maximum) et dont les caractéristiques moyennes sont les suivantes:

Longueur	60 à 100 m
Puissance voisine de	1 m
Teneur moyenne en uranium	0,5%

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<sup>1</sup> Roche broyée à la suite de contraintes tectoniques.

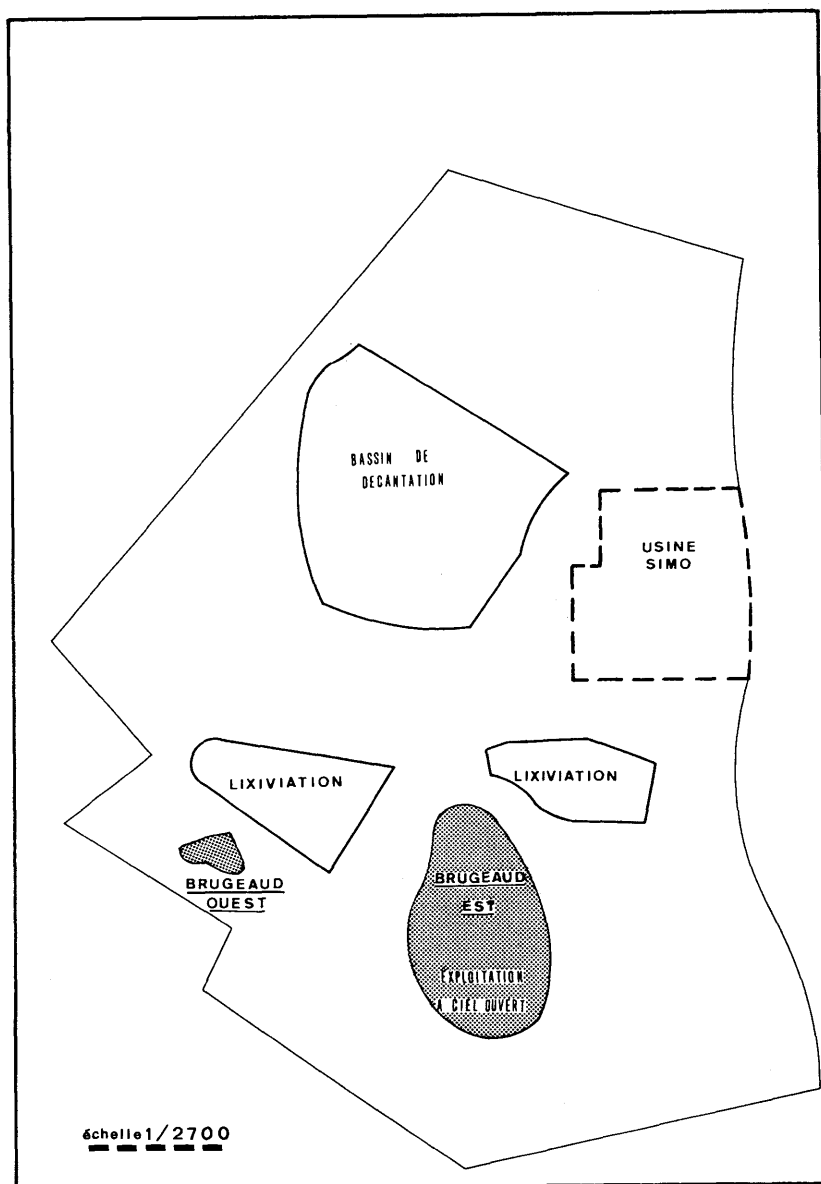


FIG.1. Plan de situation.

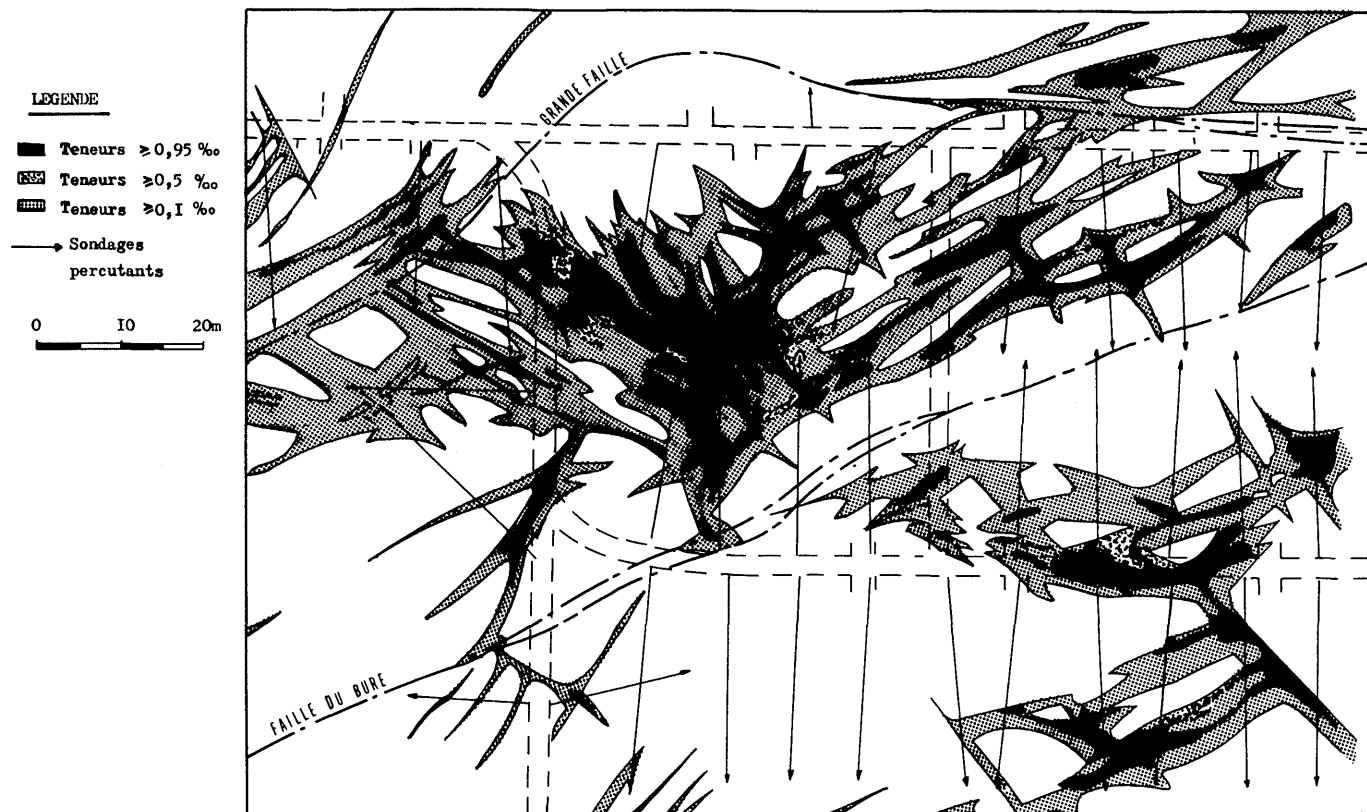


FIG.2. Le Brugeaud Est: zonéographie des teneurs dans l'amas «centre sud».



A l'origine, ces filons étaient de bonnes teneurs, mais les remaniements et les dispersions tectoniques ont conduit à l'éloignement des masses minéralisées et à l'abaissement corrélatif des teneurs.

— Les *amas*, qui représentent l'essentiel du minerai:

80% au niveau	— 30 m
75% au niveau	— 65 m
90% au niveau	— 95 m
80% au niveau	— 125 m

avec des teneurs plus faibles, souvent inférieures à 0,1%.

Les espèces minéralogiques que l'on rencontre le plus souvent sont la pechblende et les minéraux de remaniement: parapechblende, néopechblende, coffinite (produits noirs), puis gummite, uranotile et autunite.

Les ébranlements tectoniques posthumes, bien que d'ampleur réduite, sont fréquents; ils ont favorisé la diffusion des produits uranifères de remaniement dans les réouvertures de fissures voisines, accusant ainsi l'aspect de minéralisation en amas d'imprégnation. L'abondance des produits noirs décroît avec la profondeur, mais ils ne disparaissent jamais totalement; leur domaine privilégié est celui des amas très fracturés, donc très fragiles. Les remaniements actuels conduisent à des espèces minérales secondaires.

## 2. EXPLOITATION MINIÈRE — CHRONOLOGIE DES OPERATIONS (fig. 3)

Les premiers indices relevés sur le site des Brugeauds remontent, comme nous venons de le voir, à juin 1949. Des sondages furent immédiatement entrepris à partir de la surface, qui pour la plupart se révélèrent positifs. On décida alors de continuer les recherches en creusant une galerie à flanc de coteau à partir de laquelle furent effectués de nouveaux sondages. Ces travaux s'étalèrent sur la période 1951–1953.

A partir de cette date, l'existence d'un gisement était pratiquement acquise. Les relevés radiométriques montrèrent à la fois une évolution croissante des teneurs de la surface vers les zones profondes et une structure du gisement extrêmement complexe. Il fut alors décidé, et ceci au départ dans un but de reconnaissance et d'évaluation des réserves, de construire un puits de mine. Cette construction démarra dès 1953.

Le minerai extrait, en faible quantité au départ, fut expédié à l'usine Simo voisine pour la partie riche: la teneur moyenne en uranium était d'environ 1000 ppm avec une coupure qui a, bien sûr, évolué en fonction des contraintes économiques du moment, mais qui a toujours été comprise entre 400 et 600 ppm.

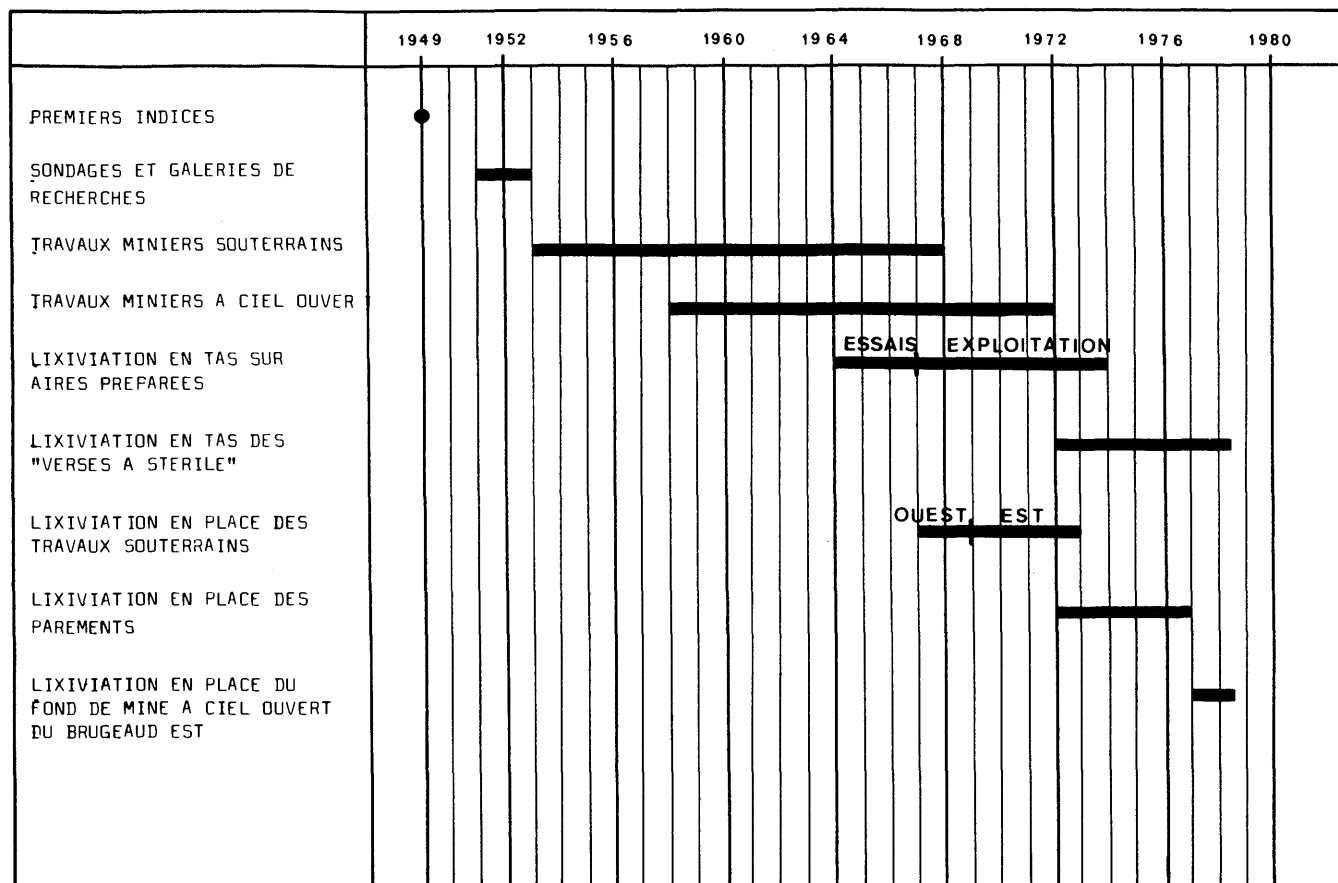


FIG. 3. Chronologie des opérations.

A cette période, le minerai obligatoirement extrait dont la teneur était inférieure à ces valeurs a été stocké sur de «grandes versées». Il était considéré comme stérile et ne devait, par conséquent, faire l'objet d'aucun traitement. Nous verrons par la suite qu'il en a été autrement.

Les travaux souterrains durèrent quinze ans, de 1953 à 1968. Parallèlement, mais à partir de 1958 seulement, furent entrepris les travaux miniers à ciel ouvert (Brugeaud Ouest et Est), qui se sont achevés en 1971.

Ce rappel n'est pas inutile car la lixiviation en place qui fait l'objet de la présente communication n'a pu être réalisée que grâce aux travaux miniers existants pour la récupération des liqueurs, ce qui a bien évidemment imposé une chronologie des opérations.

### 3. LIXIVIATION EN PLACE SUR LE BRUGEAUD OUEST ET LE BRUGEAUD EST

#### 3.1. Généralités

Nous avons vu précédemment l'importance des fractures dans le gisement, sur lesquelles s'est fait le dépôt de l'uranium. Ceci laissait présager une bonne aptitude à la lixiviation. Une étude plus fine a montré que ce réseau de fractures se ramifiait presque à l'infini, donnant une véritable porosité à la roche: des essais de laboratoire ont confirmé la «mouillabilité» du granite et par conséquent le contact facile entre l'uranium en dépôt sur ces diaclases et la solution lixiviante.

Il faut ajouter également que des essais effectués sur des granulométries grossières (30 mm) avant le démarrage de la lixiviation en tas avaient donné d'excellents rendements de récupération (en général supérieurs à 85% pour des teneurs moyennes de 450 ppm).

Ces raisons — observation préalable sur le granite, vérification à petite échelle sur du minerai peu fragmenté — nous semblaient amplement suffisantes pour envisager des opérations de récupération complémentaire d'uranium en place.

Au préalable, à la fin des travaux miniers souterrains, c'est-à-dire en 1967, on procéda au noyage des galeries de mines. C'est donc à cette époque que fut prévue toute l'installation de lixiviation qui sera décrite plus loin (système d'arrosage, pompes, tuyauteries, etc.). Cette opération se termina en 1971 et fut suivie par la lixiviation en place des parements et du fond de mine à ciel ouvert du Brugeaud Est.

#### 3.2. Installation de lixiviation (fig. 4)

A partir de deux bassins de capacité unitaire voisine de 750 m<sup>3</sup>, revêtus de caoutchouc et alimentés directement en acide sulfurique, on véhicule l'eau

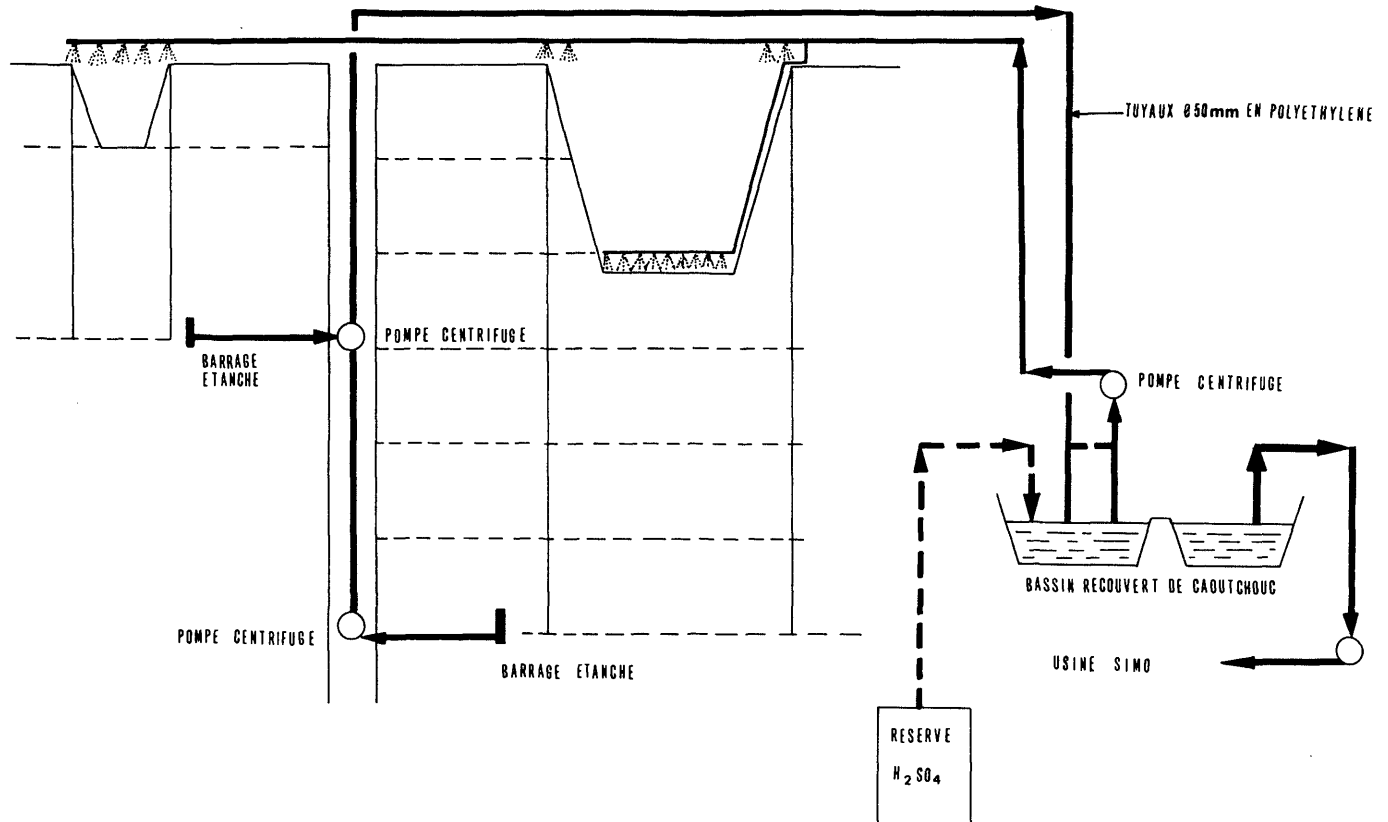


FIG. 4. Installation de lixiviation.

acidulée à 10 g/l par l'intermédiaire de pompes centrifuges. Un réseau de tuyauteries en polyéthylène de 50 mm de diamètre sur lesquelles on fixe en extrémité des tubes en acier inoxydable pour mieux positionner les points d'arrosage (au sommet des parements en particulier) a été disposé:

- dans les galeries des mines pendant le noyage de celles-ci,
- sur le pourtour des mines à ciel ouvert,
- sur le fond de la mine à ciel ouvert du Brugeaud Est.

L'eau acidulée chemine par gravité dans la masse de granite. Suivant la zone lixiviée, les liqueurs circulent sur les parements, le socle de la mine à ciel ouvert, les galeries souterraines, et arrivent dans la galerie de base (à - 120 m pour le Brugeaud Ouest, à - 245 m pour le Brugeaud Est). C'est donc à partir de ces niveaux, et ceci quelle que soit la zone lixiviée, que la récupération des liqueurs s'effectuera. Pour éviter un cheminement anarchique de celles-ci au niveau de base, on a construit un mur en béton en constituant de petits bassins à partir desquels deux pompes centrifuges en série remontent les solutions uranifères vers la surface. Suivant les cas, ces liqueurs arrivent dans les bassins de stockage qui servent de tampon (une partie de celles-ci tourne en rond, l'autre partie est dirigée vers l'usine Simo), ou bien elles sont recyclées en totalité pendant une certaine période, puis soutirées en discontinu vers l'usine.

On peut encore ajouter quelques précisions:

- L'alimentation des bassins en acide sulfurique est assurée à partir d'un réservoir de stockage d'acide pur; on a parfois utilisé les effluents acides de l'usine issus de l'extraction par solvants (raffinats).
- Malgré la différence de taille entre les deux parties du gisement Ouest et Est, il y a une grande similitude d'exploitation (mines souterraines, mines à ciel ouvert, etc.). A noter cependant deux différences pour la lixiviation en place: a) la lixiviation en place des galeries souterraines, des parements et du socle de la mine à ciel ouvert a été faite séparément pour le Brugeaud Est, globalement pour le Brugeaud Ouest; b) des travaux miniers complémentaires ont été nécessaires pour effectuer la lixiviation du fond de mine à ciel ouvert (Brugeaud Est).

### 3.3. Lixiviation en place sur le Brugeaud Ouest des galeries souterraines, des parements et du fond de mine à ciel ouvert (fig. 4 et 5)

Les dimensions approximatives de la mine à ciel ouvert sont les suivantes:

Profondeur	35 m
Diamètre au fond	30 m
Diamètre au sommet des parements	70 m
Circonférence au sommet	
des parements, environ	210 m
Surface d'arrosage	3850 m <sup>2</sup>

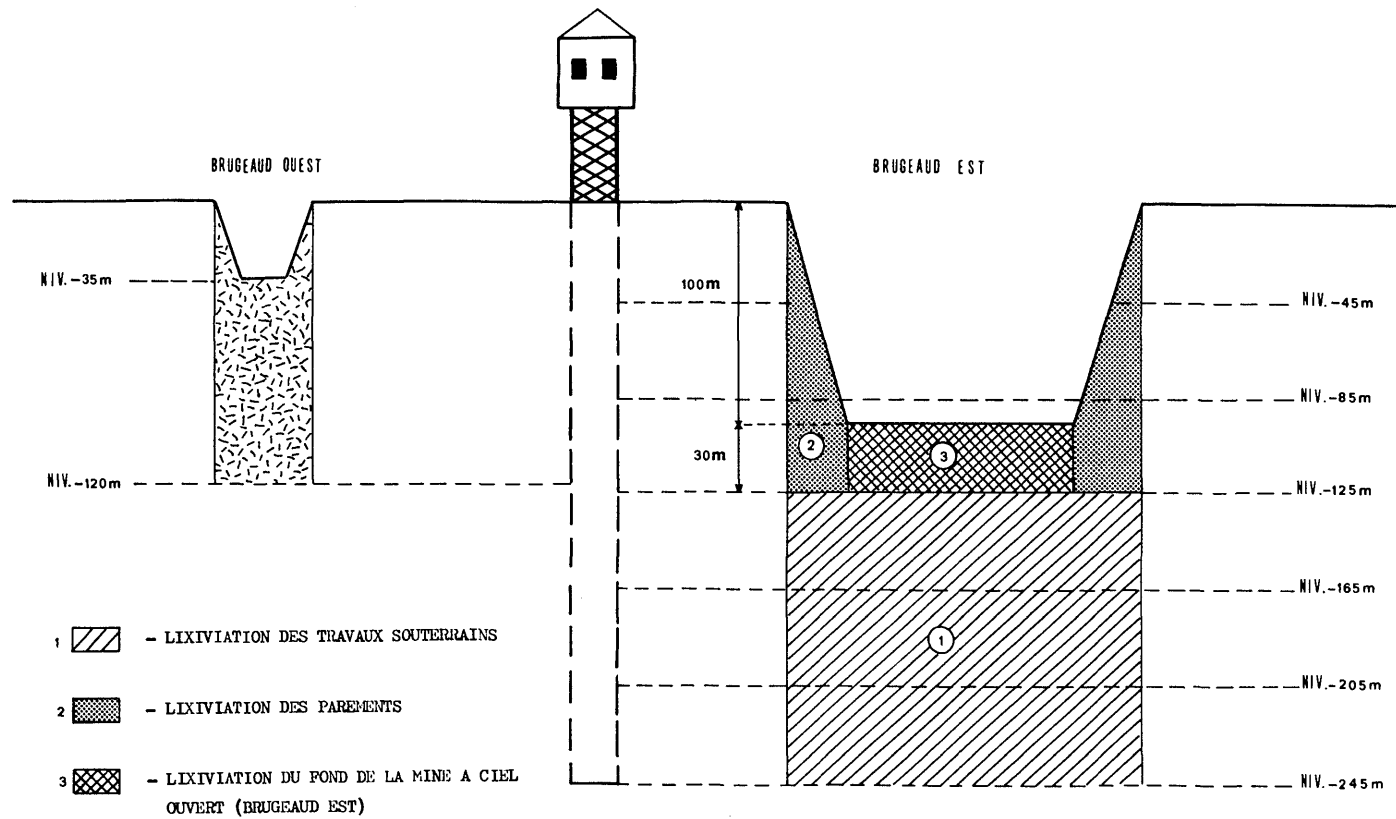


FIG.5. Zones lixivées en place.

L'arrosage avec de l'eau acidulée a été effectué sur toute la surface supérieure pour lixivier à la fois parements et fond de mine. Les liqueurs ont été récupérées au niveau -120 m où l'on a construit le mur en béton et installé les pompes centrifuges. La teneur moyenne en uranium des liqueurs expédiées à l'usine Simo était d'environ 70 mg/l et le volume total de 400 000 m<sup>3</sup>, ce qui a permis la récupération de 27,5 tonnes de métal.

### 3.4. Lixiviation en place sur le Brugeaud Est

#### 3.4.1. Lixiviation des parements (fig. 5)

Rappelons les dimensions approximatives de la mine à ciel ouvert:

Profondeur	100 m
Fond en forme d'ellipse de demi-axes	30 et 50 m
Surface du fond de mine à ciel ouvert, environ	5000 m <sup>2</sup>
Circonférence au sommet des parements, environ	1400 m

Les opérations de lixiviation ont débuté en 1971 pour s'achever en 1977; 1 200 000 m<sup>3</sup> de liqueurs uranifères ont été expédiées à l'usine Simo. La teneur moyenne en uranium était comprise entre 60 et 70 mg/l, ce qui a permis la récupération de 75,3 t d'uranium pour une consommation d'acide de 111,5 kg/kg U.

#### 3.4.2. Lixiviation en place du fond de mine (fig. 5 et 6)

Le fond de mine a une forme elliptique (demi-axes 30 et 50 m) et une superficie voisine de 5000 m<sup>2</sup>. Le noyage des galeries souterraines avait été fait précédemment, ce qui limitait le volume à lixivier (voir fig. 5) à la partie comprise entre le fond de la mine à ciel ouvert et la galerie souterraine située immédiatement dessous. La différence de profondeur entre les deux était de 30 m, conduisant à un volume global voisin de 150 000 m<sup>3</sup>.

Pour récupérer l'uranium dans cette partie, on n'a pas pu se contenter d'un arrosage simple sur le fond de mine. En effet, l'aspect du terrain et la présence de fractures importantes laissaient craindre un cheminement préférentiel des liqueurs et par conséquent une imprégnation irrégulière. C'est pourquoi, eu égard aux quantités d'uranium contenu, on décida de fragmenter la roche pour mieux récupérer l'uranium.

Des trous verticaux (fig. 6) d'environ 150 mm de diamètre furent percés pour loger les explosifs nécessaires à l'ébranlement. Cette opération complémentaire se solda par une dépense qui a, bien sûr, augmenté le prix de l'uranium

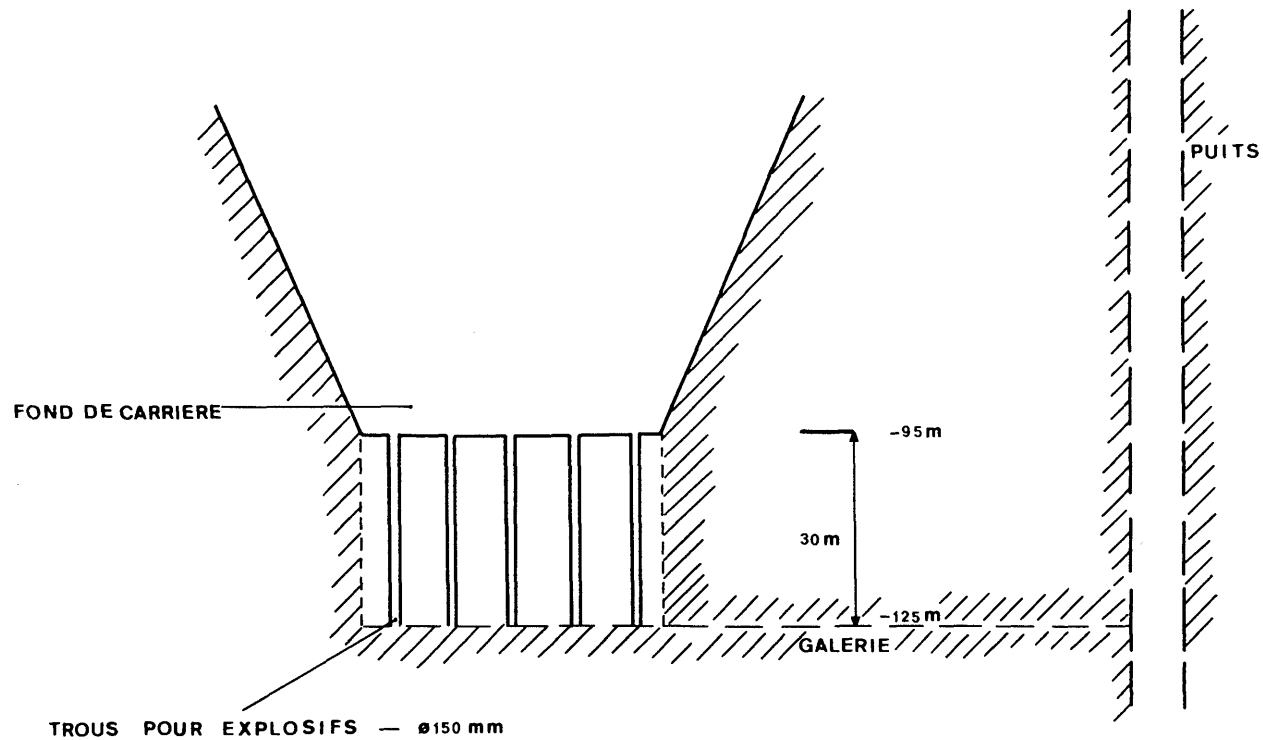


FIG. 6. Le Brugeaud Est: ébranlement du fond de mine à ciel ouvert.



recupéré, mais celui-ci est resté, comme nous le verrons, à un niveau raisonnable. Après explosion, on n'observa qu'un léger soulèvement du socle.

La lixiviation de cette partie commença en 1977 pour s'achever en juin 1978. Le volume de liqueurs expédié à l'usine fut de 400 000 m<sup>3</sup>, la teneur en uranium de 70 mg/l et la consommation d'acide de 50,3 kg/kg U. On récupéra ainsi 25,2 t d'uranium.

#### 4. BILAN

##### 4.1. Quantités d'uranium récupérées

L'exploitation et la récupération d'uranium à partir du gisement des Brugeauds sont terminées depuis la mi-juin 1978. Il est donc possible de faire un bilan précis et de mesurer la contribution fournie par la lixiviation en place.

La quantité totale d'uranium récupérée à partir des deux gisements s'élève à 2306,6 t. Elle se répartit ainsi:

##### *Uranium traité en usine*

— à partir de minerais souterrains	304 t
— à partir de minerais des mines à ciel ouvert	1416 t
Soit un total de	1720 t

##### *Uranium récupéré par lixiviation en tas*

— sur aires non préparées (verses)	95,3 t
— sur aires préparées	296 t
Soit un total de	391,3 t

##### *Uranium récupéré par lixiviation en place*

— noyage des galeries souterraines (y compris parements et fond de mine à ciel ouvert du Brugeaud Ouest)	94,8 t
— parements du Brugeaud Est	75,3 t
— fond de mine à ciel ouvert du Brugeaud Est	25,2 t
Soit un total de	195,3 t

ce qui représente pour la lixiviation en place environ 9% du total.

TABLEAU I. DEPENSES (BRUGEAUD EST)

	Uranium récupéré (kg U)		Dépenses de traitement		
	Parements	Fond	Totales (FF courants)	Par kg U (FF courants)	Par kg U (FF de 1978)
1972	10 058				} 57 <sup>a</sup>
1973	11 621				
1974	10 854				
1975	13 193		800 252	60,7	79
1976	18 198		571 828	31,0	37
1977	11 405	9 372	1 037 519	50,0	55
1978		15 841	601 856	38,0	38

<sup>a</sup> Estimé.

#### 4.2. Dépenses

On trouvera au tableau I le détail des dépenses annuelles relatives aux parements et au fond de mine du Brugeaud Est.

##### *Analyse des coûts*

Le coût moyen de traitement pour la lixiviation des *parements* est de 57 FF/kg U (FF de 1978). Ce prix est essentiellement lié à la consommation d'acide et la teneur en uranium des liqueurs expédiées vers l'usine. Celle-ci a pu évoluer d'une année à l'autre et explique les écarts observés par rapport à la moyenne.

Pour la lixiviation de *fond de mine* du Brugeaud Est, le coût moyen du traitement est de 44 FF/kg U (FF de 1978). Il faut y ajouter les dépenses occasionnées par les travaux miniers (ébranlement), qui s'élevaient à 861 000 FF (FF de 1977) ou 947 700 FF (FF de 1978) ou 35 FF/kg U, soit un total de 79 FF/kg U (FF de 1978).

En définitive, on peut retenir deux chiffres (FF de 1978):

- prix de revient de l'uranium récupéré par lixiviation en place sans travaux miniers préalables 57 FF/kg U (ou 22 FF la livre de  $U_3O_8$ )
- prix de revient de l'uranium récupéré par lixiviation en place avec travaux miniers préalables 79 FF/kg U (ou 31 FF la livre de  $U_3O_8$ )

## 5. CONCLUSION

Si nous avons tenu à présenter cette communication, ce n'est pas pour porter à la connaissance des participants une expérience très originale apportant des éléments nouveaux sur les méthodes d'exploitation minière ou de lixiviation, mais plutôt pour mettre l'accent sur la simplicité, l'efficacité de la lixiviation en place sur le site des Brugeauds et l'intérêt économique qui en découle.

*Simplicité:* L'équipement nécessaire à la lixiviation est rustique et de faible valeur. En outre, nous avons eu la chance de profiter d'une part des travaux miniers souterrains, d'autre part de la proximité immédiate d'une usine de concentration.

*Efficacité:* La récupération complémentaire d'uranium par lixiviation en place conduit à des quantités importantes, près de 200 t sur un total de 2300 t. Ce n'est pas négligeable.

*Intérêt économique:* Nous n'avons eu à subir pour une partie (Brugeaud Ouest et parements du Brugeaud Est) que les coûts de traitement chimique de concentration et purification; pour l'autre partie, des coûts analogues auxquels il faut ajouter les dépenses relatives aux travaux miniers (ébranlement). Les prix de l'uranium récupéré dans ces conditions s'élèvent à 57 ou 79 FF suivant les cas, ce qui démontre l'intérêt économique de l'opération.

## DISCUSSION

B.S.I. MARENGWA: Is the genesis of the ore deposits in the veins found in the granites hydrothermal or metamorphic? In other words, was the uranium introduced into the veins in hydrothermal solutions, or was it concentrated in the veins from the granites?

G. LYAUDET: I am not a geologist, but perhaps Mr. Ziegler can answer your question.

V. ZIEGLER: The genesis of the uranium deposits in the Limousin has been studied for a long time and various theses have been written on the subject in the last fifteen years, especially at the Petrography and Geochemistry Centre in Nancy, the most recent of which is by Mr. Leroy. Although knowledge of the genesis of these deposits has not yet moved beyond the hypothetical stage, it is now thought that it is closely connected with the last phase of formation of the Saint Sylvestre leucogranite and, more especially, with the deuteric phase, which caused the disseminated mineralization of uranite from this granite, the

basic uranium content of which reaches some 20 ppm and the thorium-uranium ratio of which is much lower than the usual one of 3:4. However, the formation of economically significant mineralization is linked with a subsequent hydrothermal phase, the effect of which was to remobilize and reconcentrate the uranium disseminated from the leucogranite into mineralized bodies with uranium concentrations worth mining.

M. ANGEL: In-situ leaching sometimes presents problems in relation to protection of the groundwater. Can you say what protective action was taken and what type of monitoring was performed?

G. LYAUDET: As I said in the introductory remarks of my oral presentation, the main factor which favoured in-situ leaching was the presence of a network of underground galleries below the area to be leached and from which the acid solutions were recuperated. This underground network therefore constituted in itself a safety factor as it prevented any extensive leakage of liquid. Also regular checks were made on waters in the surrounding area for acidity and uranium and radium contents; all these proved negative.

U. CORDERO DI MONTEZEMOLO (*Chairman*): After the end of in-situ leaching, what treatment was given to the leached zone?

G. LYAUDET: When uranium extraction had finished we continued to recuperate the liquors and to process them. This operation was continued until the pH of the liquors was the same as that of the normal groundwater. The high rainfall in this part of France made the operation particularly fast and effective.

## **POSSIBILITY OF USING BACTERIA IN THE URANIUM-SOLUTION MINING METHOD \***

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### **Abstract**

#### **POSSIBILITY OF USING BACTERIA IN THE URANIUM-SOLUTION MINING METHOD.**

In recent years mining by in-situ leaching, or solution mining, has become more attractive and has received increasing attention in the mineral industry because this method often makes it feasible to exploit low-grade and deep mineral reserves, at the same time minimizing the impact on the environment. But sometimes the high consumption of leaching acid media resulting in increased leaching cost causes this method to become uneconomic. Therefore, in many countries research work has been conducted to solve this problem. At the Indonesian Directorate of the Geological Survey the possibility of using ferric ion generating bacteria to produce the leaching media in situ is being examined to obtain lower leaching costs. Such studies have been based on information given in the literature regarding suitable conditions for bacterial growth and activity, and adjusting them to the characteristic conditions of in-situ leaching/solution mining.

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\* Since the results of this work are not yet completed, only the Abstract is published in these Proceedings; the research workers hope to complete and publish their results in the near future.



# POTENTIAL BY-PRODUCT URANIUM PRODUCTION IN THE UNITED STATES OF AMERICA

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## Abstract

### POTENTIAL BY-PRODUCT URANIUM PRODUCTION IN THE UNITED STATES OF AMERICA.

When evaluating uranium resources, one must not overlook the possibilities of recovering uranium as a by-product of other industries. Currently in the United States of America there are four plants in operation and three plants under construction to recover uranium as a by-product of the phosphate fertilizer industry. These plants will have a total combined capacity to produce about 1500 t U/a. Another plant has been constructed in Canada, and several other countries are investigating the technology. Another source of by-product uranium is the copper industry where acid leach liquors often contain significant quantities of uranium. There is one plant in the United States in operation, and two other production facilities are being constructed. Indications are that by-product uranium production from both domestic sources could expand substantially during the next several years and could total about 90 000 t U by the year 2000. The technology for recovering by-product uranium from phosphoric acid has been known for many years, but only recently have market conditions encouraged production. Even with recent improvements in process technology, many problems are still encountered in scale-up from laboratory and pilot plant operation to commercial production. Some of these problems are discussed in this review of the process technology. A recent USDOE study shows a total capital cost of US \$12 million for a by-product uranium plant designed to recover 40 to 60 t U/a from 30%  $P_2O_5$  acid provided by a phosphoric acid plant with a capacity of 155 000 t  $P_2O_5$ /a. Operating costs were estimated at US \$50 to 65/kg U (\$20–25/lb  $U_3O_8$ ), assuming the phosphoric acid contains 120 to 160 mg U/l and a raffinate of 13 mg U/l. The quantity of uranium recovered and the operating costs are highly sensitive to the uranium content of the acid. Constraints to by-product uranium production, besides low uranium content of the acid are small-size phosphoric acid plants and high strength (>40%  $P_2O_5$ ) acid. Similarly the technology for recovery of uranium from copper waste dump leach liquors has been known for several years. It was developed in the mid-1960s by the US Bureau of Mines in co-operation with Kennecott Copper Corporation at its Bingham Canyon mine near Salt Lake City, Utah. Commercial application was first realized when an ion-exchange plant to recover about 55 t U/a commenced operation in 1978.

## INTRODUCTION

When evaluating uranium resources, one must not overlook the possibilities of recovering uranium as a byproduct of other industries. In the United States of America, total byproduct uranium production has been modest so far, amounting to 1000 to 2000 t U. However, with the projected increasing demand for uranium and favorable market prices, byproduct sources of uranium are receiving considerable attention. This is especially true of uranium from wet-process phosphoric acid and from porphyry copper leaching solutions.

Currently in the United States there are four plants in operation and three plants under construction to recover byproduct uranium from phosphoric acid. One plant has been constructed in Canada and several other countries are investigating the technology. Indications are that additional phosphoric acid producers will commence uranium production in the next several years.

There is one plant in operation in Utah to recover byproduct uranium from copper leach liquors and construction of two other facilities in Arizona has been announced. Ion exchange resins are favored for recovery of byproduct uranium from copper leach liquors, whereas solvent extraction processes are used in recovering uranium from phosphoric acid.

Although the technologies for recovering uranium from both wet process phosphoric acid and from copper leach liquors has been known for many years, there still are many problems encountered in commercial application. This paper presents information on potential production, process technology, and the economics that may assist in evaluating similar resources in other countries. Major emphasis is given to uranium from phosphoric acid because that technology is more advanced and represents a greater potential for uranium product

## URANIUM FROM PHOSPHATES

### Current and Future Production

For 70 years it has been known that marine phosphorites are several times more radioactive than the average rocks of the earth's crust. Such deposits, estimated in the United States alone to contain about 4 million t U, have been considered potential sources of uranium and for more than 30 years investigators have sought means for economically recovering the uranium. Although the marketable phosphate rock recovered from these deposits usually contains only 50 to 200 ppm U, the large quantity of rock produced (currently about 50 million t/a in the United States) makes it an attractive and continuing source of uranium.



TABLE I. MARINE PHOSPHORITES RESERVES AND URANIUM CONTENT (CATHCART [1])

<u>Location</u>	<u>Reserves</u> <sup>1</sup>	<u>Uranium Average</u>	<u>(Range)</u> <sup>2</sup>
Central Florida	2.1 x 10 <sup>9</sup>	Concentrate 0.011) Pebble 0.015)	(0.003-0.030)
North Florida - South Georgia	0.3 x 10 <sup>9</sup>	0.006	(0.004-0.011)
North Carolina	2.0 x 10 <sup>9</sup>	0.006	(0.004-0.011)
Idaho, Montana, Utah, Wyoming (Phosphoria)	6.0 x 10 <sup>9</sup>	0.009	(0.002-0.021)

<sup>1</sup> Tonnes recoverable product containing at least 24 percent P<sub>2</sub>O<sub>5</sub>.

<sup>2</sup> Uranium in percent of phosphate product.

TABLE II. PAST, PRESENT AND FORECAST REGIONAL PHOSPHORIC ACID CAPACITY (REF. [3])

<u>Year</u>	<u>North America</u>	<u>Latin America</u>	<u>Western Europe</u> ( X10 <sup>3</sup>	<u>Eastern Europe</u> tonnes P <sub>2</sub> O <sub>5</sub> )	<u>USSR</u>	<u>Africa</u>	<u>Asia</u>
1968	5,768	113	2,782	297	678	463	1,032
1978	9,395	1,008	4,587	1,661	4,633	2,338	3,127
1985	9,395	2,151	4,687	2,475	6,763	3,219	5,306

The term "phosphate rock" as used in this presentation refers to the marketable product usually produced by physical beneficiation of the mined ore or "matrix" as it is called in the land-pebble area of central Florida. As shown in Table I, the land-pebble phosphate rock of central Florida, which constitutes about 75 percent of the total domestic production, contains somewhat more uranium than western, northern Florida, and Carolina phosphate rock [1]. Hence, most research and process development effort to date has been directed toward uranium recovery from deposits in the central Florida area. No process has been found that will selectively extract uranium directly from phosphate rock, usually typi-

TABLE III. PLANTS FOR RECOVERY OF URANIUM FROM PHOSPHORIC ACID

COMPANY	LOCATION	CAPACITY (TONNES/YEAR)		STATUS
		P <sub>2</sub> O <sub>5</sub>	U	
UNC Recovery Corporation (@ W. R. Grace & Co.)	Bartow, Florida	300,000	110	Operational
Freeport Uranium Recovery Co.	Uncle Sam, Louisiana	680,000	265	Operational
Wyoming Mineral Corporation (@ Farmland Industries)	Pierce, Florida	450,000	160	Operational
Gardinier Inc.	Tampa, Florida	500,000	170	Operational
International Minerals & Chemical Corporation (New Wales Plant)	Mulberry, Florida	760,000	290	Under Construction Late 1979 Start-up
International Minerals & Chemical Corporation (@ C. F. Industries, Inc.)	Bartow, Florida and Plant City, Florida	1,190,000	485	Under Construction Two Primary Recovery Units to Start up by Late 1980
Earth Sciences, Inc. (@ Western Cooperative Fertilizers, Ltd.)	Calgary, Alberta, Canada	145,000	40	Starting operation
	TOTALS	4,025,000	1,520	

fied as tricalcium phosphate,  $\text{Ca}_3(\text{PO}_4)_2$ . Alkaline leach methods are ineffective and in acid leaching the uranium extraction is proportional to the dissolution of the calcium phosphate. Therefore, the only commercial processes for uranium recovery are those based on complete digestion of the rock, such as digestion of rock with sulfuric acid to make "wet-process" phosphoric acid, which is an intermediate stage in the production of high analysis fertilizers and other phosphate chemicals. Currently about 80 percent of the phosphate rock consumed in the United States is converted to wet-process phosphoric acid; the remainder is used in other industrial and agricultural products from which the uranium is not presently recoverable [2]. During 1978 about 27 million t of phosphate rock containing an estimated 3,000 t of U were converted into wet-process phosphoric acid. This was about twice the quantity of acid manufactured 10 years ago. Recent forecasts are that phosphoric acid production is likely to expand very little in North America (Canada and the United States), but substantial increases are forecast for Latin America and other regions as shown in Table II [3].

There are practicable limitations on the quantity of byproduct uranium that might be recovered from phosphoric acid. Current indications are that uranium recovery probably is not yet economically attractive for small phosphoric acid plants (less than about 100,000 t  $\text{P}_2\text{O}_5/\text{a}$ ) or for those plants digesting lower uranium grade rock. Also, the technology for recovering uranium from more concentrated acid (>40%  $\text{P}_2\text{O}_5$ ) has not yet been fully developed. Considering these constraints, it is currently estimated that domestic uranium production from phosphoric acid could expand to about 3000 t U/a by 1985. Production after that will depend upon the manner and rate of growth of the fertilizer industry.

As previously mentioned, four byproduct uranium recovery plants are in operation and three are under construction in the United States. One plant has been constructed in Canada. These plants are listed in Table III along with the locations and the announced annual uranium production capacities. Although the processes in use or to be used at each plant have not been revealed in detail, it is understood that most plants will use solvent extraction processes, or modifications thereof, similar to those developed over the past several years by the Department of Energy's Oak Ridge National Laboratory (ORNL) [4]. These processes are significant improvements over the earlier solvent extraction processes used when uranium was recovered from phosphoric acid in the late 1950's and early 1960's.

### Process Technology

The following variables or process steps require careful laboratory and pilot plant evaluation in order to determine the

feasibility of recovering byproduct uranium from wet-process phosphoric acid.

- o Uranium content of the rock or phosphorites
- o Solubilization of uranium
- o Acid cooling
- o Uranium valence adjustment
- o Acid clarification
- o Uranium solvent extraction
- o Uranium purification
- o Removal of residual organic from the acid

Because of the variability of the uranium in apatite and phosphorite deposits, it is essential to analyze the phosphate rock used to make wet-process phosphoric acid. The uranium content of the rock, and the acid produced therefrom, significantly affects the economics of byproduct uranium production. Sometimes rock is calcined to decompose contained organic materials, some of which are soluble in the phosphoric acid. Calcined rock produces a cleaner acid for subsequent processing but, unless calcining is carefully controlled, as much as 30 percent of the uranium may become insoluble [5]. Acid produced from calcined rock is sometimes referred to as green acid, whereas uncalcined rock makes black acid.

Ordinarily when phosphate rock containing about 0.01 percent U is digested with sulfuric acid at 75-85°C, over 90 percent of the uranium in the rock reports in the phosphoric acid. The extraction can be increased to over 95 percent by the addition of a small quantity of nitric acid during digestion. Uranium lost with the insoluble calcium sulfate (gypsum) appears to be an actual part of the crystal lattice although some soluble uranium can also be lost due to inefficient washing of the gypsum filter cake. The phosphoric acid strength is maintained at about 30% P<sub>2</sub>O<sub>5</sub> and usually contains 100 to 180 mg U/l. Higher strength acid (40-50% P<sub>2</sub>O<sub>5</sub>) is produced in plants using the "hemihydrate" process but to date this process has been used very little in the United States. Currently, process technology for recovery of uranium from hemihydrate acid is still in the development stage by ORNL.

Phosphoric acid received at a uranium recovery plant is quite hot, generally about 65°C. Cooling or lowering the acid temperature prior to processing is necessary in order to improve the

extraction coefficient, to decrease the solubility of the solvent in the acid, and to reduce the fire hazard. Cooling the acid has disadvantages, such as increasing the viscosity of the acid thus requiring longer phase separation times in solvent extraction, crystallization of salts from the acid that tend to coat heat exchanger surfaces, and the necessity to reheat the acid before returning it to fertilizer production. The optimum temperature of the cooled acid depends on the process used but usually is in the range of 40-50°C to achieve good extraction efficiency and phase separation.

The second step in acid preparation is the adjustment of the acid to an oxidation state (EMF) suitable for the solvent extraction reagent being used. Proper acid EMF is essential to assure that the uranium is of proper valence and to maximize the extraction coefficient and thereby decrease the uranium recovery plant operating and capital costs. For systems where uranium oxidation is required, air, oxygen, ozone, hydrogen peroxide, or sodium chlorate may be used. Chlorate, however, introduces chloride ions into the acid and may not be acceptable due to potential corrosion of equipment. For systems where uranium must be reduced to effect recovery, iron, ferrous salts, hydrogen sulfide, or sulfur dioxide may be used. Scrap iron reduction has been favored in most commercial operations to date.

Following cooling and valence adjustment, it is desirable to remove suspended solids (gypsum, organics, etc.), otherwise the solids tend to build up and create problems during solvent extraction. The usual practice is to use a thickener to settle the solids and to polish the thickener overflow through a filter. The washed filter cake and thickener underflow contain sufficient  $P_2O_5$  and uranium to warrant recycle to the acid plant. Acid soluble and colloidal organics in the acid can be especially troublesome and several methods have been suggested to accomplish their removal. Carbon adsorption and scrubbing the acid with a liquid hydrocarbon have been reported as effective methods for removing organics [6].

Solvent or liquid-liquid extraction is used to remove the uranium from phosphoric acid. Currently, only three extractants are utilized commercially in the United States. They are octyl pyrophosphoric acid (OPPA), di (2-ethylhexyl) phosphoric acid (DEPA) with a synergistic additive such as trioctylphosphine oxide (TOPO), and octylphenyl phosphoric acid (OPAP). The extractant is dissolved in an aliphatic diluent such as kerosene to extract the uranium. There are advantages and disadvantages to the use of each of these extractants that must be evaluated in deciding on a process. Earth Sciences, Inc., concluded in its recent phosphate study for the Department of Energy (DOE) that

there is a significant cost advantage for the OPAP extractant flowheet, if the organic losses in the raffinate are kept below 50 ppm [6]. Gardinier, Inc., Tampa, Florida, opted to use the OPPA process in its new plant because of its past experience with that process. The OPPA process offers lower capital and operating costs but a higher uranium conversion cost since the product, an impure  $UF_4$ , requires special handling at the  $UF_6$  conversion facility. All other companies are using or plan to use the DEPA-TOPO process with modifications developed through laboratory and pilot plant operations.

Except for the OPPA process, a second stage of solvent extraction is used for uranium purification. Depending on the choice of extractants and strip solution, uranium valence adjustment (oxidation) may be required ahead of the second stage. The DEPA-TOPO circuit is favored although OPAP, OPPA, and Amines have been suggested for second stage use. The advantage of the second stage of solvent extraction or the purification circuit is that the uranium product readily meets the concentrate specifications of the  $UF_6$  conversion facilities. The usual practice is to strip the second stage solvent with a carbonate solution, acidify it to a pH of 6.5 and precipitate the yellowcake with any strong base, such as  $NH_3$ ,  $MgO$ , or  $NaOH$ . The yellowcake is thickened, filtered or centrifuged, dried, and calcined as in conventional uranium ore processing plants.

Uranium recovery from the acid is about 90 percent for phosphoric acid containing 140 mg U/l. Often the recovery is expressed in relationship to the quantity of  $P_2O_5$  contained in the acid. For example, Wilkinson reported that the U.S. Phosphoric Products plant, which operated on central Florida rock in the late 1950's, recovered 0.86 pounds of  $U_3O_8$ /ton (short)  $P_2O_5$  or the equivalent of 0.36 Kg U/t  $P_2O_5$  [7].

Prior to returning the phosphoric acid to fertilizer production it is necessary to assure that entrained and dissolved organics are removed or minimized. Besides recovering the expensive solvent, acid cleanup prevents organic-induced deterioration of rubber-lined equipment in the acid plant. Typically the raffinate acid contains about 300-400 ppm of entrained and 100-200 ppm dissolved organics. The entrained organics will coalesce and can be removed by settling in a properly designed coalescer. Dissolved organics can be removed by froth flotation using conventional air-sparged flotation cells. The cleaned acid contains less than 50 ppm organics and is heated to about 70°C for return to fertilizer production.

Accountability for the total quantity of  $P_2O_5$  in the acid going to and returning from the uranium recovery plant is essential. This is usually accomplished by measuring flowrates and sampling

incoming and exit streams although some operators prefer to use weight tanks to measure quantities rather than flowrates. The  $P_2O_5$  losses in the uranium recovery plant are charged to uranium production costs.

It is essential in studying the process characteristics of phosphoric acid to use freshly produced plant acid. Aged acid can produce significantly different results than fresh acid. Also, one cannot assume that the acid in one plant will be similar to that from another plant. Each acid appears to have unique characteristics that must be evaluated using fresh acid. Finally, evaluation studies need to cover a time period sufficiently long to account for the usual fluctuations in plant operations that affect the quality of acid produced.

#### Capital and Operating Costs

As part of its DOE study on recovery of uranium from phosphates, Earth Sciences, Inc., developed capital and operating cost estimates (1979 dollars) for hypothetical byproduct uranium recovery plants [6]. These plants were designed to treat 30%  $P_2O_5$  acid from 155,000 t  $P_2O_5$ /a phosphoric acid plants. Two locations, western and southeastern United States, were considered along with whether the acid were green (made from calcined rock) or black (uncalcined rock). Estimates were based on two uranium concentrations 120 and 160 mg U/l of feed acid and a raffinate concentration of 13 mg U/l.

The total capital investment ranged from \$10.7 to \$12.2 million with western plants being more expensive due to the colder climate. The total capital investment is the sum of direct, indirect, and working capital. Direct capital represents the cost of equipment and installation, buildings, and the initial chemical inventory for the solvent extraction circuits. Indirect capital costs are for engineering and construction management, interest, start-up expenses, and contingency. Working capital is assumed to be 25 percent of the direct annual operating costs.

The operating costs ranged from \$52 to \$65/KgU (\$20 to \$25/lb.  $U_3O_8$ ). These costs include such items as chemicals, power, fuel, labor, maintenance, operating supplies, insurance and taxes, home office expense, and depreciation. Depreciation was straight-line over 20 years and represents about 20 percent of the total operating costs.

As a function of phosphoric acid plant capacity, capital investment was estimated to range from about \$5 to \$18 million for plants ranging in size from 45,000 to 270,000 t  $P_2O_5$ /a. Similarly,

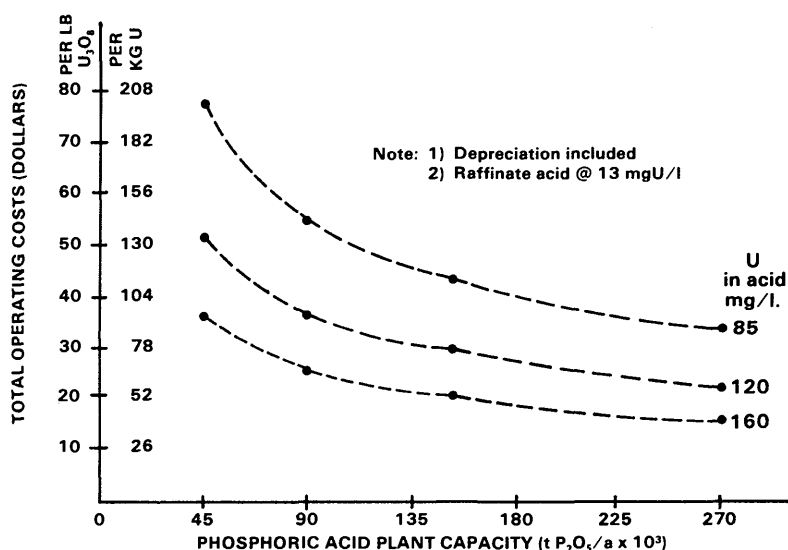


FIG.1. Operating costs as a function of plant capacity and uranium content of acid [6].

total operating costs, including depreciation, as a function of acid plant capacity and the uranium content of the acid were shown to range from about \$44 to \$203/KgU (\$17 to \$78/lb. U<sub>3</sub>O<sub>8</sub>). As shown in Fig. 1, the operating costs are quite sensitive to plant size and the uranium content of the phosphoric acid processed. There was little difference in costs between a western and a south-eastern plant location. Green acid, which is cleaner than black, is somewhat less costly to process.

Earth Sciences, Inc., also calculated the uranium selling price required to obtain a specified discounted cash flow return on investment (DCFR01). Fig. 2 shows the uranium selling prices required to yield a DCFR01 of 15 percent (after tax) as a function of phosphoric acid plant capacity and uranium content of acid. This analysis shows that at a selling price of \$104/KgU (\$40 per/lb. U<sub>3</sub>O<sub>8</sub>) one must have acid containing at least 120 mg U/l and an acid plant with a minimum annual capacity of about 150,000 t P<sub>2</sub>O<sub>5</sub>. However, a plant treating acid containing 160 mg U/l and 270,000 t P<sub>2</sub>O<sub>5</sub>/a would realize a 15 percent DCFR01 at a selling price of about \$68/KgU (\$26 per/lb. U<sub>3</sub>O<sub>8</sub>).

These estimates are for a separate (stand-alone) uranium recovery plant without any sharing of utilities, services, facilities, or administrative and management functions. Savings in capital costs could be as great as 15 percent and in operating costs



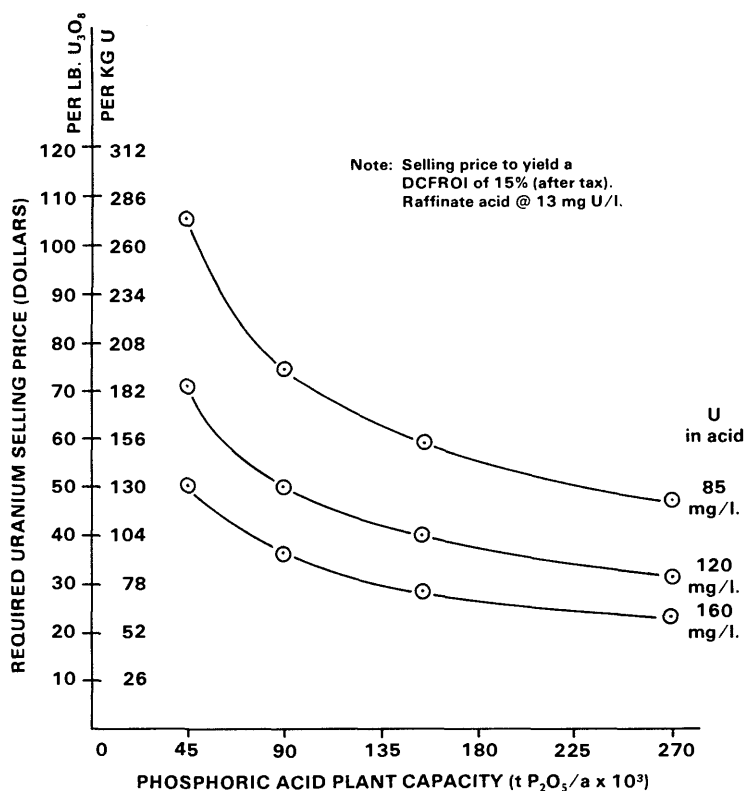


FIG.2. Required uranium selling prices as a function of phosphoric acid plant capacity, and uranium content [6].

of \$2.60 to \$10/KgU (\$1 to \$4/lb. U<sub>3</sub>O<sub>8</sub>), depending on plant size, if the plant could be fully integrated with the fertilizer complex.

Other opportunities for improved economics include a higher uranium recovery (reduction in the uranium content of the raffinate), decrease in reagent consumption, and use of a single centralized facility to treat strip solutions produced by the first uranium solvent extraction circuits at nearby phosphoric acid plants.

#### URANIUM FROM COPPER LEACHING

A survey conducted by the U.S. Bureau of Mines in 1965 revealed that uranium is present in most solutions from the acid leaching of oxidized ores and waste dumps to recover copper [8].

Although the uranium content was found to range from only 1 to 10 ppm, the total potential production was estimated at 800 to 900 t U/a if the major process streams from the larger copper mines in the western United States were treated. In 1967, the Bureau, in cooperation with Kennecott Copper Corporation, demonstrated the feasibility of uranium recovery during a 6-week pilot plant test at the Bingham Canyon mine in Utah. Feed solution for this test was taken after copper cementation, and absorption or resin loadings were evaluated in a multiple-compartment counter-current ion exchange column designed by the Bureau of Mines.

Subsequently, Kennecott constructed and operated an 800 gal/min pilot plant to further develop the ion exchange and uranium recovery technology. In 1974 Kennecott contracted with Wyoming Mineral Corporation (WMC) to confirm the economic viability of uranium production at the Bingham Canyon mine [9]. WMC designed and constructed a plant to treat 6700 gal/min of leach solution with a nominal production rate of 55 t U/a. Some mechanical problems were encountered during plant start-up in 1978, but these have been corrected. With the successful operation of this production-size plant and a continuing strong market for uranium, we can look forward to the construction of similar facilities at other copper leaching operations. Already two Arizona copper producers have announced plans to construct uranium recovery facilities. With continuing success we believe that 500 to 900 t U/a could be recovered from copper leach liquors by the mid-1980's. No data have been released on the economics of uranium recovery from copper leach liquors but the fact that additional producers are entering the field would indicate it is considered a profitable venture at current market prices.

## CONCLUSION

Uranium resource evaluation should always include an analysis of the possibilities of recovering uranium as a byproduct of other industries. Significant progress has been made the last few years in the commercialization of technology to recover byproduct uranium in the fertilizer and copper industries. Other possible sources of byproduct uranium are being studied but no production is anticipated in the near term. As the new byproduct uranium plants gain operating experience and establish firm capital and operating costs, it is expected that additional plants will be built, both in the United States and elsewhere in the world. Byproduct uranium producers in the United States have indicated a willingness to share their process technology and innovations. Some companies are requesting a royalty for patent use whereas other companies offer to design, construct, and operate uranium recovery plants either for a fee or a portion of the uranium produced.

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## DISCUSSION

G. BIXBY ORDÓÑEZ: What legislation is there in the United States of America governing the recovery of uranium as a by-product?

F. E. MCGINLEY: There is no legislation or other Government requirement making the recovery of uranium from phosphoric acid or copper leach liquors compulsory. All uranium produced in the United States is marketed commercially for nuclear power production and any company may produce uranium for sale.

There are, however, Government licensing and environmental protection regulations with which producers must comply.

P. D. TOENS: How much uranium is being spread on the ground each year as a constituent of fertilizer?

F. E. McGINLEY: If we consider only the current production of phosphate rock in the United States, this would amount to about 5000 t of uranium per year, most of which is contained in fertilizers that are spread over farmlands. Because this represents a continuing loss of a natural resource, there are some individuals who believe that uranium recovery, at least from phosphoric acid, should be made mandatory.

P. D. TOENS: Do you think this spreading of uranium could ever become an environmental hazard?

F. E. McGINLEY: There has been at least one environmental study in the United States concerned with the quantities of uranium and radium that are contained in fertilizers. I understand that this is not believed to present a serious environmental problem. The potential problems of accumulation in soils and leaching, with resulting contamination of both surface waters and groundwaters, have also been studied and are reported in the literature.

G. LYAUDET: Do the plants at present in operation in the United States process phosphoric solutions containing identical quantities of uranium?

F. E. McGINLEY: There are some differences in the uranium contents of the acids being treated for uranium recovery. Usually, the concentration will range from 100 to 160 mg/l U, depending on the uranium grade of the phosphate rock digested to make the acid. Sometimes it may be possible to segregate different concentrations or to purchase higher-uranium-grade phosphate rock for acid production, as one plant (US Phosphoric Products) did in the 1950's.

G. LYAUDET: Do losses of solvents such as trioctylphosphine oxide (TOPO) have much influence on operating costs?

F. E. McGINLEY: Losses of solvent have a considerable effect on operating costs, so it is important to keep these losses to a minimum.

B. BOYD: I understand that there are problems with the final step, i.e. with the cleaning of solvent from phosphoric acid. What work has been done on this aspect?

F. E. McGINLEY: There are still occasional problems, usually because of careless operation, but it appears that properly designed coalescers or settlers for removing most of the entrained organic matter and flotation with conventional air-sparged machines for removing the remainder of this organic matter are successfully coping with these problems. Organic solvents must also be removed to prevent deterioration of rubber-lined equipment in the phosphoric acid plant. Usually, the acid must be cleaned to ensure that there are less than 50 ppm organic matter, but this specification is easily met in a properly designed and operated uranium recovery plant.

B. BOYD: Is there a problem with the two-stage process, whereby a uranium-rich portion of phosphoric acid is shipped to a separate plant for the final extraction of uranium?

F. E. MCGINLEY: There is no difficulty with the two-stage process and we will undoubtedly see more of such operations. The Uranium Recovery Corporation was the first to operate a facility of this type and the new plant being constructed by International Minerals and Chemical Corporation at its New Wales fertilizer complex in Florida will also serve as such a facility. It will perform second-stage solvent extraction for three first-stage uranium recovery plants, one at New Wales and two at nearby facilities owned by C. F. Industries Inc. This mode of operation should enable producers to save on both capital and operating costs.

U. CORDERO DI MONTEZEMOLO (*Chairman*): How large must the capacity of a phosphoric acid plant be in order to justify the capital expenditure and operational costs involved in uranium recovery? I was rather surprised to see from your Table III that a plant will soon be started up with a  $P_2O_5$  capacity of only 145 000 t/a and from which uranium recovery is not expected to exceed 40 t/a.

F. E. MCGINLEY: At one time we thought that the minimum capacity of an acid plant would be about 150 000 t/a of  $P_2O_5$  (100%). Because of improved economics, a better figure for today would perhaps be 100 000 t/a of  $P_2O_5$ . However, the uranium content of the acid is also very important. One must look not only at the size of the acid plant, but also at the uranium content of the acid. Even a very large acid plant with a very low uranium concentration in the acid would be uneconomic.

K. CHITUMBO: What do you think is the lower limit in terms of the grade of the rock when deciding whether to recover uranium as a by-product?

F. E. MCGINLEY: With current technology and economic conditions it would not be worthwhile to attempt the recovery of uranium from phosphate rock containing less than about 50 ppm U. However, there are other factors, such as the percentage of the uranium solubilized and the size of the phosphoric acid plant, that are also important. Obviously, the higher the uranium grade of the rock, the more attractive by-product uranium recovery becomes.



# HIGH-PRECISION AUTOMATED BETA/GAMMA TECHNIQUE FOR URANIUM ORE ANALYSIS

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## Abstract

### HIGH-PRECISION AUTOMATED BETA/GAMMA TECHNIQUE FOR URANIUM ORE ANALYSIS.

A fully automated instrument has been developed (AURORA – Automatic Uranium Ore Analyser), which employs the beta/gamma technique for the analysis of uranium ore. The following parameters can be printed out: (a) Sample No.; (b) uranium concentration; (c) Ra/U equilibrium state; (d)  $^{232}\text{Th}$  concentration; and (e) potassium concentration. The system once set up can be left unattended for the analysis of up to 100 samples at a time. The beta/gamma technique has the advantage over gamma spectrometric techniques in that it compensates for disequilibrium conditions such as radon loss and different Ra/U ratios. However, the accuracy obtained by early instrumentation was mediocre ( $\pm 25\%$  above 1000 ppm uranium). With AURORA, the errors of early instruments have now been reduced to the stage where the accuracy obtained is comparable to XRF and chemical analysis. Other advantages are: the sample can be read immediately; simple and quick sample preparation; equilibrium state is given; and large sample mass (50 – 100 g) improves sampling statistics. The key features of the AURORA are: (a) The single detector is a stabilized CaF/NaI phoswich (phosphor sandwich) detector, with a high sensitivity to both betas and gammas; (b) Sample preparation consists of grinding to less than 100 mesh and filling the specially designed sample container; (c) A constant and efficient counting geometry is achieved by placing the detector directly beneath the base of the sample container; (d) Automatic corrections for density effects and thorium and potassium interferences; and (e) Systems operation and data analysis are controlled by a microprocessor, and accurate results are achieved with even low-grade ores. A manual version (GEMINI) has been developed for lower through put requirements.

## 1. INTRODUCTION

### 1.1 Requirement

The major mining houses in South Africa had a requirement for an automatic instrument for mine management and exploration. The requirement was for the rapid analysis of uranium ores with a high

precision, especially at the lower levels encountered in Southern Africa. The preparation of the sample had to be rapid and simple, for in mine management hundreds and in some cases, several thousand samples were required to be analysed daily. The instrument itself had to be simple in operation as the equipment had to be run by relatively unskilled personnel. It was decided that a fully automatic machine was required for mine management, but a manual version having the same specification, would be extremely useful for exploration analysis and low throughput applications. Naturally, the system had to be reliable and relatively inexpensive.

In addition to the uranium concentration, useful information could be obtained from other parameters such as the Ra/U disequilibrium ratio and thorium and potassium concentrations.

## 1.2 Selection of Beta - gamma technique

Beta/gamma counting [1 - 3] was the only technique with the potential of meeting all of the above requirements. An X-Ray fluorescence analyser has a high capital cost, a longer sample preparation time, and does not give the disequilibrium factor. Chemical analysis has a very long sample preparation time and would be extremely difficult to automate. Gamma spectroscopy can give large errors due to radon loss, and a 20 day waiting period is necessary to avoid such errors. Gamma counting without spectroscopy gives very large errors due to the various states of disequilibrium encountered from both radon loss and natural leaching effects. Thorium interference is a serious problem with the pure gamma techniques.

The beta/gamma method has a number of advantages over the other techniques. Minimum sample preparation is required and samples can be read immediately following grinding. Accuracy is relatively independent of the Ra/U ratio and radon loss, and the Ra/U disequilibrium ratio can also be obtained. A large sample mass (50 - 100g) improves the sampling statistics and allows lower uranium ore concentrations to be analysed. The technique can be readily automated and the thorium is not a serious interference. In this paper it is shown that the various sources of error common to earlier instruments have now been reduced to the state where the accuracy obtained is comparable to that obtained by XRF and chemical analysis.

## 2. THEORY

### 2.1 U - 238 decay chain, and disequilibrium states.

Uranium - 238 is the parent isotope of a natural decay chain terminating at lead - 206, which is stable. Table I shows the relative abundances of the significant betas and gammas, when the series is in its natural (secular) equilibrium.

Geological action can lead to large disturbances in the ratio of uranium to radium and radium daughters, since the uranium compounds are more transportable. Thus ore deposits are often found either enriched or depleted in uranium relative to natural equilibrium conditions. Since most of the gammas emitted by the decay chain come from isotopes following radium (Ra-226), the gamma count alone is not necessarily related to the uranium concentration. With the beta-gamma technique, the uranium measurement is based on the Pa-234m beta count (block 1 Table I). Due to the relatively short half-life of Th-234 and Pa-234m, the latter isotope is generally in equilibrium with U-238, and in the event of any disturbance, would return to equilibrium within about 150 days. The



TABLE I.  $^{238}\text{U}$  SERIES (ILLUSTRATING  $\beta - \gamma$  TECHNIQUE).

Isotope	Half Life	Significant	
		$\beta$ ( $\geq 1,5$ MeV)	$\gamma$ ( $> 150$ keV)
U - 238	4500 My	-	-
Th - 234	24d	-	-
Pa - 234m	1,2m	1 <span style="border: 1px solid black; padding: 2px;">1,5 MeV - 9% 2,3 MeV - 90%</span>	-
U - 234	2470 ky	-	-
Th - 230	80 ky	-	-
↑ Relatively Soluble			
Ra - 226	1620y	-	186 keV - 4%
Rn - 222 (gas)	3,8d	-	-
Po - 218	3m	-	-
At - 218	1,3s	-	-
Rn - 218	0,02s	-	-
Pb - 214	27m	-	3 <span style="border: 1px solid black; padding: 2px;">242 keV - 8% 295 keV - 19% 352 keV - 36% 609 keV - 47%</span>
Bi - 214	20m	2 <span style="border: 1px solid black; padding: 2px;">1,5 MeV - 40% 1,9 MeV - 9% 3,3 MeV - 19%</span>	
↓ ↓ ↓ ↓ ↓			
Pb - 206	Stable		

total beta count includes a contribution from Bi-214 (block 2, Table I), which is in a fixed (predetermined) ratio to the gamma count from Pb-214 and Bi-214 (block 3, Table I). About 98% of the gamma count in the energy range 220-660 keV is due to these isotopes, the remainder coming from the U-235 decay chain. Thus the number of Bi-214 betas can be calculated from the measured gamma count, and subtracted from the total beta count to give the number of Pa-234m betas, which is proportional to the uranium concentration, and independent of the U-238/Ra-226 ratio.

The equilibrium of the U-238 series can also be disturbed by loss of Rn-222, which is a gaseous isotope following radium. Some loss of radon gas generally occurs when an ore sample is ground for analysis, and may significantly reduce the gamma count from the sample, since the Pb-214 and Bi-214 daughters of Rn-222 are no

longer generated. The half-life of radon-222 is 3,8 days, and the return to equilibrium takes about 5 half-lives ( $\sim 20$  days). This results in appreciable errors if only the gamma rays are counted during the disequilibrium period. However, it can be seen from consideration of the half-lives of the isotopes following Rn-222, that both Pb-214 and Bi-214 will be close to their equilibrium state with Rn-222 during the creation of this isotope by the Ra-226 parent. Hence, the beta-gamma ratio of Pb-214 and Bi-214 will be relatively constant, and uranium measurement by the beta-gamma technique will be practically independent of radon loss.

## 2.2 Determination of uranium concentration

The beta and gamma counts can be represented by two simultaneous equations, as follows:-

$$B = U \cdot (a + E \cdot c) \cdot f_1(m) \dots \dots \dots (1)$$

$$G = U \cdot (b + E \cdot d) \cdot f_2(m) \dots \dots \dots (2)$$

where

B = nett total beta count per unit time

G = nett total gamma count (220-660 keV) per unit time

U = uranium concentration, ppm

a = nett beta count in unit time from Pa-234m, per ppm U\*.

b = nett gamma count (220-660 keV) in unit time from U-235 series per ppm U\*.

c = nett beta count in unit time from Bi-214, per ppm U\*.

d = nett gamma count (220-660 keV) in unit time from Pb-214 and Bi-214, per ppm U\*.

$f_1(m)$  = beta density correction

$f_2(m)$  = gamma density correction

E = fractional content of Pb-214 and Bi-214 compared with their content in a similar uranium sample in radiochemical equilibrium.

U\* = equilibrium ore.

a, b, c, and d are experimentally determined by measurements on two calibrated ores. Firstly, a radium-free uranium compound is used to determine a and b. Secondly, c and d are found from this information and measurements on a calibrated uranium ore in radiochemical equilibrium by subtracting the two sets of measurements to yield data on the radium section of the decay chain.  $f_1(m)$  and  $f_2(m)$  are determined empirically by altering the density of the calibration standards by compression. This leaves two unknowns, U and E, and U is found by eliminating E from eqns (1) and (2), to yield an eqn of the form:-

$$U = k_1 B - k_2 G \dots \dots \dots (3)$$

where  $k_1$  and  $k_2$  are constants determined by a, b, c, d,  $f_1(m)$  and  $f_2(m)$ .

## 2.3 Errors due to counting statistics

The random nature of radioactive decay gives rise to error limits due to counting statistics. The standard deviation,  $\sigma_U$ , of the uranium measurement is given by

$$\sigma_U = [k_1^2 (B + b) + k_2^2 (G + g)]^{\frac{1}{2}} \cdot t^{-\frac{1}{2}} \dots \dots \dots (4)$$

where

b = background beta count per unit time

g = background gamma count per unit time.

and

t = time

For the instrument described in this paper, typically

$$\sigma_U = [0,9 (B + b) + 0,05 (G + g)]^{\frac{1}{2}} \cdot t^{-\frac{1}{2}} \dots \dots \dots (5)$$

TABLE II. CALCULATED STANDARD DEVIATION  $\sigma_u$  OF THE URANIUM MEASUREMENT DUE TO COUNTING STATISTICS, FOR DIFFERENT COUNTING TIMES.

ppm U	$\sigma_u$ , ppm.		
	1 min	3 min	10 min
10	7	4	2
100	17	10	5
1000	51	30	16
10000	162	94	51

From this equation, and the sensitivities and background count-rates for the detector (Table III), the counting errors can be calculated, for specified counting times.

Table II illustrates these errors for 1,3, and 10 minute counts.

#### 2.4 Measurement of Ra-226/U-238 equilibrium condition

The Ra/U ratio, relative to a value of 1 for an equilibrium ore, can provide useful geological information on the direction or uranium enrichment or depletion, when mapping out an ore body.

It is possible to substitute U, as determined by eqn (3), back into eqns (1) or (2), to find E. However, E will not accurately reflect the Ra-226/U-238 equilibrium state if there has been appreciable loss of Rn-222 gas, since Pb-214/Bi-214 will then be out of equilibrium with Ra-226.

In AURORA, the Ra/U equilibrium factor is found by counting the 186 keV gamma peak, which is due to both Ra-226 and U-235. The contribution of the latter isotope is known from the uranium measurement, assuming that the U-235/U-238 ratio is a constant, as is generally the case. Hence the fraction of the peak due to Ra-226, and the Ra/U factor, can be found.

Knowing both E and the Ra/U factor, the fractional loss of Rn-222 can also be calculated.

### 3. INSTRUMENTATION

#### 3.1 Operation

In the automatic machine (AURORA), the sample containers are stacked in a vertical magazine which can hold up to 100 samples. This procedure can be carried out away from the instrument, thus keeping down-time to a minimum. When the magazine is clipped in position, an automatic transport mechanism then presents the samples sequentially to an electronic balance and the detector. The mass and count information is fed to a programmable calculator, which automatically prints out the uranium concentration and other information in suitable units (e.g. kg/tonne, %, ppm). The sample is finally ejected into a receiving basket (figure 1).

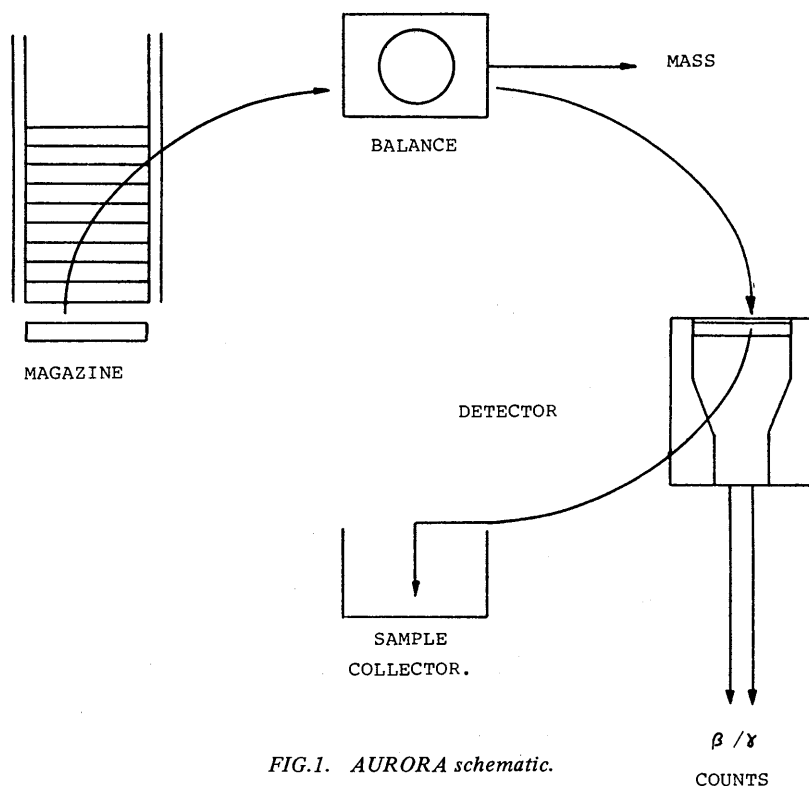


FIG.1. AURORA schematic.

In the manually operated instrument (GEMINI), the sample, in the same type of container, is first weighed and then inserted into the counting position via a drawer assembly. The mass information is inserted via a keyboard.

### 3.2 The sample container

The only preparation consists of the ore sample being ground to less than 150 microns (100 mesh) and poured into a specially designed precision moulded, plastic container. A secure plastic lid is then added to reduce spillage and contamination, and simplify sample preparation and storage. The containers have a circular cross section and are approximately 84mm diameter x 12mm high. The base has a uniform 0,8mm thickness and is a compromise due to considerations such as beta attenuation (20%), mechanical strength, and the necessity for uniform beta attenuation in a batch of sample containers.

The aspect ratio was chosen to give a high efficiency for uranium betas, by optimising the ratio of surface area to volume of the ore sample. The gamma efficiency is thereby also optimised, although this is not so critical, since the gamma count is generally higher than the beta count.

The container height is sufficient to assure an infinite thickness of ore for Pa-234m betas. This is not so for the more energetic betas from Bi-214, and a small density correction is necessary.

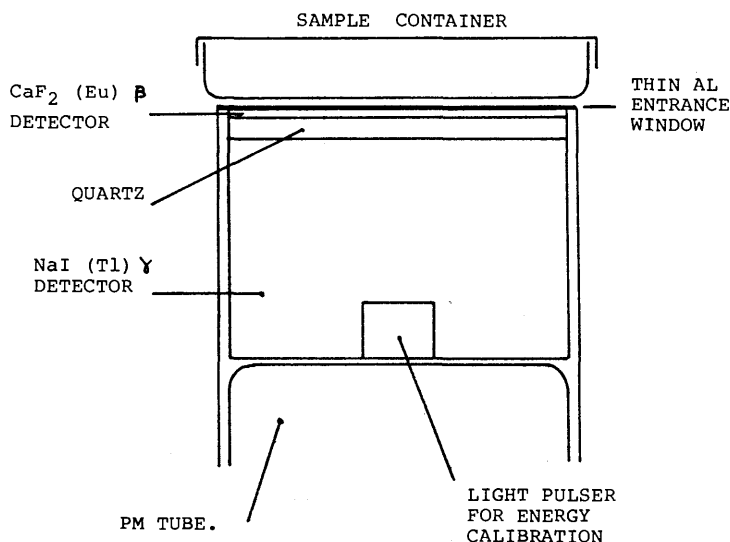


FIG.2. Detector and counting geometry.

### 3.3 Detector and counting geometry

The detector is a 75mm diameter  $\text{CaF}_2(\text{Eu})/\text{NaI}(\text{Tl})$  phoswich scintillation detector [4], which counts betas and gammas separately but simultaneously. The front crystal,  $\text{CaF}_2(\text{Eu})$ , (figure 2), is the beta detector and is only 0,25mm thick in order to minimise the beta background count, which is generally the limiting factor in the measurement of low concentrations of uranium. The quartz crystal following the  $\text{CaF}_2(\text{Eu})$  crystal absorbs the higher energy betas, to prevent them from reaching the 63mm thick  $\text{NaI}(\text{Tl})$  gamma detector. Very few gammas interact in the  $\text{CaF}_2(\text{Eu})$  or quartz, and most pass through to the  $\text{NaI}$  crystal. The decay times of the light pulses from each phosphor are different, and the beta and gamma counts can thus be separated electronically.

It can be seen from eqn. (4) that the detector must have a high sensitivity and low background count, to obtain an acceptable precision with the lower ore grades. The phoswich detector has a high sensitivity for both beta and gamma radiations in the energy range of interest. The thin  $\text{CaF}_2(\text{Eu})$  crystal minimises the beta background count, and this is reduced still further by using pulse shape analysis to reject events occurring simultaneously in both phosphors. The thinness of the  $\text{CaF}_2(\text{Eu})$  crystal also results in a low percentage of false beta counts due to the gamma radiation from the sample. The detector is surrounded with a thick lead shield to attenuate ambient gamma radiation, which causes background counts in both phosphors. Typical figures for sensitivity and background are given in Table III.

Another feature of the detector is the built-in  $\text{Am-241}$  light pulser, which is used as a stabilizer to greatly reduce gain variations in the detector and electronics. Such variations are primarily due to gain drift in the photomultiplier tube, caused by temperature changes and ageing.

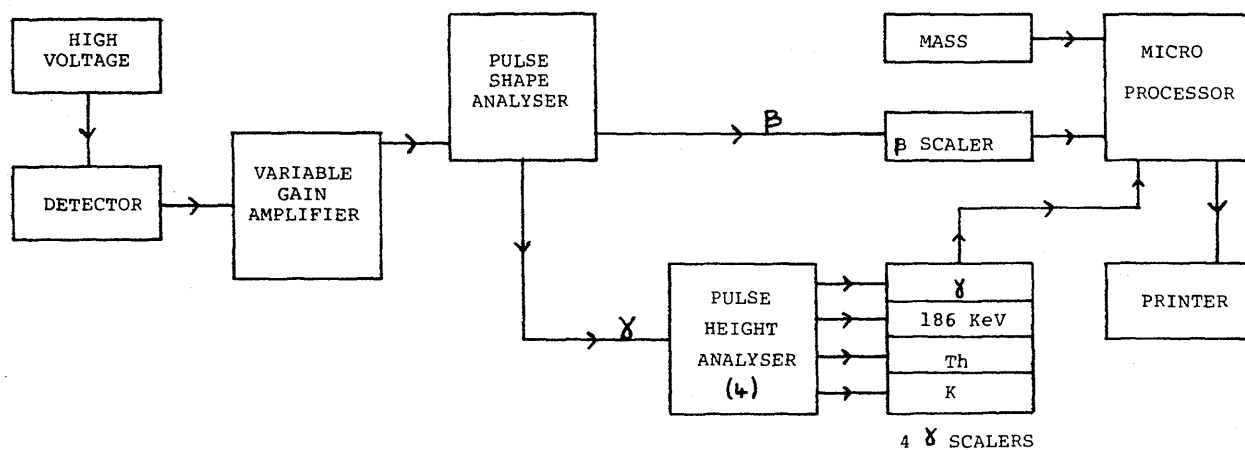


FIG.3. AURORA electronics.

TABLE III. TYPICAL CHARACTERISTICS OF  $\text{CaF}_2$  (Eu)/NaI (TI) DETECTOR

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Beta sensitivity :	2,5 cpm per ppm U (equilibrium ore).
Beta background :	10 cpm
Gamma sensitivity:	6 cpm per ppm U (equilibrium ore).
Gamma background :	300 cpm.
Gamma counts in beta channel:	<1%
Beta counts in gamma channel:	<1%

---

### 3.4 Density effects

The gamma count for a specific uranium content is almost linearly proportional to the sample density (i.e. sample mass, as the sample volume is constant.) Since the density of ores investigated varied by as much as a factor of three, it was clearly necessary to weigh each sample and apply corrections. It was found that the sample mass varied between about 30 - 90g, depending on the ore density. Corrections are made by applying an empirically determined equation of the form:-

$$G_0 = G \exp (k (M_0 - M) )$$

where

G =measured gamma count

G<sub>0</sub> = corrected gamma count

k =a constant

M<sub>0</sub> = a standard mass to which the system is calibrated

M =sample mass.

A similar, but much smaller correction is applied to the beta counts.

### 3.5 Electronics

The electronic schematic is shown in figure 3. The beta and gamma pulses are separated by a pulse shape analyser, the gamma pulses having a much shorter decay time than the beta pulses. The beta pulses are fed directly into a scaler. The gamma pulses are separated by pulse height analysis into 4 separate scalers:- (a) the main gamma counting channel between about 220 - 660 keV, (b) the 186 keV channel, which is used to obtain the Ra/U disequilibrium factor, (c) a thorium channel, and (d) a potassium channel. The information from the 5 scalers is fed together with the sample mass, into the microprocessor.

The sample is counted over a number of fixed time intervals of 10 seconds. The microprocessor can be programmed to repeat the time interval as necessary to achieve either a preset counting accuracy or until a preset time has been reached. Counting to a preset accuracy considerably shortens the counting times of high activity samples. The flexibility offered by the microprocessor results in significant time saving with a high throughput of samples. For example, if the user is only concerned with concentrations above 100 ppm, grades lower than this can be confidently rejected after a 20 or 30 second count. If required, the rejected low grade samples can be loaded into another magazine and counted at night for longer periods.

The microprocessor controls both the system operation and the data analysis.

TABLE IV. RADIOCHEMICAL DISEQUILIBRIUM  
IN PROSPECTING SAMPLES

SAMPLE NO	ppm U <sub>308</sub>		
	$\gamma$	XRF	$\beta-\gamma$
25048	87	864	870
25060	790	25	24
28552	38	548	538
28555	2110	93	114

### 3.6 Correction for interferences

The Thorium-232 series of naturally occurring radioisotopes interfere with both the beta and the gamma count. However, it can be seen that if the beta/gamma ratio of the Th-232 series is made exactly the same as that for Ra-226 and daughters, then the system will completely compensate for this interference. The width of the main gamma channel is adjusted to achieve this balance. With the gain stabilised system, fractions of gamma peaks can be included, so that a U:Th ratio of 1:1 will result in an error of less than 1% in the uranium measurement.

Interference from potassium-40 may become noticeable at ore concentrations of less than 100 ppm uranium. A potassium concentration of 1% will cause the measurement to be 13 ppm higher. The 1,46 MeV gamma peak from potassium-40 can be counted, and appropriate compensation applied. Due to the low counting efficiency, a counting time of the order of 10 minutes would be required for an accurate measurement of 1% potassium.

## 4. RESULTS

The system was calibrated using Canadian reference ore type B1 - 3 [5], which is close to equilibrium, and a 1% standard of a radium-free uranium compound. Good agreement was obtained with measurements by XRF, chemical analysis, and delayed neutron counting for ore samples in the range 20 - 4000 ppm. The errors due to counting statistics were found to be slightly lower than the theoretical predictions of Table II.

The accuracy achievable under extreme disequilibrium conditions is illustrated by Table IV, which shows some results from a Southern African deposit in which the Ra/U ratio varies from 20 to 0, relative to 1 for an equilibrium ore. Values obtained from the gamma count reflect the disequilibrium conditions. About 98% of the gammas are due to radium and daughters, and give very little indication of the uranium content. In contrast, good agreement is obtained between XRF and the beta-gamma technique.

## CONCLUSIONS

Improvements have been made to the beta-gamma technique such that the accuracy attainable is now comparable to that obtained with



XRF or chemical analysis. The lowest detectable concentration is about 20 ppm for a 1 minute count. This can be extended downwards by longer counting times. An instrument has been developed which is capable of analysing up to 100 samples automatically. Correction is provided for density effects, and information on the Ra/U equilibrium state and thorium and potassium interferences is also obtained.

A manual version has been developed for lower throughput requirements.

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## DISCUSSION

K. CHITUMBO: To what extent is your beta/gamma technique compatible with the delayed neutron technique — especially in the presence of thorium?

C.J. SHARLAND: The AURORA is very well compatible with the delayed neutron technique even in the presence of thorium. With equal amounts of thorium and uranium the error is less than 1%.

G.N. KOTEL'NIKOV: How many elements can be analysed simultaneously in one sample?

C.J. SHARLAND: The primary element determined is uranium ( $U_3O_8$ ), but both potassium and thorium are measured to correct for any interference from these isotopes.

G.N. KOTEL'NIKOV: What is the productivity of your device, i.e. how long does it take to analyse one sample?

C.J. SHARLAND: The time taken to analyse a sample depends on its activity and on the accuracy required. The accuracy is predetermined by the operator, and the instrument counts in ten-second increments until the required accuracy is achieved. Thus, a high-activity sample is counted in a very short time (10–20 s) and a low-activity sample takes correspondingly longer, up to a maximum set by the operator (typically 2 min). If the sample is very low in activity (e.g. less than 50 ppm) the AURORA will count for 20 s and then reject it as being too low. Again this lower limit can be set by the operator. The average counting time is usually about 1 min.



# ESTUDIO DE EXPLOTACION CUERPO “TIGRE III”. DISEÑO A CIELO ABIERTO Y ESTUDIO DE UN METODO DE EXPLOTACION SUBTERRANEA DE ALTA RECUPERACION PARA EL CUERPO “TIGRE III”

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## Abstract–Resumen

EXPLOITATION STUDY OF THE ORE-BODY “TIGRE III”. OPEN-CUT DESIGN AND STUDY OF A HIGH-RECOVERY UNDERGROUND MINING METHOD FOR THE TIGRE III ORE-BODY.

The paper first carries out an analysis for the purpose of determining the limiting sterile/ore ratio for open-cut and underground mining in the specific case of Tigre III. In this connection it considers a high-recovery method of underground mining (involving the use of cemented hydropneumatic filling chambers), a general mine plan covering access, transport, ventilation and removal of ore as well as auxiliary services relating to the Tigre III ore body as a whole. The costs of this method of mining are determined for purposes of comparison with the open-cut method. Similarly, the limiting sterile/ore ratio is taken as the basis for an analysis of different types of pit and a design suited to the limiting ratio is adopted. As a final solution the paper favours a method which combines open-cut and underground mining. It proposes the use of the open-cut method up to the limiting ratio (in accordance with the pit design chosen) and of underground method (by the filling chamber method) for the rest of the area.

ESTUDIO DE EXPLOTACION CUERPO “TIGRE III”. DISEÑO A CIELO ABIERTO Y ESTUDIO DE UN METODO DE EXPLOTACION SUBTERRANEA DE ALTA RECUPERACION PARA EL CUERPO “TIGRE III”.

En el trabajo se hace primeramente un análisis para determinar la relación estéril/mineral límite para explotación a cielo abierto y subterránea, en el caso particular de Tigre III. Con este fin se estudia un método de explotación subterránea de alta recuperación (por cámaras de relleno hidroneumático cementado), un trazado general de mina para acceso, transporte, ventilación y evacuación del mineral y servicios auxiliares para todo el cuerpo Tigre III, y se determinaron costos de explotación para comparar con cielo abierto. Así mismo, en base a la relación estéril/mineral límite se analizaron diferentes alternativas de canteras, adoptando finalmente un diseño conforme a la relación límite. Como solución final se adopta un método de explotación combinado, por cielo abierto y subterránea, proponiendo explotar a cielo abierto hasta la relación límite, conforme al diseño de cantera seleccionado y el área restante del cuerpo en forma subterránea, por el método de cámaras de relleno.

## 1. DESCRIPCION GENERAL DEL YACIMIENTO

### 1.1. Ubicación

El yacimiento Tigre III constituye un satélite del yacimiento Tigre I – La Terraza (el depósito más importante de los registrados en el área de Sierra Pintada, 16 000 t de reservas de  $U_3O_8$ ), del que se sitúa a unos 2 km al sur (Fig.1).

Su área se localiza a aproximadamente 30 km al este de la ciudad de San Rafael (Mendoza), enclavada en la porción austral del denominado “Bloque San Rafael”, del cual forma parte Sierra Pintada. El “Bloque San Rafael” es una unidad geomorfológica constituída por una faja serrana de dirección meridional (150 km de larga por 90 km de ancha y 1 800 m s/nm), separada al oeste de la Cordillera Principal por una zona deprimida y limitada al este por una amplia llanura, constitutiva de la unidad geomorfológica “La Depresión Externa de Mendoza”. El yacimiento se ubica a los  $68^{\circ}40'$  de longitud oeste y  $34^{\circ}40'$  de latitud sur.

### 1.2. Geología y reservas

#### 1.2.1. Estratigrafía

En el sector Tigre III del yacimiento “Dr. Baulfés” aparecen los miembros: Areniscas Atigradas y Toba Vieja Gorda, pertenecientes a la Formación Cochicó (Pérmico), con las mismas características litológicas y faciales que presentan en el resto del ámbito del yacimiento.

El Miembro Areniscas Atigradas (portador de la mineralización) está constituído por areniscas de color gris amarillento, de grano fino a grueso, de composición arcósica (plagioclasas, feldespato potásico, cuarzo, líticos) y cemento calcáreo. Presentan estratificación entrecruzada (los geólogos Rodríguez y Valdivieso –1970– de la CNEA, Argentina, le asignan origen eólico fluvial).

El espesor de este conjunto sedimentario oscila entre 100 m y 130 m y en su tercio superior se distingue una intercalación de toba o tufita de marcada importancia en la correlación y que puede considerarse como “piso” de la zona mineralizada de interés.

El miembro Toba Vieja Gorda se superpone al anterior. Se trata de una toba de color gris morado a violeta, de textura porfírica en la que se distinguen clastos de cuarzo, feldespatos, biotita, rocas volcánicas, comagmáticas y otros pertenecientes a la formación La Horqueta (Devónico).

#### 1.2.2. Tectónica (Fig.2)

El área Tigre III forma parte del flanco occidental del braquianticlinal del Tigre. En este sector los estratos tienen un rumbo general N-S a NNO-SSE, con buzamientos que oscilan entre  $15^{\circ}$  y  $20^{\circ}$  al oeste y OSO respectivamente.

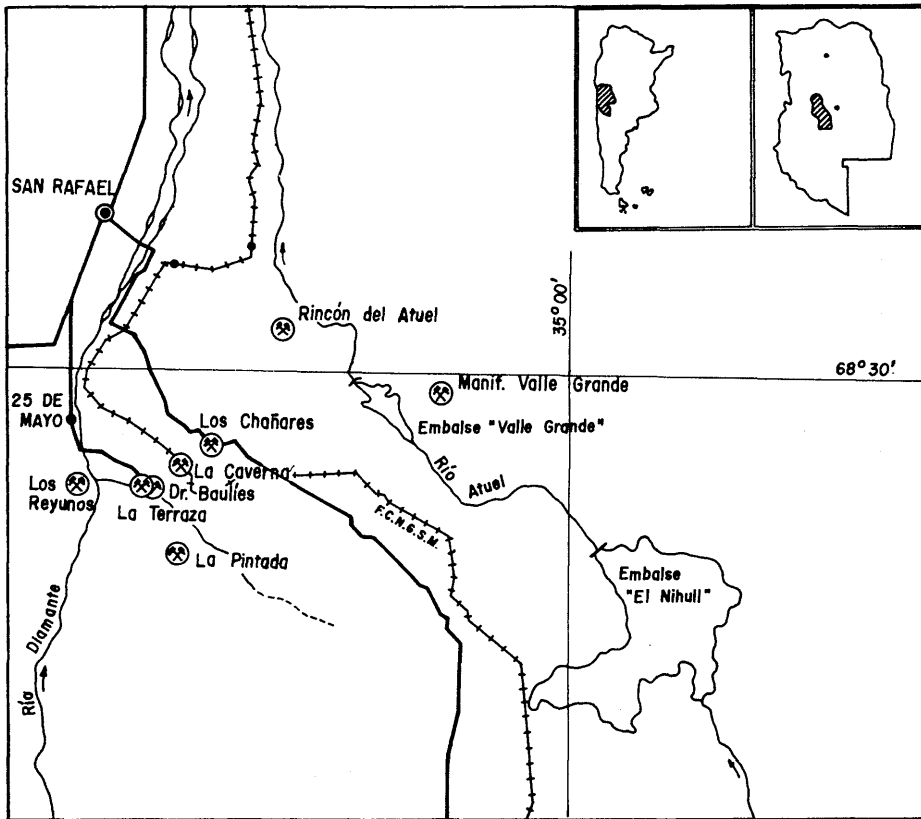


FIG.1. Plano de ubicación del yacimiento.

Todo el conjunto sedimentario, incluida la mineralización, se encuentra afectado principalmente por fallas directas de rumbo aproximado E-O que producen un escalonamiento en bloques que se alzan hacia el norte.

Dos de estas fallas principales delimitan el cuerpo mineralizado: hacia el norte, en bloque bajo, limita con el sector denominado Tigre II y hacia el sur en bloque elevado, está separado de la zona llamada Media Luna.

Excluyendo estas dos fracturas de fuerte rechazo (30 m a 40 m) las restantes fallas mapeadas presentan resaltos cercanos a los 5 m.

La distinta respuesta mecánica de tobas y areniscas a la deformación determina que en las últimas fallas principales sean acompañadas de numerosas fallas menores o manifestarse como diaclasas. En muchos casos estas fracturas se presentan rellenas de calcita.

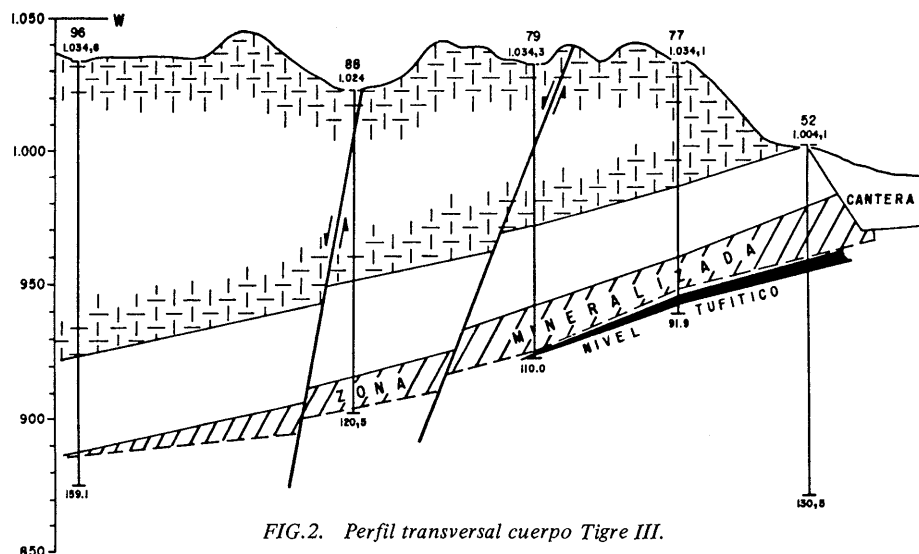


FIG.2. Perfil transversal cuerpo Tigre III.

### 1.2.3. Mineralización

El cuerpo mineralizado, como ya se mencionó, se aloja en Areniscas Atigradas, en su tercio superior, entre 20 m y 35 m aproximadamente por debajo del contacto toba-arenisca, y se extiende por una superficie de 80 000 m<sup>2</sup>.

Hacia el norte y sur los límites son de carácter tectónico; en dirección oeste, sobre buzamiento, la mineralización va perdiendo potencia a la vez que disminuyen los tenores de U<sub>3</sub>O<sub>8</sub>, y hacia el oeste, el mineral ha sido extraído ya en su mayor parte en explotaciones anteriores.

La mineralización es de carácter peneconcordante a la estratificación y morfología lenticular; las areniscas mineralizadas, aunque desde el punto de vista litológico no presentan un techo y piso definido, adquieren una tonalidad rojiza debido a la presencia de óxidos de hierro a los que se encuentran asociados.

La potencia mineralizada no es constante, acunándose por lenticularidad hacia los bordes del cuerpo, alcanzando un valor máximo de 18,25 m en el sondeo 87; la potencia promedio es de 7,45 m (considerando un tenor de corte de fondo de 0,3‰). La zona de máximo espesor tiene un rumbo aproximado NE-SO.

Los tenores son variables tanto en sentido horizontal como vertical, situación que dificulta la correlación entre sondeos.

Los valores mínimos se registran en el sondeo 97 (0,4 ‰ de U<sub>3</sub>O<sub>8</sub>) con un máximo de 1,49‰ de U<sub>3</sub>O<sub>8</sub> en los sondeos 602 y 81, siempre considerando un tenor de corte de fondo de 0,3‰ de U<sub>3</sub>O<sub>8</sub>. La ley media de la totalidad del cuerpo mineralizado es de 0,71‰ de U<sub>3</sub>O<sub>8</sub>. En la Fig.3 está representada la zoneografía de acumuladas con un tenor de corte de fondo de 0,3‰.

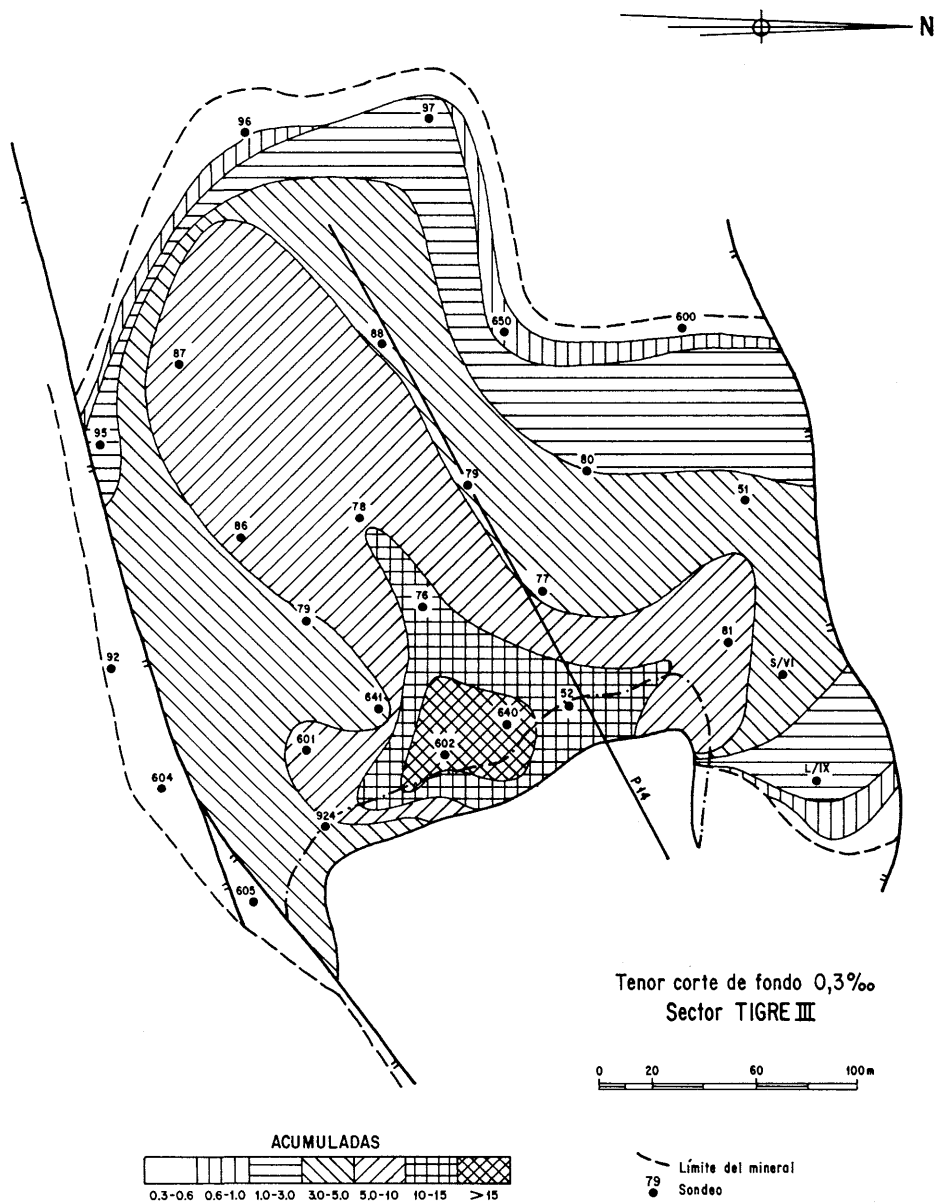


FIG.3. Zoneografía de acumuladas.

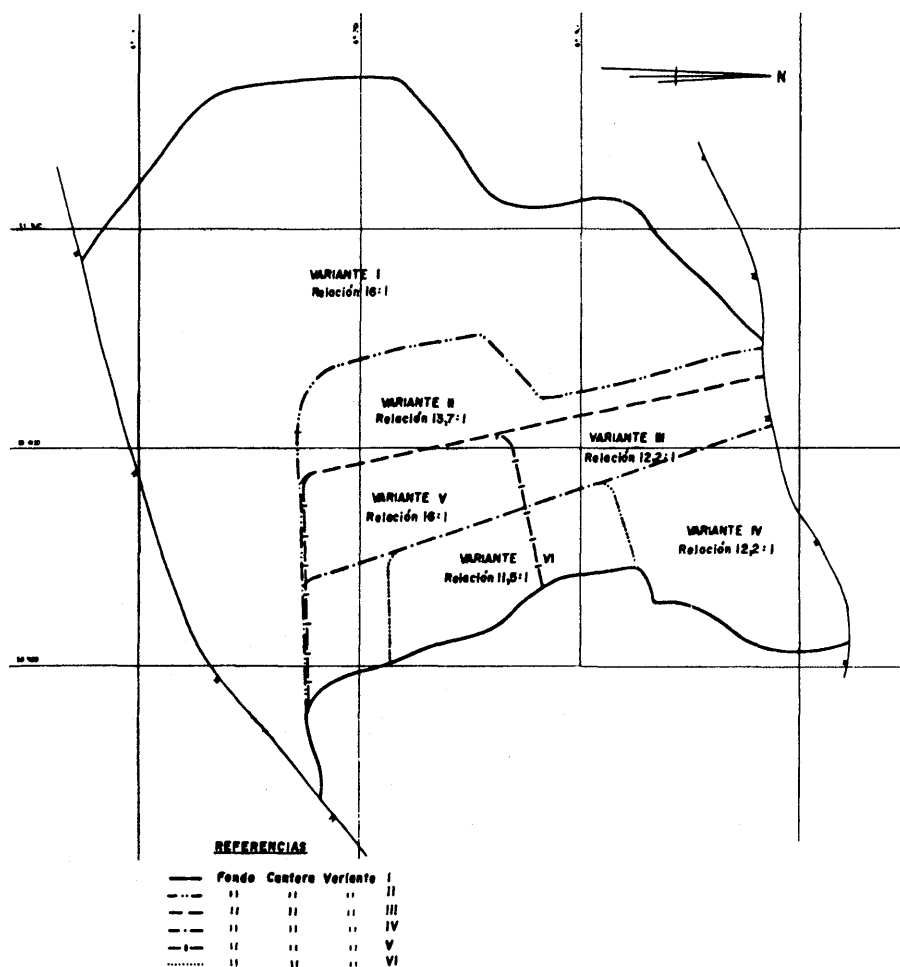


FIG.4. Alternativas de explotación a cielo abierto.

#### 1.2.4. Reservas

La cubicación se realizó por el método del área extendida, construyendo polígonos alrededor de cada sondeo y ponderando los valores de los mismos con las áreas respectivas. Posteriormente los resultados fueron ratificados calculando los volúmenes por el método de los perfiles. Las cifras obtenidas que se indican a continuación corresponden a un tenor de corte de fondo de 0,3‰ de  $U_3O_8$ .



# CUADRO I. COMPARATIVO DE VARIANTES DE EXPLOTACION A CIELO ABIERTO

Variante Cantera	Area de cuerpo aproximadamente considerada	Relación de destape obtenida	Costo estimado por t de mineral en dóls.
I	100%	16 : 1	32,3
II	50%	13,7 : 1	28
III	25%	12,2 : 1	25
IV	12%	12,2 : 1	25
V	10%	16 : 1	32,3
VI	5%	11,5 : 1	23,7

Volumen:	596 231 m <sup>3</sup>
Potencia media:	7,45 m
Ley media:	0,7‰ de U <sub>3</sub> O <sub>8</sub>
Mineral:	1 490 578 t
Fino:	1 058 t

## 2. METODOS DE EXPLOTACION

### 2.1. Introducción

El presente estudio de explotación del cuerpo Tigre III se llevó a cabo tras un análisis previo de factibilidad de alternativas de explotación a cielo abierto y subterránea, dada la alta relación de encape del cuerpo para su explotación a cielo abierto y la posibilidad de utilizar métodos subterráneos competitivos con esa tecnología o combinados.

Se consideraron seis variantes de explotación a cielo abierto (figura 4) con 45° de talud final en todos los casos, obteniendo los resultados del Cuadro I.

Se estudió un método de explotación subterránea con acceso, trazado de mina y metodología de arranque que permitiera fundamentalmente una explotación sencilla, valiéndose de métodos aplicables en función de las características

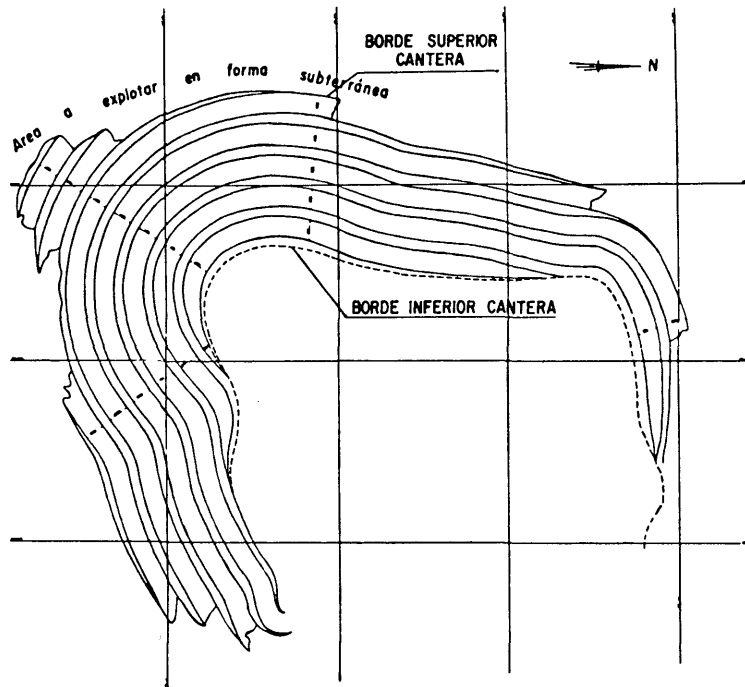


FIG.5. Diseño a cielo abierto.

geológico-estructurales del yacimiento, de bajo costo y alta recuperación, con utilización de una mecanización versátil, moderna, sobre neumáticos que fuera utilizable tanto para las operaciones de preparación como para las de explotación y de empleo conocido en obras civiles de cierta similitud (construcción de túneles en obras viales o diques, excavaciones, etc.). Asimismo, para la construcción de los túneles principales y la excavación de las cámaras de explotación se estudiaron técnicas y metodología de trabajo con las que estuvieran familiarizadas las empresas de obras mineras y civiles existentes en el país, a efectos de posibilitar su participación en la licitación de la obra y la efectiva y eficiente concreción de ésta.

El método de explotación estudiado (descrito en el capítulo 5) es por cámaras con relleno neumático cementado, con un costo real por tonelada estimado en dóls. 15. Las pérdidas totales estimadas son del orden del 20% y las relativas respecto a cielo abierto del 10%, con lo que el costo a los efectos de comparación con este sistema se eleva a dóls. 20, cargándole al costo real las pérdidas relativas.

En base a este costo y a las variantes de contorno de cantera analizadas en primera instancia, se estudió un prediseño de cantera con 50 grados de talud final aproximadamente que diera la relación límite de destape estimado como frontera

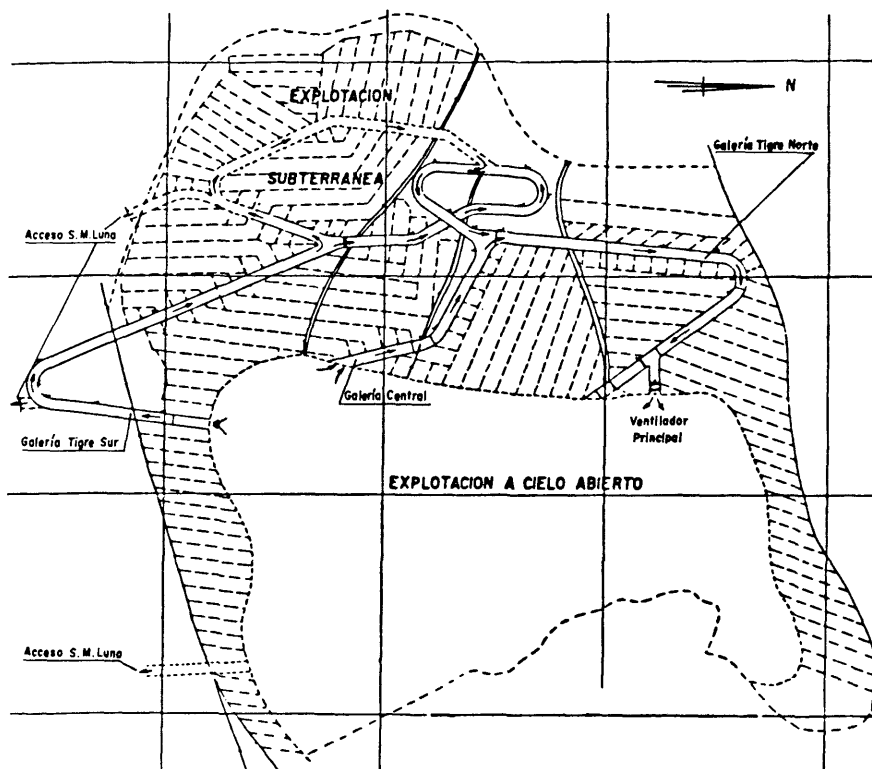


FIG.6. Trazado general de mina subterránea. Distribución de cámaras. Circuito principal de ventilación.

económica entre explotación a cielo abierto y subterránea para este caso particular y en las condiciones establecidas, resultando un contorno inferior de cantera (límite inferior) conforme a lo indicado en las figuras 5 y 6. Se adoptó este prediseño como definitivo para explotar primero parte del depósito a cielo abierto, hasta la relación límite de destape estimada y explotación subterránea en el área restante, cuya relación de destape sobrepasa la relación límite (figuras 5 y 6).

## 2.2. Determinación de la "Relación Límite" de destape para la limitación de las áreas a explotar a cielo abierto y por métodos subterráneos

La relación media de destape del cuerpo Tigre III, considerado un talud del borde terminado de  $45^\circ$  es de 16 : 1, y con borde de  $50^\circ$  de 14 : 1 aproximadamente.

El coeficiente E/M indica en forma directa la economía de una explotación a cielo abierto, y su importancia se explica haciendo un breve análisis:

Llamemos E al número de t de estéril "in situ" que debemos remover para explotar M mineral.

Llamemos p al precio de costo por t movida, considerando a efectos del cálculo el mismo valor para la t de estéril que para la de mineral.

El costo C de la tonelada de mineral a explotar estará dado entonces por la expresión:

$$C = \frac{p(E + M)}{M} = p \left( \frac{E}{M} + 1 \right)$$

Es evidente que para un costo C determinado, la relación E/M depende de p, cuanto más se disminuye p tanto más puede aumentar E/M.

Tomando como base un costo de remoción promedio por t, estimado en dóls. 1,9; un costo subterráneo por tonelada estimado en dóls. 20 (incluyendo perdidas relativas de mineral) y un precio de dóls.40 la lb de  $U_3O_8$  (1978), la relación límite de destape entre explotación subterránea y a cielo abierto, para el caso que estamos considerando, se deduce de la fórmula anterior igualando dicha expresión al costo de explotación subterránea y despejando E/M.

$$20 = 1,9 \left( \frac{E}{M} + 1 \right)$$

De ésta despejamos E/M y obtenemos:

$$\frac{E}{M} = \frac{20}{1,9} - 1 = 9,5$$

Es decir, para el equipamiento actual disponible de acuerdo a la magnitud de la posible cantera Tigre III, el actual precio del uranio y comparando con el método de cámaras con relleno neumático, Tigre III es explotable a cielo abierto al más bajo costo, hasta una relación aproximadamente 9,5 : 1; a partir de esta relación aproximadamente, es más económica la explotación subterránea.

### 2.3. Plan general de trabajo

La explotación total del cuerpo Tigre III se realizaría en forma general en tres etapas que totalizan aproximadamente 6 años, conforme al siguiente detalle:

Etapla 1: Explotación a cielo abierto = 3 años (850 000 t de mineral)

Etapla 2: Preparación subterránea = 7 meses

Etapla 3: Explotación subterránea = 2 años (640 000 t)

### 3. PRE-DISEÑO DE CANTERA PARA LA EXPLOTACION A CIELO ABIERTO

#### 3.1. Generalidades

En la figura 6 se representa el pre-diseño de cantera que se dedujo aplicando la relación E/M límite, un talud final de borde de cantera terminado en  $50^\circ$  y los demás parámetros de cantera adoptados (ancho de bermas y altura final de bancos).

En el área contorneada para extraer mineral se han cubicado aproximadamente  $340\,000\text{ m}^3$  (850 000 t). El estéril encerrado dentro del pre-diseño de cantera adoptado es del orden de  $3\,500\,000\text{ m}^3$ . De estas cifras se deduce una relación E/M (estéril/mineral) = 10.29.

#### 3.2. Parámetros de cantera

Basándose en los datos obtenidos de las explotaciones experimentales a cielo abierto realizadas en Sierra Pintada (Tigre III y Tigre I), comportamiento del terreno, características estructurales de la roca de encape, y rendimientos operativos que se aspira obtener, se han adoptado los siguientes parámetros de cantera:

— Altura de bancos de trabajo en estéril	10 m
— Altura de bancos de trabajo en mineral	2 m
— Talud de los bancos de trabajo	$75^\circ$
— Talud máximo de cantera de trabajo	$40^\circ$
— Ancho mínimo de plataforma de trabajo	18 m
— Berma de trabajo	9 m
— Talud final promedio del borde de cantera terminado	$50^\circ$
— Altura de los bancos del borde terminado	20 m
— Angulo de talud de los bancos del borde terminado	$75^\circ$
— Berma final del borde de cantera terminado	11,6 m
— Ancho mínimo de rampas de acceso (camino)	10 m
— Gradiente medio de las rampas de acceso	$8^\circ$

- Gradiente máximo de las rampas de acceso 12°
- Gradiente máximo en zonas de curva o giro 4°

### 3.3. Control geológico

Con el fin de clasificar el mineral durante el arranque, extracción y acopio en planchadas de mineral, se han previsto los siguientes controles:

- Sondeo y perfilaje radimétrico de avanzada banco mineralizado, en malla aproximada de 20 X 20 m a efectos de elaborar una zoneografía general del área a explotar.
- Perfilaje radimétrico de los bancos para voladuras a fin de elaborar una zoneografía de detalle del área a arrancar.
- Perfilaje radimétrico de sondeos, en malla de 10 X 10 m, en la zona de sobrecarga de estéril inmediatamente en contacto con la mineralización, a efectos de detectar con precisión el techo del mineral, y limitar las voladuras de destape del banco.
- Control de orden de carga del material volado, conforme a las zoneografías obtenidas, tanto en el sentido horizontal como vertical. Se ha previsto limitar las voladuras de mineral en bancos de 2 a 3 m y, eventualmente, 1 m. La carga en caso de necesidad puede llegar a realizarse en capas de 1 m, con equipo de carga adecuado.
- Control radimétrico de camiones de mineral provenientes de cantera, en túnel radimétrico, a efectos de la tipificación del mineral y su colocación en planchadas de stock, en la pila correspondiente conforme a su ley a fin de lograr un “blending” adecuado, de acuerdo con los requerimientos de las plantas de lixiviación en pilas y convencional.

A tal efecto se han previsto las siguientes calidades de mineral:

- a) *Mineral para planta convencional:* Ley media 1,1 kg/t, tenor superior a 0,8 kg, con 3 subclases: 0,8 a 1 kg; 1 kg a 1,3; mayor de 1,3 kg.
- b) *Mineral para lixiviación:* de 0,4 a 0,8.
- c) *Mineral de baja ley para stock en previsión de eventual tratamiento futuro:* 0,2 a 0,4.

Se prevé la necesidad de afectar al transporte del mineral proveniente de cantera camiones de 15 t aproximadamente con geometría uniforme de las cajas volquetes, a efectos de que el área de captación del túnel sea siempre la misma.

- Muestreo de camiones de mineral (uno cada 50 aproximadamente), con el fin de ejecutar y controlar la recta de correspondencia radimetría-tenor.

### 3.4. Plan de producción

Para satisfacer los requerimientos de  $U_3O_8$  de la C.N.E.A. durante los años 1979, 1980 y 1981, se ha fijado para San Rafael una producción de 850 000 t de mineral, en ese período. Dicha producción sería provista en su totalidad, en principio, por la explotación a cielo abierto del cuerpo Tigre III. Conforme a ello surge el siguiente rango de producción de mineral y movimiento de estéril:

Producción anual de mineral	283 000 t
Producción diaria de mineral (300 días/años)	940 t
Movimiento anual promedio de estéril	2 900 000 t
Movimiento diario promedio de estéril (300 días/años)	9 600 t
Movimiento día estéril + mineral	10 540 t

### 3.5. Equipo necesario para la explotación a cielo abierto

Se estimó conforme a las siguientes consideraciones básicas:

Razón de destape promedio	10.3 : 1
Producción por día mineral promedio	940 t
Producción por día estéril promedio	9600 t
Tres turnos de 8 horas de operación	(estéril y mineral)
Seis días de operación por semana	(300 días/año)

Se ha seleccionado el siguiente equipo principal como el más conveniente:

#### 3.5.1. Equipos principales

- Carga del estéril: 2 cargadores frontales con balde roquero y dientes de protección, de 4,6 m<sup>3</sup> de capacidad de balde, montados sobre neumáticos, y una unidad de reserva.
- Carga de mineral: 1 cargador frontal con balde roquero de 2,3 m<sup>3</sup> de capacidad, montado sobre neumáticos, más una unidad de reserva.
- Transporte del estéril: 9 camiones roqueros volcadores de 30 t de capacidad útil de transporte, más 3 unidades de reserva.
- Transporte del mineral: 3 camiones roqueros volcadores de 15 t de capacidad útil de transporte, más 2 unidades de reserva.
- Perforación del estéril: 3 carros roto-percusivos del tipo Crawler-Drill montados sobre orugas, más una unidad de reserva, con los correspondientes equipos de aire comprimido.

- Perforación del mineral: 2 carros perforadores roto-percusivos del tipo Crawler-Drill, montados sobre orugas, más 1 unidad de reserva, más los respectivos compresores o planta de aire comprimido.

### 3.5.2. Equipos auxiliares

Para la preparación de accesos a los bancos, construcción o mantenimiento de caminos, movimiento de rocas y servicios auxiliares, se considera necesario el siguiente equipamiento:

- 2 Topadoras tipo CAT D 9
- 4 Perforadoras neumáticas livianas
- 2 Compresores neumáticos portátiles
- 1 Cargadora frontal de 2,3 m<sup>3</sup>
- 1 Motoniveladora
- 1 Camión para combustible
- 1 Camión lubricador
- 1 Camión para agua
- 1 Camión plano
- 1 Camión grúa
- Equipos varios

## 4. PREPARACION SUBTERRANEA

### 4.1. Trazado de las galerías de preparación

El trazado general de las labores principales que se proponen se representa en la Fig.6.

Mediante el trazado de galerías el cuerpo mineralizado ha quedado dividido en áreas (paños) que se han numerado de I a IX. Cada área de explotación tiene, por razones de seguridad, dos salidas a superficie.

### 4.2. Fundamentos del trazado

Conforme a la información existente sobre las condiciones estructurales del yacimiento Tigre III, el cual se encuentra afectado por la presencia de abundantes fallas y un intenso diaclasamiento, en especial la arenisca portadora de la mineralización que a su vez constituye el techo del sector mineralizado, condiciones que pueden observarse en la arenisca expuesta en la explotación a cielo abierto realizada durante los años 1976 y 1977 en dicho yacimiento, nos



encontramos, para una explotación subterránea, con un techo inestable que es necesario asegurar debidamente para lograr un arranque del mineral en forma segura y económica.

Por tal motivo se consideró más conveniente la aplicación de un método de explotación en descenso (o rebaje), asegurando previa y definitivamente el techo. Para facilitar el acceso a la zona superior de mineral en todas las áreas, de baja y gran potencia (especialmente en estas últimas), se previó emplazar las labores principales en el techo de la mineralización (techo de la galería coincidente con el techo de la mineralización), lo que permite preparar las cámaras con acceso directo desde las galerías principales, ahorrando la ejecución de labores preparatorias secundarias para llegar a la zona superior de las cámaras y simplificando asimismo la posterior metodología de arranque. Esta es la razón fundamental por la cual las galerías principales van en dicho sector constituyendo asimismo parte del método de explotación, dado que las cotas de las mismas no son definitivas, sino que se ha previsto variarlas en descenso, correlativamente con la explotación de las cámaras, para facilitar el arranque de las mismas por banqueo y posibilitar la carga del material desde el nivel que previamente se va a ir dando a la galería principal inferior. Este punto se detalla en el capítulo 5.

El azimut que se propone para las galerías principales en cada tramo se prevé que reúna los siguientes factores: pendiente de las rampas dentro de los límites aceptables en función de los equipos mineros que para tal fin se fabrican actualmente, división del yacimiento en áreas de explotación que posibiliten la construcción de cámaras de longitudes compatibles con la mecanización a utilizar; la formación de un circuito de ventilación, acceso a las áreas a explotar y transporte de mineral, materiales y personal.

#### 4.3. Características constructivas de las labores

Las galerías GTN (Galería Tigre Norte) y GTS (Galería Tigre Sur) se han previsto con una sección de  $22,5 \text{ m}^2$  de 5 metros de ancho por 5 metros de alto.

Tal sección se considera suficiente para la circulación holgada de los equipos (cargadoras frontales, camiones, carros perforadores, etc.) y para satisfacer las necesidades de ventilación de la mina, con rangos de velocidad del aire dentro de los valores aceptables (máximo 300 metros por minuto). Así mismo, con 5 metros de altura se puede montar una tubería de ventilación de 1 metro de diámetro necesaria para la construcción de los túneles.

Dado el intenso fracturamiento de la arenisca en que se construirán los túneles, se ha previsto soportarlos con los siguientes elementos:

- Bulones de 2 a 4 metros de longitud.
- Malla metálica de 10 cm X 10 cm aproximadamente, que se fijará a las paredes y techo de la galería mediante las plaquetas de los bulones.

- Eventualmente gunitado, de un espesor de 3 cm aproximadamente.
- Los elementos de fortificación se regularían conforme a las características del terreno.

#### **4.4. Equipamiento necesario para la preparación**

El avance en roca se realizará mediante arranque con voladuras.

Para la operación de perforación, las diversas alternativas dependen de las ofertas de las diferentes empresas que participen en la licitación de la obra, pudiendo utilizarse Jumbos con martillos hidráulicos o neumáticos, o bien camiones con plataformas y martillos portátiles, con las ventajas e inconvenientes característicos de cada caso particular.

La carga del material se prevé hacerla con cargadoras frontales accionadas con motores gasoleros, cuyas características quedan asimismo libradas a las propuestas de los oferentes. Se consideran convenientes cargadoras del orden de 2,5 m<sup>3</sup> de capacidad.

La evacuación del mineral deberá realizarse en camiones con caja roquera, aptos para el trabajo en interior de mina, dependiendo también su tipo de las ofertas de las empresas participantes en la subasta. Se consideran convenientes camiones del orden de 7 m<sup>3</sup> de capacidad.

### **5. METODO DE EXPLOTACION SUBTERRANEA**

#### **5.1. Método de explotación**

El método de explotación estudiado, descrito en forma concisa es por cámaras con arranque en forma descendente, con sostenimiento del techo y paredes mediante abulonado y malla metálica, con posterior relleno neumático cementado del espacio vacío dejado por las cámaras explotadas.

#### **5.2. Metodología de trabajo**

##### *5.2.1. Potencias de 4 a 20 metros (Figs 7 y 8)*

Para los casos de potencias comprendidas entre 4 y 20 metros se ha estudiado una metodología en la que se cumplan las siguientes etapas:

- (1) Construcción de las galerías de preparación principal emplazándolas en el techo del mineral (cota techo galería coincidente con cota techo mineral).

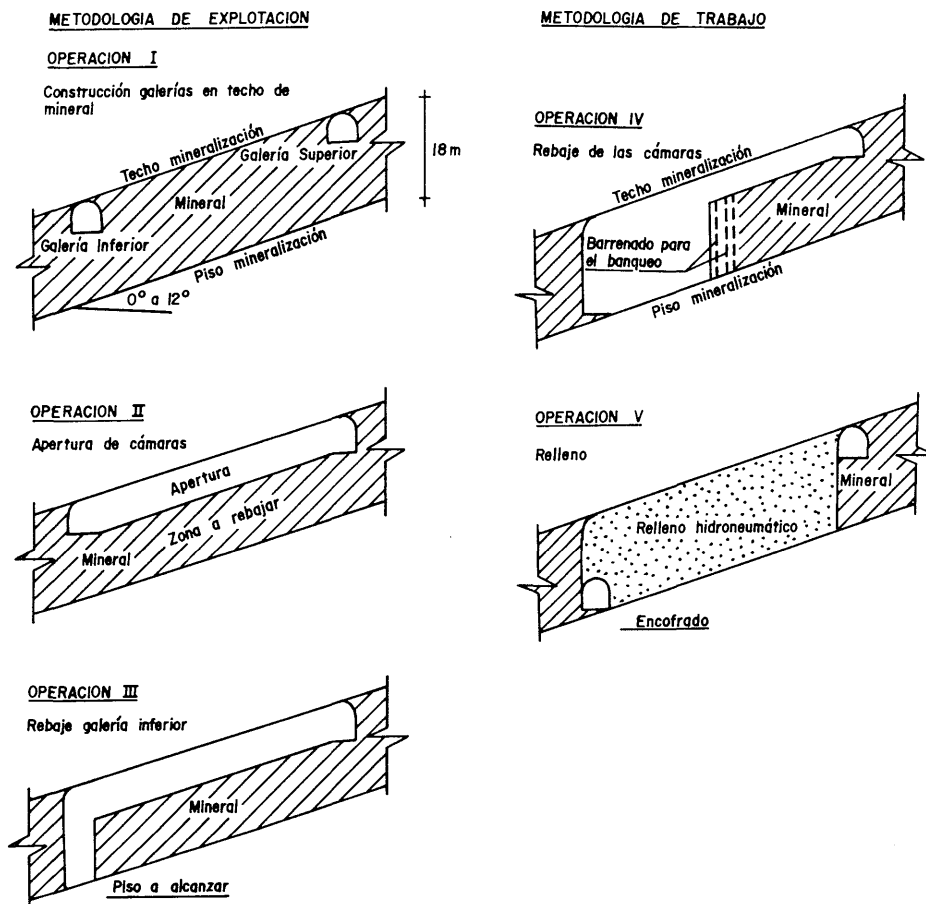


FIG. 7. Metodología.

- (2) Apertura de la cámara a explotar a partir de la galería inferior y superior de la misma. A medida que avanza la apertura de la cámara se procede en forma simultánea a bulonar el techo y las paredes de la misma. La densidad del bulonado y alternativa de colocar malla o no, se regulará según el comportamiento y características de la roca expuesta.
- (3) Rebaje de la galería inferior de la cámara (o como alternativa la superior, si por razones de planificación general de la preparación y explotación es más conveniente esta variante) en toda la longitud del ancho de la misma, a efectos de crear una cara libre que posteriormente permita el banqueo de explotación. La operación de rebaje de la galería inferior irá acompañada con bulonado y enmallado de las paredes.

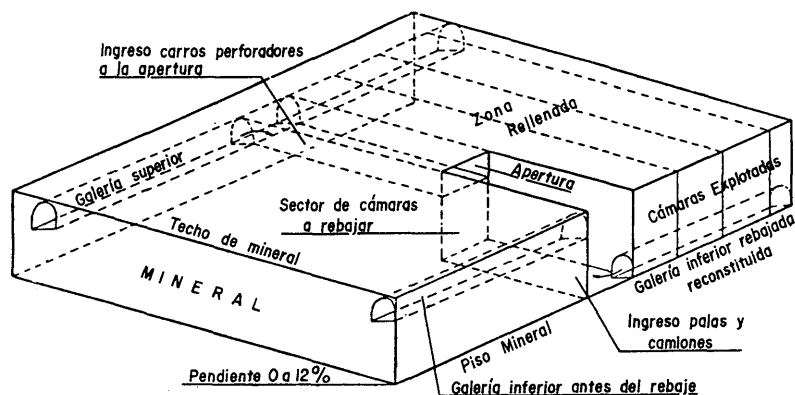


FIG.8. Perspectiva. Método de explotación.

- (4) Banqueo de la cámara (rebaje) hasta la cota en que se rebajó la galería inferior, con progreso de la cara libre del banco desde la galería inferior hasta la superior. Por la galería inferior se entrará a la cámara con la pala cargadora a efectos de evacuar el material volado. Por la parte superior de la cámara accederá el carro perforador para efectuar el barrenado que posibilitará el rebaje por bancos. Durante la operación de rebaje de cámara se procederá a asegurar las paredes de la misma mediante bulonado y enmallado. Los trabajos de fortificación (bulonado y enmallado) se adecuarán en todos los casos a las condiciones del terreno.
- (5) Relleno: Efectuado el rebaje total de la cámara hasta el piso de la mineralización, se procederá al relleno de la misma. Antes de la operación de relleno se armará un encofrado abovedado coincidente con el contorno de la galería inferior, en la cota del piso del mineral a efectos de soportar el relleno y lograr que posteriormente, cuando fragüe el mismo, la galería quede reconstruida en éste. La figura 8 representa el método de explotación en perspectiva para una mejor comprensión del mismo.

#### 5.2.2. Potencias menores de 4 m

Para las potencias menores de 4 metros se suprimen las etapas 3 y 4 descritas precedentemente.

### 5.3. Características constructivas de las labores

Apertura: La apertura de la cámara tendría las siguientes características:

- Ancho: 7,5 metros.
- Alto: 4,0 metros.
- Largo: Según el sector donde esté emplazada la cámara. El largo medio será aproximadamente 40 metros. El máximo 120 metros.
- Pendiente: Según el sector donde esté emplazada la cámara y su orientación. La pendiente máxima no sobrepasará el 12%.
- Bulonado del techo: Se prevé el empleo de bulones de Fe tipo tortal de 20 mm de diámetro por 2 a 4 metros de largo aproximadamente, cementado.
- Enmallado del techo y paredes: Se prevé el empleo de tela metálica romboidal o cuadrada, de una malla de 10 X 10 cm aproximadamente. La misma irá fijada por medio de las plaquetas de los bulones.
- Bulonado de las paredes: Se prevé el empleo de bulones de similares características que los del techo, a razón de uno cada 2 m<sup>2</sup> aproximadamente.
- Alto total de la cámara: Será variable de acuerdo con la potencia del manto del sector que se esté explotando.

### 5.4. Relleno hidroneumático

#### 5.4.1. Características generales

El relleno consistirá en una mezcla de arena, cemento y agua que se transportaría a las cámaras por medio de tuberías, impulsado en forma neumática.

El porcentaje de agua respecto a los sólidos sería aproximadamente del orden del 15 al 20% y tiene por finalidad posibilitar el fraguado de la mezcla. El porcentaje de sólidos sería entre el 80 y 85%.

En la medida de lo posible, la arena deberá ser muy calcárea y contener poco sílice, a fin de disminuir el desgaste interno de las tuberías y bombas por abrasión. El contenido de arcilla deberá ser reducido a nulo a los efectos de obtener un relleno de buena resistencia. El porcentaje de gruesos deberá ser mínimo (no más del 10%–15% con tamaño comprendido entre 10 y 3 mm, el resto será de menor tamaño).

De acuerdo a los ensayos llevados a cabo en los Laboratorios de Agua y Energía (presa Los Reyunos – Mendoza) sobre morteros realizados con distintas relaciones cemento-arena, se adoptó la relación 1 : 20 (cemento-arena) a los efectos

de calcular costos y, en la medida en que se conozca el comportamiento de la roca mientras se desarrolle la explotación de las cámaras, esta relación podría aumentarse haciendo más económico el método.

El contorno de las galerías reconstituídas con un espesor de 1 m debería llevar una relación cemento-arena de 1 : 10 a los efectos de obtener mayor resistencia y además se le agregarían aceleradores de fraguado. En el contorno de la galería se debe lograr en corto tiempo (4—5 días) una resistencia que soporte el peso del relleno sobre ella, con el fin de no retardar el ciclo de minado.

#### *5.4.2. Instalaciones y operaciones para el relleno*

Con objeto de disponer en las cámaras de un relleno que satisfaga las características generales descritas en la subsección 5.4.1, se debería contar con los siguientes elementos:

- Una cantera, o canteras, en la zona del yacimiento que provea el material arenoso y/o utilizar las colas de planta, cuando ésta esté en funcionamiento en el complejo San Rafael, y/o material de desmonte triturado.
- Una instalación de clasificación de arena, compuesta de tolva y zaranda en caso de que la granulometría del material proveniente de las canteras no satisfaga las especificaciones requeridas.
- Una planta de preparación de la mezcla consistente en forma resumida de silo para cemento, silo para la arena, tanque de agua, mezcladores de arena, cemento y agua.
- Equipo de bombeo neumático que puede estar ubicado en la superficie o interior de la mina, en caso de que sea alimentado por gravedad.
- Sala de compresores, de características adecuadas a la capacidad de la planta de preparación de relleno a efectos de proveer de aire a las bombas neumáticas.
- Red de cañerías entre plantas de mezclado y bombas, y entre éstas y el sector explotación. Las cañerías entre mezcladores y bombas pueden ser de 6 a 8 pulg. y entre bombas y cámaras en explotación de 6 pulg, construídas en acero o P.V.C. de resistencia adecuada a las presiones de trabajo (mínimo 60 lbf/pulg<sup>2</sup>).

#### **5.5. Capacidad de la planta**

La capacidad de la planta de relleno ha sido estimada para un volumen diario de 400 m<sup>3</sup>, equivalente a la producción día que se aspira obtener.

#### **5.6. Distribución de las cámaras**

La figura 6 nos muestra la distribución de las cámaras proyectadas para toda el área a explotar en forma subterránea.

### 5.7. Equipamiento necesario para la explotación

Para lograr altos rendimientos operativos y bajos costos se considera que se debe emplear el siguiente equipo:

- Operación de apertura de cámaras y bulonado del techo y paredes: Jumbos hidráulicos o neumáticos.
- Operación de rebaje o banqueo en las cámaras: carros perforadores sobre orugas.
- Operación de carga del material volado: cargadoras frontales, sobre neumáticos, de 2,5 m<sup>3</sup> de capacidad de cuchara.
- Transporte de mineral a superficie: camiones roqueros, aptos para trabajar en interior de mina, de 7 m<sup>3</sup> de capacidad aproximadamente.

## 6. VENTILACION

### 6.1. Circuito principal de ventilación

El circuito principal de ventilación recorrería las galerías principales con entrada de aire por la galería Sur, descenso del mismo hasta la progresiva 241 metros, empalme con la galería Norte, ascenso por ésta en todo su recorrido, hasta salir a superficie (figura 6).

Desde el circuito básico se distribuiría el aire a los distintos sectores en explotación o preparación mediante un sistema de compuertas y/o ventiladores auxiliares.

### 6.2. Ventilación en las labores preparatorias

La ventilación en las labores de desarrollo se haría en forma impelente, utilizando tuberías de P.V.C. u otro material adecuado.

La estimación del volumen de aire necesario durante el avance de las galerías preparatorias se realiza teniendo en cuenta los gases producidos en el interior de éstas, ocasionados por los explosivos y la combustión de los motores diesel.

### 6.3. Ventilación durante la explotación

La necesidad de aire durante la explotación ha sido estimada teniendo en cuenta el número de cámaras necesarias para satisfacer la producción prevista, y considerando los requerimientos de un eventual avance en una galería en desarrollo. La ventilación en las cámaras se establecerá derivando el aire del circuito principal a éstas mediante portones o cortinas de ventilación, cuando

la cámara tenga los dos extremos comunicados a las galerías de ventilación. En el caso de cámaras ciegas, la ventilación se realizaría en forma forzada con ventiladores axiales y tuberías.

Se ha estimado que el ventilador principal montado sobre la galería de ventilación debería aspirar un caudal de aproximadamente 3000 m<sup>3</sup>/min.

## DISCUSSION

**J. MARTIN-DELGADO TAMAYO:** First, I would like to congratulate you on your excellent paper.

Secondly, it seems to me that, in open-cast mining, the main problem you will have, with a waste-ore ratio of 10 : 1 and an average grade of 700 ppm, is how to avoid dilution of the ore by waste; in other words, you will have to be as selective as possible in mining. In the underground part of the mine, for which the mining technique used (descending chambers with supports placed in advance and total hydraulic filling) is an expensive one, this problem will be less serious. But, where the open-cast part is concerned, I would like to ask you what steps you intend to take to improve selectivity, apart from radiometric probing of blast-holes and of driving trucks through the radiometric tunnel.

**J.C. BALUSZKA:** For the scale of the open-cast mining work performed on the Tigre III ore-body we believe that the selection processes used at present are sufficient. These processes are as follows: first, by means of grade control drilling in the roof of the ore, the cut-off between waste and ore is determined with fair accuracy. A 10 m × 10 m drill-hole grid is then made in order to draw contour maps showing the grade ranges of the ore to be blasted. This information is supplemented by logging of the blastholes in the ore, which are made in an even smaller grid. It has been possible to control ore blasting in such a way that ejection does not occur, so that the in-situ contour map of the ore grade does not change. Since this detailed contour map is available, selection can be performed effectively, in both the vertical and horizontal directions, while the ore is being loaded. Ore is selected in the vertical direction while being loaded in one-metre layers with a front-end loading machine having caterpillar tracks which enable it to move on top of the broken rock. This classification performed at the mine face can then be checked with the radiometric tunnel, whereby ore that has been diluted or enriched in the marginal zones between one category and another can be detected.

**L. DEL CASTILLO GARCIA:** Do you think that the radiometric tunnel is really sufficient on its own, or have you considered using an ore classifier?

**J.C. BALUSZKA:** Radiometric tunnels for classifying ore in trucks, used to supplement the original classification during ore loading, have proved sufficient for our needs.



L. DEL CASTILLO GARCIA: Your mining costs seem rather high to me.

J.C. BALUSZKA: As can be seen from section 2 of the paper, the cost of mining per tonne of material (waste or ore) removed, with open-cast mining, is approximately US \$1.90, with a waste : ore ratio of 10 : 1. Under present operating conditions this gives us a cost per tonne of ore extracted of US \$20.90 (which is equivalent to a cost of US \$30/kg of  $U_3O_8$ ). This figure represents the total cost and includes overheads (20%), utilities (10%), value-added tax (16%), gross-receipts (turnover) tax (1.6%) and financing costs (7%).

Underground mining costs (cost of filling plus cost of development plus cost of mining) are calculated at approximately US \$9.46/t, which rise to US \$14.61 when overheads, value-added tax, gross-receipts tax and financing costs are added.



## Invited Review Paper

# DEVELOPMENT OF THE ALLIGATOR RIVERS URANIUM DEPOSITS

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### Abstract

#### DEVELOPMENT OF THE ALLIGATOR RIVERS URANIUM DEPOSITS.

The Alligator Rivers Uranium Province in the Northern Territory of Australia has proven uranium deposits at Jabiluka, Ranger, Koongarra and Nabarlek which contain more than 80% of the country's low-cost reasonably assured uranium resources estimated to be 290 000 t U. Following the Government's decision in 1977 to proceed with the further development of Australia's uranium resources, the region is destined to become a major producer of  $U_3O_8$  for export. At the time of the decision provision was made for strict controls to protect the environment, the granting of Aboriginal land rights and the creation of a major national park. The paper describes the progress made to achieve these objectives. The open-cut mining methods to be used at Ranger, Koongarra and Nabarlek are described, as well as the underground mining operations proposed for Jabiluka. Each of the treatment plants will use the conventional acid leach, solvent extraction purification process for uranium recovery. The characteristics of the treatment operations are outlined. The water-management schemes, tailings disposal methods and procedures for environment protection are also discussed. The proposed initial production capacities of the operations are: Jabiluka 2540 (expanding to 7630 in the fifth year of production); Ranger 2540 (expanding to 5080 when commercially practicable); Koongarra 850; and Nabarlek 920 t U/a. Both Nabarlek and Ranger have been granted Government development approval and construction is proceeding at each site with the expectation that normal commercial production will commence towards the end of 1980 and 1981, respectively. Planning for the Jabiluka and Koongarra projects has reached an advanced stage; each are undergoing environmental procedures and will have to reach agreement with the Aborigines on environmental and other matters before site work can commence.

## 1. INTRODUCTION

The first exploration phase for uranium in Australia was from 1944-61 and resulted in the production of 7,800 tonnes U from five mining centres during 1954-71 [Warner 1]. The second exploration phase began in 1966 [Battey & Hawkins 2], and as a consequence the country's reserves (i.e. reasonably assured resources recoverable at a cost below US\$80/kg U) increased

from 6,200 tonnes in 1967 to 290,000 tonnes U by June 1978. The major Australian uranium deposits and their geology, together with the geologically favourable areas for the occurrence of uranium have been described recently [Ryan 3, OECD-NEA/IAEA 4]. In mid-1978 almost 18% of the Western World's uranium reserves were located in Australia [AAEC 5] .

The recent discoveries of significant uranium resources, the potential for further finds and the fact that nuclear power generation is unlikely to be introduced in Australia before the 1990's suggested that Australia could become a major exporter of uranium. This was affirmed by the Commonwealth Government when it announced in the Parliament on 25 August 1977 its decision to proceed with the further development of Australia's uranium resources on a carefully regulated and controlled basis having full regard to the protection of the environment and the welfare of the Aboriginal people[6]. This decision was made after detailed consideration of the principal findings and recommendations in the reports of the Ranger Uranium Environmental Inquiry (RUEI) [7, 8].

At present uranium concentrate is produced in Australia only at Mary Kathleen in north-west Queensland. The open-cut mine and treatment plant were re-commissioned late in 1975 and the future production rate is expected to be about 600 tonnes U per annum. The new uranium mining projects in Australia are expected to commence operation first in the Alligator Rivers Province. This paper presents an outline of the proposed developments in this area, which on existing knowledge is destined to become the principal uranium production area in the country, and indeed one of major significance in the context of world nuclear fuel supplies.

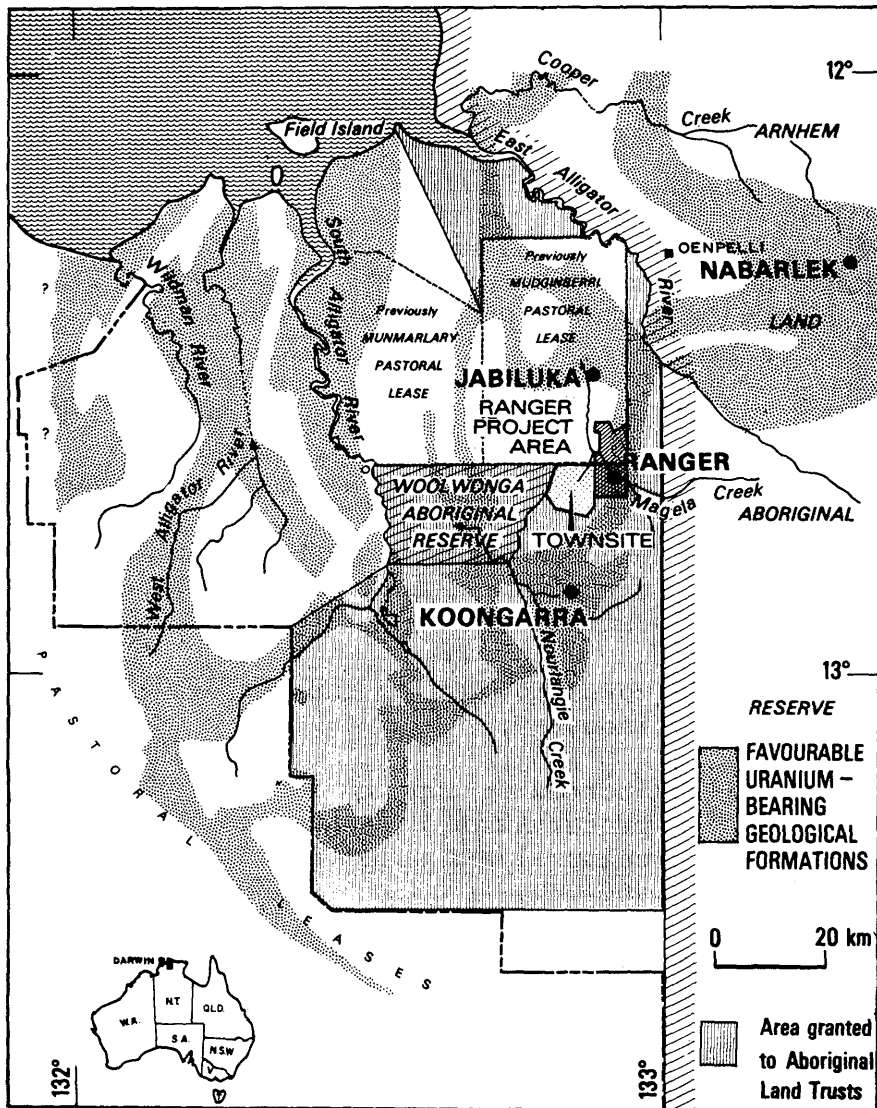


FIG.1. Prospective uranium-bearing formations in the Alligator Rivers province.

## 2. THE PROVINCE IN PERSPECTIVE

The Alligator Rivers Uranium Province is about 200 km east of Darwin in the Northern Territory of Australia and covers an area of about 25,000 km<sup>2</sup>. It includes the catchments of the East, South and West Alligator Rivers, Wildman River, and Cooper, Magela and Nourlangie Creeks. (Figure 1).

Previously the region was remote and little developed, but from about the mid-1960's considerable interest and activity arose, deriving from:

- (i) awareness of its scenic, recreation and tourist attractions, and its diverse wildlife and native flora;
- (ii) proposals for a large part of the region to be declared a National Park;
- (iii) moves to develop the pastoral industry;
- (iv) increasing concern for the preservation of Aboriginal sites of cultural and scientific significance; and
- (v) discovery of very large resources of uranium.

In 1972 fact finding studies were initiated jointly by the Government and the mining companies covering the physical and biological environment, sites of Aboriginal culture, economic resources, land use alternatives and their effect on the environment, and the need for further investigations. [Christian & Aldrick 9].

Today about 800 Aborigines live in the region, principally at Oenpelli, and the total population is rather more than 1,200. In common with much of northern Australia the area has a monsoon-like climate. Virtually the entire rainfall, which can vary considerably but averages about 1,350mm annually, occurs in the wet season generally confined to the November-March period. Evaporation exceeds rainfall in most years, averaging about 2,200mm.

## 2.1 Geology and Uranium Resources

The region comprises complexes of granitoid rocks, gneisses and migmatite mantled and surrounded by a sequence of Lower Proterozoic sediments which are overlain by Middle Proterozoic sandstone and interbedded volcanics to the east and south. Mesozoic sandstone and Cainozoic sand and alluvium cover much of the central and northern parts of the region. The uranium deposits occur in the metamorphosed Lower Proterozoic sediments [Battey & Warner 10]. Because a large percentage of the potential host rocks is covered by younger sandstones or by sand and alluvium, exploration to date is considered to have located only a portion of the uranium resources in the province. The extent of prospective uranium-bearing formations is shown in Figure 1.

The Alligator Rivers Uranium Province contains some 83% of Australia's uranium reserves. Data on the deposits are given in Table 1.

## 2.2 Aboriginal Aspects

Arnhem Land has been a major Aboriginal reserve since 1931; the Nabarlek deposit is located in this reserve. On the basis of the RUEI recommendation the Government has granted to Aborigines title to traditional land in the

TABLE IALLIGATOR RIVERS URANIUM DEPOSITS

Orebody	Discovery Date	(a) Company Announced Resources (tonnes U)	Average Ore Grade % U <sub>3</sub> O <sub>8</sub>	Company/ Organisation Involved
Jabiluka No.1	1971	2,900	0.25	Pancontinental Mining Ltd. and Getty Oil Development Co. Ltd.
Jabiluka No.2	1973	173,000	0.39	
Ranger Nos.1 & 3	1970	85,000	0.22-0.33	Peko-Wallsend Operations Ltd., Electrolytic Zinc Aust. Pty.Ltd. and Australian Atomic Energy Commission
Koongarra	1970	11,300	0.27	Noranda Aust. Ltd.
Nabarlek	1970	8,100	2.35	Queensland Mines Limited
Note: (a) The figures refer to in-situ resources except Koongarra which are for mineable resources at a cut-off grade of 0.018% U <sub>3</sub> O <sub>8</sub> .				

Region (Figure 2), including the southern half of the Ranger Project Area and the Koongarra mining area, under the Aboriginal Land Rights (Northern Territory) Act 1976. Legislative changes have been made to enable Aboriginal land to become part of the proposed National Park. The Government has acquired the Mudginberri and Munmarlary pastoral properties (the former of which includes the northern half of the Ranger Project area and the



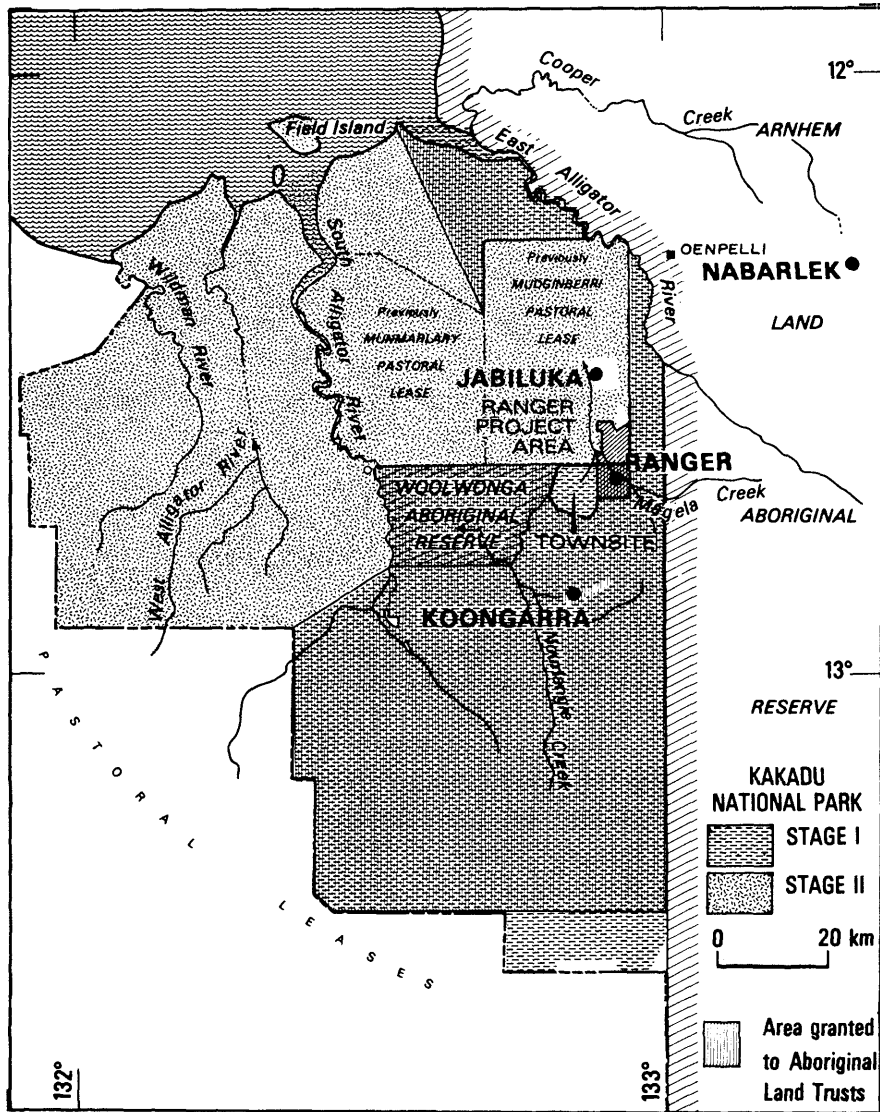


FIG.2. Kakadu National Park. Stage one and two areas.

Jabiluka mining area) to afford an opportunity for Aboriginal land claims to be made and determined and for these areas to become part of the National Park.

The granting of land claims will allow the Aboriginals to negotiate on terms and conditions for mining and allow them to obtain financial benefits under the above Act. The recommendations of the RUEI have been adopted on measures to deal with the impact of development on Aboriginal society including employment, education, race relations, health and alcohol. Provision has been made for the participation by Aboriginals in the management of the National Park and for related training and employment.

### 2.3 National Park

The Kakadu National Park will ultimately cover an area of 12,500 km<sup>2</sup>. Stage One covering approximately 6,500 km<sup>2</sup> was gazetted in April 1979; the balance in Stage Two will be declared later (Figure 2).

No exploration, development or mining is permitted at present in the area declared as Stage One; any future exploration will be subject to the Plan of Management of the park.

The area to be included later as Stage Two of the National Park is to be declared a Conservation Zone and managed in a manner consistent with its future use as a National Park. Exploration, mining and development may be permitted in this area under special controls. However, the Government has stated that until Aboriginal claims over land in the area have been determined it will not permit granting of mining interests over that land without prior consultation with and the agreement of the Aboriginal people.

The Jabiluka, Ranger and Koongarra deposits will be excluded from the Kakadu National Park, but included in Aboriginal land following processes of the land rights legislation. Mining which proceeds in the park area will be on the basis that financial arrangements have been made to ensure that once mining ceases the site will be rehabilitated and then included in the park.

#### 2.4 Township

In order to provide for the needs of the mining companies, administration of the National Park and other Governmental activities a new township is to be established in the area (Figure 2). The town, which will not be part of Aboriginal land, will be within the park, and will be a 'closed' town with a population of not more than 3,500. Late in 1978 the Legislative Assembly of the Northern Territory passed the Jabiru Town Development Act which, inter alia, created the Jabiru Town Development Authority. This legislation gives to the Authority the powers to develop and maintain the town and related functions.

### 3. PROPOSED DEVELOPMENT OF THE RESOURCES

The RUEI recommended a sequence of development of mines in the Alligator Rivers Uranium Province at appropriate intervals. The Government decided to allow the Ranger Project to proceed on the basis of the environmental controls recommended by the RUEI. It also decided that the development of other new mines in the region should take place only when the requirements of the Environment Protection (Impact of Proposals) Act 1974 have been complied with in each case, and that it is satisfied as to the acceptability of the impact of the development on the environment and the Aboriginal people, having regard to the Region as a whole.

The Government noted that these requirements and others such as negotiation between the mining companies and the Aboriginal land owners would, of themselves, lead to a sequence of development. As a result, the Government decided that it would not specify a sequence, either as to order or timing.

Recently significant progress has occurred towards achieving production at Ranger. In November 1978 the Commonwealth Government and the Northern Land Council, acting on behalf of the traditional Aboriginal land owners, reached agreement on terms and conditions as required under the Aboriginal Land Rights (Northern Territory) Act 1976 [11]. In January 1979 the Commonwealth Government and the Ranger Joint Venturers (refer Table 1) signed an agreement on the terms and conditions under which the deposits will be developed [12]. At that time the Minister for Trade and Resources issued an authority under the Atomic Energy Act 1953 for the mining of the deposits, and a management agreement was executed between the three Joint Venturers and Ranger Uranium Mines Pty. Ltd., giving the latter company responsibility for the mining and related operations on the Ranger Project Area. [12]. In March 1979 the Government gave approval for the Nabarlek Project to proceed subject to applicable administrative and legal requirements. These were satisfied later in that month by the signing of the agreement on terms and conditions between the Northern Land Council and Queensland Mines Ltd., and the issue by the Northern Territory Government of a Special Mining Lease to the company.

The source of the information given below generally has been the draft and/or final environmental impact statements published by the companies [13, 14, 15, 16, 17] and other documents in respect of Ranger [8, 12, 18]. In the following discussion

TABLE IIALLIGATOR RIVERS MINEABLE RESERVES

Orebody	Mineable Ore (Tonnes)	Cut-off Grade (%U <sub>3</sub> O <sub>8</sub> )	Contained Reserve (Tonnes U)	Average Grade (%U <sub>3</sub> O <sub>8</sub> )
Jabiluka Nos 1 & 2 <sup>(a)</sup>	53,300,000	0.05	175,900	0.39
Ranger No 1 <sup>(b)</sup>	17,253,000	0.10	43,900	0.30
Koongarra No 1 <sup>(c)</sup>	4,937,000	0.018	11,300	0.27
Nabarlek	494,000	0.10	7,700	1.84
<p>Notes: (a) Jabiluka No 2 orebody has not yet been fully delineated; it contains <math>1.1 \times 10^6</math> tonnes of gold-bearing ore averaging 10.7 g Au/tonne.</p> <p>(b) In addition, Ranger No 3 orebody is estimated to contain 41,600 tonnes U; it has not been fully delineated; it will be developed on completion of mining No 1 orebody.</p> <p>(c) Development of No 2 orebody is not planned at present.</p>				

it should be noted that at present only the Ranger and Nabarlek Projects have received development approval from the Government. The Jabiluka and Koongarra Projects are undergoing environmental procedures and will have to reach agreement with the Northern Land Council on environmental and other matters. It should be realised, also, that some alterations may occur to the current proposals given below for the operations at each project.

### 3.1 Mining Operations

The estimated mineable reserves are given in Table II. Except for Jabiluka the orebodies

TABLE IIIPIT DIMENSIONS AND PARAMETERS

Orebody	Final Pit Dimensions (m)			Wall (a) Slope (degrees)	Bench Heights (b) (m)		
	Major Axis	Minor Axis	Depth		Ore	Waste	Final
Ranger No 1	680	680	175	40	7	14	14
Koongarra No 1	525	275	100	45	9	9	9
Nabarlek	335	185	72	45	6	6	12
Notes: (a) Actual slopes will be established as experience on stabilities of wall rocks is gained.  (b) Proposed heights are chosen to facilitate separation of ore and waste.							

given in Table II will be mined by the conventional open-cut technique of drill, blast, load and haul. The planned final pit dimensions and other parameters are shown in Table III. It is proposed that underground mining methods will be used at Jabiluka.

Approximate initial annual ore production rates will be: Jabiluka 1,050,000; Ranger 1,150,000; and Koongarra 300,000 tonnes. Based on present "firm" planning of future production rates of  $U_3O_8$  these orebodies will have lives of 27, 15 and 12 years respectively. It is proposed to mine fully the high-grade Nabarlek orebody within 29 weeks to assist in subsequent blending and grade control and to allow the pit to be used for permanent

disposal of tailings. The waste to ore ratios for the open pits are Ranger No.1 3.4 to 1; Nabarlek 2.2 to 1; and Koongarra 1.5 to 1. The low cut-off grade used for Koongarra accounts for the favourable waste to ore ratio.

### 3.1.1 Open Pits

Primary production drilling will be by rotary blast hole drills; the actual drill patterns will be influenced by the geological conditions of the rock mass. Blasting will be carried out using ammonium nitrate fuel oil (ANFO) explosive in dry holes and in wet holes using a waterproof plastic liner with ANFO or an ammonium nitrate based water resistant slurry explosive. It is possible that blasting may be contracted to an explosives manufacturer and that deliveries could be made directly from a magazine central to the uranium district. Blasting will be restricted to those times when atmospheric inversions are not present. Secondary breaking will be accomplished by either a mobile hydraulic hammer or by "pop" holes drilled by a track-mounted percussion drill with this blasting occurring at scheduled times.

Grade control programs will be adopted in each pit to minimise dilution and to ensure minimum head grade fluctuations. All blast holes at Ranger will be logged with a radiometric probe and advance information for planning will be obtained by drilling every fourth hole 14 m below collar and probing to that depth. The readings will indicate the locations and grades of ore and waste. Nabarlek will define ore and below ore grade uranium bearing material by radiometric scanning, sampling and assaying.

At each operation broken ore will be loaded into trucks by rubber-tyred or track front-end loaders as these allow greater flexibility for selective mining. The larger Ranger mine plans to

use loaders with a bucket capacity of  $11.5 \text{ m}^3$ , whereas Koongarra intends to use  $4.6 \text{ m}^3$  loaders. Waste materials will also be handled by front-end loaders at Koongarra and Nabarlek, while at Ranger electric-powered shovels with a capacity of  $8.4 \text{ m}^3$  will be used.

All material mined will be removed from the pits by rear-dump, off-highway trucks ranging in capacity from 30 to 77 tonnes. Haul roads within the pits will be from 20 to 25 m wide with an average gradient of 10%. These unsealed roads will be watered by truck mounted sprays to control dust.

Ranger and Nabarlek intend to evaluate the radioactive content of each truck load by means of a scanning device. This scanner will determine the destination of each load of material according to its grade. Blast zones at Koongarra have been computer planned taking account of bench elevation, material type, tonnes of ore and grade for each zone and constant plant feed grade is expected to be maintained by mining on as many as three benches simultaneously.

Depending on the grade of the material hauled, ore will go direct to the crusher or appropriate stockpiles and waste will be sent to suitably located dumps. Ranger plans to stockpile ore of various grades adjacent to the primary crushers. Appropriate blending will assist in maintaining a reasonably constant head grade to the mill. Koongarra proposes to have a low grade ore stockpile of weathered material mined during pre-production stripping. This stockpile will be segregated into two areas with average grades of 0.037% and 0.110%  $\text{U}_3\text{O}_8$ . The higher grade material from this stockpile will be blended with the primary ore from the pit. The lower grade portion is considered to be potential ore and will be treated independently.



At Nabarlek because of the very short mining period (29 weeks) the broken ore will require covering with a suitable protective material to prevent leaching and atmospheric dispersion during the wet and dry seasons respectively.

Auxiliary equipment to support the production units in the pits will consist of dozers, road graders, water tankers and mine service vehicles.

### 3.1.2 Underground

Access to the underground mine at Jabiluka will be provided by two vehicular declines at a gradient of 1 in 10. Ore will be delivered to the treatment plant from the underground primary crushing station by a conveyor located in a separate 1 in 4 decline.

Three mechanised underground mining systems will be used. In the thicker and higher grade sections of the orebody, long hole open stoping and vertical crater retreat stoping will be used for primary stoping and pillar recovery. The narrower lenses will be recovered using cut and fill techniques. All stopes will be filled to prevent surface subsidence, facilitate the control of airflow and limit the production of radon gas. Rockbolting and meshing with gunite cover will provide ground support where necessary. Rubber tyred diesel or electrically powered equipment will be used for drilling, loading, trucking, ground support and service operations to provide maximum mobility.

To meet the requirements imposed by the presence of radon gas, diesel fumes and dust, the ventilation system has been designed to ensure that fresh air passes through the working places and out of the mine by the shortest route and as

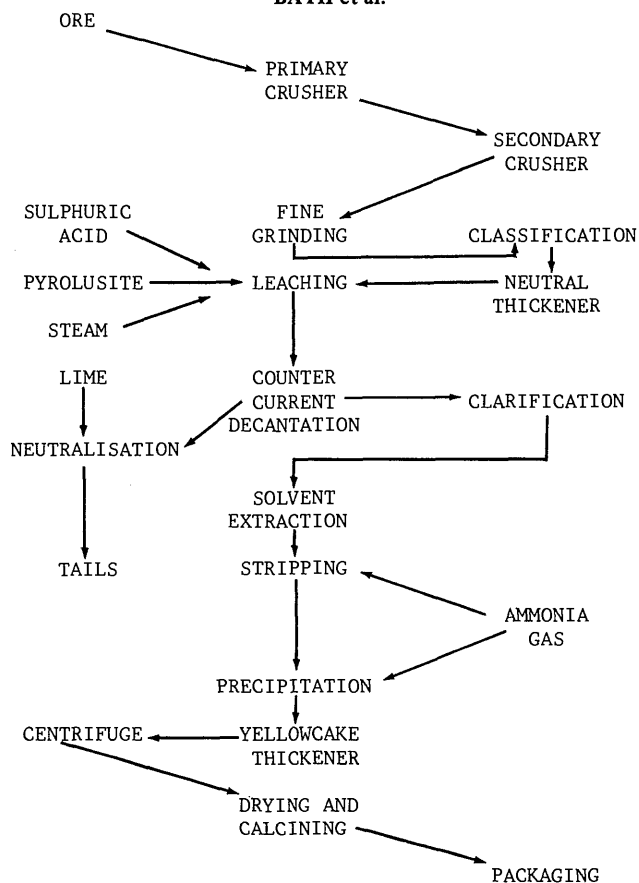


FIG.3. Generalized process flow-sheet – Alligator Rivers mills.

quickly as possible. Ventilation shafts will be developed by raise borers.

Surface facilities will be located adjacent to the vehicular declines portal and will include the mine workshop, warehouse, offices and changerooms.

### 3.2 Treatment Operations

Extensive metallurgical testing of samples from the deposits has shown that the ores

are amenable to the conventional acid leach process for uranium recovery. The treatment process is shown diagrammatically in Figure 3; process parameters are given in Table IV.

At Ranger, Koongarra and Nabarlek run-of-mine ore will be delivered to the crushing plants either directly from the mine pits and/or from stockpiles. Treatment of Jabiluka and Ranger ores will involve both primary and secondary crushing; for Ranger gyratory and cone crushers will be used respectively. Fine grinding will be achieved in a two-stage circuit consisting of a rod mill in open circuit followed by a ball mill in closed circuit with hydro-cyclone classifiers. Primary ore from Koongarra will be crushed in a jaw crusher and then blended with weathered ore before semi-autogenous milling. The discharge from this mill will be classified and the classifier sands sent to a ball mill. Nabarlek intends to use semi-autogenous grinding, classification of the resultant slurry, and ball milling of the coarser fraction.

The wet process of semi-autogenous milling minimises radon and dust emissions, and since Nabarlek ore is amenable to this grinding technique, considerable advantage is gained by adopting it when handling such a high grade material. In the other plants using dry crushing methods dust control will be achieved by the use of water sprays, ventilation fans and high efficiency scrubbers.

Sulphuric acid leaching of the ores is to be carried out in air agitated pachucas (mechanically agitated at Nabarlek) with oxidation potential control by the addition of manganese dioxide as a pyrolusite/water slurry and at temperatures up to 50°C to reduce leaching times.

All uranium treatment plants will use conventional thickeners for counter current decantation (CCD) to separate solids from the

TABLE IV. TREATMENT PLANT DATA

ITEM	JABILUKA	RANGER	KOONGARRA	NABARLEK
<u>GENERAL</u>				
Ore treatment rate (t/a)	1,050,000 (a) 2,100,000 (b)	1,150,000	365,000	61,800
Head Grade (% U <sub>3</sub> O <sub>8</sub> )	0.25 (a) 0.45 (b)	0.30	0.30	1.84
Uranium recovery (%)	95	88.3	94.5	94.4
Plant Availability (%)	90	90	90	90
Concentrate Production (t U/a)	2,540 (a) 7,630 (b)	2,540	890	920
Ammonia (kg/kg U) (c)	0.49	0.51	0.41	0.55
Lime (kg t <sup>-1</sup> )	37	20.0	11.2	30.0
<u>LEACHING</u>				
No. of Pachucas	2 x 7 (a)	10	6	6
Consumption H <sub>2</sub> SO <sub>4</sub> (kg t <sup>-1</sup> )	63	48	30	66
pH	1.8	1.8-2.0	1.6	1.5
Consumption MnO <sub>2</sub> (kg t <sup>-1</sup> )	6.8	7.3	8.0	4.0
Temperature (°C)	45	50	50	40
Retention time (h)	36	29	20	>24
Pulp density (% solids w/w)	56	55-60	50	49
<u>COUNTER-CURRENT DECANTATION</u>				
No. of stages	6	6	6	8
Flocculant (kg t <sup>-1</sup> )	0.12	0.11	0.30	0.36
Under flow (% solids w/w)	55	54	45	45
<u>EXTRACTION AND STRIPPING (d)</u>				
Extraction phase ratio (aqueous to organic)	0.63:1	2.26:1	2.5:1	0.61:1
Stripping phase ratio (aqueous to organic)	1:3	1:11	1:9	1:6

Notes: (a) Data are for initial plant operation.

(b) Data are for treatment during the fifth year of production and subsequently.

(c) Data are total ammonia consumption, i.e. for stripping and precipitation

(d) All projects have four stages for each of extraction and stripping.

uranium-bearing solutions. The thickener underflows are to be neutralised with lime slurry generally to pH of about 8 before disposal to the tailings area at Ranger, Koongarra and Nabarlek. Neutralised tailings from the Nabarlek plant (pH 8.5 - 10.0) will be further treated with barium chloride solution to facilitate precipitation of radium before disposal of the tailings. The partially neutralised tailings at Jabiluka will be treated in a flotation circuit to produce a concentrate which will be treated in a 600 tonne per day cyanidation plant for gold recovery. Flotation tailings will be neutralised before pumping to the mine cemented fill preparation plant. About 50% of the tailings as sands will be returned to the mine as fill. The remainder as slimes will be passed to the tailings disposal area.

The pregnant solutions from the CCD circuits will be clarified in thickeners followed by sand filters before entering the solvent extraction circuits. Uranium concentrations in the feed are expected to be: Jabiluka 1.6, Ranger 0.7, Koongarra 1.0, and Nabarlek 3.6 g U l<sup>-1</sup>.

Solvent extraction is to occur in conventional mixer settler equipment employing counter-current flow and using for the organic phase a tertiary amine (typically Alamine 336, 2.5-4% by volume) and a modifier (isodecanol, about 2% by volume) in kerosene as the diluent (Napoleum 470-B for Koongarra). Stripping of the loaded solvent will be achieved by counter-current flow of ammonium sulphate solution with pH control by addition of ammonia. The stripped solvent will be regenerated with 5 - 10% sodium carbonate solution.

Ammonium diuranate (yellowcake) will be precipitated by addition of ammonia, after which

the slurry will be thickened, washed to remove ammonium sulphate and centrifuged to give a material containing about 60%  $U_3O_8$ . Final drying and calcining up to temperatures of  $650^{\circ}C$  will be performed in oil-fired multihearth driers. The exhaust gases from the calciners are to be scrubbed to control emission and recover  $U_3O_8$  dust.

Either rolls or trommel type crushers will reduce the size of the calcined products to less than 10 mm, which will be stored in bins prior to drumming for despatch. Dust control in the packaging area will be important in equipment selection and operation.

### 3.3 Tailings Disposal

Tailings disposal at the various open pit operations will utilise surface storage dams, storage in the mined out pits or a combination of both.

An environmental requirement for the Ranger Project is that tailings will be returned to the No. 1 Pit on conclusion of operations in that pit unless otherwise mutually agreed by the Commonwealth Government, the Northern Land Council and the Ranger Joint Venturers [12]. The short mining life of Nabarlek means that the pit will be available for the direct discharge of tailings to it during the treatment operations. Koongarra presently propose to use an above surface storage dam. Tailings disposal at Jabiluka has been described earlier.

The dam walls have been designed to minimise seepage and to remain stable under all likely climatic and seismic conditions. At Jabiluka, Koongarra and Ranger, the dams will be earth-rockfill structures, constructed of selected

impervious and semi-pervious materials derived from waste rock and borrow areas. At Koongarra, the borrow<sup>1</sup> area for dam construction material will be within the tailings pond area and will provide essentially sub-surface storage of the consolidated tailings. The Nabarlek pit will be sealed where necessary to reduce the flow of groundwater through the stored tailings.

Final rehabilitation of the tailings disposal areas is not known at this stage, but will depend on agreements to be reached between the parties concerned - for Ranger as indicated above, and for the other three projects the Northern Territory Government, the Northern Land Council and the company involved.

Distribution of tailings in surface dams will use systems of floating discharge lines. A cover of 1 to 2 m of water will be maintained over the tailings during the treatment plant operations to reduce radon emanation and to prevent drying and dusting.

#### 4. SERVICES

Manpower at the various projects will reach a maximum during the construction periods, when the expected workforces will be: Jabiluka 1,600, Ranger 600, Koongarra 250 and Nabarlek 450. Temporary construction facilities will be provided at each site.

Permanent accommodation for the employees of Jabiluka, Ranger and Koongarra will be provided at Jabiru Township, which is to be located 10 km west of the Ranger site and will include all residential, commercial, community and

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<sup>1</sup> "borrow" = to take the soil from one area to another.

recreational facilities. Queensland Mines Limited plans to have accommodation at both Nabarlek and Darwin with air transportation for employees provided between the two centres. A roster system will be implemented so that permanent employees at Nabarlek will spend only working days on site and rest days out of Arnhem Land. The estimated permanent workforces are Jabiluka 785 initially (rising to about 880 in the fifth year of production), Ranger 340, Koongarra 150, and Nabarlek 100 (only 60 of whom would be on site at any one time).

The general service facilities at each project will allow for a high degree of independence of operation. All electrical power requirements will be generated in diesel power stations at the sites. Installed plant capacities will be Jabiluka 35 MW initially (increasing to 40 MW by the fifth year), Ranger 13.8 MW (includes 4.3 MW for Jabiru Township), Koongarra 4.0 MW and Nabarlek 3.8 MW.

Potable water for the Jabiru township and Ranger will be drawn from boreholes on the west side of the South Alligator River and will be delivered by a pipeline routed alongside the Arnhem Highway. Jabiluka will obtain its water requirements from ground water sources within the project area. Water for Koongarra will be from a bore system located approximately 1 km from the plant site. At Nabarlek, water will be pumped from bores located to the east of the mine site and will be stored at the plant site.

Sulphuric acid will be manufactured at each site (excepting Nabarlek) in package plants employing double catalysis and double absorption techniques to minimise emission of sulphur oxide to the atmosphere. The plants will have rated capacities of 450, 185 and 50 tonnes per day at Jabiluka, Ranger and Koongarra respectively.



Elemental sulphur will be imported probably through the port of Darwin. The small amount of acid required at Nabarlek will be purchased and transported to the site by road.

Liquid ammonia will be delivered by road in pressurised tankers and stored at each site. Jabiluka may install an ammonia recovery plant to treat effluent solutions from the yellowcake precipitation process and recycle the ammonia. Ranger may build an ammonia recovery unit depending on experience gained after production has commenced. Pyrolusite, lime, fuel oil, kerosene and other consumables will be brought in by road and on-site storage will be sufficient to meet plant requirements generally for up to four months.

Because road transport will be the principal means of bringing bulk equipment and materials into all mines in the Alligator Rivers Province, sealed roads from the Arnhem Highway will service Jabiluka, Ranger and Koongarra. During the dry season access to Nabarlek will be by a graded road from the end of the main highway and to satisfy a requirement to bring in and take out equipment during the wet season a barge landing site near Laterite Point, 82 km north of the mine, will be used.

An airstrip located near the Jabiru township will provide air access to three of the projects with Nabarlek having its own airstrip providing virtual all weather access.

## 5. ENVIRONMENT AND HEALTH PROTECTION

The Environment Protection (Alligator Rivers Region) Act 1978 provides for the appointment of a Supervising Scientist and the establishment of a Co-ordinating Committee and a

Research Institute for the purposes of co-ordinating and integrating environmental control in the region. The control will be exercised by the Director, Australian National Parks and Wildlife Service and other Government agencies including the Northern Territory Government. The Co-ordinating Committee and the Research Institute, headed by the Supervising Scientist, will assist in the implementation of controls, the development of standards, and the monitoring of the impact on the environment of mining operations.

The environmental conditions for the Ranger Project have been established and are contained in the Authority to Mine the deposits [12]. One of the provisions relating to technology states that "Taken as a whole, and in their component parts, the plant and the mine shall be designed, and the mining, milling and related operations within the Ranger Project Area shall be carried on in accordance with best practicable technology." The Authority to Mine defines in detail the concept of "best practicable technology". The Ranger Project also requires authorisations issued under the Uranium Mining (Environmental Control) Act 1979 of the Northern Territory.

As with the Nabarlek project, it is intended that the Jabiluka and Koongarra projects will take place under leases granted to the companies issued in accordance with the Northern Territory Mining Act (1939). This Act and other relevant Northern Territory legislation will be the means of establishing environmental control over the operations at Jabiluka, Koongarra and Nabarlek.

### 5.1 Water Management Systems

A key element to protect the quality of the environment of the region will be the

establishment of sophisticated water management schemes at each operation to ensure that the release of contaminants to natural water systems is prevented as far as possible.

Jabiluka will operate a 'no release' water management system. Clean and contaminated catchments will be segregated by topography, cut off drains and embankments. Natural run off will be diverted away from or around the contaminated catchment areas, and where possible will be discharged direct to Magela Creek. Contaminated run off will be collected and held in sumps and containment ponds for subsequent evaporation or use in the ore treatment plant. Seepage from containment structures will be minimised by use of impervious clay, membrane liners and concrete as appropriate in their construction. Mine waters, from groundwater inflow, fill dewatering, drilling, washing down and dust suppression will be collected in underground sumps and pumped to the surface to the process make-up water reservoir. Other sources of process make-up water will include the tailings pond, the containment ponds and groundwater sources.

The water management system to be adopted at Ranger is given in the Authority to Mine [12]. The total area of operations at Ranger will be designated a "Restricted Release Zone" and release of liquid water, other than the natural sub-surface flow of groundwater, will only be with the approval of the relevant supervising authority (i.e. Government agency) and in accordance with standards set by that authority. Initially, the water management system will not allow any intentional releases, and before these may be made investigations into the flow, mixing and dispersion characteristics of the Magela Creek system are to be undertaken. A series of three ponds will be used to collect water from the plant area, the pit and the waste dump. Water from the plant area and the waste dump will be used as

treatment plant process water and any excess will be released later if approved. Mine water will be treated with flocculants and will be used where possible in the treatment plant or pumped to the tailings dam.

At Koongarra water management is designed to achieve a no release objective throughout the construction and operation phases. Retention and evaporation ponds will be constructed for control of mine and plant waters and the objective is to be realised by evaporating all water which enters the system and which is surplus to process requirements. A particular feature of the Koongarra system will be the use of dewatering holes to pump out groundwater in the schists surrounding the pit excavation. By this means the surrounding water level will be kept below the mine working bench, thus virtually eliminating inflow of sub-surface waters. The zone of influence of the cone of depression of the water table will include the tailings dam and evaporation ponds. Thus any seepage to deep aquifers from these structures will be drawn into the dewatering holes and will be retained in the water management system. Decommissioning of the system after production ceases is expected to take about five years and will include complete evaporation of all residual waters. Releases will only be necessary in the event of a 1 in 100 year return three year wet cycle.

At Nabarlek, the water management scheme caters for the possible occurrence of a 1 in 10,000 year high rainfall. The system will have retention and evaporation ponds and disposal of tailings into the pit from the commencement of ore treatment. Containment of all waters from the plant area, sewage treatment, ore stockpile and waste dumps will be achieved. A retention pond will act as a buffer storage for excess plant requirements. At the finish of treatment

operations it is expected that ten years will elapse before evaporation of stored waters will be complete. During operation and/or decommissioning application may be made to the appropriate supervising authority for the release of water which satisfies water quality standards for the region.

## 5.2 Health Aspects

The Commonwealth Department of Health has published a "Code of Practice on Radiation Protection in the Mining and Milling of Radioactive Ores" [19]. This code of practice is in force in the Northern Territory, having been promulgated in June 1978 in Regulations under the Mines Regulation Act (1939). All aspects of radiation protection of employees at the operations in the Alligator Rivers region will therefore be enforceable under this legislation.

All drill rigs, loaders, trucks, etc. to be used in the open-pit mining operations will have pressurised cabs fitted with filters and air-conditioners for the protection of the operators. Also, in the case of the Nabarlek operations, to satisfy the radiation exposure level limits, it can be expected that rotation of personnel will be necessary to restrict time spent in the pit and in certain areas of the treatment plant which have higher than usual levels of gamma radiation.

The stopping methods to be used at Jabiluka will minimise exposure to gamma radiation from higher grade ore as operators will spend most of their working time in non-mineralised rock zones. Part of the dose control strategy will also be the rotation of personnel between the various stopping systems. The mine ventilation system will be engineered to keep the radon daughter concentrations below Code of Practice standards.

## 6. CONCLUSION

The mining and milling of the uranium ore deposits in the Alligator Rivers Province will depend on open-cut and underground mining operations and well established treatment techniques for which no unforeseen technical difficulties should arise. The design and construction phases will employ sound engineering practice and with careful control during production the uranium concentrates are expected to meet the most stringent specifications in respect of further processing in the nuclear fuel cycle.

The Ranger Project has Government development approval and major site construction has commenced following the end of the 1978/79 wet season. Present indications are that normal commercial production of 2,540 tonnes U per annum should occur towards the end of 1981. The Ranger Joint Venturers have stated that when it is commercially practicable production will be increased to 5080 tonnes U per annum. The Nabarlek Project also has development approval and production is expected to commence towards the end of 1980 at approximately 920 tonnes U per annum. As mentioned earlier the Jabiluka and Koongarra Projects have not yet been given Government development approval and therefore dates of commencement of production from these resources cannot be specified. However, Figure 4 summarises published information on indicative periods of construction and commissioning for each project once approval is obtained and the schedule of nominal annual production capacities planned for the development of the resources.

Since the August 1977 announcement by the Government to proceed with development of Australian uranium it will be apparent that considerable progress has been achieved toward the establishment

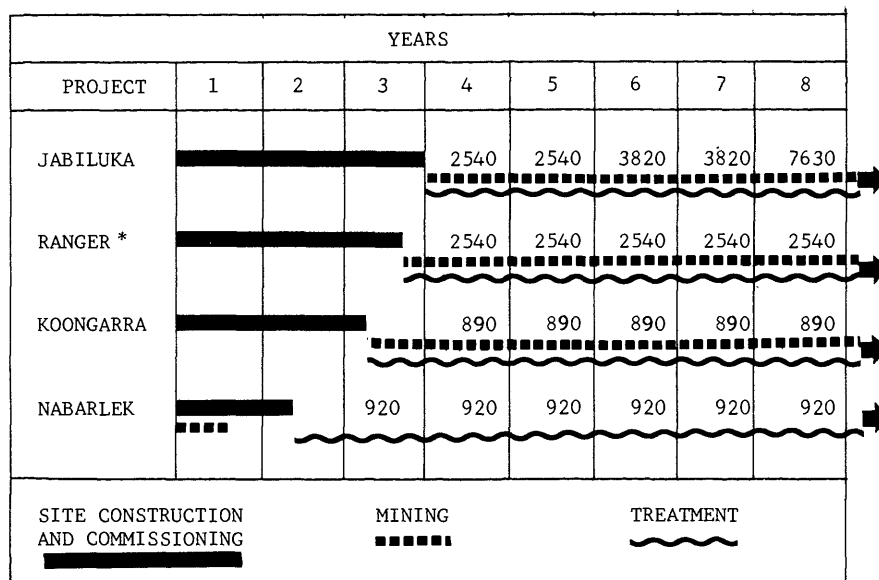


FIG.4. Alligator Rivers uranium projects – proposed construction and commissioning periods, and nominal annual production capacities.\*

Note: At present only Ranger and Nabarlek have Government approval for development; construction of the other projects cannot commence until approval is given.

Production capacities are in t U/a; no estimate is given of the output in the year in which commissioning occurs.

\* Ranger will increase production to 5080 t U/a when commercially practicable.

of a major new uranium mining industry in the Alligator Rivers region of the Northern Territory. Marketing of the product from these projects is outside the scope of this paper; the conditions applying to the export of Australian uranium have been dealt with recently in Ministerial statements [20, 21, 22].

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## DISCUSSION

A.E. BELLUCO: You mentioned the presence of brannerite as a mineral associated with pitchblende. What is the origin of the brannerite and what effect does its presence during processing have on recovery of the ore?

R.K. WARNER: Brannerite has been identified only as a minor constituent of the uranium mineralization of the area. Therefore, its specific origin has not been investigated in detail. Under the acid leaching conditions used, which are designed to give optimum recovery when ores containing mainly pitchblende or uraninite are being treated, it is likely that most of the uranium in brannerite will not be recovered.

K. CHITUMBO: Could you give us some indication of the cost of the recovery methods used, in the light of your estimates of recovery tonnages?

R.K. WARNER: The companies involved in the development of the resources do not make data on the cost of production available to the public.

G.N. KOTEL'NIKOV: In view of the climatic conditions of the Ranger and Nabarlek mines (1350 mm precipitation per annum) I would like to know something about the prospecting aspect of opening up the deposit. On one of your

slides you showed ore which consisted of pitchblende and hydrated uranium oxides, and this demonstrates that leaching processes in the hypergenesis zone have been insignificant. At what depth was the sample taken and what prospecting techniques were used both when the deposit was first discovered and for subsequent prospecting? What is the age of the host rocks and the absolute age of the mineralization?

R.K. WARNER: The Ranger and Nabarlek deposits were found in the course of investigating very strong anomalies discovered in airborne radiometric surveys. The anomalies were investigated by means of a series of trenches and drilling programmes. Both ore-bodies cropped out and the zone of weathering extended to a depth of 15 to 30 metres. The depth of the sample from Nabarlek shown on the slide was 5 metres from the surface. The ore-bodies occur in the Lower Proterozoic Cahill Formation, which is of the order of 2200 to 2400 million years old. It is difficult to quote an absolute age for the mineralization, as it is considered to have a complex depositional history of successive phases of uranium deposition, metamorphism and metasomatism. Ages of 900 to 1600 million years have been recorded at Ranger and 900 million years at Nabarlek.

J.J. SCHANZ: Could you say, in a general way, what your expectations have been for the resources in the area from the moment at which detection of anomalies first occurred up to the present, when the area has been partially developed?

R.K. WARNER: During the period from the time of detection of anomalies (late 1960) to June 1978, reasonably assured resources recoverable at a cost below US \$90/kg U (i.e. reserves) of 241 000 t of uranium were established in the Alligator Rivers area. Estimated additional resources have not been quantified: much of the host formation is covered by sandstones, sand and alluvium, and only about 15% of the prospective area has been explored adequately. The province is considered to have a large potential for further discoveries, and economic resources may possibly be 5–10 times those already proven.

J.A. PATTERSON (*General Chairman*): What is the nature and extent of current exploration and resource assessment activities being performed by industry and the government in the Alligator Rivers area?

R.K. WARNER: At present no exploration is being undertaken in the area of the national park. In Stage One, which has been gazetted, future exploration will be subject to the Plan of Management of the park. Stage Two is to be declared a Conservation Zone in the interim period before its inclusion in the national park. It is expected that exploration in this part will be permitted under special controls. Outside the park areas exploration is being carried out.

J.A. PATTERSON (*General Chairman*): What prospects are there of finding uranium in the Alligator Rivers area in new deposits, in the light of the changing status of territory in the Aboriginal lands and in national parks? Although the Alligator Rivers area may be potentially the world's largest uranium district, the likelihood of the development of these areas seems to be very uncertain.

R.K. WARNER: The development of further deposits discovered in the region will take place when the requirements in respect of environmental protection and of the impact on the culture and society of the local Aborigines have been met under the relevant legislation. The Ranger and Nabarlek projects have been pioneers in this respect and demonstrate the government's firm intention to ensure orderly development of the uranium resources of the region.

# **ESTIMATION OF UNDISCOVERED URANIUM RESOURCES**

**(Sessions VI and VII)**



## EVALUATION OF URANIUM RESOURCES IN ANTARCTICA

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### Abstract

#### EVALUATION OF URANIUM RESOURCES IN ANTARCTICA.

The continent of Antarctica comprises roughly nine per cent of the total land surface of the earth and is the only large land area that has been left almost totally unexplored for uranium resources. In 1976 the first systematic uranium resource evaluation, entitled Antarctica International Radiometric Survey, was started as a part of the US Antarctic Research Program. This project was staffed jointly by scientists from the University of Kansas and the Bundesanstalt für Geowissenschaften und Rohstoffe of the Federal Republic of Germany. The survey has continued for three antarctic field seasons and an extension of operations for the next four years has been approved. Two areas in the Transantarctic Mountains and one part of Marie Byrd Land have been surveyed by airborne gamma-ray spectrometric methods. The work that has been conducted demonstrates clearly that radiometric surveys can be performed successfully under the rigorous climatic conditions in Antarctica, and that significant and reproducible data can be obtained. So far no substantial concentrations of uranium have been detected but deposits of thorium minerals have been found.

### INTRODUCTION

Although one of the primary objectives of the Antarctic International Radiometric Survey is to provide a comprehensive assessment of the potential uranium resources of the antarctic continent, it is obvious that this goal can not be realized in the near future. The total area of Antarctica is 13.5 million km<sup>2</sup> but only about 2% is ice-free [1]. Approximately 260 000 km<sup>2</sup> are actually accessible for direct airborne or surface radiometric measurement. Survey activities are further complicated by the fact that many of the individual outcrops are widely separated and large contiguous ice-free areas are not common. The long distances between outcrops, the high relief that characterizes most of the ice-free terrain, the extreme polar climate and the remoteness from inhabited areas combine to make radiometric survey activities both difficult and costly. Work has to be limited to the antarctic summer season with a maximum operational period of about three months a year.

In the three years during which the radiometric survey has been in progress, about 50 000 km<sup>2</sup> were accessible by helicopter from the three camp sites. Within this area, the percentage of exposed rock varied from less than 5% in Marie Byrd Land to roughly 20% at the Darwin Glacier and almost 30% in South Victoria Land. Since operations began in 1976 it is estimated that the radiation detectors measured the radiation from a strip approximately 9000 km long and roughly 50 m wide. Since parts of the flight path are over ice, or had to be conducted at altitudes that do not provide significant data, only about half the actual flight path provides useful radiometric information. In view of these facts, only about 225 km<sup>2</sup> have actually been subjected to direct radiation measurement. This is less than 0.1% of the total exposed land area of the antarctic continent. For this reason it is premature to attempt any kind of comprehensive estimate of the uranium resource potential of Antarctica. The discussion which follows is intended to indicate the nature of the activities, the general geology of the survey areas, and a summary of the results obtained to date.

## AIRBORNE SURVEY FLIGHT OPERATIONS

To provide an understanding of survey activities in the special circumstances of the antarctic environment, it is appropriate to include a discussion of logistics and the particular problems which are encountered. The Antarctic International Radiometric Survey (AIRS Project) is one of a number of other scientific activities that are conducted from camps on the continent and no individual aircraft are devoted exclusively to the radiometric survey. Furthermore, the aircraft can not be hangared and must remain outside at all times. This means that all radiation detection apparatus must be mounted in the aircraft before each flight and removed after the end of the flight. Since the detector crystals are thermally regulated, they must be kept on electrical power at all times. Special insulation is provided to shield the detectors from thermal shock. When not in flight operation, the spectrometer, detectors, and electrical power supply are kept in a maintenance hut to provide some shelter from the low outside temperatures.

### Aircraft

US Navy UH-1/N helicopters, which are identical to the civilian model 212 Bell, were used for all the helicopter survey operations. This aircraft is a twin-engine turbine-type with each engine capable of delivering 900 shaft horsepower. The average fuel consumption at cruising speed is approximately 600 lb (273 kg)/h. The standard fuel capacity of 1400 lb (636 kg) can be augmented by the use of an auxiliary fuel bladder with a capacity of 900 lb (409 kg). Total flight time can thereby be extended to nearly four hours. The US Navy recommends



1600 lb (727 kg) as net useful load and this limitation applies to all field operations of this helicopter in Antarctica. Maximum service ceiling is 15 000 ft (4600 m) above mean sea level (MSL) and maximum airspeed is 130 knots (245 km/h) at sea level. As with all types of aircraft, these limits may be reduced or changed as a result of weather conditions or at the pilot's discretion. The excellent safety record in Antarctica clearly reflects aircraft reliability, high quality of pilots and good maintenance. Twin-engine characteristics undoubtedly add significantly to this record.

### **Counter location in the aircraft**

The detector crystals were located in the centre of the cabin directly behind the avionics console. The aircraft structure at this point provides the least attenuation for gamma radiation. This location is forward of the main fuel tanks and the transmission and only two thin sheets of aluminium alloy separate the detector package from the outer surface of the aircraft.

### **Flight operation in high relief areas**

Owing to the high relief, which is characteristic of the terrain in which the surveys are conducted, high-performance helicopters are clearly the most practical vehicles for making radiation surveys. Under favourable conditions, helicopters can be operated at forward speeds of 50 to 100 km/h and at ground clearances of 15 to 30 m. In calm air, cliff faces and steep slopes can be approached without difficulty. In particular, however, the ability of helicopters to land in rough terrain makes them especially well suited to this type of survey.

### **Modification to standard operations required in high relief areas**

The Navy UH-1/N is a high-performance helicopter with good stability characteristics; yet at the high altitudes and steep relief in the study areas the pilots are pressed close to the safety limitations much of the time. Because of these factors, standard flight operations had to be modified. The optimum ground clearance of 100 m and constant airspeed cannot be maintained. Radar altimeters are of use only in a few areas of low slope, and height measurement by visual estimate is most satisfactory in the very high relief areas. Experimentation proved that a ground clearance of approximately 15 m could be held with maximum accuracy and ease, and this was maintained unless turbulence warranted a greater clearance for reasons of safety.

### Survey equipment and methods

The airborne gamma-ray spectrometer currently in use for these surveys consists of a GAX 512 NaI (Tl) detector with a total volume of 8390 cm<sup>3</sup>. Thermal stabilization is provided by a GR-900 interface which also furnishes the high-voltage power to the photomultipliers. Detector output is supplied to a GR-800A gamma-ray spectrometer and six channels of the spectrometer output are fed to a GAR-6 six-channel analog recorder. All electronic components were manufactured by GeoMetrics Inc. This equipment has proved to be entirely satisfactory for helicopter flight operations and it has demonstrated its ability to withstand the rigours of the antarctic environment.

Because of the special conditions which are encountered in making radiometric surveys in Antarctica, a system of airborne crew operation had to be developed that would permit the spectrometer output to be related accurately to the geology of the terrain over which the helicopter was flying. All crews consisted of a minimum of three persons. The spectrometer operator was seated on the left side of the helicopter at the window of the left door. A geologist was also seated at the right door window and a crew member responsible for aircraft position recovery was placed with a view out of the front windows of the helicopter. Survey and flight crew members were equipped with head-sets plugged into the aircraft intercom system.

When in flight, the geologist calls out the geological or geographic feature over which the aircraft is flying and provides a brief description. The spectrometer operator places a fiducial mark on the recorder chart and assigns a consecutive number to each mark. He also makes a short note of the description provided by the geologist. The crew member in charge of position recovery plots the position of the numbered fiducial mark on his map of the terrain so that a continuous flight path is generated.

If vertical cliffs are being scanned an attempt is made to orient the aircraft so that the cliff face is on the right side of the aircraft. When this is not possible, the spectrometer must perform both his normal task and the function of the geologist. When anomalies are encountered, the spectrometer operator may request turns or re-runs over the flight path. If the anomaly is sufficiently strong he may request landing for a ground survey. Substantial anomalies that are detected during flight are always checked on the ground with portable ground survey counters.

## GEOLOGY OF THE SURVEY AREAS

### South Victoria Land and the Darwin Glacier

The survey areas in South Victoria Land and surrounding the Darwin Glacier are essentially contiguous and form a strip approximately 500 km long in the central

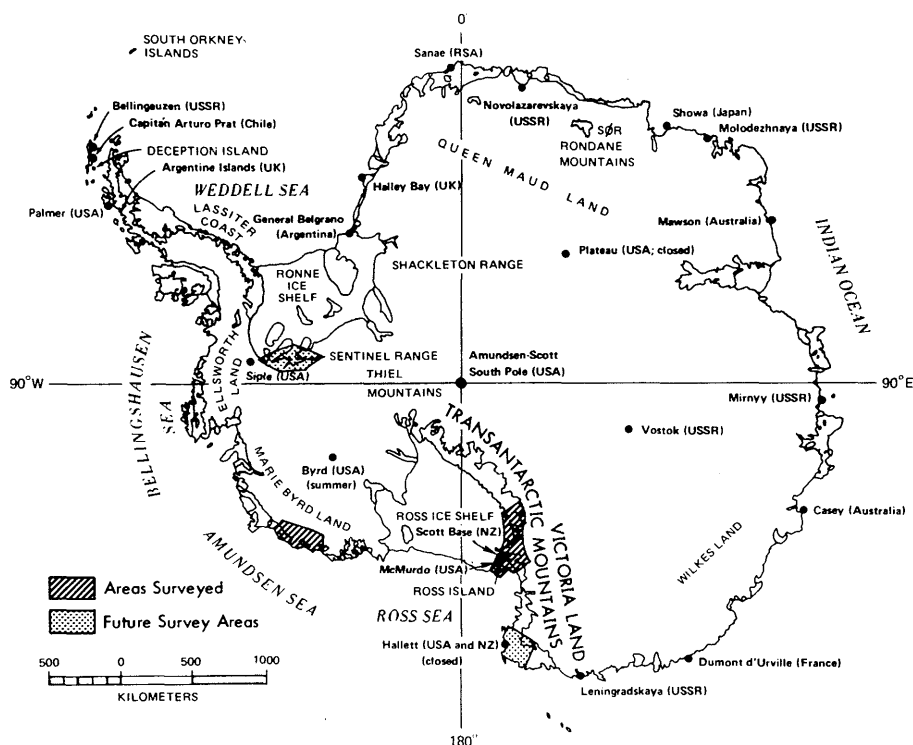


FIG.1. Index map showing areas of Antarctica for which airborne gamma-ray surveys have been completed and areas that will be surveyed in the next three years.

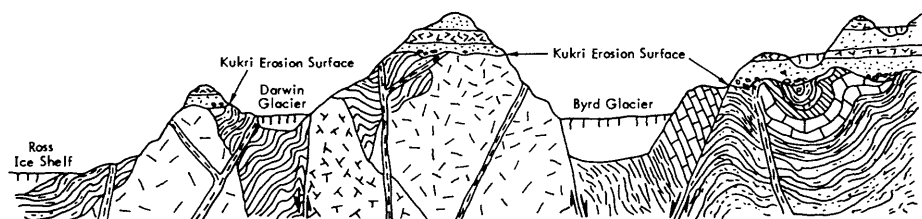


FIG.2. Cross-section showing typical geological structure of the Transantarctic Mountains in the Darwin Glacier area. This section is oriented diagonally across the Transantarctic Mountains from north-east to south-west.

Transantarctic Mountains (Figs 1 and 2). In both areas the oldest rocks outcrop along the coast and they consist of a metamorphic complex of gneisses, schists, phillites, and marbles. These rocks have been named the Ross Supergroup Metasediments and they range in age from late Proterozoic to upper Cambrian. *Archaeocyathus* has been found in some of the marbles and limestones in the southern part of the region. The total thickness of the Ross Supergroup Metasediments is thought to be about 9000 m and they are generally considered to be geosynclinal deposits. However, massive limestones and dolomites outcrop in the mountains just south of the Byrd Glacier. The presence of these sediments suggests that some relatively stable sedimentary basis were developed in this part of the region.

Deposition of the original sediments was followed by the Ross Orogeny. During this period of tectonic activity a number of masses of granite, granodiorite and diorite were intruded and widespread folding and regional metamorphism occurred. The igneous bodies were designated as the Granite Harbor Intrusives by Gunn and Warren [2], and they are clearly closely associated with the metamorphics. A post-tectonic phase with true intrusives can be readily separated from the syntectonic phase that consists mainly of gneisses. The Larsen Granodiorite is syntectonic and has been classified as an intrusive largely because it forms a batholithic body nearly 400 km in length parallel to the coast of South Victoria Land. Actually it commonly shows gneissic and foliated texture and is probably the product of high-grade metamorphism and partial anatexis. The smaller granitic bodies are post-tectonic. They are undeformed and discordant and are sometimes intruded as sills. The Hope Granite in the south and the Irizar Granite in the north are examples of these types of intrusive. They are substantially higher in radioactivity than the Larsen Granodiorite and its equivalents. Large pegmatite zones occur in several parts of the survey area and these frequently show elevated radioactivity levels but no significant concentrations have been discovered [3]. The average geological ages measured on these metamorphic and igneous rocks range from 525 to about 470 million years.

The Ross orogeny and the post-tectonic intrusive phase were followed by a period of uplift and erosion during which the Kukri Erosion Surface was developed. This surface truncates all the underlying structures and intrusive bodies and obviously constitutes the most extensive sedimentary hiatus in the Palaeozoic sequence. The Kukri Erosion Surface is of interest because of the discovery of local basins containing substantial concentrations of thorium minerals, rare earths, and tin in the lowest beds of the Beacon Supergroup which lie directly on top of the surface.

The Beacon Supergroup [4] can be separated into two major divisions, the Taylor Group of Devonian age and the Victoria Group which is Permo-Triassic. So far, radioactive minerals have been found only in the base of the Taylor Group. Carbonaceous zones and channel sandstones are common in the Victoria Group and they should provide ideal geochemical conditions for the accumulation of uranium deposits but to date none have been detected. The Beacon Supergroup is overlain in some areas by the Jurassic Kirkpatrick Basalts and is extensively

intruded by the Ferrar Dolorites also of Jurassic age. In the southern part of the survey area in the Transantarctic Mountains a layered intrusive has been found. This body is probably one of the intrusive centres for the Ferrar Dolorites.

The last major magmatic event in this portion of the central Transantarctic Mountains was the eruption of a series of highly alkaline volcanic rocks. The volcanism probably began in late Tertiary and was associated with faulting along the Ross Sea margin. It continues to the present time in the activity of Mt. Erebus. The extrusive rocks include olivine basalt, phonolite, trachyte and kenyte [5].

### **Marie Byrd Land**

The geology of Marie Byrd Land is much less well known than that of the Transantarctic Mountains [6]. Exposures are very limited and the high precipitation rate coupled with the frequent occurrence of severe storms makes flight operations unusually difficult. The dark gneisses and schists that outcrop at Cape Burks on the Hobbs Coast are probably the oldest rocks in the area but the geological age has not been determined. Small outcrops of arenaceous sediments containing plant fragments were found in 1977 in the nunataks on the Ruppert Coast in the western portion of the survey area. These beds are probably age equivalents of the Beacon Supergroup in the Transantarctic Mountains. Felsic intrusive rocks of Cretaceous age occur at a number of localities where pegmatites and aplites associated with these rocks show somewhat elevated radioactivity levels. The youngest rocks in the area are Cenozoic volcanics that make up the Flood and Executive Committee Ranges. Fumarolic activity continues at Mt. Berlin suggesting that these volcanos may have been active in the recent past.

### **Thorium deposits on the Kukri Erosion Surface**

The strongest radiation anomaly that has been detected in three seasons of airborne surveying occurs in sandstones and conglomerates immediately overlying the Kukri Erosion Surface. This surface was most probably formed in late Ordovician or Silurian. In the central Transantarctic Mountains there is clear evidence of intense weathering in the rocks underlying the erosion surface. At Roadend Nunatak on the Darwin Glacier, the surface is developed on the Hope Granite and the feldspars are completely converted to kaolin for at least 10 m downward into the granite. Similar evidence of deep chemical weathering is also observed at Tentacle Ridge, 30 km northwest of Roadend Nunatak. The most intense and extensive anomaly occurs at a site designated AIRS No. 1, which is located on a high ridge about equidistant from the foot of the Bartrum Glacier and the east side of the Touchdown Glacier. Analyses of samples from AIRS No. 1 and Roadend Nunatak are listed in Table I.

TABLE I. ANALYSES BY X-RAY SPECTROMETRY  
*Estimated accuracy  $\pm 10\%$  of the amount present*

Element – ppm	AIRS No. 1	Roadend Nunatak
Th	1200	3000
U	31	18
Zr	200	750
Y	110	230
Ba	550	180
La	110	190
Ce	180	430
P	150	610
Sn	140	190

The actual deposits of the thorium-bearing minerals apparently occur as placer deposits in the basal beds of the Beacon Supergroup. Following the stratigraphy proposed by Plume [7], these deposits are present in the Brown Hills Conglomerate Member of the Windy Gully Sandstone. At Roadend Nunatak, the Windy Gully Formation is overlain by dark shales and siltstones that are undoubtedly the equivalents of the Terra Cotta Siltstone at its type section east of Windy Gully 200 km to the north on the Taylor Glacier [8].

#### SUMMARY OF RESULTS

Although no deposits of uranium minerals have been found in Antarctica that would be of commercial value even if located in populated areas, much has been learned about the detailed geology of those parts of the continent that have been surveyed. Local areas with generally higher than average radioactive mineral content have been found in some of the crystalline basement rocks near the Koettlitz Glacier in South Victoria Land. The variations in uranium content are consistent with what would be expected in similar rocks on other continents. The existence of locally enriched regions suggests that normal geochemical concentration mechanisms have been in operation and that with sufficient time and effort it may be possible to define uraniferous provinces.

The sediments of the Beacon Supergroup show significant concentrations of radioactive minerals in the Darwin Glacier area but they contain mainly thorium

with only minor concentrations of uranium. The thorium deposits appear to be limited to the lowest part of the Beacon Supergroup rocks. The lithological characteristics of the upper part of the Beacon sequence would appear to be especially favourable for the accumulation of uranium. Highly permeable channel sandstones are abundant and carbonaceous fragments are frequently scattered throughout the channel fillings. In Marie Byrd Land the outcrops are very limited and no concentrations were observed. In the Transantarctic Mountains these rocks have been the subject of very careful radiometric surveys but no significant radiation anomalies have been found. The few localized anomalies that have been detected in the upper part of the Beacon Supergroup rocks always appear to be related to thin bentonite beds or to layers of tillite that are irregularly distributed throughout the upper part of the section. None of these occurrences constitute evidence of any secondary enrichment and they appear as anomalies mainly because they stand out against the unusually low radiation background of the associated sandstones. There is little evidence of uranium concentration in any of the numerous coal beds that are present in the upper Beacon rocks. It is true, however, that the Beacon Supergroup rocks are Karroo equivalents and Karroo beds are known to contain uranium deposits in India and South Africa. [9–11].

The Cenozoic and Recent volcanics vary widely in composition, but the trachytes that are present on Ross Island and in parts of South Victoria Land are anomalously radioactive. Unfortunately, the dry permafrost conditions and the extremely low rate of chemical weathering has apparently limited the mechanisms for further concentration. If these rocks could be found in metamorphic contact with favourable host rocks the potential for significant deposits might exist.

### ACKNOWLEDGEMENT

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# AN ESTIMATION OF THE URANIUM POTENTIAL OF FINNISH BEDROCK

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## Abstract

### AN ESTIMATION OF THE URANIUM POTENTIAL OF FINNISH BEDROCK.

Only about 3% of the bedrock in Finland (337 000 km<sup>2</sup>) is exposed; the rest is covered by transported glacial or glaciofluvial drift. About 10% of the drift is covered by lakes and 30% by peat land. The thickness of the drift averages 7 m. The bedrock is composed of Archean and Proterozoic formations, almost all of which were metamorphosed in the Precambrian era. Uranium occurs commonly in refractory minerals but also as uraninite and its secondary minerals in a wide variety of host rocks. The known uranium deposits are associated with quartzites and conglomerates, carbonatites, pegmatites, albitites and shear zones in different rock types. The first stage of uranium exploration consists of reconnaissance survey by airborne radiometrics and/or lake sediment geochemistry. The follow-up stage comprises car-borne scintillometry, terrain geophysics, detail geochemistry and diamond drilling. About US \$700 000 are spent annually on uranium exploration. The four best occurrences might produce a total of 2700 t U at a cost of US \$80–130/kg U. By 1985 Finland will be needing 2130 t natural uranium for its four power plants.

## 1. NUCLEAR POWER PLANTS AND THEIR FUEL DEMAND

At present one Soviet 420 MWe (PWR) reactor and one Swedish 660 MWe (BWR) reactor are operative in Finland, a pair of similar reactors is under construction and will go into production in 1980. Plans are underway for building a third PWR reactor (1000 MWe) which should be operative some time after 1985.

By 1985 these plants will need 480 t UO<sub>2</sub>, which is about 2130 t natural uranium or 2500 t yellow cake. The annual demand would be 64 t UO<sub>2</sub>, 285 t natural uranium or 335 t yellow cake. By the year 2000 all five plants would need about 2200 t UO<sub>2</sub>, 10200 t natural uranium or 12000 t yellow cake. After 1985 the annual demand would be 106 t UO<sub>2</sub>, 500 t natural uranium or 590 t yellow cake.

If this were taken from a deposit of 0.1% U (1000 ppm U), the total demand by 1985 would be 2 130 000 t of ore and the annual output 285 000 t of ore. By 2 000 the total demand would be 10 200 000 t of ore and after 1985 the annual output should be 500 000 t of ore.

## 2. THE BEDROCK, ORES AND DRIFT IN FINLAND

The Finnish bedrock consists of: 1) Archean meta-volcanics and metasediments (over 2 800 m.y. in age), and conformable intrusives of granitic to ultrabasic rocks (> 2 500 m.y.) all in the old craton of the eastern Fennoscandian shield; 2) middle-Precambrian intrusives and metamorphic schists (1 800 ... 2 000 m.y.) divided into strongly migmatized mica gneiss and granodiorite-rich Svecofennian formations and not so strongly metamorphosed phyllite, mica schist and granite rich Karelian formations, see Table I; 3) post-orogenic rapakivi-granite massifs (1 550 ... 1 700 m.y.) and almost unmetamorphosed Jotnian clay- and sandstones with cutting diabases (1 300 m.y.); and 4) small remnants of Paleozoic limestone, sandstone and slate as well as still younger small carbonatite and alkaline rock massifs (450 ... 600 m.y.).

Several banded iron formations and minor uranium and molybdenum occurrences are found in the oldest gneissose bedrock unit occupying about one third of Finland. The main part of the bedrock consists of group 2, that is of schists and intrusive rocks metamorphosed in the Svecokarelian orogeny. Several economic sulphide deposits (about 2 200, 2 000 and 1 900 m.y. old) are associated with these formations; some oxide occurrences (2 000 m.y.) are also found in this group [1, 2]. Most of the known uranium showings have been found in the quartzites and phosphorous dolomites of the Karelian formations.

Only about 5% of Finland is occupied by the younger granites and Jotnian sedimentary formations, and only some fractions of per cent of the bedrock consists of Paleozoic or younger rocks. The content of uranium in P-rich carbonatites is clearly above the average content in rocks but is still far from an economic grade.

Table I. The composition of the Karelidic (eastern and northern Finland) and Svecofennidic (southern and Central Finland) bedrock [3].

	Karelides	Svecofennides
Percent of the total area of schists:		
phyllite, mica schist	45.2	-
mica schist, mica gneiss	-	79.0
quartz-feldspar schist	2.8	6.5
quartzite	26.4	0.3
dolomite	0.3	-
crystalline limestone	-	0.3
metabasalt and amphibolite	25.3	13.0
Percent of the total area of intrusives:		
peridotite and gabbro	3.3	5.9
quartz diorite and granodiorite	23.6	56.5
granite	73.1	37.6

The total area of Finland is 337 000 km<sup>2</sup>, and its median elevation is 150 m. Most of the area lies less than 200 m above sea level; only in the north, in Lapland, are there higher rounded hills, some of which are over 1 000 m in elevation. The peneplanated country was totally covered by Pleistocene glaciations, which not only removed most of the ancient drift and polished the bedrock surface but also pressed the earth's crust down. Today the rate of land upheaval is highest, 1 cm/year, on the west coast.

There is usually a sharp boundary between Quaternary drift and bedrock; only in sheltered depressions are there thicker in situ deposits of weathered bedrock. No more than about 3 % of the bedrock is exposed, and the mean thickness of the drift is about 7 meters. Over 90 % of the drift is glacial till 2 to 5 meters in thickness and locally crossed by sand and gravel eskers up to 100 m in thickness. In the southwestern parts of the country the till is covered by late- or postglacial silt and clay up to 50 m in thickness, and in the central parts by thinner beds of silt. About 10 % of the mineral drift is covered by lakes and about 30 % by peatlands.

Table II. Uranium prospecting at the Geological Survey of Finland

	Areal stream and lake sediment geochemistry km <sup>2</sup>	Aerial radiometric surveys altitude 30 ... 50 m km <sup>2</sup>	Diamond drilling of U prospects m
1972	-	4 320	-
1973	-	4 010	1 286
1974	1 500	8 020	1 529
1975	7 700	8 430	833
1976	14 400	7 640	1 685
1977	10 000	8 160	659
1978	20 700	10 300	1 225

### 3. URANIUM EXPLORATION

#### 3.1. History

Systematic prospecting for uranium was put in motion in the early 1950's in the southern and eastern parts of the country by the Geological Survey, Imatran Voima Power Co. and Atomiennergia, a private company. Initially Geiger counters, boulder tracing and car-borne scintillometers were used as were also geochemical methods such as U-determination of till and radon measurement of soil air [4]. These activities led to the discovery of some small deposits, e.g. the uraninite-bearing dykes in Askola, south Finland, and the uraniferous quartzite zone of Koli, east Finland. In the latter region the deposit of Paukkajanvaara was mined by Atomiennergia Co. from 1960 to 1961. Later Outokumpu Co. (base metals) and Rautaruukki Co. (steel), two state-owned mining companies, also began prospecting activities. Uranium exploration, which from 1950 to 1975 was carried out by several private and state-owned organizations, is now concentrated almost solely at the Geological Survey of Finland (Table II).

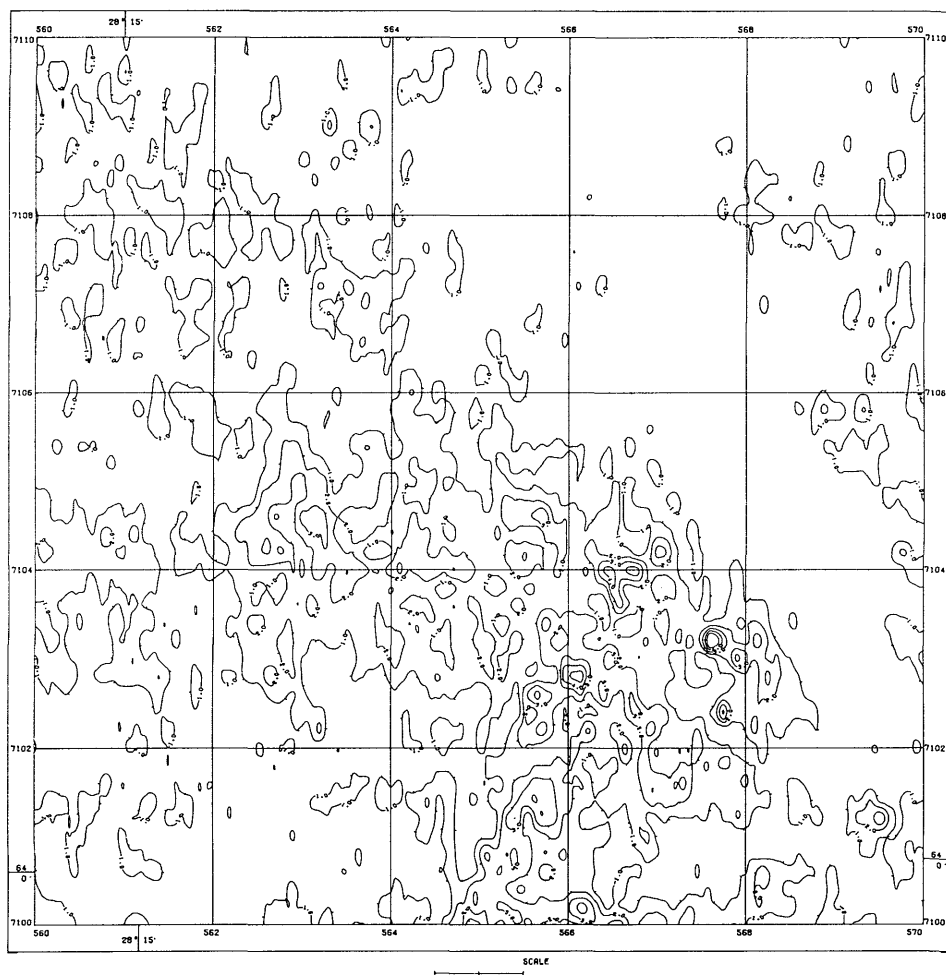
#### 3.2. Regional prospecting

In Finland the general public have traditionally been encouraged to go in for prospecting and to send samples of

"ore" for investigation. To GSF 15 000 samples are received annually; of these, 10-15 % contain ore minerals and about 0.2 % lead to more extensive field studies. The Geological Survey has helped enthusiastic private prospectors by lending them geiger-counters; thus, about 1 % of the samples obtained are radioactive. The state-owned and private mining companies engage in similar activities.

Until 1972 air-borne radiometrics was carried out from an altitude of 150 m with a line spacing of 400 m by a small scintillometer that measured total  $\gamma$ -radiation. Since 1972, these have been replaced by low altitude (30-50 m, 200 m line spacing) air-borne geophysical measurements employing a more sophisticated radiometric unit (4 Scintiflex NaJ-crystals, total volume  $27.3 \text{ dm}^3$ ; Nokia spectrometer, 54 channels) that measures U-,  $\text{Th}$ -, K- and total  $\gamma$ -radiation intensities. About 50 000  $\text{km}^2$  or 15 % of the land area have already been covered by these flights. The results are corrected for altitude variations, for Compton scattering and background radiation. Maps of U, Th, K and total radiation intensity as well as radioelement ratio maps are then automatically drawn (Fig. 1). The uranium maps and particularly multielement maps are useful in exploration. Because of the marked effect on the results from the thickness and the moisture content of soil and the number of rock exposures per unit area the results, anomalies, must be checked by measurements in the field. The granitic exposures are shown by total and potassium  $\gamma$ -counts but, because they usually contain refractory minerals, by uranium radiation as well. The uraninite bearing exposures do give strong anomalies e.g. Askola red granite, U-channel 283 c/s (background 50 c/s) and Kesänski quartzite, U-channel 70 c/s (background 10 c/s). Uranium rich erratics do not usually give anomalous intensities, but springs with radioactive water do [5].

In 1973 reconnaissance uranium geochemical mapping with the aid of uranium analysis of organic lake sediments and - in lakeless areas - organic stream sediments got underway [6]. The mean sample density is  $0.2/\text{km}^2$  in the lake sediment study and  $0.5/\text{km}^2$  in the stream sediment study. The lake sediment grid now covers roughly 30 000  $\text{km}^2$  and the stream sediment grid about 26 000  $\text{km}^2$ , together about 17 % of the area of Finland. Lake sediment sampling



*FIG.1. Airborne radiometric map, displaying equivalent concentration of uranium. Contour interval 1 ppm eU, flight altitude 30 m. Empty spaces are due to peat bogs and a lake.*

(through the ice) is done in winter and stream sediment sampling in summer. The samples are dried, ashed and analysed for U by delayed neutron counting in the Reactor laboratory of the State Technical Research Center with their own new automatic equipment. Anomalies found are checked by sampling other materials like humus or glacial till and analysing them by the same method. The very mobile

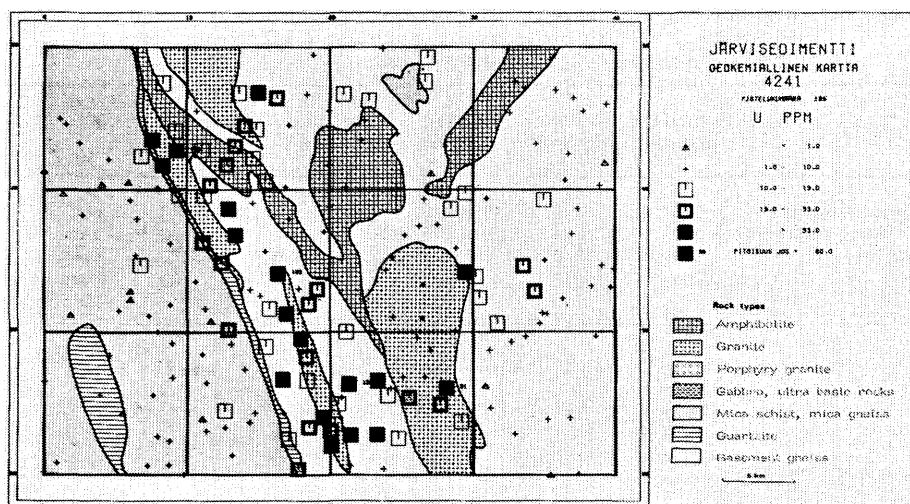


FIG.2. Uranium content in lake sediments, ppm in ash. The bedrock west of the narrow diagonal quartzite horizon is of Karelian schists (1800 m.y. in age), the eastern part of the area is part of the Pre-Karelian basement complex.

uranium goes easily to ground water solution and forms hydro-morphic anomalies in till, which is why the source for such an anomaly in valley or hill side originates from horizons situated above [7, 8].

The trace metal contents of lake sediments depend upon the size of the lake and the contents of manganese and organic material. A more representative picture of the distribution of an element may therefore be obtained if results of only one size class of lakes, e.g.  $\phi$  0.5 to 1 km, are given on the map, and the absolute concentrations are converted to relative values by dividing them with the L.O.I.<sup>1</sup> or Mn- contents of the same samples. Correction with manganese content or with L.O.I. have not proved necessary for uranium. Figure 2 shows an example of uranium geochemical maps. Table III gives a summary of the results from some 1200 km<sup>2</sup> map sheet areas, showing that there are conspicuous variations in the uranium contents between different areas and different materials.

<sup>1</sup> L.O.I. = Loss of ignition.

Table III. Mean uranium contents of different sampling materials, variation of medians, anomaly threshold ( $\Sigma$  %95) and maximum contents in different map sheet areas (1200 km<sup>2</sup> each)

Material	Number of samples	Mean content ppm	Median contents ppm	Variation of		Obs.
				Threshold contents ppm	Maximum contents ppm	
Lake sediment	5 003	3.8	2.1-7.2	7.9- 55.1	24-5020	ash
Stream sediment	4 098	3.3	3.1-7.7	56.0-122.5	857- 953	ash
Humus	5 796	2.6	1.8-3.1	12.7- 36.9	45-1270	ash
Till	5 265	3.7	3.0-4.5	6.1- 13.6	19- 100	-0.06 mm fraction

### 3.3. Local exploration

All uranium indications (aeroradiometric, geochemical (Fig. 3), geological and erratics) are investigated by a special group from the Exploration Department of the Geological Survey. In 1977 this group made follow-up studies on 28 locations (Fig. 4) by different methods of ground geophysics, boulder tracing, radon measurements and other geochemical investigations as well as diamond drilling. The targets could be divided according to the indication, which in 9 cases was a radiating rock exposure found by geologists or the public, in 7 cases a U-bearing boulder, in 4 cases an aeroradiometric anomaly, in 3 cases a geochemical anomaly, and the rest follow-up investigations on known occurrences.

In 1978 field observations were performed over an area of 14 000 km<sup>2</sup> as a result of aeroradioactive indications and over an area of 1 500 km<sup>2</sup> as a result of geochemical anomalies. Most of the aeroradiometric anomalies were caused by granitic rocks containing somewhat more U and Th than in average. In Kankaanpää, southwestern Finland, and in Sukeva, eastern Finland, high uranium contents occurred as nests in pegmatite. In Sotkamo, eastern Finland, a sathrolitic horizon of the older basement showed conspicuous amounts of uranium. The investigations are continuing.



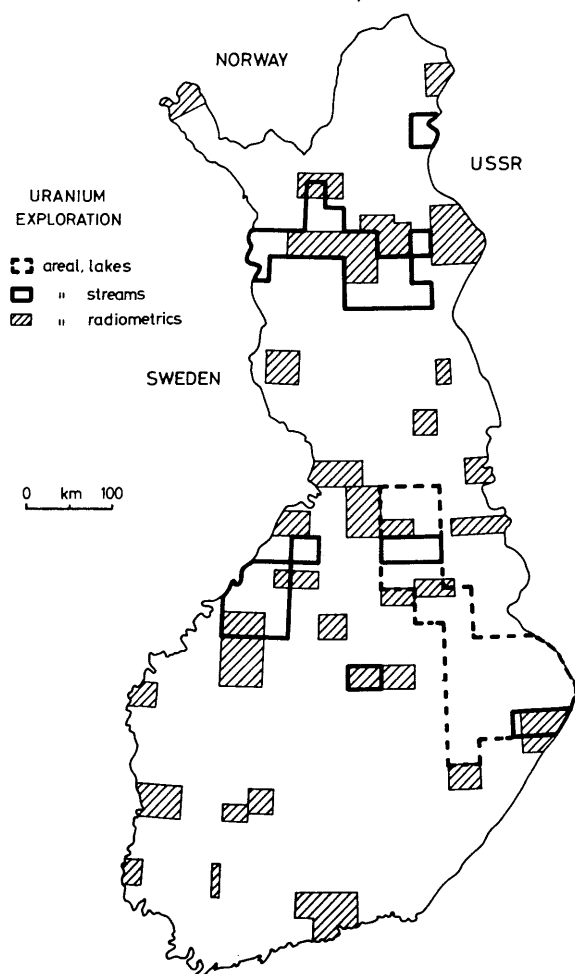


FIG.3. Areal reconnaissance surveys for uranium: lake and stream sediment geochemistry; low-altitude airborne radiometrics; from 1972 on.

Some older findings have been restudied and, for example, in Askola, southern Finland, a hematitized uraniumiferous shear zone cutting granitic formations was located by VLF-sounding and radon measurements. The deposit is soon to be subjected to diamond drilling.

Diamond drilling (1124 m) at Revonkylä, eastern Finland, showed that the mineralisation, located in the silicified parts of mica gneiss in the Pre-Karelian basement,

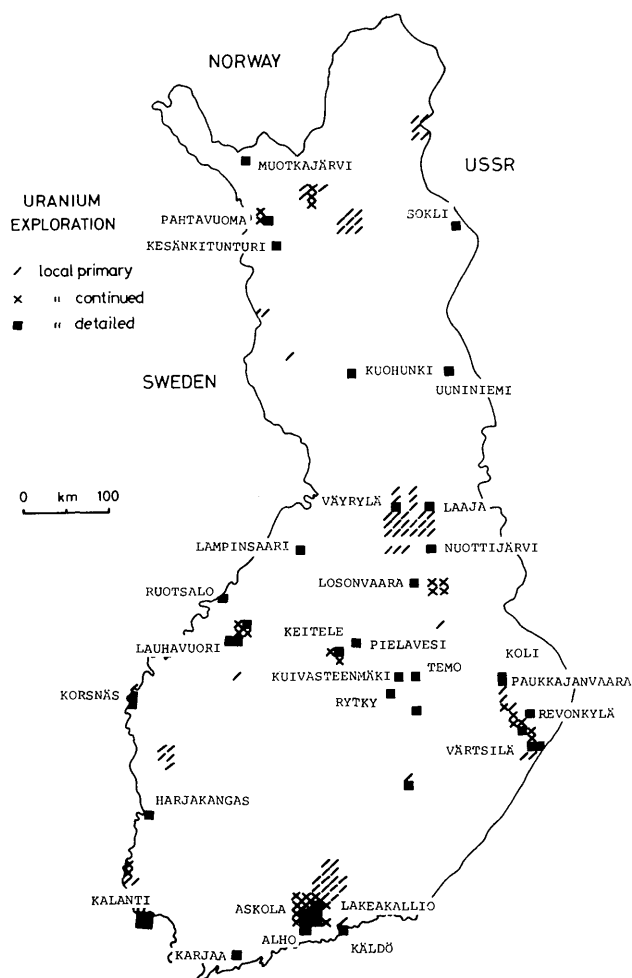


FIG.4. Known uranium occurrences and fields of follow-up uranium exploration by the Geological Survey from 1972 on.

consisted of horizons 2 to 4 m thick, with an uranium content of only 300–400 ppm (0.03–0.04 %). Some rather small new showings were found by geochemical assay and radon measurement in the environment of the Revonkylä deposit. Follow-up studies are being carried out by car-borne  $\gamma$ -radiation counting over an area of 1700 km<sup>2</sup>.

#### 4. URANIUM OCCURRENCES

The known uranium occurrences of Finland [9,10] can be classified as follows:

1) disseminated uraninite or secondary U-minerals in Jatulian (lower Karelian) epicontinental quartzites and quartz-pebble conglomerates just above the Archaean basement; for example, the deposits along a 20 km long belt in the Koli region, eastern Finland, (including the exhausted Paukkajänvaara deposit 30 t  $U_3O_8$  from an ore of 0.12 % U), Ruunanmäki (184 t; 0.14 %), Ipatti (69 t; 0.08 %) Martinmonttu (23 t; 0.10 %) and more to the south the small occurrence of Värtsilä; as well as Kesänkitunturi, Kolari, Lapland, consisting of 2 lenses 100–150 m long 15–20 m thick in quartzite (1120 t; 0.06 %).

2) uraninite in phosphorus rich Marine Jatulian dolomite and skarn horizons overlying Jatulian quartzite and below Kalevian geosynclinal (graphitic) mica schists; for example, Nuottijärvi, Paltamo 1100 t  $U_3O_8$  (0.04 % U, 3.5 %  $P_2O_5$ ) and Losonvaara, Sotkamo 0.5–3 m thick beds with 0.04 % U and 4 %  $P_2O_5$  northern Finland; Lampinsaari, Vihanti above Zn ore body a horizon with 0.03 % U and 3–5 %  $P_2O_5$  and Harjukangas, Noormarkku 10 m thick horizon 0.02 % U, Ruotsalo, Kälviä as well as Korsnäs, all from western Finland; the 4 km long low grade U-P-deposit of Temo, Muuruvesi and the small occurrences in skarn of Pielavesi and Rytty, Kuopio as well as that of Kuivasteenmäki, Siilinjärvi, in dolomite all in eastern Finland.

3) in Sokli alkaline intrusion (450 m.y. in age) Savukoski, Lapland a U-Ta-pyrochlore mineralization in carbonate is estimated to contain several millions of tons of low grade ore, 0.01–0.02 % U.

4) impregnation of uraninite in spilitic dykes (metabasite, albitite or albite diabase) or in rocks adjacent to Karelian schists; for example, Pahtavuoma, Kittilä, Lapland, uraninite in albitite associated with 2 Mo-bearing shear zones cutting black schists and greenstones of Lapponium formations, 0.3–0.5 % U together with 700 t  $U_3O_8$ ; the occurrence of Uuniniemi, Kuusamo 0.1 % U and 1.5 % Th in northern

Finland; uraninite and secondary U-minerals bearing albite diabase in the Mårtenson ore body, part of the exhausted Paukkajanvaara deposit,

5) rich patches in hematitized or silicified shear and breccia zones of Archean gneisses and granites; for example Revonkylä 2-4 m thick belts with 0.03-0.04 % U in Kiihtelysvaara, eastern Finland; the small occurrences in migmatite at Alho, Askola, southern Finland and Kuohunki, Rovaniemi, Lapland, also belong to this category,

6) small scattered patches of U and Th in granitic neosome of migmatites the paleosome being amphibolite and kizigite. These occur throughout Finland but mainly in the south; for example, Kälkö, Pernaja, a 100-200 m wide belt in which spotty concentrations of U vary from 0.5% to 10%,

7) pegmatitic dykes such as Muotkajärvi, Enontekiö, Lapland or pegmatite granite Laaja, Suomussalmi and Väyrylä, Puolanka northern Finland, but especially those close to the contact of rapakivi granite, e.g. Lakeakallio, Askola, southern Finland.

The four best deposits may produce about 2 700 t U at a cost of less than \$ 130/kg U.

## 5. THE FUTURE

The goal of uranium exploration has been to find in Finland enough fuel for the nuclear power plants in operation and planned. The likelihood of finding uranium deposits ought to be rather good considering the age and type of bedrock, since large uranium deposits are known in the same type of rock formations in other parts of the world. Despite all our efforts, however, no occurrences rich and large enough to be economically mined have yet been found.

Uranium exploration will be continued and intensified by designating priority areas critical for uranium and adapting all known reconnaissance and detail prospecting methods [ 11, 12 ].

Air borne radiometric mapping will be continued by low altitude flights with modernized equipment. The 41-year-old DC-3 will be replaced by a de Havilland Twin Otter with a new improved instrumentation on board. An increase of the annual coverage of measurements by about 20% has been planned.

Geochemical reconnaissance studies on lake and stream sediments will be continued at the same rate of coverage as geophysics, that is 10 000 to 12 000 km<sup>2</sup> annually. Also hydrogeochemical methods will be tested. The statistical treatment of multi-element analytical data will be developed to give more information on element combinations and thus to augment knowledge of metallogenetic units in bedrock.

In 1979 the uranium group of the Exploration Department which consists of 4 geologists, 1 geophysicist, 1 technician, 10 assistants and 5 to 6 undergraduate summer assistants, will spend \$ 175 000 on salaries and \$ 150 000 on drilling and other field investigations. According to plans, the total sum would be doubled during the next five years; nearly as much will be spent on uranium exploration by other institutions in Finland.

The uranium group selects its targets with the aid of information provided by the forementioned reconnaissance surveys, by geological mapping and analogies and by boulder tracing performed by geologists and the public. All the classical prospecting methods are in use: car-borne scintillometry, boulder tracing, geochemical and geophysical measurements, pneumatic and diamond drilling. Newer methods will be tried out, developed and, if possible, invented. Possibilities also exist for future international cooperation in the form of the IUREP project of the IAEA/NEA.

## ACKNOWLEDGEMENTS

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## DISCUSSION

A. HATTON: I noticed that you did not mention in your oral presentation that any analysis of water for uranium or radium had been performed. Was this because it was considered that such an analysis would not give useful results in Finland?

L. K. KAURANNE: We have in fact analysed groundwater from soil and rock wells. The background values were 1–3 ppb U and 5–7 nCi/l Rn, the anomalies being 30–2500 ppb U and 10–50 nCi/l Rn respectively. The maximum contents found in groundwater in Finland are 1500 ppb U and 1200 nCi/l Rn. A five-year programme on groundwater analysis covering the whole country was begun in 1978.

A. E. BELLUCO(*Chairman*): Of the general prospecting techniques used (airborne radiometry and geochemistry), which has given the best results in Finland in view of the large areas to be surveyed?

L. K. KAURANNE: Both airborne radiometry and geochemistry (lake and stream sediments and groundwater) are used for reconnaissance in uranium exploration, and we find anomalies with both methods. The geophysical anomalies are usually easier to study and to interpret in the follow-up stage of exploration. But both methods have their drawbacks. Water or a water-saturated peat layer only half a metre thick blocks the gamma radiation, which explains why vast areas do not give any response, and some of the anomalies found radiometrically are geochemical (hydromorphic). U, Rn and other elements have been transported in water solution, and so the difficulties of interpretation with this method are the same as in geochemistry. In some areas better results are obtained with airborne radiometry, in others with geochemistry. It is impossible to say which is better overall. Geophysics may be somewhat cheaper when large areas are studied, depending of course on the geochemical sampling density. In any case, we in Finland use both methods together.





# THE URANIUM POTENTIAL OF GREENLAND – A GEOLOGICAL ANALYSIS OF FAVOURABILITY

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## Abstract

### THE URANIUM POTENTIAL OF GREENLAND – A GEOLOGICAL ANALYSIS OF FAVOURABILITY.

The potential for uranium deposits in Greenland is estimated on the basis of the following criteria: (1) the land areas, not covered by inland ice are 340 000 km<sup>2</sup>, and geologically extremely well exposed; (2) the geological history ranges from the early Archaean to the Quaternary, and a wide range of geological environments and lithologies are present; (3) the level of geological information is generally of a high quality, but the information is heterogeneous; (4) uranium exploration up to the present is mainly of a reconnaissance nature and it has only covered a part of Greenland; (5) only one major uranium deposit is at present known. It is concluded that the area with the highest potential is found in south Greenland and encompasses the early to middle Proterozoic mobile belt and the middle Proterozoic suite of Gardar alkaline igneous plutons. The high favourability of these units is supported by data from recent exploration work in the area as well as by radiometric data of an earlier date. In particular Ketilidian metasedimentary units, and the Ketilidian and post-Ketilidian country rocks of some of the Gardar complexes appear favourable. In view of the limited exploration work carried out, the early Proterozoic mobile belt in north central Greenland as well as the Proterozoic crystalline basement and the overlying suprastrata of the Thule Group in north-west Greenland are classified as being of medium favourability. Further exploration is required within these areas.

## 1. INTRODUCTION

A review of the geology of Greenland forms the basis for the definition of the geological associations favourable for uranium mineralisation. The present stage of geological knowledge and the present results of uranium exploration is evaluated, and when possible compared with characteristic geological criteria of major types of uranium deposits [1, 2 & 3].

International organisations such as OECD/IAEA as well as several larger geological surveys are using considerable effort to predict undiscovered uranium resources in order to evaluate the world uranium potential. This work is based on certain geological assumptions and uses either simple geological evaluation or

statistical methods of probability. The undiscovered resources are categorised in a number of ways, each reflecting the confidence in their existence. It is hoped that the presentation below will contribute to this evaluation work. The present analysis will, however, focus on the recognition of areas generally favourable for uranium mineralisation rather than on the establishment of conceptual models. The paper applies the terminology for resource classification as used by the NEA/IAEA.

## 2. PRESENT STAGE OF EXPLORATION

Greenland is the largest island in the world with a size of 2 175 000 km<sup>2</sup>. The central part is covered by an extensive ice sheet and only 340 000 km<sup>2</sup> along the margins of the island are available for exploration in the classical sense. The climate is sub-arctic and arctic, the vegetation is sparse and generally the geological exposure is extremely good.

Politically Greenland was administered by Denmark up to May 1979, when "home rule", governed by a local council, was introduced. However, Greenland is part of the Danish Kingdom and under the Danish constitution. It is together with Denmark a member of the EEC.

Traditionally, exploration for uranium in Greenland has been carried out by the Danish state, whereas private mining companies undertake the major part of the exploration for other minerals.

Prospecting for uranium in South Greenland commenced in 1955 within the magmatic complex of Ilímaussaq. Within this complex a low grade deposit, the Kvanefjeld Deposit, was discovered. On the basis of geological and metallurgical research during the following years, this project has now reached a detailed evaluation stage. Environmental, political and legal problems are yet to be solved. The deposit contains 27 000 tons U classified as reasonably assured resources and 16 000 tons U classified as estimated additional resources. The average grade is 340 ppm U. Undiscovered resources with a grade of more than 150 ppm U are estimated to 600 000 tons U.

Regional exploration for uranium in Greenland has been performed since 1971 [4, 5 & 6]. At present approximately 190 000 km<sup>2</sup> has been explored by aerial radiometry supplemented by drainage sampling and ground gamma-ray scintillometry and spectrometry.

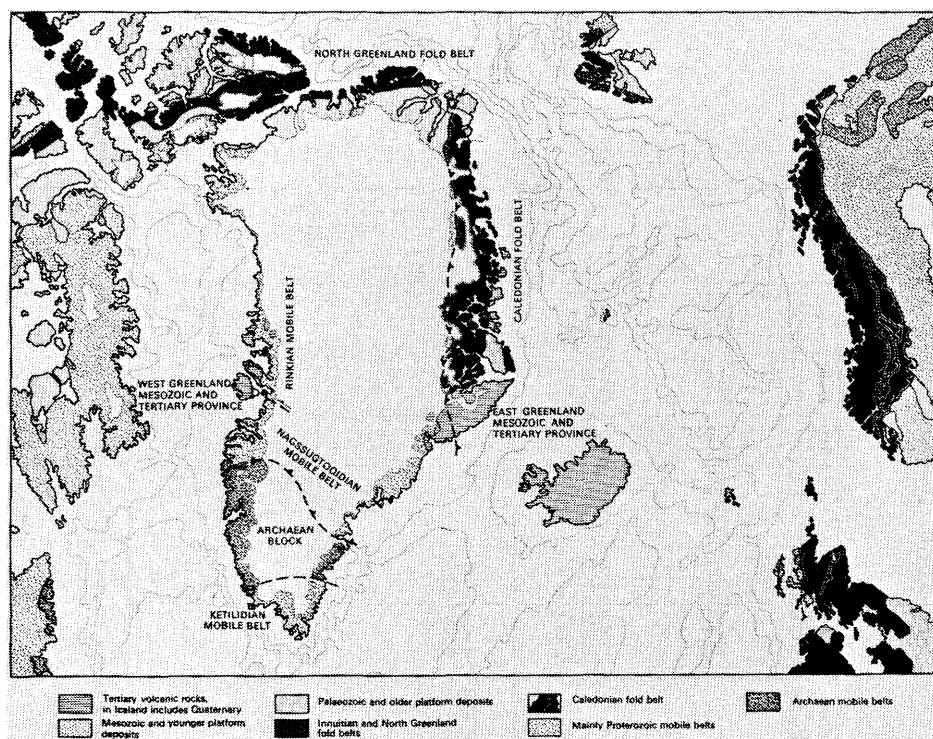


FIG.1. The structural divisions in Greenland in relation to the neighbouring land areas [7].

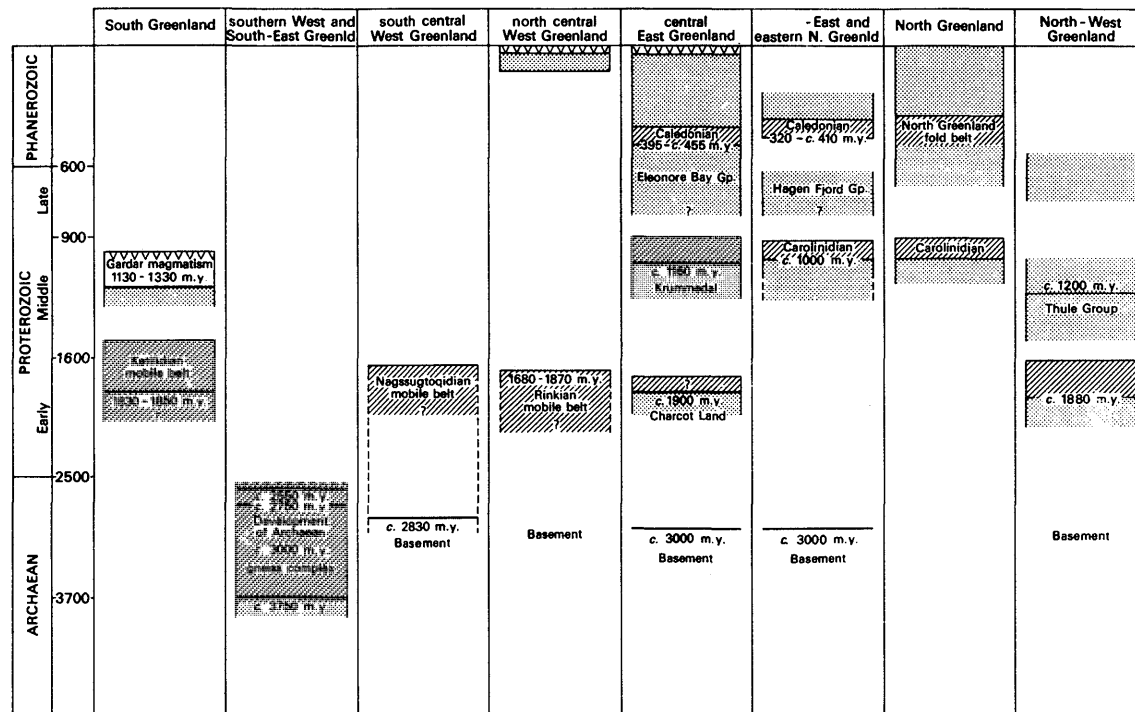
### 3. SUMMARY OF THE GEOLOGY OF GREENLAND

The major geological units in Greenland can be matched with similar units in either North America or northern Europe (Fig. 1). The structural divisions comprise, 1) a Precambrian shield in West, South, and South-East Greenland, 2) two Palaeozoic fold belts and younger sedimentary platform areas in North and North-East Greenland, and 3) areas of Cretaceous and Tertiary sediments, and magmatic rocks in central West and central East Greenland.

The general geology and its evolution is outlined in Table I, and the following description is mainly based on Escher and Watt [7].

1) The Precambrian shield consists of an old Archaean block bordered to the north and south by the early Proterozoic Nagssugtoqidian and Ketilidian mobile

TABLE I. SCHEMATIC GEOCHRONOLOGICAL REPRESENTATION OF THE MAJOR GEOLOGICAL EVENTS AFFECTING GREENLAND [7].



Divisions in the Precambrian are used informally

— End of thermal activity (K-Ar and Rb-Sr mineral ages)

— Major magmatic/metamorphic event (Rb-Sr whole rocks and U-Pb zircon ages)

Deposition

Orogeny

Cratogenic igneous activity

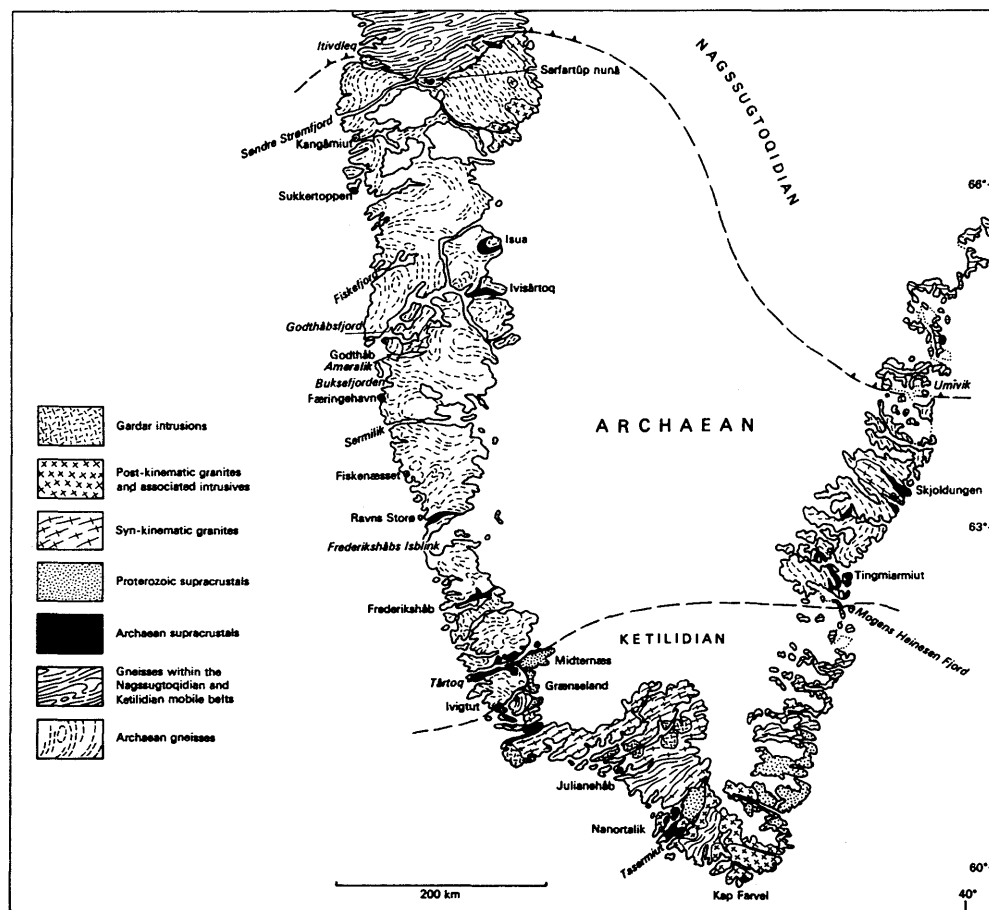


FIG.2. Simplified map of the Archaean and Ketilidian complexes of Greenland [7].

belts (Fig. 2). Another mobile belt, the Rinkian is found to the north of the Nagssugtoqidian, against which it forms a strong structural contrast. In middle Proterozoic time cratogenic conditions prevailed and supracrustal rocks were deposited in South Greenland and in parts of North and East Greenland. In South Greenland the deposition of sediments and lavas was accompanied by the intrusion of dykes and alkaline plutons, one of which is the Ilímaussaq complex.

2) The East Greenland fold belt is part of the Caledonian orogenic belt which occurs on both sides of the North Atlantic Ocean. In East Greenland the fold belt contains an appreciable amount of Archaean and Proterozoic gneisses and sediments involved variously in the Caledonian folding. The orogenic period was followed by continental sedimentation and magmatic activity during the Devonian and Carboniferous with the upper Permian transgression marking the onset of marine sedimentation along the eastern flank of the fold belt.

The North Greenland fold belt is a continuation of the Inuitian orogeny in arctic Canada [8]. Sedimentation took place during the early Palaeozoic and it was followed by uplift and deformation during the Devonian. In contrast to the Caledonian belt in East Greenland there are no recognisable Precambrian rocks involved in the fold belt in North Greenland, and the metamorphic grade is generally lower. Devonian volcanism and late Phanerozoic tectonic activity accentuated and modified the orogenic structure and tectonic pattern in North and East Greenland.

3) The Tertiary volcanism resulted in the deposition of extensive piles of lava flows with a total thickness of 7000-8000 m in central West and central East Greenland. In West Greenland the lavas were extruded on Cretaceous sediments overlying the eroded Rinkian mobile belt. In East Greenland the volcanism was followed by the emplacement of several igneous complexes with a wide range in composition; the Skaergaard intrusion is one of the complexes.

To complete the onshore geological description of Greenland one must mention the extensive ice sheet which was formed as a result of climatic deterioration during the Quaternary.

#### 4. THE URANIUM POTENTIAL OF GREENLAND

An estimate of the uranium potential of Greenland must be accepted with the realisation that there is considerable information on the geology in one area and almost none in another. The geological framework covers a wide range in lithology, stratigraphy and time, and in the following evaluation each major geological division and their minor units will be looked at separately following the division of Table I.

#### 4.1. Precambrian shield

The central Archaean gneiss block has, as shown by Kalsbeek [9], a low content of uranium; generally less than 0.5 ppm U in composite sand samples. During an aerial radiometric survey in 1975-1976 [10 & 11] areas with radioactive anomalies were located in late and post-tectonic granites within the gneiss. The anomalies were obviously related to allanite bearing pegmatites. Uraninite has been identified in some pegmatites, but in insignificant amounts, and the uranium potential of the Archaean gneiss terrain is estimated as negligible [2].

A higher potential is attributed to two Phanerozoic carbonatite complexes, emplaced into the Archaean terrain.

The carbonatites are in strong radiometric contrast with the surrounding gneiss and mineralisations of uranium, thorium and rare earths are widespread in the carbonate rocks and in the fenites. Narrow zones, 10-100 m long with a concentration of 0.5-1% U, explained by an extremely high concentration of uranium rich pyrochlore, have been reported from the Sarfartôq carbonate complex [12]. The niobium content in these zones varies between 10 and 40%. Uranium concentrations of 400 ppm are found in carbonatites at Palabora in South Africa [2]. Uranium is here recovered as a by-product and the importance of uranium in carbonatites is closely linked with the general mineral wealth of some of these complexes.

To the north of the Archaean block the Lower Proterozoic Nagssugtoqidian mobile belt is composed of granodioritic and enderbitic gneiss. Within the gneiss of granodioritic composition a large number of radiometric anomalies have been found. These are dominated by thorium, primarily controlled by monazite-bearing pegmatites (K. Secher, personal communication). Concentration of uranium in pegmatites or granites is believed to be the only target of some interest as remnants of supracrustal rocks are very scarce within the Nagssugtoqidian mobile belt. It may be classified as belonging to a category of small uranium potential.

The lower Proterozoic Rinkian mobile belt is situated to the north of the Nagssugtoqidian, and in contrast with this, the Rinkian area comprises an extensive cover of metasedimentary and metavolcanic rocks. The supracrustal rock series, which overlies the gneiss unconformably, has a thickness of more than 5 km. The deformation of the Nagssugtoqidian belt was strong

whereas the Rinkian gneisses and supracrustals are characterised by open gneiss domes and nappe structures. No regional uranium exploration has been carried out in the area, but from an analytical programme, carried out by the author on earlier collections of rock and sand samples, clusters of samples with more than 5 ppm uranium were found close to the border zone between gneiss and pelitic schists as well as in the area of the large synorogenic Prøven granite. In view of the inadequate exploration the Rinkian area has a good potential for unconformity related deposits, and a minor interest for disseminated igneous uranium resources.

The Ketilidian mobile belt south of the Archaean block consists of geological units ranging from supracrustal rocks of varying metamorphic grade to migmatites and late orogenic granites (Fig. 2). The geochronology of the Ketilidian mobile belt is outlined in Table II. In the Ivigtut area on the northern side of the mobile belt an unmetamorphosed supracrustal sequence of sediments including quartz pebble conglomerates and volcanic rocks overlies the Archaean gneiss unconformably. Towards the south the gneiss and supracrustal rocks are progressively involved in the Ketilidian metamorphism and the southern part of the area consists of a folded and metamorphosed series of gneiss and migmatized metasediments and metavolcanics. Late tectonic intrusive granites occur in the central and southernmost part of the area.

Proterozoic sediments resting unconformably on the Archaean basement are an obvious favourable target for uranium exploration and some attention was given to this during the geological mapping of the area in the late nineteen fifties. Uranium bearing veins and thorium mineralised shear zones were reported by Berthelsen and Henriksen [13] from the gneiss area. However, no evidence has been found of uranium mineralisation along the border zone with the supracrustals [14]. An appreciable uranium potential still exists in the gneiss area and exploration with more modern techniques is being undertaken. The extremely low metamorphic grade of the sedimentary cover rocks is indicative of limited reworking with mobilisation of uranium being fluids.

In contrast to the Ivigtut area the metasediments, metavolcanics and migmatites of the southern area near Tasermiut fjord are of considerably higher metamorphic grade. They consist of pelitic and semipelitic gneiss overlain by quartzites, arkoses and sediments of acid volcanic origin [15]. The importance of the sediments is stressed, being originally deposited as a red coloured



TABLE II. CHRONOLOGY OF SOUTH GREENLAND [7].

	POST-GARDAR	Dykes (coast-parallel swarms)	
	GARDAR	Dykes, alkaline intrusions, faulting, sandstones and lavas of Eriksfjord Formation	1000-1300 m.y. —
KETILIDIAN		Thin tholeiitic dykes in persistent swarms	
		Closure of K-Ar and Rb-Sr mineral systems	1500
		Late granites: foreland granites, intrusion of central granite zone (reactivation), rapakivi suite and appinitic suite	1770-1810
		Deformation (two or three fold phases, trending NE or ENE; thrusting and faulting in the foreland), metamorphism (greenschist-granulite facies), migmatitisation and formation of early granites	1840
		(Is there a still older major thermal event?)	
PRE-KETILIDIAN		Northern border zone	
		Zones of migmatites (Tasermiut, Lindenows Fjord, Kap Farvel)	
		Qipisarqo Group: sedimentary	
		Sortis Group: volcanic	volcanic
		Vallen Group: sedimentary	sedimentary
			(age relations unknown)
		major unconformity	
		Extensive basic dyking (Iggavik dykes)	ESE NE N-NW ESE
		Folding, thrusting, metamorphism and migmatitisation	2600
		Tartoq Group: volcanic and sedimentary rocks	
		Ilordleq Group (position uncertain)	
		Ikermiit supracrustal series?	
		Older basement (most of the regional gneisses): four units of quartz dioritic and granodioritic gneiss with characteristic lithology and structural position (gabbro-anorthositic rocks, metavolcanic and metasedimentary rocks)	

sandstone in the littoral zone of a geosyncline. The sandstone area displays a steep metamorphic gradient; it has a large aerial extent and a thickness of 150-500 m. Also the presence of graphitic and pyritic horizons in the semipelitic gneiss is noteworthy resembling the metamorphosed Proterozoic Aphebian rocks at the floor of the Athabaska basin in Canada. As a target for uranium exploration the area seems highly favourable, and this has been confirmed by the ongoing field work in the area as many anomalies were found during the exploration. The ore type to be looked for during future exploration will be of high grade vein type or unconformity related deposits.

Extensive areas of late and post-tectonic intrusive granites make up a characteristic feature of the Ketilidian (Fig. 2, Table II). Most of the granite plutons are believed to be reworked pre-Ketilidian basement and a fair potential for uranium from such an environment might be argued with the formation of small deposits of a granitic-anatectic character. Preliminary radiometric data are unfortunately limited and a better judgement of the attractiveness of the area must await results from the present regional exploration programme.

Following the closure of the Ketilidian evolution and the cratogenic period with deposition of sandstones and lavas, a major suite of alkaline igneous complexes was emplaced in what is known as the Gardar igneous province (1330-1140 m.y.) in South Greenland [16], (Fig. 2). The central complexes, which were accompanied by a large number of basaltic dykes, range in composition from granites and quartz syenites to highly undersaturated peralkaline nepheline syenites. Gabbros, syenogabbros and augite syenites occur within both oversaturated and undersaturated complexes. The uranium potential generally attributed to alkaline igneous rocks and late magmatic differentiates has been presented by Sørensen [17 & 18] and Armstrong [19]. Some of the factors contributing to uranium concentration in alkaline igneous rocks are described by Bohse et al. [20], Steenfelt and Bohse [21] and Adler [22]. Sørensen et al. [23] have described the disseminated uranium deposit in peralkaline nepheline syenite of the Ilímaussaq intrusion, and together with the uranium deposit at Rössing in Namibia [24] these occurrences form a well documented type of magmatic syngenetic uranium mineralisation.

Although no uranium occurrences have been located up till now within the seven other major Gardar complexes, these plutons and their surrounding country rocks are regarded as most favourable exploration targets for low grade disseminated deposits and small high grade vein deposits. Attention has already been devoted to favourable source rock - host rock associations at the border zone of two of the complexes, the Ilímaussaq and the Igaliko complex. Alkaline rest liquids and volatiles in the vicinity of both complexes were introduced along lineaments, mainly joint zones in the surrounding lavas, sandstones, and granite. The emplacement is followed by an enrichment in radioactive elements, the character of which has not yet been studied. A number of alkaline veins, mainly rich in thorium extending several kilometres from the border of the Ilímaussaq intrusion, has been reported by Hansen [25].

The existence of possible fluidisation breccias in the vicinity of Ilímaussaq has lately been recognised by the author, and the conceptual model for the formation of uranium deposits in shallow intrusive environments, as proposed by Tilsley [26] with uranium deposits formed by degassing of highly differentiated alkaline intrusives might be valid for a larger part of the Gardar magmatic province.

#### 4.2. The East Greenland and North Greenland Palaeozoic fold belts

The uniform character of the geological environment in many uranium deposits and the time relation of most major deposits is being increasingly appreciated [27]. No major uranium deposits have been found in Orogenies of this age outside Greenland, and the Caledonides of East Greenland and the North Greenland Palaeozoic fold belt (*sensu stricto*) may be classified as being of limited interest for uranium mineralisation. A more detailed geological analysis will therefore be concerned mainly with the elements of older or younger rock units within the fold belts variously influenced by the orogeny.

As pointed out earlier Precambrian rocks have not been recognised as being involved in the North Greenland fold belt, and the only region of any significance for uranium exploration include areas of late Precambrian and lower Palaeozoic suprastrata overlying a crystalline basement, supposed to be of Proterozoic age. Such areas are confined to the western part of North Greenland from Melville Bugt to Inglefield Land [8]. No exploration for uranium has been carried out in North Greenland, but the region is presently being geologically mapped and results from that, and from drainage samples, will provide a better basis for an evaluation of the uranium potential. The exploration target will be unconformities in the Proterozoic rock sequence.

A substantial part of the Caledonian regime in East Greenland was explored for uranium from 1971 to 1977 [28 & 29]. The survey confirmed that rocks of Palaeozoic age have a low potential for uranium. Uranium mineralisations of minor importance (C. Hallenstein, personal communication) have, on the other hand, been found within elements of early Proterozoic crystalline rocks, with a Rb-Sr isochron age of 1725 m.y. [30]. Uranium mineralisation has also been found in acid volcanic rocks of Middle Devonian age [31] and in minor amounts in fault zones active through the Caledonian to the Tertiary. Stratabound uranium occurrences in phosphates, similar to the uranium mineralisation in the

TABLE III CATEGORIES OF URANIUM POTENTIAL WITHIN GEOLOGICAL UNITS IN GREENLAND. FOR EXPLANATION OF CATEGORIES AND SETTING OF URANIUM DEPOSITS, SEE TEXT IN "CONCLUSION" CHAPTER.

Geological division	Age	Category of U-potential	Geol. setting of U-deposits
<u>South Greenland</u>			
Gardar igneous province	Middle Proterozoic	I	C,D
Ketilidian mobile belt	Early to middle Proterozoic	I	B,C,D
<u>Southern West and South-East Greenland</u>			
Central gneiss block	Archaean	III	C
Carbonatite complexes	Phanerozoic	II	C
<u>South central West Greenland</u>			
Nagssugtoqidian mobile belt	Early Proterozoic	III	C
<u>North central West Greenland</u>			
Rinkian mobile belt	Early Proterozoic	I	B,C
<u>Central East Greenland</u>			
Precambrian sediments and crystalline rocks	Age of units not distinguished	I	B,D
Caledonian regime sensu lato	Late Proterozoic to Permian	III	C,D,F
Tertiary intrusive centres	Tertiary	I	C,D
<u>North-East and Eastern N-Greenland</u>			
Precambrian sediments and crystalline rocks	Age of units not distinguished	I	B,D
Caledonian regime sensu lato	Late Proterozoic to upper Palaeozoic	III	C,D,F
<u>North Greenland</u>			
North Greenland fold belt sensu lato	Late Proterozoic to upper Palaeozoic	III	D,F
<u>North-West Greenland</u>			
Crystalline basement and Thule Group	Early to middle Proterozoic	I	B,D

Old Red Sandstone of northern Scotland [32] have been located in Devonian red beds on Wegener Halvø [20]. On the basis of the well known geological framework and the results of exploration up to the present, the Archaean and Proterozoic formations preserved in East Greenland seem to be only of medium favourability, and mainly for the discovery of vein type deposits.

#### 4.3. Tertiary sediments and magmatic rocks

The Tertiary volcanogenic province of basaltic lavas in central East and central West Greenland is regarded as having no potential for uranium, nor are the marine lower Cretaceous to Lower Tertiary sediments underlying the lava pile in West Greenland. In the southernmost part of the sedimentary basin the depositional facies is fluviatile deltaic, often with coal seams, and this is the only part of the basin which deserves moderate attention in future exploration plans. No prospecting has yet been carried out in these areas.

The Tertiary igneous complexes in East Greenland are shallow intrusions more or less differentiated along an alkaline trend. Small pneumatolytic or hydrothermal mineralisations with uranium and thorium have been found in and around two of the most differentiated intrusions, which were included in the radiometric survey [29]. Many of the Tertiary igneous intrusions have not been explored for uranium, but the potential of the complexes in general is probably moderate. Deposits to be expected would be disseminated magmatic or vein type. A hydrothermal uranium mineralisation belonging to the latter category, and associated with fluorite and barytes has been found by Nordisk Mineselskab A/S in a major fault zone between late Caledonian granites and upper Permian sandstones. The mineralisation, which is small, is controlled by the fault and its source is probably the nearby Tertiary Werner Bjerre Intrusion.

## 5. CONCLUSIONS

The potential for uranium within the various geological units in Greenland has been described qualitatively on the basis of the available geological knowledge combined with the results of the uranium exploration up to the present. Only a preliminary type of analysis has been possible, firstly because the present geological knowledge varies considerably from one region to another, and secondly because the amount of uranium

exploration carried out up to now is not only limited but has usually been performed as second priority to contemporaneous geological mapping by the Geological Survey of Greenland.

In realistic terms, however, an improvement of the scale of exploration in Greenland, and presumably of many other places, is an interactive process where positive or negative results partly control the next steps to be taken. The present analysis is accordingly not more than a single step in a very long sequence of events.

Table I summarises the general geology and the geological evolution of Greenland while Table III gives an estimation of the potential of the various geological units for uranium deposits, using the ranking, as used by the NEA/IAEA Steering Group on Uranium Resources in the IUREP report in a slightly modified version. The ranking scheme which only considers deposits which can be exploited at costs less than \$ 130/kg U is:

- I Areas with good potential but insufficient geological knowledge and exploration,
- II Areas with good potential but insufficient exploration,
- III Areas with limited potential and interest.

Table III also lists the types of deposits most likely to be found within each area according to the following categories, as used in the EUREP report:

- A. Quarts pebble conglomerate deposits
- B. Proterozoic unconformity-related deposits
- C. Disseminated magmatic, pegmatitic and contact deposits in igneous and metamorphic rocks
- D. Vein deposits
- E. Sandstone deposits
- F. Other types of deposits.

It is concluded, that the highest potential is connected with the Proterozoic rocks within the extensive Precambrian regime of Greenland. In particular the Ketilidian metasediments in South Greenland as well as the Gardar intrusive complexes and their surrounding Ketilidian country rocks are attractive. The high favourability of this area is strongly supported by encouraging results of earlier and present exploration work. The Proterozoic rocks of the Rinkian mobile belt in West Greenland, the basement of the Thule Group in North-West Greenland and the Proterozoic crystalline rocks associated with the Caledonides in East Greenland may be classified as having medium favourability.

The classification in Table III is presented with the realisation that the state of exploration in each subarea is different. Future exploration will therefore undoubtedly result in reclassification.

#### Acknowledgment

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**DISCUSSION**

R.N. CROCKETT: Can you say whether, in view of the difficult geographical and climatic conditions obtaining in Greenland, there is a high probability that even quite large deposits of uranium will remain undiscovered for longer than might be the case elsewhere?

B.L. NIELSEN: In general, the physical conditions for exploration in Greenland are not very difficult compared to most other areas. Unlike in many places, the rate of exposure in Greenland is very high, and you do not always have to rely entirely on indirect exploration methods. So I do not believe the physical conditions in Greenland place particular constraints on the discovery of uranium deposits.

## ИСПОЛЬЗОВАНИЕ ОПРЕДЕЛЕНИЙ ПРОТАКТИНИЯ ПРИ ОЦЕНКЕ УРАНОВЫХ РУДОПРОЯВЛЕНИЙ НА ПОИСКОВОМ ЭТАПЕ

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### Abstract—Аннотация

#### USE OF PROTACTINIUM DETERMINATIONS IN EVALUATING URANIUM ORE OCCURRENCES AT THE EXPLORATION STAGE.

A method based on the use of protactinium determinations is proposed for preliminary rapid evaluation of anomalous sectors and uranium ore occurrences. Three groups of uranium ore manifestations are considered on the basis of uranium-protactinium ratios with a view to deciding whether further exploration is called for or whether work should be stopped. In the first case, with a U/Pa ratio close to one, the ores on the surface and in the deeper horizons have about the same content. In the second case, with a U/Pa ratio larger than one, the uranium has been introduced from elsewhere and the manifestation does not merit further investigation. In the third case the U/Pa ratio is less than one; the uranium on the surface has been leached out, but at some depth one could expect the percentage concentration in the ores to increase. The use of this method makes it possible to save time and other resources which would otherwise be expended on unproductive manifestations and at the same time guarantees that deposits with weak radioactivity and low concentrations on the outcrops in the hypergenesis zone will not be passed over.

#### ИСПОЛЬЗОВАНИЕ ОПРЕДЕЛЕНИЙ ПРОТАКТИНИЯ ПРИ ОЦЕНКЕ УРАНОВЫХ РУДОПРОЯВЛЕНИЙ НА ПОИСКОВОМ ЭТАПЕ.

Предлагается метод предварительной экспресс-оценки аномальных участков и урановых рудопроявлений с использованием определений протактиния. По соотношению урана и протактиния рассматриваются три группы урановых рудопроявлений с целью их оценки для дальнейшей разведки или прекращения работ. 1-ый случай, когда отношение  $U$  к  $Pa$  примерно равно единице. Руды на поверхности и на глубоких горизонтах имеют примерно одинаковое содержание. 2-ой случай, когда отношение  $U$  к  $Pa$  больше единицы. Уран привнесен извне. Рудопроявление не заслуживает разведки. 3-ий случай, когда отношение  $U$  к  $Pa$  меньше единицы. Уран на поверхности выщелочен, но на глубине следует ожидать увеличения процентного содержания в рудах. Применение метода позволяет сократить время и средства за счет отбраковки бесперспективных рудопроявлений и гарантирует от пропуска месторождений со слабой радиоактивностью и низкими содержаниями на выходах в зоне гипергенеза.

В поисковой практике встречается группа урановых рудопроявлений, обязанная своим происхождением процессам выщелачивания урана и радия из рудных залежей и последующего переотложения этих элементов в приповерхностных слоях. Рудопроявления этого типа часто обладают высокой радиоактивностью и кондиционными содержаниями на выходах, но быстро затухают с глубиной.

Такие рудопроявления, называемые "бескорневыми", "инсоляционными", "ложными" и т.д., трудно диагностируются с поверхности. Определение их бесперспективности связано с большими затратами средств и времени, необходимыми для осуществления проходки горных выработок.

С другой стороны, в определенной геохимической обстановке на выходах кондиционных рудных тел, затронутых процессами гипергенеза, наблюдается интенсивный вынос урана и радия. В этом случае на поверхности сохраняются лишь слабые гамма-аномалии, а опробование показывает содержания урана, не соответствующие реальным содержаниям в первичных рудах.

Эта группа рудопроявлений чаще всего, особенно при мелкомасштабных поисках, пропускается или относится к категории бесперспективных. В лучшем случае они вскрываются горными выработками в последнюю очередь.

Особенности поисковых работ и методы оценки аномальных участков с нарушенным равновесием между ураном и радием освещены в литературе, но для рассматриваемых рудопроявлений эти методы не всегда применимы [1].

В настоящей работе рекомендуется более эффективный метод первичной оценки рудопроявлений, основанный на комплексном определении протактиния, урана и радия в пробах, отобранных на аномальных участках.

Протактиний является продуктом распада  $^{235}\text{U}$  и находится с ним в определенных количественных соотношениях. Период его полураспада равен  $3,248 \cdot 10^4$  лет. В отличие от урана и радия протактиний обладает значительно меньшей миграционной способностью и лучше сохраняется на выходах рудных тел даже в условиях интенсивно проявленных процессов гипергенеза, что позволяет использовать его в качестве "репера" и определять интенсивность и направление перемещений урана и радия. Представление о направленности миграционных процессов (привнос – вынос) является важным фактором при оценке рудопроявления и выдаче заключений о целесообразности проведения разведочных работ.

Все многообразие индивидуальных особенностей различных урановых рудопроявлений в конечном итоге сводится к трем типам урано-протактиниевых соотношений, характеризующих направление и интенсивность миграционных процессов, что и определяет их предварительную оценку.

Характеристика радиологических особенностей рудопроявлений различного типа, обнаруженных с поверхности в процессе поисковых работ, приведена в табл. I.

На рудопроявлениях первого типа содержания протактиния близки к содержаниям урана и радия, что свидетельствует о слабом выносе этих элементов.

Разведка рудопроявлений такого типа показала, что содержания урана на поверхности и на глубоких горизонтах примерно одинаковы.

ТАБЛИЦА I. РАДИОЛОГИЧЕСКИЕ ОСОБЕННОСТИ РУДОПРОЯВЛЕНИЙ В ЗАВИСИМОСТИ ОТ НАПРАВЛЕНИЯ МИГРАЦИИ РАДИОАКТИВНЫХ ЭЛЕМЕНТОВ

Тип рудопроявления	Содержание радиоактивных элементов (%)			Отношение содержаний		
	уран	протактиний	радий	урана к протактинию	радия к протактинию	радия к урану
1	0,070	0,075	0,060	0,93	0,80	0,86
2	0,07	0,05	0,08	1,40	1,60	1,15
3	0,05	0,17	0,08	0,29	0,47	1,6

Примечание: содержания протактиния и радия приведены в единицах равновесного урана.

Рудопроявления второго типа характеризуются преобладанием урана над протактинием, что свидетельствует об интенсивном выносе урана. В этом случае содержания урана на глубоких горизонтах ниже, чем на поверхности, и часто приближаются к кларковым.

На рудопроявлениях третьего типа содержания протактиния значительно выше, чем урана. Это указывает на интенсивный вынос урана и на более высокие его содержания в первичных рудах. В скважинах и горных выработках глубоких горизонтов содержания урана оказались в несколько раз выше, чем на поверхности. Аналогично оценивается направление процессов миграции радия.

Полученные результаты дают основание рекомендовать определение протактиния на всех аномальных участках, обнаруживаемых при поисках, с целью установления истинных содержаний урана в первичных рудах до их разрушения на выходах.

Предлагаемый способ оценки урановых рудопроявлений при поисках стал возможен после разработки И.П.Шумилиным радиометрического метода и датчиков для экспресс-анализов проб на уран, радий, радон, протактиний и торий [2].

Анализы могут выполняться на стандартной, серийной аппаратуре. Датчик представляет собой кристалл NaJ (Tl) размером 100×100 мм или 80×60 мм, над которым расположены два или три ряда по 6 бета-счетчиков СТС-6 с дополнительными съемными фильтрами, толщиной 0,2 г/см<sup>2</sup>.

Измельченные образцы от 20 до 150 г измеряются в одной или (для повышения точности бета-измерений) в нескольких кюветах площадью 120 см<sup>2</sup> в промежуточных или насыщенных для бета-лучей слоях. Измерения выполняются одновременно по 6 каналам.

По жесткому бета-излучению выделяется излучение группы урана, а по общему гамма-излучению с пороговой дискриминацией 300 кэВ — группы радона.

Излучение группы протактиния выделяется в двух участках гамма-спектра — в области энергий 82 и 270 кэВ, что позволяет дублировать анализы на протактиний.

Излучение радия выделяется в области энергий 186 кэВ, а ряда тория — 2620 кэВ.

На основании этих измерений составляется система из 6 уравнений и определяется уран, радий, радон, протактиний (два результата) и торий. Для повышения точности анализов пробы можно прокалывать при температуре 800-1000° С. При этом проба теряет около 90% радона и интенсивность мешающего излучения группы радона уменьшается на порядок. Продолжительность комплексного анализа равна 10-15 мин.

### ЛИТЕРАТУРА

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- [ 2 ] ШУМИЛИН, И.П., Возможности применения Ge (Si)-детекторов для анализов по естественной радиоактивности, Ат. Энерг. 37 (1974).

### DISCUSSION

V. ZIEGLER: Could protactinium analyses be used for strategic geochemical reconnaissance of the stream-sediment or soil types in climatic conditions in which the uranium has undergone surface leaching?

G.N. KOTEL'NIKOV: The protactinium technique is used at the second stage of prospecting work, i.e. when radiometric prospecting (airborne or ground) has been completed and when anomalous sectors or ore deposits have already been found. It is used to answer the question whether they are worth further prospecting work (trenches, shafts and drill-holes) or should be rejected.

We have not used the technique in the tropics, but encouraging results have been achieved when evaluating deposits with outcrops of ore bodies that have completely disintegrated near the surface. We in fact make protactinium analyses of samples taken from anomalies irrespective of the climatic conditions in which they are found.

B.L. DICKSON: I was surprised to hear you state that you see a disequilibrium between  $^{238}\text{U}$  and  $^{234}\text{Pa}$ , since these two isotopes reach equilibrium in about 100 days. In fact the beta-gamma method of ore analysis is based on the assumption that  $^{238}\text{U}$  and  $^{234}\text{Pa}$  are in equilibrium. Do you have a model which will take into account the rapid displacement of uranium that must occur if the disequilibrium you mention between  $^{238}\text{U}$  and  $^{234}\text{Pa}$  is to be observed?

G.N. KOTEL'NIKOV: Our method is also based on the assumption that uranium and protactinium are in equilibrium. But the point we made in our paper related not to a shift in equilibrium between these two elements, but to their separation as a result of hypergenesis.

# EXECUTION OF THE UNITED STATES NATIONAL URANIUM RESOURCE EVALUATION PROGRAMME

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## Abstract

### EXECUTION OF THE UNITED STATES NATIONAL URANIUM RESOURCE EVALUATION PROGRAMME.

The National Uranium Resource Evaluation (NURE) programme is directed by the US Department of Energy with the major goal of establishing reliable estimates of the uranium resources of the United States of America. The NURE programme has been organized into several elements to develop and compile geological, geochemical, geophysical and other information that will contribute to assessing the distribution and magnitude of uranium resources. The extensive efforts now under way will result in a systematic collection and compilation of data which will be used to prepare a comprehensive report in 1980 covering certain priority areas, an interim report in 1983 of additional priority areas, and a final report covering all of the United States in 1985. The principal programme elements are quadrangle evaluation, quadrangle assessment, aerial radiometric and magnetic surveys, hydrogeochemical and stream-sediment surveys, topical geological studies, world-class and intermediate-grade studies, and technology development. The Bendix Field Engineering Corporation is the principal contractor responsible for the execution of the NURE programme. Substantial responsibilities have been assigned to the US Geological Survey, to various state surveys, and to private firms.

## 1. INTRODUCTION

The National Uranium Resource Evaluation (NURE) program is funded by the Government of the United States and directed by the U.S. Department of Energy (DOE). Bendix Field Engineering Corporation (a subsidiary of The Bendix Corporation) has been employed by the DOE as the principal contractor and has been assigned major responsibility for execution of the NURE program. The goals of the NURE program are to assess the nuclear fuel resources of the United States; reduce uncertainties about the estimated extent, availability and economics of domestic and foreign nuclear fuel resources; and make available to industry technology and resource information for use in the

development and production of uranium resources. Principal activities of the NURE program include aerial radiometric and magnetic surveys, geochemical surveys, and on-the-ground geologic studies of the United States.

The NURE program is, in simple terms, a two-phase program--data collection and data evaluation. When one wants to determine some characteristic of a population a statistical evaluation is commonly made to determine the proper sample size, the sample is taken, the characteristic is measured, and a projection is made as to the total characteristic expected in the entire population. A statistician can readily determine, if several sets of samples are taken, the probability that the projection is correct. Unfortunately, the information desired for the NURE program, the quantity of recoverable uranium in the United States (given certain prices and costs), cannot be sampled because information is available concerning only the discovered deposits and little is known concerning the size of the total population. Sampling is therefore not a viable technique. The entire United States, the total area in which the population may be found, must be examined. The extensive effort now underway in each NURE program element will result in a systematic collection and compilation of data which will be used in preparing a comprehensive uranium-resource report in 1980 covering certain priority areas, an interim report in 1983 on additional priority areas, and a final report on all the conterminous United States and Alaska in 1985.

A program the size of the NURE program must be divided into work units if it is to be properly scheduled and administered. The basic work units, geographic areas, are the one-degree latitude by two-degrees longitude (three-degrees longitude in Alaska) quadrangle areas delineated by the National Topographic Map Series (published at a scale of 1:250,000). There are 603 of these quadrangles in the NURE program. The average area of a quadrangle is approximately 18,500 square kilometers.

## 2. SUBPROGRAMS

The NURE program has been divided into several subprograms for the development and compilation of geologic, geophysical, geochemical, and other information which will contribute to the evaluation and assessment of the uranium reserves and potential resources of individual quadrangles. These subprograms are:



- a) Aerial radiometric reconnaissance surveys.
- b) Hydrogeochemical and stream-sediment reconnaissance surveys.
- c) Subsurface studies.
- d) Geologic maps.
- e) Topical geologic studies.
- f) Technology development.
- g) World-class and intermediate-grade studies.

## 2.1. Aerial radiometric reconnaissance surveys

The objective of this subprogram is to conduct a high-sensitivity aerial gamma-ray-spectrometer and magnetic reconnaissance survey of the conterminous United States and Alaska. Aerial reconnaissance surveys began in 1974 and are scheduled to be completed in 1982. Nine million square kilometers will be surveyed, with flight line spacings ranging from 1.6 kilometers to 10 kilometers. Two million line kilometers of data will be acquired. All of the surveys are flown by private industry subcontractors.

Rigid specifications and quality control procedures must be met by the subcontractors. Concrete pads with measured amounts of uranium, thorium, and potassium have been constructed at the Grand Junction, Colorado, airport to facilitate system calibration and normalization. A 6.5 kilometer test range for in-flight calibration and normalization has been established on the south shore of Lake Mead, Arizona. The specifications and requirements of aerial survey contracts are available from Bendix Field Engineering Corporation.

Results of individual aerial surveys (reports and map folios) are placed on open-file by the DOE. Results are also transmitted to field geologists and to a BFEC group at Grand Junction concerned with further data presentation and interpretation.

## 2.2. Hydrogeochemical and stream-sediment reconnaissance surveys

The objective of this subprogram is the systematic determination of the distribution of uranium and associated trace elements in the conterminous United States and Alaska. Three DOE laboratories are currently conducting these surveys: the Savannah River Laboratory in both the eastern and western parts of the United States; the Los Alamos Scientific Laboratory in the Rocky Mountain States and Alaska; and the Oak Ridge Gaseous Diffusion Plant in the central United States. Although each of the laboratories is an independent organization, Bendix has been given responsibility for overall coordination of and specifications for data collection, analyses, and reports.

Multi-element analyses are made of approximately 2000 samples collected in each quadrangle. Depending upon the hydrology and cultural development of the quadrangle, samples may be taken of surface waters, ground waters, wet stream and lake sediments, and dry stream sediments (dry stream sediments are collected only in arid regions of the western United States). Although this subprogram is in direct support of the NURE program, the results, multi-element analyses, will be of value in exploring for other minerals and in defining geochemical provinces. The reports of individual surveys are available and treated in the same manner as the aerial-survey reports.

### 2.3. Subsurface studies

There are several, not necessarily mutually exclusive, activities in the subsurface-studies subprogram. All activities have the same objective--improving the reliability of potential resource estimates beyond that which could be achieved by surface studies alone.

#### *2.3.1. Department of Energy drilling projects*

Drilling projects directed by the DOE are for the purpose of improving the reliability of previous estimates of potential uranium resources credited to small areas of high priority, generally near known uranium deposits or areas of active exploration. Drilling is subcontracted to private firms and supervised by Bendix, as are all drilling activities. Drill cores are cut where appropriate, and all holes are logged by private logging companies or by Bendix-operated logging equipment, or by both, depending upon the availability of logging equipment and the logs desired.

#### *2.3.2. Quadrangle-evaluation drilling*

Geologists examining quadrangles may request drilling for the purpose of acquiring subsurface information. Budget limitations prohibit meeting all requests, requiring that requests be evaluated, and only those of highest priority are drilled. Much effort is required to assure the maximum return of information from a minimum number of holes. Coring and logging is accomplished as in 2.3.1.

#### *2.3.3. Other drilling*

Drilling projects may be proposed in support of other subprograms, such as topical studies or world-class and intermediate-grade studies. The proposed drilling projects are evaluated and supported as funds allow.

#### 2.3.4. *Well-logging programs*

Well-logging programs, other than logging required for support of drilling projects, are designed to acquire additional subsurface information on an "as available" basis. Private firms may request that their exploration holes be logged by DOE-owned, Bendix-operated gamma-ray-spectrometer logging equipment. These requests are honored whenever possible.

A subprogram currently under study, the national logging program, is designed to provide logging of any well in an area of interest. The well may have been drilled for oil, gas, water, or minerals other than uranium.

Substantial use is made of well logs run by private firms for private mining companies. Many mining companies supply the DOE with copies of logs of uranium-exploration drill holes. Bendix does not participate in this activity.

To help assure consistent and meaningful interpretation of private logs, test holes containing uranium of known concentration and distribution are maintained at several locations in the United States. All private logging firms use these facilities, without charge, to calibrate and normalize individual logging probes and related truck-mounted equipment.

#### 2.4. Geologic maps

Geologic maps are an obvious requirement for any mineral-resource-evaluation program. At present suitable geologic maps are available for only 150 of the 603 quadrangles. Prior to 1977, if a geologic map of a quadrangle to be aerially surveyed was not available from either the U.S. Geological Survey or one of the state surveys, the map was compiled by the aerial-survey subcontractor. Since 1977 geologic maps have been procured from private geologic subcontractors. Time and funds do not allow geologic mapping, in the ordinary sense, by geologists walking the area. The quadrangle maps are compiled by using geologic maps of suitable scale available from published reports, graduate theses, or other reliable sources. Areas not covered by such maps are mapped using aerial photograph interpretation techniques. Care is taken to assure, where possible, that the interpretation is accomplished by geologists familiar with the geology of the area.

#### 2.5. Topical geologic studies

This subprogram, accomplished largely through subcontractors, has two prime objectives: (1) to identify the criteria

that best explain the ore genesis of the world's significant uranium deposits; (2) to investigate recent developments in geochemistry, geostatistics, and the processing of remote-sensing data for possible application in the determination of potential uranium resources. Projects initiated in support of objective (1) are not mere descriptions of ore bodies. Subcontractors are required to make judgements on a number of key geologic factors, including geologic history, sources of uranium, transport systems, the nature and cause of uranium deposition, and post-depositional events, including those that tend to preserve the deposit.

## 2.6. Technology development

The major goal of technology development is to provide new and improved instrumentation and techniques for uranium exploration, exploitation, and resource assessment. The subprogram utilizes a number of geoscience-technology areas, including conventional and nuclear geophysics, geology, geochemistry, geostatistics, and remote sensing.

Activities are conducted by Bendix through a combination of in-house and subcontracted projects. The mix of activities includes some high-risk, low-cost projects which, if successful, could have a significant impact on the NURE program. A full list of projects is beyond the scope of this article, but some of the projects completed or in progress are: uranium borehole logging with prompt-fission neutrons; design of a delayed-fission neutron-logging probe; californium-252-based borehole-logging system; solid-state photomultiplier tube; large-volume germanium gamma-ray detector; and design and construction of a radon emanometer.

## 2.7. World-class and intermediate-grade studies

In the summer of 1978, the DOE identified a requirement for a report, to be submitted by October 1980, describing the potential uranium resources in those quadrangles likely to have the largest resources. The NURE program schedule was changed to meet this requirement. It was also decided that special efforts should be made to locate and evaluate geologic environments similar to those of non-sandstone world-class deposits not now known in the United States. Over 80 percent of U.S. uranium production is from sandstone deposits, while over 80 percent of the world uranium production is from non-sandstone deposits. The world-class studies, a search for major non-sandstone deposits in the United States, were initiated. At present, the principal effort is directed toward evaluation of Precambrian quartz-pebble conglomerates.

The existence of large intermediate-grade (100-500 ppm  $U_3O_8$ ) deposits has been indicated by private exploration activity. A decision was made to make special efforts to locate and evaluate these deposits and, if located, to determine the size, grade, and economics of mining and milling. Both the world-class and intermediate-grade studies were started late in 1978.

### 3. INTRODUCTION TO QUADRANGLE EVALUATION AND QUADRANGLE ASSESSMENT

Quadrangle evaluation, for the purposes of the NURE program, comprises those data-collection activities that attempt to identify and quantify certain geologic characteristics of individual quadrangles through the use of geophysical, geochemical, and geologic field methods. Quadrangle assessment is the act of assessing the results of quadrangle evaluation to determine the potential uranium resources of the quadrangle. Most of the previously described subprograms directly support quadrangle evaluation and assessment.

The overall objective of evaluation and assessment is to determine the probable location and quantity of uranium resources in the United States. This will be accomplished by comparison and extrapolation. Quantities of uranium per unit area, unit volume, or unit length will be extrapolated from control areas (areas of known uranium mineralization) to areas with similar geologic environments.

#### 3.1. Definitions

Several phrases will be used in the following discussions of evaluation and assessment. To avoid confusion or misinterpretation rather strict meanings have been assigned to these phrases.

##### 3.1.1. *Potential uranium resources*

The recoverable quantities of uranium in hypothetical uranium deposits assumed to be present in potential resource areas, deposits that could be mined at a profit under various given prices and costs.

##### 3.1.2. *Potential resource area*

That geographic area delineated by the field geologist as having geologic characteristics identical to or nearly identical to those as determined by the application of recognition criteria of a control area.

### *3.1.3. Control area*

An area with all of the following attributes: a geographic area delineated by the DOE or Bendix; an area with relatively uniform, although not necessarily simple, geologic characteristics; a uranium-producing area or a well-explored area for which data are available concerning the quantity, distribution, and grades of the uranium mineralization present; an area for which recognition criteria have been determined.

### *3.1.4. Recognition criteria*

The criteria a NURE geologist is required to use to identify a geologic environment that may have favorably influenced the formation of uranium deposits.

### *3.1.5. Uranium endowment*

The total quantity of uranium postulated to exist in a control area or in a potential-resource area, with the stipulation that the quantity include only that uranium postulated to occur in concentrations with a grade exceeding 100 ppm (0.01%)  $U_3O_8$ . In practice, several quantities may be proposed for a single area, depending upon probability distributions. For example, there may be a 5 percent probability that the uranium endowment is at least 100 tonnes  $U_3O_8$  and a 95 percent probability that the uranium endowment is not greater than 1000 tonnes  $U_3O_8$  in a single potential resource area.

### *3.1.6. Subjective assessment*

Any determination of uranium endowment or potential uranium resources that uses one or more mathematical factors selected by personal judgement or intuition.

### *3.1.7. Mineralization factor*

The uranium endowment per unit area in a control area. It is expressed as tonnes of  $U_3O_8$  per square kilometer.

### *3.1.8. Area, volume, and length*

'Volume' or 'length' may be substituted for 'area' in the definitions. For example, the mineralization factor for a roll-front sandstone deposit can be given in tonnes per cubic kilometer of sandstone, tonnes per square kilometer of sandstone, or tonnes per kilometer along the length of the roll front. Care must be taken when making calculations to assure that the

same units, volume, area, or length, are used for the control area and the potential-resource area.

### 3.2. Uranium deposit classification and recognition criteria

The evaluation and assessment strategy selected for the NURE program requires the comparison of areas to be assessed with control areas of known mineralization. Standards had to be established to assure uniform comparison and selection techniques. A classification of uranium deposits provides the most convenient framework for selecting control areas and making comparisons. A preliminary classification has been developed [1] and provided to the field geologists. The guiding philosophy in developing the classification system was to provide the most useful framework for assessment rather than a theoretically complete genetic classification. Although the principal classifications are divided on the basis of rock type, the classifications make substantial reference to ore genesis. Deposits whose origins have not been convincingly resolved are described by observable geologic features rather than inferred genesis. Revision of the classifications is likely as new information becomes available, but revisions will be made with care to ensure consistent evaluation of all quadrangles during the life of the NURE program.

Each uranium deposit class or subclass has several unique features that distinguish it from other classes. Many of these features are the result of the geologic environment that existed at the time the deposits were formed. These environmental features are of particular importance to the field geologist, as he is unlikely to discover uranium deposits; instead he will discover environments which are favorable for uranium deposits but which lack any surface shows of uranium mineralization. To assure uniform and consistent interpretations by the hundreds of geologists working on the NURE program, descriptions of the environments (Geologic Characteristics of Environments Favorable for Uranium Deposits) [2], have been provided for use by field geologists. This publication contains substantial detail concerning all the classes of uranium deposits currently known in the world. The many criteria by which a geologist may recognize favorable environments are listed. A field manual that briefly describes the recognition criteria has also been prepared [3].

### 3.3 Control areas

Control areas will be selected and defined by the Grand Junction, Colorado, staff of the Resource Division, DOE, and the Geology Division of Bendix. A perfect control area would

be one in which all of the uranium had been recovered; in which knowledge was perfect concerning the grades, quantity, distribution, and mode of occurrence of all of the uranium deposits in the area; and in which the geology was uniform throughout the area. In other words, a perfect model. No such control areas have yet been identified. Nevertheless, control areas will be selected and the necessary factors determined where possible, and projected or postulated where not possible. Comparison of an imperfect known with an unknown is preferable to simple speculation regarding the unknown. Selection of the many control areas required for the NURE program is currently in progress. Control areas for classes of deposits not now known in the United States will be selected from foreign deposits for which the required criteria is available. As new or better information is acquired regarding control areas, changes will be made in the earlier assessments of individual potential-resource areas.

#### 4. QUADRANGLE EVALUATION

Quadrangle evaluation, the field examination of geologic features and the final interpretation and utilization of data obtained by the various subprograms, is the responsibility of the field geology team. To accomplish part of this work, Bendix has established field offices in Anchorage, Alaska; Spokane, Washington; Reno, Nevada; Casper, Wyoming; Grand Junction, Colorado; Albuquerque, New Mexico; Austin, Texas; Atlanta, Georgia; and Pittsburgh, Pennsylvania, with an average of 10 geologists per field office. It was obvious that organizations having specific knowledge of the geology of certain quadrangles should be encouraged to participate in the evaluation work. To that end, a joint working agreement has been made between the DOE and the U.S. Geological Survey, and subcontracts have been issued to several state geological surveys and private subcontractors. The 116 quadrangles to be evaluated before October 1980 are assigned as follows: U.S. Geological Survey, 23; state geological surveys, 14; private subcontractors, 36; Bendix, 43. All quadrangle-evaluation studies are directed and monitored by Bendix.

Scheduling of the quadrangle evaluation program is based on assigning two geologists for two years to quadrangles of average difficulty. To ensure a uniform approach by the many geologists working on quadrangle evaluation, the work on each quadrangle is divided into three phases: Phase I, planning, 6 months; Phase II, field investigation, 15 months; Phase III, folio preparation, 3 months. The time constraints of the



NURE program are inflexible. Variations in the effort required to evaluate the many quadrangles are accommodated by varying the amount of geologic work expended on each quadrangle.

#### 4.1. Phase I, planning

The object of Phase I is the preparation of a plan for evaluating the quadrangle. The geologist must develop a knowledge of the geology of the quadrangle (including previous geological, geochemical, and geophysical studies), a knowledge of the mode of occurrence of uranium deposits, and a knowledge of application of uranium-environment-recognition criteria. To ensure that quadrangle evaluation is planned and executed with maximum efficiency and thoroughness, the geologist is required to prepare the following materials during Phase I.

##### *4.1.1. Preliminary list of annotated references*

The annotated list of references is evidence that the geologist has read and considered the principal literature concerning the quadrangle. An exhaustive reference list is not required, the objective being a knowledge of the geology of the quadrangle rather than acquisition of a long list of titles.

##### *4.1.2. Preliminary table and map of uranium occurrences*

Knowledge of the environment and distribution of uranium occurrences within a quadrangle is essential to quadrangle investigation. With this information, the geologist can select those recognition criteria best suited for the discovery of similar environments elsewhere in the quadrangle.

##### *4.1.3. Geologic-map index*

The geologist is required to compile a geologic-map index, delineating the extent and availability of geologic maps, for the same reasons he is required to compile an annotated list of references.

##### *4.1.4. Generalized land-status map*

Land status may affect the feasibility of geologic investigations within parts of the quadrangle. The preparation of this map is required to call attention to any special considerations required by the current legal status of land areas within the quadrangle.

#### 4.1.5. *Work plan*

Upon completion of the foregoing items, the geologist must submit a work plan that describes the work to be accomplished in Phase II (field investigations). The work plan is divided into four sections containing the following information:

4.1.5.1. *Surface-study plan.* The surface-study plan shall include: (1) the locations of areas to be studied in detail; (2) the geologic characteristics of these areas, including, but not limited to, specific lithologic units, structural elements, and environments of deposition; (3) the rationale used in selecting these areas; (4) the methods to be used to investigate these areas, including sampling programs; (5) the methods to be used to study any uranium anomalies detected by the hydrogeochemical or aerial surveys; and (6) the personnel to be used to accomplish this study.

4.1.5.2. *Subsurface-study plan.* The subsurface-study plan must include all of the surface-study elements which relate to investigation of subsurface geologic environments. Maximum use shall be made of existing subsurface data. Additional drilling may be requested, but the geologist must not depend on the additional drilling for evaluation of the quadrangle.

4.1.5.3. *Estimate of manpower and resources required.* The geologist is required to submit estimates that will allow budgeting of available funds, manpower, and equipment. If the estimates are too large, the scope of the work is reduced.

4.1.5.4. *Milestone chart.* A milestone chart is submitted to illustrate the duration and expected completion date of each major activity included in the work plan, thereby allowing monitoring of progress when Phase II (field investigation) is in progress.

#### 4.2. Phase II, field investigations

All geologic environments within the quadrangle to a depth of 1524 meters will be evaluated by surface investigations, aerial gamma-ray-spectrometer surveys, hydrogeochemical and stream-sediment analyses, and analyses of subsurface data. Following evaluation, each environment must be assigned to one of three categories: (1) favorable for the occurrence of uranium deposits which could contain at least 90 tonnes of  $U_3O_8$  with an average grade not less than 100 ppm  $U_3O_8$ ; (2) unfavorable for the occurrence of such deposits; or (3) unevaluated (environments with insufficient information to categorize).

#### *4.2.1. Information used in categorizing environments*

A classification of uranium deposits [1], descriptions of environments favorable for uranium deposits [2], and a field manual listing recognition criteria [3], are given to each NURE geologist. These publications are the principal guides for categorizing environments. The nature of individual environments within quadrangles is determined by field investigations, aided by the aerial and hydrogeochemical surveys and subsurface studies. Geologists are required to examine the significant uranium occurrences in or adjacent to the quadrangle under study. A uranium-occurrence report form is supplied by Bendix to ensure a thorough description of each occurrence.

#### *4.3. Phase III, folio preparation*

The final phase of quadrangle evaluation is the preparation of a folio consisting of a text, maps, and other illustrations, which present the quadrangle-evaluation data and interpretations. A style manual [4] is supplied to each geologist to eliminate inconsistencies between reports; to answer style questions for geologists, draftsmen, and typists; and to ensure adherence to a standard format. Details of the style requirements are not presented in this article, but items of importance to be included in the folio are described in the following sections.

##### *4.3.1. Environments favorable for uranium deposits*

A geologic environment judged to be favorable shall be described as to: (1) the name and age of the formation, member, or other geologic unit in which it occurs; (2) the geographic location; (3) the depth, thickness, volume and projected surface area; (4) a geologic description of the favorable environment; (5) the specific recognition criteria observed; (6) the class or classes of uranium deposits observed or anticipated; (7) the observed or inferred mineral assemblage of the deposit or deposits.

Proper application of the uranium-environment-recognition criteria and uranium-deposit classification is essential for uniform and consistent evaluation and subsequent assessment. The geologist must include in item (5) above a detailed description of the extent to which the favorable environment exhibits the uranium-environment-recognition criteria identified for the class of deposit described in item (6). In item (6), the geologist shall classify the deposits observed or anticipated according to the classification provided [2]. Deposit mineralogy is important because it may affect economic recovery

of uranium and the geologist must describe the observed or inferred uranium and gangue mineral assemblage for the class of deposits anticipated.

#### *4.3.2. Environments unfavorable for uranium deposits*

For each environment judged to be unfavorable for the occurrence of uranium deposits, the geologist must describe the environment and the reasons for judging it unfavorable. The absence of known uranium occurrences or geochemical or radiometric anomalies does not necessarily indicate the absence of subsurface uranium deposits. The geologist's evaluation should confirm that the geologic environment does not correspond to any of the favorable environments described by the uranium-environment-recognition criteria.

#### *4.3.3. Unevaluated environments*

Investigations should be conducted to acquire sufficient information for evaluation, but in some cases such investigations may not be possible within the time constraints of the NURE program. Over-use of this category will defeat the objectives of the NURE program. The geologist must supply statements defending the use of this category and describe the location of and the reasons why, the environment was unevaluated.

### 5. QUADRANGLE ASSESSMENT

Quadrangle assessment is the act of assessing the results of quadrangle evaluation to determine the potential uranium resources of individual quadrangles. It is the prelude to the determination of the potential uranium resources of the United States. Rene Descartes (1596-1650), the father of modern philosophy, gave advice that well fits the technique selected for quadrangle assessment: "It is truth certain that, when it is not in our power to determine what is true, we ought to follow what is most probable." The assumption implicit in the selection of the assessment method to be used is that "what is most probable" is that additional uranium resources will be discovered in the same geologic environments that exist at known uranium deposits and that the quantities of uranium likely to be present are proportional to the sizes of the individual environments. This assumption omits the probability that uranium will be discovered in presently unknown geologic environments. The seriousness of this omission is obviously unknown.

It is acknowledged that not every favorable geologic environment will contain uranium deposits and that the quantity of uranium present will not always be directly proportional to that of similar environments. Too little is known of the success-versus-failure ratios experienced by exploration groups to allow a statistical determination of the probability that a particular favorable geologic environment contains uranium deposits. The only approach now apparent is to request the field geologist to estimate, on a subjective basis, the range of probabilities. An investigation in progress will compare the known uranium resources of various geologic environments in an attempt to establish a statistical basis for estimating the probable range in the quantities of uranium likely to be present in similar geologic environments.

### 5.1. Quadrangle-assessment methodology

The assessment requirements of the NURE program impose two problems: (1) the estimation of the uranium endowment (see 3.1.5.) of a potential-resource area (see 3.1.2.) and; (2) the estimation of the potential uranium resources (see 3.1.1.). Estimation of the uranium endowment is accomplished by analogy; the potential-resource area is assumed to contain the same quantity of uranium per unit area, volume, or length as does the control area (see 3.1.3. and 3.3.). The assumed comparison is subject to modification by the assessor. The uranium endowment of the potential-resource area is initially calculated by the following formula:

$$E_r = V_r \cdot M_c \cdot C_{rc}$$

where:

$E_r$  = uranium endowment of the potential-resource area.

$V_r$  = volume of the potential-resource area (or area or length; see 3.1.8.).

$M_c$  = mineralization factor, the uranium endowment per unit volume (or area or length) in the control area.

$C_{rc}$  = the correction factor subjectively derived by the assessor to adjust for any differences between the control area and the potential-resource area.

The volume ( $V_r$ ) of the potential-resource area may not be perfectly known. The field geologist is required to establish upper and lower limits with the instruction that the

limits be correct 19 out of 20 times (a 5 percent to 95 percent probability range). If the values presented by the field geologist are too extreme additional field work is indicated.

The mineralization factor ( $M_c$ ) is imperfectly known. Confidence limits will be established during study of the individual control area. As more complete information becomes available concerning control areas, the assessments can be made more precise. As might be expected, many approaches have been advanced as regards the determination of the mineralization factor. The factor includes all uranium mineralization found with a grade greater than 100 ppm  $U_3O_8$ . Mine cut-off grades are substantially greater, and knowledge is imperfect concerning the quantity of uranium present in subeconomic grades.

The selection of the boundaries of control areas obviously affects the magnitude of the mineralization factor. The inclusion of large unmineralized areas will reduce the mineralization factor. The boundaries of control areas must be selected using the same criteria as used for selecting the boundaries of the potential-resource areas. If this is done with care, some uniformity or regularity, a relationship, may be discovered between mineralization factors--at least between similar classes of deposits and perhaps between dissimilar classes.

The correction factor ( $C_{rc}$ ) presents difficult problems. The opinion of the geologist, geologic intuition, has often been of value in mineral exploration, but the correction factors submitted by perhaps 603 geologists for the 603 quadrangles in the NURE program are likely to be uniform only if rigorously determined. The geologist is required to list, defend, and weight the various factors he uses in arriving at the final correction factor. The final factor should not be of a value that will grossly affect the results. If it does, it is likely that the geologist has compared the potential-resource area with a mismatched control area, or that the potential-resource area is improperly defined. Further, the field geologist is most likely to select a value less than one, under the assumption that no currently nonproductive area is likely to be as productive as the control area. A comparison of deposits in Canada or Australia with those known 10 years ago illustrates the danger of such assumptions.

The potential uranium resources of a potential-resource area are those quantities of uranium that can be recovered, at a profit, in conventional mining operations. The costs of mining and milling and the sale price of uranium concentrates must be considered.

The cost of mining is affected by the depth, size, shape, and grade-tonnage distribution of the ore bodies. This information is obviously unknown, or at best poorly known, in most potential-resource areas. For purposes of assessment the characteristics of individual ore bodies are assumed to be identical to those of the control area unless firm data acquired in the potential-resource area indicate otherwise. Probability distributions will be developed (various techniques are currently under study) to provide a range in the probable quantities of potential-uranium resources assigned to individual potential resource areas.

The sale price of  $U_3O_8$ , as well as mining and milling costs, determines the cut-off grade for mining operations. Present plans require the determination of potential uranium resources at  $U_3O_8$  forward costs of \$15.00, \$30.00 and \$50.00 per pound  $U_3O_8$ .

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#### DISCUSSION

A.E. BELLUCO (*Chairman*): In Argentina we have used the same principles as you have for extrapolating from uranium geology and the same models of the various conditions in which deposits are typically found. We use these principles and models for delineating and for giving priority to what are called "prospecting units". The latter are quantified in terms of "indices of favourable uranium geology", which in turn are used for estimating the speculative resources of the country.

I have undertaken an IAEA technical assistance mission to Peru for the purpose of delimiting and selecting "prospecting units", and for quantifying the "indices of favourable uranium geology" using this system.

A programme supported by the IAEA and IANEC is currently being carried out in Latin America for evaluating and quantifying “indices of favourable uranium geology”, applying the same principles on the basis of regional “prospecting units”. This programme should make it possible to prepare an Operational Project for prospecting work under the various geological and environmental conditions found in Latin America.



# RESEARCH ON INTERACTIVE GENETIC-GEOLOGICAL MODELS TO EVALUATE FAVOURABILITY FOR UNDISCOVERED URANIUM RESOURCES\*

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## Abstract

### RESEARCH ON INTERACTIVE GENETIC-GEOLOGICAL MODELS TO EVALUATE FAVOURABILITY FOR UNDISCOVERED URANIUM RESOURCES.

Current methods of evaluating favourability for undiscovered uranium resources are unduly subjective, quite possibly inconsistent and, as a consequence, of questionable reliability. This research is aimed at reducing the subjectivity and increasing the reliability by designing an improved method that depends largely on geological data and their statistical frequency of occurrence. This progress report outlines a genetic approach to modelling the geological factors that controlled uranium mineralization in order to evaluate the favourability for the occurrence of undiscovered uranium deposits of the type modelled. A genetic model is constructed from all the factors that describe the processes, in chronological sequence, that formed uranium deposits thought to have a common origin. The field and laboratory evidence for the processes constitute a geologic-occurrence base that parallels the chronological sequence of events. The genetic model and the geologic-occurrence base are portrayed as two columns of an interactive matrix called the "genetic-geologic model". For each column, eight chronological stages are used to describe the overall formation of the uranium deposits. These stages consist of (1) precursor processes; (2) host-rock formation; (3) preparation of host-rock; (4) uranium-source development; (5) transport of uranium; (6) primary uranium deposition; (7) post-deposition modification; and (8) preservation. To apply the genetic-geological model to evaluate favourability, a question is posed that determines the presence or absence of each attribute listed under the geologic-occurrence base. By building a logic circuit of the attributes according to either their essential or non-essential nature, the resultant match between a well-documented control area and the test area may be determined. The degree of match is a measure of favourability for uranium occurrence as hypothesized in the genetic model. This process of geological decision analysis results in a series of favourability maps that can be combined into a final composite favourability map.

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\* Work done in co-operation with the US Department of Energy.

## INTRODUCTION

The quantitative estimation of undiscovered uranium resources is a relatively new but increasingly important activity. Most accepted methods of estimating are based on evaluating favorability for the presence of undiscovered uranium deposits from geologic analogy, which is a valid basis. To date, however, these methods have been highly subjective and of questionable reliability. The need for improved uranium resource assessment methodology to lessen subjectivity and to increase reliability of estimates is widely recognized. With this need in mind we undertook research in 1978 to develop a new methodology that would rely more heavily on orderly application of geologic data to a carefully conceived model rather than upon the subjective estimates based primarily on the thinking and experience of one or a few experts [1,2].

This paper deals with the progress of research on the first part of the resource estimation process, namely, with the evaluation of favorability for the presence of undiscovered uranium deposits. The evaluation of favorability will be attempted by weighting various geologic attributes according to their frequency of occurrence rather than subjectively by either the model builder or user.

We have chosen a genetic approach to modeling to evaluate favorability. Use of genetic models of some sort is not new to uranium geology, but we believe that our more rigid format relating geologic evidence directly to genetic concepts is an improvement. We clearly separate interpretation from factual data. The genesis is broken down into its component parts that permit variations in genetic ideas to be easily factored into the model. The geologic evidence, which is commonly thought of as a geologic model, is aligned with the corresponding genetic concepts. This system of genetic and geologic models is the basis for questions to be asked to test the genetic processes. These questions are used to construct logic circuits that will in turn allow evaluation of favorability and the generation of favorability maps using geologic decision analysis.

## GEOLOGIC BACKGROUND FOR MODELING

The average uranium content of the Earth is probably similar to that of meteorites, but the uranium content of typical crustal rocks varies widely and nearly all these rocks are considerably enriched in uranium compared to the mantle and core. As an ubiquitous and highly mobile metallic element, uranium has been concentrated into a variety of deposits of widely different forms, rock and metal associations, and grades. Ore deposits, of course, represent local

concentrations much greater than the average crustal rocks and probably contain uranium that has been recycled many times. Every deposit, therefore, represents the culmination of a complex series of events or processes that began with the evolution of the primordial Earth.

Although the location of individual deposits is most closely related to properties of the host rock and the immediate source of the uranium, the location of uranium-rich provinces is probably related to much larger scale features that involve the gross structural and geochemical evolution of the Earth.

Four stages in the Earth's history were particularly critical to the segregation and concentration of uranium and to the types of deposits that could be formed. These were 1) segregation of the sialic crust; 2) the development of life; 3) the development of an oxygen-rich atmosphere; and 4) the development of land plants.

1) Segregation of sialic crust. This part of the crust is variably heterogenous, forms the continental cratons, and is much richer in uranium and other radioactive elements than either the oceanic crust or mantle. Presumably, this uranium enrichment was largely accomplished very early in the Earth's history by a partitioning between the mafic mantle and the more sialic crustal rocks. Most uranium deposits have probably been the result of both physical and chemical reworking of these sialic continental rocks through anatexis and differentiation, on the one hand, and through weathering, sedimentation, and leaching processes, on the other.

Early differentiation of a low-density, uranium-enriched, sialic continental crust has probably played a most important part in the creation of all uranium deposits. This differentiation was the first of a series of preconcentration processes that have probably preceded the deposition of all economic uranium deposits.

2) The development of life. Paradoxically, the development of life on the Earth created two new and opposite environments that have been extremely important in the redistribution of uranium. In mid-Precambrian time, about 2400 million years ago, the so-called oxyatmoverision occurred when life forms had developed that liberated free oxygen, probably by photosynthesis. Shortly after this time, bacterial organisms may have developed that could have reduced sulfate and other sulfur species to  $H_2S$ . The action of these bacteria created environments of highly reducing conditions very much in contrast to the action of the oxygen-producing organisms. Because uranium is highly mobile under oxidizing conditions and is generally immobile under reducing conditions at low temperatures, the importance of this aspect of certain living organisms is evident.

Living organisms also concentrated organic carbon and, when they died, their remains were incorporated in fine-grained marine sediments. Organic-carbon compounds are well-known concentrators of uranium by adsorption, reduction, and chelation. As soon as oxidation liberated uranium from rocks to meteoric solutions in the Precambrian, the uranium began to accumulate in the seas and to be entrapped by organic-rich marine sediments.

3) The development of an oxygen-rich atmosphere. As noted above, the development of an oxygen-rich atmosphere was closely linked to the development of oxygen-liberating living organisms. Once oxygen was available, meteoric surface waters became strongly oxidizing. Weathering, which had previously been controlled largely by physical processes and pH imposed by  $\text{CO}_2$ , now became influenced by oxidation processes. Uranium, which was relatively insoluble under reducing weathering conditions, became readily leachable under oxidizing weathering conditions. Placer concentrations of easily oxidized uranium minerals, such as uraninite, could not form or exist except under unusually rapid transport or cold conditions. Transport of uranium in solution by meteorically derived waters created a whole host of new varieties of uranium deposits.

4) The development of land plants. After the time of the oxyatmoverison and prior to the Devonian Period, when land plants became abundant, most of the uranium liberated by weathering conditions was carried to the sea by surface waters. Few, if any, continental sediments are known to have accumulated supergene uranium during that period of time, probably because there was little organic carbon either to concentrate the uranium directly or to serve as a nutrient for sulfide-reducing bacteria. Nearly all uranium deposits in continental sediments are associated in some way with organic materials and were formed after the Devonian Period.

The significance of this discussion to genetic modeling for favorability for uranium resources is to point out the importance of preconcentration and precursor conditions to most deposits, particularly to those that contain uranium of economic grade.

In the geochemical cycles of uranium in the Earth's development, a first genetic distinction can be made as to whether a deposit is either syngenetic or epigenetic. Syngenetic deposits were formed contemporaneously with the host rock. These uranium deposits commonly are the same shape as the host-rock body. The uranium is fairly uniformly disseminated and generally closely related to the allogenic mineral phases of the host rock. Syngenetic deposits are commonly low grade, such as uraniferous granite and marine black shale, but a few are high grade, such as uraniferous pegmatites. Genetic models of syngenetic deposits are generally simple, as mineralization itself is but a single stage,

and hence the evaluation of favorability for uranium resources is also simple.

Epigenetic deposits were concentrated after the host rock formation. They have many shapes in many geologic environments, have complex mineralogy, and have a wide range in grade. Two and commonly more stages of uranium concentration are evident in epigenetic deposits. Thus, the prediction of occurrence of unfound deposits and projection of mineralized rock into unexplored ground is far more difficult and subject to chance for epigenetic than for syngenetic uranium. More controversy surrounds the genesis of epigenetic deposits than syngenetic ones, and it is fair to state that the genesis of no single epigenetic deposit can be absolutely proven. Thus, the design of genetic models for epigenetic deposits must be flexible to accomodate variable genetic concepts. It is the epigenetic deposit at which genetic modeling is aimed, and these models should prove most useful in evaluating favorability for undiscovered uranium resources.

#### THE GENETIC-GEOLOGIC MODEL CONCEPT

Uranium has been concentrated by various processes into deposits in many different igneous, sedimentary, and metamorphic environments. For a given deposit, or group of like deposits, these processes presented in their chronologic sequence constitute what we mean by a "genetic model". Each process has left behind evidence in the form of observable features in the host rock and its general environs. These geologic features can be woven into a basic geologic story or model ("geologic base"). The genetic model and its geologic base are dependent upon one another, and the two can be molded into an interactive chronological matrix that we call the "genetic-geologic" model. It is this genetic-geologic model that we use as a framework for geologic and related data in order to evaluate favorability of an area for undiscovered uranium deposits.

#### Genetic scheme of process stages

A sequential scheme of chronological stages is formulated here to include the entire genetic history of an uranium province and its contained uranium deposits. The scheme was prompted by discussion of resource modeling by Ruzicka [3]. The eight general stages for a genetic model of uranium deposits are as follows:

- I. Precursor processes
- II. Host-rock formation
- III. Preparation of host rock
- IV. Uranium source development
- V. Transport of uranium
- VI. Primary uranium deposition
- VII. Post-deposition modification
- VIII. Preservation

A special note must be made concerning the source of uranium; it may predate the host-rock formation, and it does not everywhere fit easily into the chronological scheme.

The various stages in the genetic model are intended to be a framework in which to list the events, conditions, and processes that influenced mineralization. A few comments on each stage will clarify the kinds of concepts that are intended under each heading. Furthermore, these comments will indicate the kinds of geologic evidences that were used to develop the genetic concepts.

Precursor processes--The precursor processes generally produced regional features that describe the geologic history prior to host-rock deposition. These processes may extend back to the Earth's formation when certain parts of the crust became more rich in uranium than other parts--in other words, the creation of an uranium-rich province. Such a uranium province may have been modified later by various geologic processes during which some uranium deposits could have been formed. For many types of deposits, Precambrian shield areas or orogenic belts are important; for others, forelands are more favorable than geosynclines; and continental basement rocks are more likely to be uraniferous than oceanic rocks, except for carbonaceous shale and phosphate. The precursor processes may also be represented by small-scale features and more closely related in time to the host-rock formation, such as in the development of intermontane basins, caldera centers, and plutons.

Some precursor products may have provided a later source for the uranium. For example, deep-seated igneous activity may have produced a labile uranium source rock that was later to become a provenance for sediments, or to be exposed to supergene weathering. Ancient uraniferous marine shales have in some places acted as protore for later metamorphic uranium deposits.

The usefulness of precursor events may be limited in favorability evaluation of some areas, particularly the extension of a known uranium belt into deeper parts of the same basin.

Host-rock formation--The history of the host-rock formation bears closely on the uranium concentration process. The initial host-rock components--both reactive and inert chemicals, initial porosity and permeability, and relative stratigraphic position and geologic age are important attributes to consider. Certain rock types, because of their inherent genesis, are preferred hosts for uranium deposits as witnessed by the naming of uranium deposit types by host-rock name.

For certain types of sedimentary host rocks, for example sandstone, this stage might best be divided into three substages: 1) source of sediments, 2) transport of sediments, and 3) deposition of sediments. For igneous and volcanic host rocks, one or more magmatic stages and possibly an accompanying sedimentary stage may be required to model the uranium deposits.

Host-rock preparation--Preparation of the host rock may be the most important stage in epigenetic mineralization processes. In some host rocks, preparation begins during rock deposition or soon thereafter; in others it is much later, and there may be several stages. Diagenesis may play an important role in the preparation of sedimentary and volcanic rocks. Tectonism is important not only in some sedimentary rocks, but is vitally important in brittle igneous and metamorphic rocks that are host for fissure-vein deposits. The absence of tectonism can also be critical. Weathering, thermal activity, and metamorphism are important in soluble rocks to create void space for sites of mineralization such as replacement of limestone or dolomite by chert. Alteration that precedes introduction of uranium-bearing solution is part of rock preparation. Hence, the preparation of host rock consists of both chemical and physical changes.

Uranium-source development--As noted above, the potential source of uranium may develop before host-rock deposition, as in uranium-rich granite plutons and pelitic rocks. Other sources may develop during sedimentation of the host rock, such as contemporaneous volcanism. In still others, an even later development may provide a viable source for uranium, once again commonly volcanic in nature.

Transport of uranium--For most deposits a system is necessary to transport uranium and accompanying metals and other chemicals from their source or sources to where they accumulate. But evidence for the system and transporting fluids is most commonly faint if detectable at all. Most important to uranium movement are hydrological systems, but unfortunately too little is known about the paleohydrology of the environment of uranium deposition. Nevertheless, knowledge of present groundwater conditions, recharge points of both present and possibly past systems, a possible hydraulic-gradient condition (dip), potential conduits (location of aquavoids, faults, joints), and discharge points to the system are important pieces of evidence to list. Timing is also important; there must have been communication between the proposed source and the present site of the uranium deposits at the proper time.

Primary uranium deposition--This may be the next stage after the development (or the availability) of a source--disseminated, protore, or magma--and in many types of deposits, it is the second stage of enrichment beyond crustal abundance. Uranium minerals are

deposited by many processes, including adsorption and absorption; reduction by organic matter, gases, or sulfides; evaporation; or temperature and pressure changes in igneous and metamorphic systems. Evidence of alteration related to uranium mineralization that is more pervasive and widespread than the ore body, such as in a roll-front, is an attribute of great importance. Recognition of the mineralization processes for favorability models may not be as useful as premineralization processes because mineralization processes were commonly too localized. From the viewpoint of understanding the geochemistry of ore deposits themselves, however, genetic modeling of mineralization will be important.

Some primary uranium ores are closely related to the source and deposition of other metals, such as iron, vanadium, gold, molybdenum, selenium, chromium, nickel, and thorium, and of other substances, such as organic materials (humate). Thus, the depositional history of these materials may be important to include in the model, particularly if their presence as geochemical halos extends far beyond the known uranium deposits.

Post-deposition modification--Post-primary deposition modification can either increase or decrease the uranium grade. Some deposits have been completely destroyed, and the uranium either transported to the ocean or redeposited elsewhere, commonly as a different type of deposit. For example, some humate-related tabular deposits in the Westwater Canyon Member of the Morrison Formation (Upper Jurassic) in the San Juan Basin were partly to completely destroyed and redeposited in and adjacent to faults (stacked ore) and perhaps as a later event into roll-front deposits. Thus, there may be a need to divide this stage into two or more substages. Supergene enrichment has been proposed as a process important in some deposits. Modification related to the present-day conditions is a part of the preservation stage.

Preservation--Preservation of the deposits is essential. It is dependent upon protection by stable overburden conditions, favorable climatic conditions, the erosional cycle, groundwater conditions, and time. Some otherwise favorable ground may be unfavorable because of failure of preservation.

The geologic base and its interaction with genetic concepts

The conceptual genetic model is generated from observations of the geologic setting of a given type or group of uranium deposits with common characteristics. The development of a genetic model most logically goes from field and related laboratory analyses to the conceptual ideas on the genesis, but because of our basic



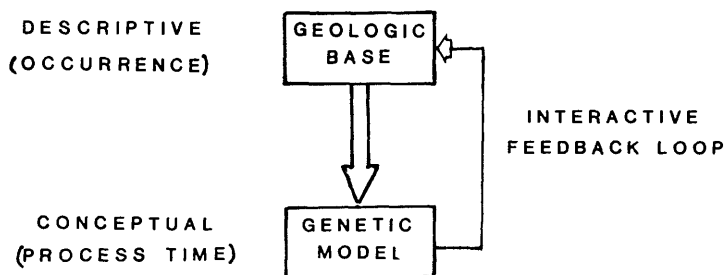


FIG.1. Relation of geological base to genetic model.

knowledge of the science of uranium geology, some genetic processes can be inferred without any field evidence. The evidence may not yet have been observed. In some processes, the evidence may have been destroyed, or be far removed from the site of uranium occurrence, but nevertheless the process must have taken place. Fundamentally, the geologic occurrence base is described first and the genetic model follows, but during the development of the genetic model an interactive feedback loop is established between the two, as diagrammed in Figure 1. This points out a strength of our system of modeling in that it forces the field geologist to think about the entire process of mineralization without leaving out a critical part.

The geologic occurrence base consists largely of the three-dimensional empirical relationships of the known deposits to their surroundings. The fourth dimension, time, is introduced by listing the geologic occurrence data in the chronological order of the genetic model. Some geologic data do, however, have a bearing on time relationships; both field evidence for timing of geologic events and laboratory evidence of mineral age are important to developing a more accurate genetic model.

## MODEL BUILDING

Ideally, a genetic-geologic model represents a distinct class of deposits that have had similar genetic histories. In practice, however, because of significant local variations and the uncertainty of genesis, a model is easier to construct for a regional representative of a class. Indeed, a model may be made for a single deposit, but the principle is the same for two or more like deposits. When we have constructed enough models for representative deposits, we may be able to generate a model for a class of deposits.

TABLE I. GENETIC-GEOLOGIC MODEL AND APPLICATION FORMAT

Stages	Genetic conceptual model	Geologic occurrence base	Application questions
I. Precursor processes,...	x	x	x
II. Host-rock formation <sup>a/</sup> ...	x	x	x
III. Preparation of host rock.....	x	x	x
IV. Uranium source development.....	x	x	x
V. Transportation of uranium.....	x	x	x
VI. Primary uranium deposition.....	x	x	x
VII. Post-deposition modification <sup>1/</sup> .....	x	x	x
VIII. Preservation.....	x	x	x

<sup>a/</sup>These stages may be subdivided where desirable.

Genetic-geologic uranium-deposit models are built of pertinent geologic data tied together by conceptual interpretations of how that data can be related in time to processes of uranium deposition. The available data concerning the geologic setting of the specific deposit or deposits are assembled in the eight-step chronologic order outlined above; this part of the model is designated the geologic occurrence base, which becomes part of the matrix format shown in Table I. From the geologic occurrence base, a more generalized process-oriented genetic conceptual model is constructed as a second part of the matrix. A uniform grammatical style is required for these two parts of the matrix that makes the model easy to construct. The geologic base should be described in the present tense because it concerns observations that can be made today. The genetic model, on the other hand, should be stated in the past tense for it describes things that have taken place in the past. In assembling a model from the geologic base, it will commonly become evident that the genetic history is incomplete. This, in turn, suggests that additional geologic data are required or that existing data have not been adequately interpreted. In this manner, a feedback loop is maintained between the two parts of the model, which can result in constant improvement of the model as new data and improved interpretations become available. The kinds of new data needed should be listed in the form of an epilogue.

With today's state of knowledge, even the best models that can be devised have some stages or processes that are controversial or speculative. If there is more than one explanation for certain data, such explanations may be left as alternatives in the model to be tested. Some important aspects may be omitted because they have not been recognized; others may be misinterpreted for various reasons. In best models that can be devised, therefore, there may be both omissions and imperfections. In addition, extraneous geologic-base observations may be included that can be neither related to nor disassociated from the genesis of deposits. They should remain as parts of the model, however, because it would be presumptuous to omit them without adequate justification.

In the actual writing of the model, the outline form is used to enter the material related to each stage. The statements should be brief full sentences that describe the essence of the ideas and evidence. Within each stage the statements should be placed in a chronological order, if possible, from the oldest to youngest event or process. An abbreviated example is as follows:

#### Genetic Conceptual Model

- I. Precursor processes
  - A. A uranium province developed in the area.
  - B. Extended marine and continental deposition occurred on stable platform.
  - C. Host-rock deposition was preceded by uplift marginal to downwarping that formed a continental basin.

#### Geologic-Occurrence Base

- I. Precursor evidence
  - A1-Basement rocks are uranium-rich.
  - A2-Uranium deposits are in older rocks.
  - A3-Basement rocks contain uranium-rich zircon.
  - A4-Isotope studies show uranium loss in basement.
  - B1-Underlying strata consist of both marine and continental rocks.
  - B2-Regional dip of both host and underlying rocks is low.
  - B3-Pre-host rock faulting is minor.
  - C1-Depositional environment of host indicates renewed marginal uplift.
  - C2-Distribution of host coincides with present basin.

Once a heading is established it should be carried to the right with appropriate corresponding statements in the next column. Generally a single genetic concept requires more than one piece of evidence and, as illustrated below, more than one question may be required to test the presence or absence of a piece of evidence. Where appropriate, some parts of a stage--either in the genetic model or geologic base--may require justification or explanation that would clutter the model. Thus, these explanatory statements should be listed in an appendix to the model and identified by the corresponding outline letter-number code such as IA2. This will also facilitate recall of explanatory notes if the model is computerized.

#### FAVORABILITY EVALUATION

Once constructed, a genetic-geologic model may be used in various ways to evaluate favorability. Most simply, it can be used to evaluate a region in a purely subjective fashion by deciding how well the region fits the model. One could go farther and subjectively assign a relative weight to each part of the model and determine a score relative to a control area. There is, in fact, a formal system called "Prospector" [4] that uses such a weighting scheme. As part of our research, we have designed a method that is somewhat similar to but different than Prospector. The method is based on a logical framework of questions that relate the factors which compose each genetic-geologic model.

#### Questionnaire and the logic circuit

The genetic-geologic model is proposed as a tool to be used to assess the favorability of a given area to contain uranium deposits of a certain grade and size--in essence, a method for estimating undiscovered uranium resources. To do this, we have devised a system of questions that correspond to the geologic base and conceptual genetic model--the third column in the matrix shown in Table I. The answers to these questions provide a way of comparing a test area with a control or model area. We have decided to pose questions in a uniform manner. The questions are asked so that they may be answered in the positive (+1), in the negative (-1), or as don't know (0). Questions that need to be answered by specifying some numerical quantity must be rephrased to conform with the above scheme.

An example of some questions asked that relate to the model described above is as follows:

##### I. Precursor questions

A1-Are basement rocks uranium rich?

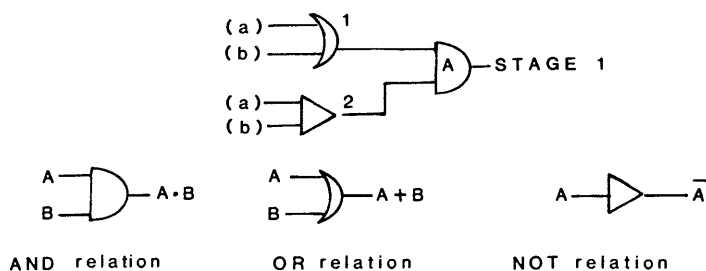


FIG.2. General logic circuit elements and an example of a general circuit.

- A2-Are there uranium deposits in older rocks?  
 A3-Do regional basement rocks contain uranium-rich zircon?  
 A4-Do isotope analyses of basement rocks show uranium loss?  
 B1-Does underlying sequence contain marine and continental rocks?  
 B2a-Is the dip of the underlying strata  $<5^\circ$ ?  
 B2b-Is the dip of the host rocks  $<5^\circ$ ?  
 B3-Is pre-host faulting minor?  
 C1-Are the host rocks of higher energy environment than that of underlying rocks?  
 C2-Does the distribution of host coincide with present basin?

The geologic attributes and their corresponding questions from a model, such as exemplified above, can be classified as 1) strictly necessary, 2) sufficient, or 3) questionable. Those questions for attributes necessary for the presence of a stage are multiplicative and are expressed in the generalized circuit by an "AND" relation (Fig. 2). Those questions for attributes sufficient for the presence of a stage are additive and are expressed by an "OR" relation. The "NOT" relation is used to include relations for which absence is either a necessary or sufficient attribute. As an alternative a question can be asked in the negative, thus avoiding the use of the "NOT" relation. Attributes that are questionable as to whether they are either necessary or sufficient are usually left out of the circuit.

Applying this kind of logic, we can construct a provisional logic circuit for the questions given above (Fig. 3). In determining a favorable precursor condition in this example, the questions are answered using available data. In the "uranium province" and "basin development" parts, which are "OR" relations, only one positive answer is required to establish their favorability. In the "stable platform" part, which is an "AND" relation, a negative answer to either question indicates an unfavorable platform condition.

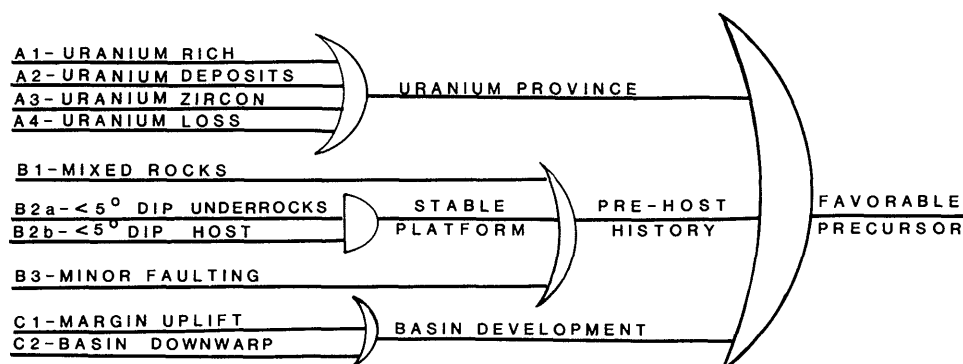


FIG. 3. A provisional logic circuit example.

### Application to a test area

The next step in our research is to apply a model to a test area using logic circuits in a geologic decision analysis. We have chosen the model being developed for humate-related tabular deposits in the San Juan Basin in New Mexico for this purpose. In order to examine the entire basin in some detail, it has been gridded into cells 4 km on a side. The data needed to answer the questions for each cell are being computerized. A draft of the questions and the logic circuit is being reviewed. The first test phase will consist of determining the relative weights for the various stages that compose the model. The weights will be calculated by making use of a control area. Upon completion of this phase, the unexplored part of the basin will be evaluated. Favorability maps will be generated for each process stage and for the composite of all eight stages, which, when appropriately weighted will constitute the overall favorability.

### CONCLUDING REMARKS

We have attempted to show that the genetic aspect of the model is interactive with the geologic evidence that supports it. Different models can also be interactive with one another in the sense that the process stages of the formation of a deposit in one environment may in part be similar to those of a deposit in a different environment. Study of the complex similar and dissimilar relationships between different models aided by the computer may result in the definition of environments not yet examined for uranium occurrences. Eventually, we hope that genetic-geologic modeling will also lead to an acceptable genetic classification of uranium deposits.

## ACKNOWLEDGEMENTS

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## DISCUSSION

M.V. HANSEN: Has the concept you describe actually been applied in practice yet, or is it still very much at the research stage?

W.I. FINCH: The concept is now being applied to the San Juan Basin. We chose this area because we have extremely abundant data on it for developing methods of application. Areas for which less data are available will present a problem, but not more than with any other method.

M.V. HANSEN: Will the characteristic analysis routine be used in this approach?

W.I. FINCH: Yes, but that system is being modified to introduce new improvements. Furthermore, probabilities of uranium occurrence and the use of grade-tonnage relationships will be incorporated for the first time. It should be noted that the presence of anomalous uranium has not been used as a factor in determining favourability, although in past and contemporary methods for evaluating favourability the presence of uranium has been a dominant factor and almost the ultimate

criterion of favourability. We have decided to use knowledge of uranium occurrences obtained, for example, by airborne radiometric surveys performed in order to detect anomalies and by prospecting and mining for estimating the initial probability of discovering economic quantities of uranium. It is conceivable that anomalous uranium might be found in environments unfavourable for economic deposits of uranium in large concentrations.

A.E. BELLUCO: Do you add up the values corresponding to each of the parameters which make up each genetic model? If so, when does the sum indicate geological favourability in each of the cells you analyse? Is the same mathematical weighting given to each of the parameters which make up the 100 points in the geological questionnaire for each cell?

W.I. FINCH: For the present we do not plan to add the values for each parameter. In the initial phase we will use the scheme of "OR" and "AND" relations described in the paper, whereby each cell will receive a value of (+1), (-1) or (0) for each attribute. A cell with a resultant (+1) value will be favourable, a (-1) unfavourable, and (0) will indicate that favourability is unknown. We do plan to carry out an experiment in the Grants Mineral belt – the control area – to develop a weighting scheme that will relate the frequency of occurrence of each parameter relative to the presence or absence of ore and to the grade-tonnage values of known ore bodies. We will probably find that only part, perhaps 10%, of the 100 attributes are needed to evaluate favourability.

A. HATTON: Many different models will be required for purposes of comparison. Who will construct the original models and, when constructed, will they be generally available?

W.I. FINCH: We now have in preparation over 20 separate models covering most known types of economic deposit. These will be published as soon as they have been completed by the United States Geological Survey.



# EL POTENCIAL URANIFERO DEL CRETACICO CONTINENTAL EN LA PATAGONIA EXTRANDINA DE LA REPUBLICA ARGENTINA

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## Abstract-Resumen

### THE URANIUM POTENTIAL OF THE CONTINENTAL CRETACEOUS OF PATAGONIA.

The geological features of the fluvial sedimentation of the Cretaceous of Patagonia – the Grupo Chubut Formation in the provinces of Chubut and Santa Cruz, underlined this environment as one of high favourability for discovering uranium deposits. The area has been explored by the CNEA who found two outstanding levels of fluvial sediments that are today the target of further studies to find new deposits. The general geological features are given, together with the results of the exploration up to date, distribution of radiometric anomalies, deposits discovered and an estimation of the uraniferous potential of the Formation.

### EL POTENCIAL URANIFERO DEL CRETACICO CONTINENTAL EN LA PATAGONIA EXTRANDINA DE LA REPUBLICA ARGENTINA.

Las características geológicas de la sedimentación fluvial del Cretácico en la Patagonia extrandina – Formación Grupo Chubut de las Provincias del Chubut y Santa Cruz, han encuadrado este ambiente como de alta favorabilidad para el hallazgo de depósitos uraníferos. El área ha sido explorado por la CNEA determinándose dos niveles preferenciales de sedimentación fluvial bien distintivos que actualmente son objeto de estudios tendientes al hallazgo de cuerpos mineralizados. Se indican las características geológicas generales de la Formación, resumiéndose los resultados de la exploración realizada hasta el presente, distribución de anomalías radimétricas, depósitos hallados y estimación del potencial uranífero de la misma.

## 1. INTRODUCCION

Desde el año 1959, la Comisión Nacional de Energía Atómica de la República Argentina explora la Patagonia en busca de depósitos uraníferos, quedando rápidamente evidenciada en los estudios iniciales de dicha exploración la importancia del área abarcada por los afloramientos de la sedimentación correspondiente a la cuenca cretácica, continental, fluvial y/o lacustre, con gran participación de depósitos piroclásticos ácidos a mesosilícicos, que constituyen el llamado Grupo del Chubut.

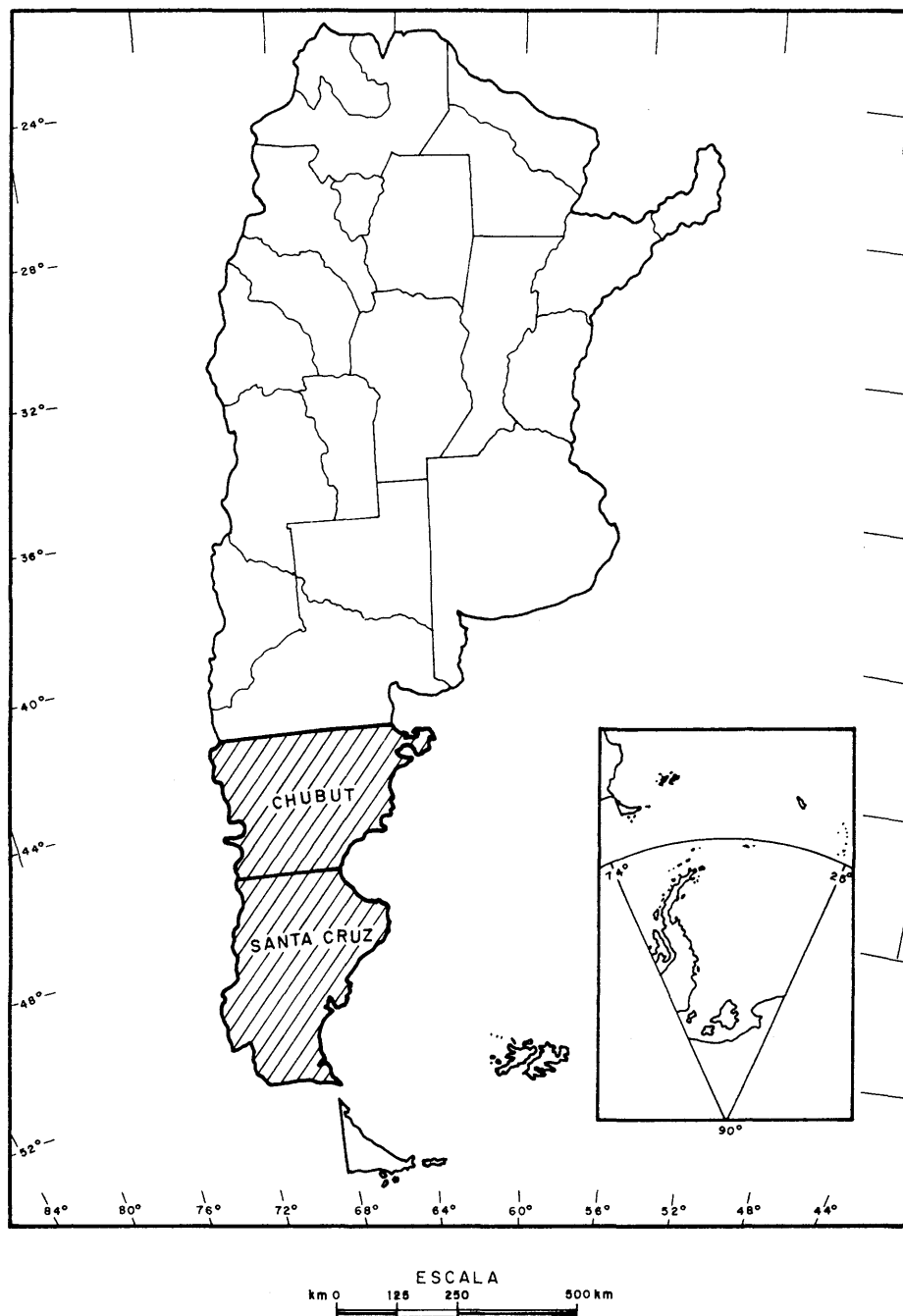


FIG.1. Mapa de ubicación de las provincias del Chubut y Santa Cruz.

Esta formación se extiende predominantemente por la provincia patagónica argentina del Chubut, donde sus afloramientos cubren 18 700 km<sup>2</sup>. En la provincia de Santa Cruz aflora aproximadamente 3450 km<sup>2</sup> ocupando así una amplia porción de la llamada Patagonia extrandina.

Las provincias del Chubut y Santa Cruz (224 686 y 243 943 km<sup>2</sup> respectivamente – Fig.1) han sido exploradas mediante prospección aérea radimétrica de contaje total en vuelos de prospección regular (con separación de 1 km) cubriéndose 51 340 km<sup>2</sup> en la primera y 24 376 km<sup>2</sup> en la segunda, y está en curso de elaboración la cobertura de 100 000 km<sup>2</sup> de prospección radimétrica aérea con discriminación, sobre el área que abarca la casi totalidad de los afloramientos de la Formación Grupo Chubut.

En el Cuadro I se indica la distribución estratigráfica de las mineralizaciones uraníferas halladas y en la Fig.2 la distribución cuantitativa de anomalías de uranio según unidades formacionales; este gráfico sintetiza los resultados obtenidos e indica la preeminencia de la formación aquí analizada en cuanto a posibilidades uraníferas siendo bien evidente su importancia con respecto a las restantes unidades estratigráficas consideradas en la Patagonia.

El análisis, en particular, de las anomalías halladas determinó que dentro del Grupo del Chubut, las mismas se distribuían en 2 posiciones estratigráficas bien definidas dentro del Cretácico continental, en la base y en el techo, referidas a sedimentos de carácter fluvial, delineándose así dos zonas tipo, de sedimentación cretácica con mineralización uranífera, la de la Sierra Pichiñán para la sección basal y Sierra Cuadrada para la cuspidal.

## 2. GEOLOGIA GENERAL

La sedimentación cretácica comienza con un conglomerado fluvial de alta energía que rellena las partes bajas del paleo-relieve elaborado sobre el sustrato pre-cretácico integrado tanto por formaciones paleozoicas como mesozoicas.

Estos conglomerados y areniscas fluviales han sido denominados en distintas partes de la cuenca cretácica como serie Matasiete, Formación Los Adobes [1], miembro inferior de la Formación C°Fortin [2] y por último miembro Arroyo del Pajarito de la Formación Gorro Frigio [3].

Se trata de sedimentitas psamo-psefíticas, con intercalación de lentes pelíticas, que exhiben estructura de sedimentación fluvial de alta energía, estratificación diagonal entrecruzada, corte y relleno, etc., cemento calcáreo, color pardo amarillento en los sectores aflorantes, restos de troncos silicificados y, en las lentes finas, restos de materia orgánica carbonizada.

Localmente para la Sierra de Pichiñán esta sección ha sido designada miembro “A” de la Formación Los Adobes.

CUADRO I. CUADRO ESTRATIGRAFICO SINTETICO DE AMBIENTE PATAGONIA Y POSICION DE LOS DEPOSITOS URANIFEROS

E D A D E S		NOMENCLATURA REGIONAL	LITOLOGIA Y FACIES	MANIFESTACIONES DE URANIO
CUARTARIO		VARIAS FORMACIONES	Depósitos continentales Niveles de pie de monte Vulcanismo básico	Minerales amarillos en sedimentos de cuen- cas s/desague en Sierra Cuadrada (Chubut).
T E R C I A R I O	PLIOCENO	RIONEGRENSE	Sedimentos continentales	
	MIOCENO	SANTACRUCENSE COLLONCURENSE	Sed. continent.areniscas blanquecinas y tobas fria- bles.	Minerales amarillos en areniscas asociados con lignitos en Valle de las Pinturas (San- ta Cruz)
	OLIGOCENO	PATAGONIANO	Sed.marinas (aren. y calcá- reos organógenos) y conti- nent.(areniscas, arcillas y tobas con niveles carbonos- os en el NW.	
	EOCENO	TOBAS DE SARMIENTO	Tobas blancas	
	PALEOCENO	RIO CHICO	Sed. contin. areniscas y ar- cillas rosadas, con niveles asfaltíferos y plantíferos.	Minerales amarillos en areniscas asociados c/restos carbon. en Pto Visser y con asfal- to en Krueger (Chubut).
SALAMANCA-ROCA		Sed. marinas,areniscas,arci- llas y limos calcáreos fosi- líf.	Minerales amarillos asociados c/asfalto en Cañadón Gato, Chubut.	
CRETACICO	SENONIANO	LEFIPAN P°DEL SAPO	Sed. marinas y continentales	
	MEDIO A INFERIOR	GRUPO DEL CHUBUT	Sed. continent.,conglomera- dos, areniscas y arcillas / con intercalaciones de to- bas y frecuentes variaciones faciales.	Minerales amarillos y negros en conglomer- ados y areniscas finas carbonosas en Dist. Los Adobes y Pichiñan Oeste (Chubut) en a- reniscas y arcillas (Sierra Cuadrada)e im- pregnaciones de baja ley en tobas (Laguna Palacios).
JURASICO	SUPERIOR	CAÑADON ASFALTO	Tobas, areniscas y lutitas amarillas y pardas.	Minerales amarillos en lutitas asfaltíferas de Cañadón Sauzal, Chubut.
	MEDIO	GRUPO LONCO-TRAPIAL	Pórfiros y porfiritas y to- bas asociadas.	

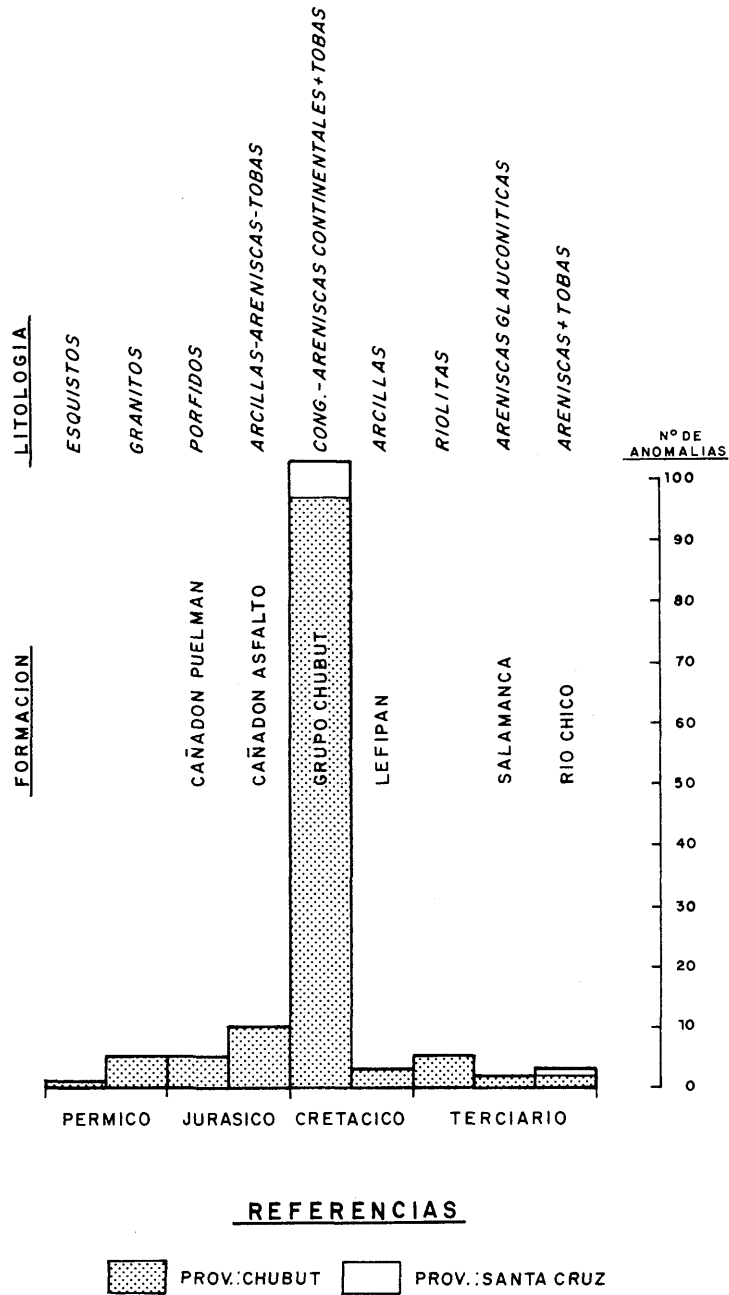


FIG.2. Distribución de anomalías en las provincias del Chubut y Santa Cruz según ambientes geológicos.

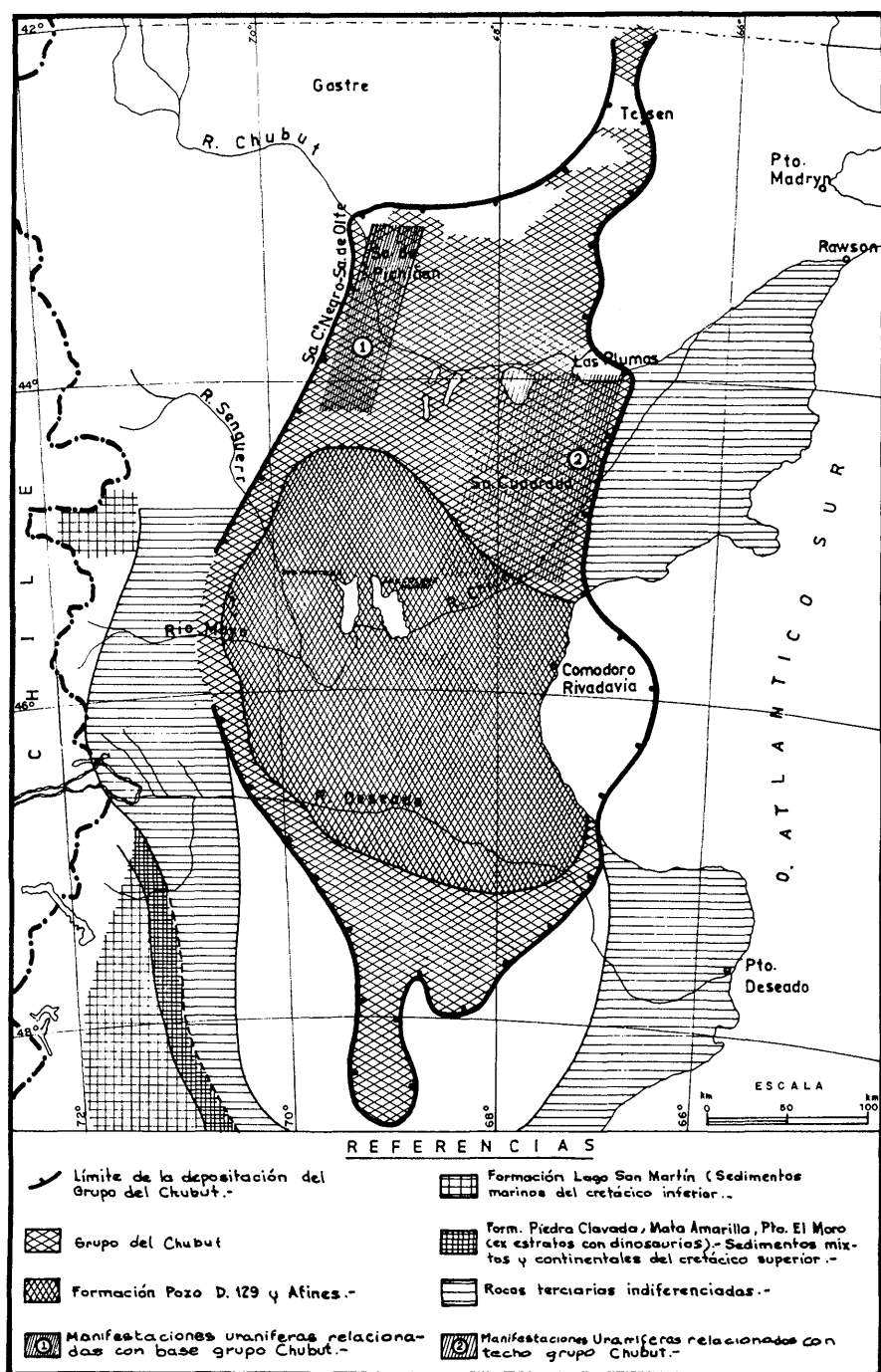


FIG.3. Distribución del Grupo Chubut. Bosquejo "Worm's Eye" sobre efusivas mesojurásicas.

Transicionalmente pasa a los términos superiores (miembro “B” de la Formación Los Adobes), representados por limos tobáceos de color rojo que presentan algunas intercalaciones areno-conglomerádicas de color pardo amarillento, cemento calcáreo y estructuras fluviales de mediana a alta energía. Equivale al miembro Bardas Coloradas de la Formación Gorro Frigio [3].

Siguen en aparente relación de concordancia, areniscas, limos tobáceos, tufitas, delgados niveles de calcarenitas y rocas silíceas de grano fino de color verde claro a gris verdoso que exhiben una sedimentación de carácter lagunar (miembro “C” de la Formación Los Adobes) equivalente a serie del Castillo o sección inf. del miembro C°Barcino de la Formación Gorro Frigio.

En forma transicional se sobrepone a los niveles anteriores una potente serie de tobas y tufitas de color castaño a rojizo con algunas intercalaciones gris-blancuecinas rematando en su porción superior con areniscas conglomerádicas de color rosado a marrón grisáceo con estratificación fluvial de mediana a alta energía que incluyen troncos silicificados de gran tamaño y cemento calcáreo (miembro “D” Formación Los Adobes), equivalentes a serie del Bajo Barreal, a sección superior miembro C°Barcino de la Formación Gorro Frigio y a la Formación Puesto Manuel Arce [3] en la zona de Sierra Cuadrada.

Todos los elementos incluidos dentro del Grupo Chubut [4] se encuentran limitados en su base por la última fase de los movimientos intermálmicos (Valanginiano) y en su techo por los interseñonianos por lo que se le atribuye una edad cretácica inferior a media.

En general el Grupo Chubut se depositó en los bajos del relieve pre-cretácico, mayormente constituido por el sustrato volcánico del jurásico medio, mediante el depósito de sedimentos fluviales de alta energía; a medida que se produce la colmatación de la cuenca, la sedimentación pierde energía pasando a una etapa de planicie aluvial (miembro “B”) y de allí a otra francamente lagunar (miembro “C”). La sedimentación continental continúa hasta rematar en un conglomerado fluvial de morfología mantiforme correspondiente a una depositación plana (miembro “D”).

El ciclo fluvial inferior (Formación Los Adobes) desarrolla aparatos fluviales de grandes dimensiones de hasta 15–16 km de ancho con espesores de hasta 100 m de depósitos clásticos, en sucesivas canalizaciones superpuestas interdigitadas y de escaso confinamiento vertical, sobre todo en la base.

El ciclo fluvial superior de planicie de inundación posee una gran distribución areal de 50 a 70 km de ancho, espesor reducido, hasta 30 m de alternancia de sedimentitas psamo-psefíticas y pelíticas, con canalizaciones definidas y muy buen confinamiento vertical (piso, techo) y lateral debido a cambios faciales.

La mineralización uranífera está asociada fundamentalmente a ambos ciclos fluviales, el basal (Formación Los Adobes) y el cuspidal (Formación Puesto M. Arce) (Fig.3). Siendo, como se dijo, el área tipo de la primera la Sierra de Pichinán y de la segunda Sierra Cuadrada.

El desarrollo de la exploración se ha centrado primordialmente en el área de la Sierra de Pichinán donde se ha ejecutado una cobertura geoquímica general de 4400 km<sup>2</sup> y desde 1968 se han realizado 42 000 m de sondeos.

En 1978 se inició otra vez la exploración del área de Sierra Cuadrada y, desde 1965, se llevan realizados 3000 m de sondeos en el cretácico alto.

Por consiguiente, la experiencia reunida por los geólogos de la CNEA se basa fundamentalmente en las condiciones geológicas mineras del cretácico basal explorado en la Sierra de Pichinán.

### 3. LOS SEDIMENTOS MINERALIZADOS

#### 3.1. Ciclo fluvial inferior (Formación Los Adobes, miembros "A" y "B")

En el flanco de la Sierra Pichinán ha sido posible establecer dos condiciones bien definidas de las sedimentitas fluviales:

- las sedimentitas reducidas con coloración gris a gris verdosa;  
pirita;  
materia carbonosa no oxidada;  
mayor grado de cementación;
- sedimentos oxidados con color ocre amarillento a ocre rojizo;  
ausencia de pirita;  
óxidos e hidróxidos de Fe;  
menor grado de cementación.

El pasaje de las sedimentitas oxidadas a las reducidas es vertical y horizontal.

Las fases verticales son en general periclinales a los afloramientos del relieve pre-cretácico.

Las fases horizontales son groseramente paralelas a la expresión topográfica.

Estamos en presencia de un área mineralizada en ambiente reducido en vías de removilización por descenso del nivel de base local.

En esta zona se ha hallado el depósito de Los Adobes con 130 t de U<sub>3</sub>O<sub>8</sub>, de 100 m de largo por 75 m de ancho y 5 m de espesor medio totalmente oxidado (uranofano, schroenkingerita y fosfuranilita), preservado de la erosión por una arcilla bentonítica superior, un limo de base y una falla frontal y ubicado sobre la margen derecha del aparato fluvial. Sobre el mismo borde se ha hallado otro pequeño cuerpo oxidado denominado C°Cóndor, de una superficie de 4 ha y con reservas de 107 t de U<sub>3</sub>O<sub>8</sub> y ley 0.57% U<sub>3</sub>O<sub>8</sub>, preservado entre la discordancia Cretácico-Jurásico y una falla mediante la cual el cuerpo ascendió poniéndose fuera del alcance de la removilización.



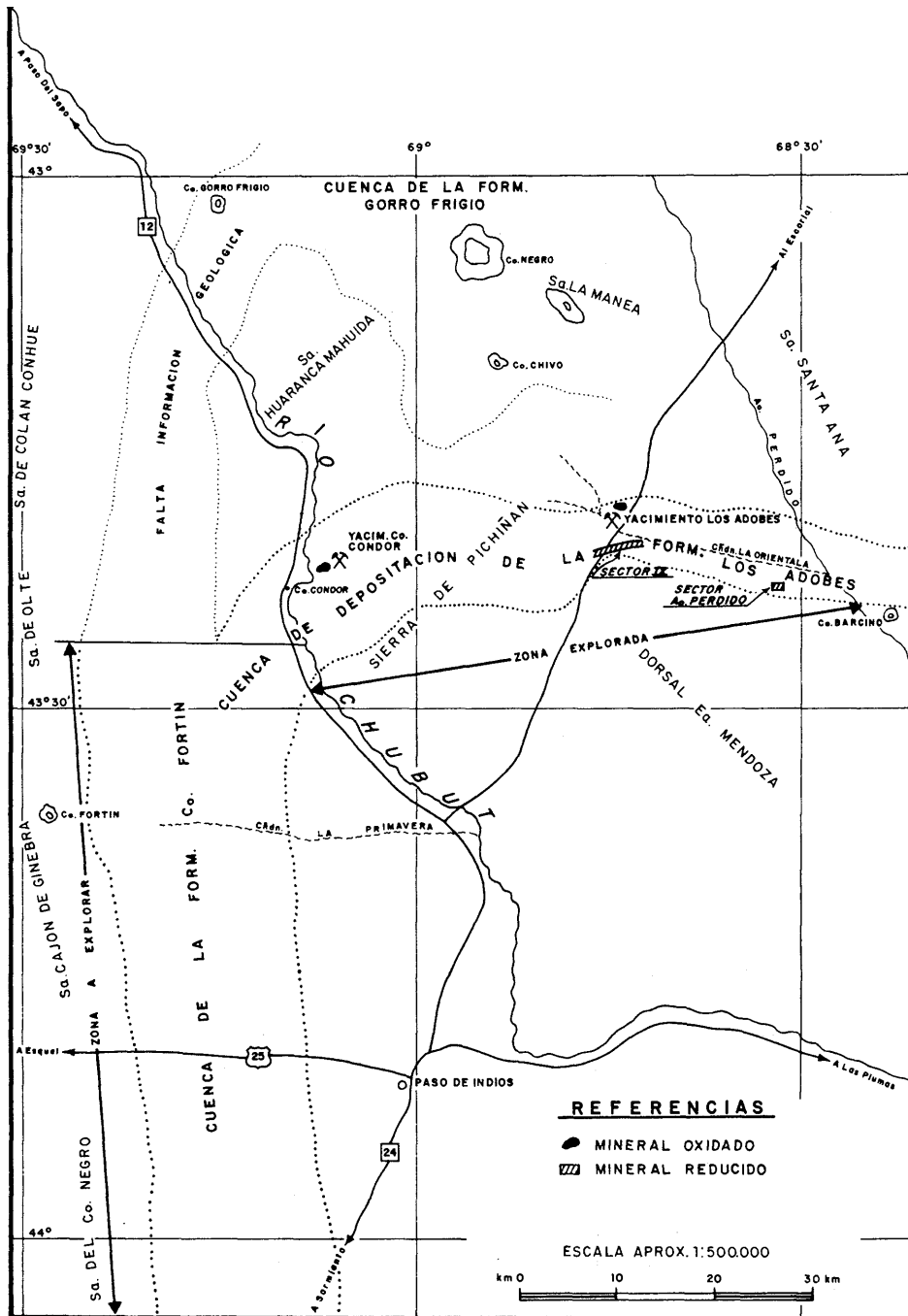


FIG.4. Desarrollo cuencas fluviales sincrónicas con la Formación Los Adobes.

Se encontró el resto de la mineralización en íntima asociación con materia orgánica y en ambientes reducidos. La faja mineralizada actualmente en vías de exploración sobre la margen izquierda del aparato fluvial está en sectores reducidos entre 65 y 125 m de profundidad, habiéndose determinado hasta ahora 50 t sobre una superficie explorada de 20 ha y ley media 0,55%  $U_3O_8$ ; la mineralización está constituida por compuestos urano-orgánicos (aún no determinados con precisión) y uraninita y escasa participación de vanadio.

La exploración del aparato fluvial se ha extendido 20 km al este con una malla de 1 500 a 2 000 m, detectándose 2 sectores mineralizados en Arroyo Perdido a los que dado el estado incipiente de la exploración realizada se le atribuyen 7 t de  $U_3O_8$  por 0,01 ha de superficie.

Resumiendo, no se ha hallado un frente areal definido de avance del proceso de oxi-reducción, sino islotes en estado reducido, en un ambiente oxidado, con mineralización en los sectores reducidos y en sectores oxidados preservados. Solamente se han hallado cuerpos lenticulares peneconcordantes.

En las interfases de oxi-reducción no se ha determinado ningún aumento sensible de mineralización que permitiera la inferencia de un cuerpo del tipo "roll".

En la Fig.4 se observa en forma esquematizada la disposición del ciclo fluvial cretácico basal tal como se la conoce hasta el presente. Se indican los sectores hallados con mineralización oxidada y en ambientes reducidos.

El aparato fluvial de Los Adobes es el que se ha explorado y ofrece las mejores posibilidades económicas (canalización E-W).

La canalización del C° Gorro Frigio y el canal principal no han sido explorados aún.

### 3.2. Ciclo fluvial superior (Formación Puesto Manuel Arce)

Este ciclo fluvial cuspidal cretácico yace inmediatamente debajo de la discordancia intersenoniana que trajo como consecuencia la invasión marina atlántica del Salamanquense (Terciario inferior). Es conocido desde el año 1958 como uranífero pero su exploración se restringió a los niveles superficiales oxidados en el frente N de Sierra Cuadrada. La exploración del área fue iniciada de nuevo en 1978.

Los sedimentos fluviales se depositaron sobre una gran planicie aluvial de unos 50 km de ancho, formando depósitos mantiformes con ocasionales cursos canalizados, y elevado aporte de materia orgánica (troncos en los lechos de los ríos y materia orgánica finamente dividida en los bordes).

Se ha determinado en el flanco N de Sierra Cuadrada un máximo de 3 ciclos de sedimentación fluvial separados por limos producidos por la alteración de sedimentos tobáceos.

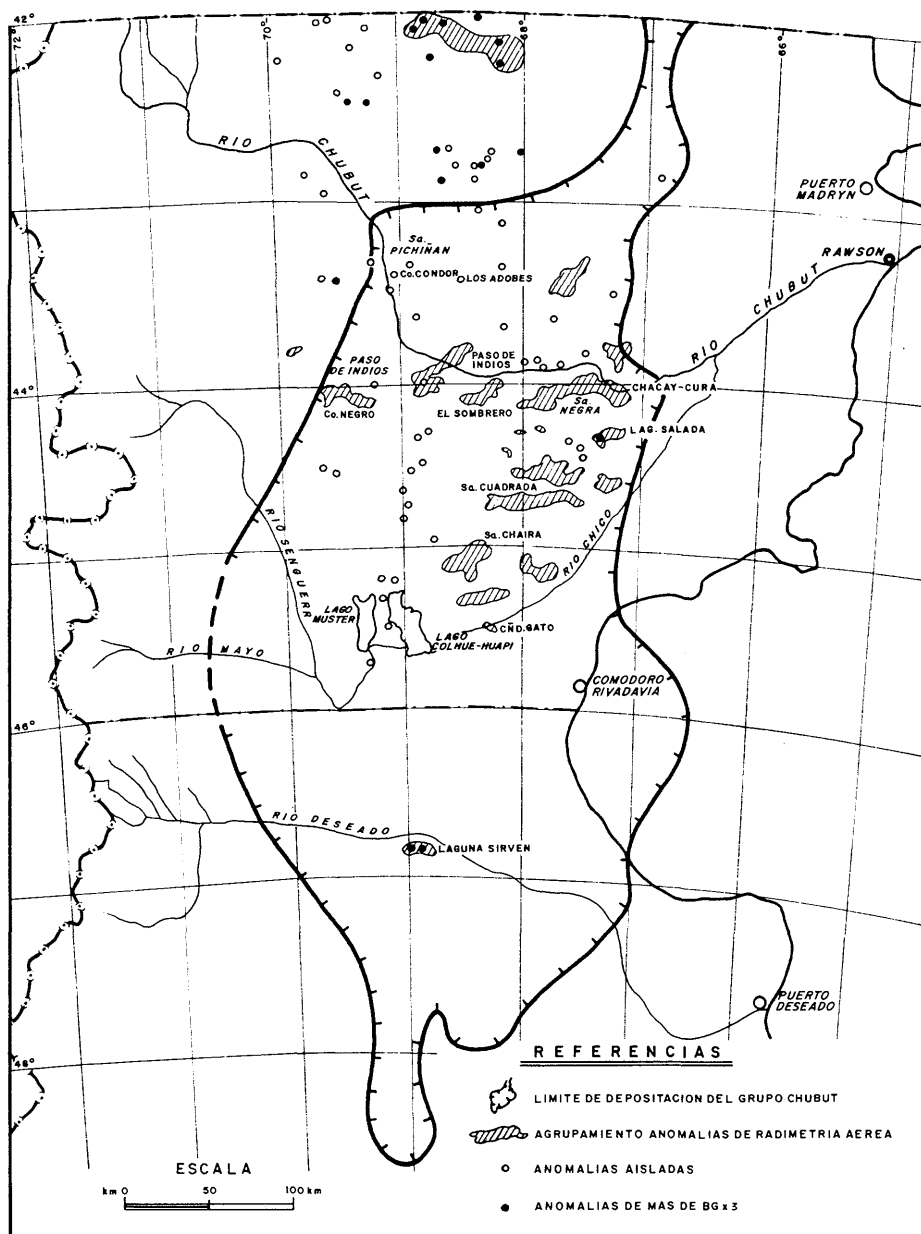


FIG.5. Distribución de anomalías radimétricas aéreas dentro de los límites de deposición del Grupo Chubut.

El ciclo fluvial comenzó depositándose sobre un sustrato tobáceo muy diagenizado hasta el grado de constituir arcillas impermeables que confinan perfectamente la base del mismo.

El primer nivel fluvial es de naturaleza psamo-psefítico y alcanza un espesor máximo de 10 m, es canalizante muy cementado por carbonato en su base y de alta energía, siendo el de mayor importancia por la mineralización presente.

Arrastra grandes troncos que interiormente carbonizados y silicificados retienen en sí la mineralización uranífera y como halo en las areniscas circundantes.

Se han determinado las siguientes especies de mineralización uranífera: carnotita, schroeningerita y antunita con elevada participación de vanadio.

A continuación del depósito de un nivel tobáceo se presenta otro nivel en general psamítico y ocasionalmente un tercero, siendo todo ello culminado por un nivel de pelitas de origen tobáceo de 25 m que confina totalmente el ciclo fluvial de unos 30 m de espesor sobre el que se depositan los sedimentos marinos del Salamanquense (Terciario).

El buzamiento en general es del orden de 1 a 2° al S, constituyendo Sierra Cuadrada presumiblemente un sinclinal suave en el que se depositaron basaltos cuartarios que hoy coronan la Sierra y que constituyeron el elemento que preservó la misma.

El ciclo fluvial, principal aportante de mineralización, es en su base, como ya se dijera, muy cementado con calcita lo que determina su afloramiento sobre grandes áreas como relicto erosivo, configurando un área muy extendida de anomalías radiactivas.

El nivel sigue con dirección S a SO ocultándose bajo la Sierra Chaira y Pampa Pelada y extendiéndose al N hasta algo más allá de la altura de la localidad de las Plumas.

Presenta pues condiciones típicas y favorables para el hallazgo de un frente de oxi-reducción y formación de depósitos tipo "roll".

En el año 1961 se exploró el borde del flanco N de Sierra Cuadrada, determinándose la existencia de cuerpo mineral de 0,3 ha con 14 t de  $U_3O_8$  — 0,70%  $U_3O_8$  de ley media.

En la Fig.5 se ha esquematizado la distribución de las anomalías radiométricas aéreas dentro del ambiente de depositación del Grupo Chubut.

### 3.3. Resumen

En el Cretácico de la Patagonia extrandina se han determinado hasta el presente los siguientes depósitos:

Formación Los Adobes	Yac. Los Adobes	130 t de $U_3O_8$	0,75 ha
	Yac. Los Adobes (Sec.IX)	50 t de $U_3O_8$	19,55 ha
	A°Perdido	7 t de $U_3O_8$	0,01 ha
	C°Cóndor	107 t de $U_3O_8$	4,00 ha
Formación P.M. Arce	Yac. Sierra Cuadrada	14 t de $U_3O_8$	0,57 ha
	TOTAL	308 t de $U_3O_8$	24,88 ha

o sea, como dato de base tenemos 308 t de  $U_3O_8$  en una superficie de 0,25 km<sup>2</sup>.

#### 4. DETERMINACION DEL POTENCIAL URANIFERO

La tentativa de determinación del potencial uranífero de una formación conduce en cierto modo a un salto al vacío por cuanto no existe una metodología de cálculo aceptada y probada suficientemente sobre todo para el caso de áreas con escaso desarrollo de exploración. Por ello se entiende que la fórmula de cálculo puede asentarse fundamentalmente sobre la metodología descrita en el trabajo de la referencia [5], donde:

$$P = D \times S \times F$$

P = Potencial de la formación

D = Densidad de la mineralización del área de control (t de  $U_3O_8$  por km<sup>2</sup>).

S = Superficie por km<sup>2</sup> del área inexplorada considerada favorable dentro de la formación evaluada.

F = Factor de favorabilidad geológica: favorabilidad del área que se evalúa relativa al área explorada basada en la geología, los resultados de la exploración, presencia de mineralización e intensidad y extensión de la radiactividad anómala.

La aplicación de la fórmula en áreas con reducido desarrollo de exploración, insistimos, tropieza con la dificultad de determinar el valor S pues no siempre se ha completado la exploración geológica hasta el punto de conocer cuál es el área inexplorada considerada favorable dentro de la formación, a pesar de que en ocasiones se conozcan los límites de ésta, siendo por lo tanto necesario introducir en la fórmula un factor de corrección basado en los datos estadísticos de países con gran desarrollo exploratorio de uranio tal como los EE.UU. Este factor es el "P" de probabilidad de mineralización.

#### 4.1. Determinación del factor P de probabilidad de mineralización

Para esta determinación se ha recurrido a los interesantes datos estadísticos publicados en 1970 por J.A. Patterson (“Carácter de los recursos uraníferos de los Estados Unidos”) [6] y a los valores de recursos de los EE.UU. en 1977 [7].

El desfase de 7 años entre una y otra publicación no ha podido ser obviado por falta de conocimiento de una posible actualización de la información estadística.

Hasta el año 1977 los recursos razonablemente asegurados (RRA) y los recursos adicionales estimados (RAE) de los EE.UU. eran:

$$1\,995\,000 \text{ t de } \text{U}_3\text{O}_8$$

correspondiendo el 82% de estos recursos a 2 áreas: el “Plateau del Colorado” (40,5%) y la cuenca de Wyoming (41,5%) de geología similar a la del Grupo Chubut.

De los mapas publicados sobre el área abarcada por cada una de las zonas llegamos a:

Colorado	—	355 000 km <sup>2</sup>	—	807 975 t U <sub>3</sub> O <sub>8</sub>
Wyoming	—	174 000 km <sup>2</sup>	—	827 925 t U <sub>3</sub> O <sub>8</sub>

Según Patterson [6] para el mineral de 10 a 15 dóls./lb el espesor medio era de 2,7 m y la ley de 2,7‰ U<sub>3</sub>O<sub>8</sub>.

Como nosotros queremos calcular el potencial a 50 dóls./lb, podríamos estimar que para 1,50 m de espesor la ley debería ser 1,21‰ U<sub>3</sub>O<sub>8</sub> (se cuadruplica el precio, se divide por 4 el mineral contenido).

Con estos valores de tonelaje U<sub>3</sub>O<sub>8</sub>, espesor (1,50 m) y ley (1,21‰ U<sub>3</sub>O<sub>8</sub>) calculemos en cada caso qué extensión superficial tendrían las reservas dadas, asumiendo una densidad de 2 g/cm<sup>3</sup>.

$$S = \frac{\text{kilogramos de } \text{U}_3\text{O}_8}{\text{espesor} \times \text{ley} \times \text{densidad}}$$

$$\text{Caso Colorado} = \frac{807\,975\,000}{1,5 \times 1,2 \times 2} = 222\,582\,640 \text{ m}^2 = 222,6 \text{ km}^2$$

$$\text{Caso Wyoming} = \frac{827\,925\,000}{1,5 \times 1,2 \times 2} = 228\,078\,510 \text{ m}^2 = 228,1 \text{ km}^2$$

Luego la probabilidad de mineralización es:

$$\text{Colorado} = \frac{222,6}{355\,000} = 0,00063\%$$

$$\text{Wyoming} = \frac{228,1}{174\ 000} = 0,00132\%$$

$$\text{Ambas} = \frac{450,7}{529\ 000} = 0,000852\%$$

$$P = 0,000852$$

Adoptaremos este último como valor P.

#### 4.2. Determinación del factor de favorabilidad geológica F

Los geólogos de CNEA al evaluar el potencial del país y ante la necesidad de dar un valor a F, decidieron que fuera un dígito de valor 0 a 1 que comprendiera las principales características de las áreas favorables o no para presentar posibilidades de mineralización uranífera.

Así se llegó al Cuadro II y la clasificación de cada formación se establece mediante la sumatoria de la puntuación de cada una de las características analizadas, correspondiendo una sola puntuación por característica y en casos mixtos la máxima.

Así una formación con características óptimas (caso de la Formación Grupo Chubut) involucra un factor F igual a 1.

#### 4.3. Fórmula de cálculo

Finalmente la fórmula  $P = D \times S \times F$  se modifica ligeramente:

$$P = \frac{R \times S \times F \times p}{s}$$

donde P = Potencial de la formación

R = Recursos descubiertos (RRA + RAE) t de  $U_3O_8$

S = Superficie ( $km^2$  de la formación) (Aflorante)

F = Factor favorabilidad geológica

p = Factor de probabilidad de mineralización

s = Superficie de los recursos R ( $R/s$  = densidad del área de control)

Por lo tanto en el caso del Grupo Chubut tendremos:

$$P = \frac{308 \times 22\ 150 \times 1 \times 0,000852}{0,25} = 23\ 251\ t\ de\ U_3O_8$$

CUADRO II. FACTOR DE FAVORABILIDAD GEOLOGICA. TABLA DE PUNTUACION PARA UNIDADES GEOLOGICAS

[illegible]



Los geólogos de la CNEA y los autores en particular son conscientes de que este ensayo de establecimiento del potencial uranífero de una formación portadora no pasa de ser tal y que el factor de favorabilidad geológica debería ser definido mejor incorporando elementos no considerados aquí (tales como estabilidad del área, cercanías y volumen de las fuentes de aporte, etc.). Pero para lograrlo es necesario contar más a menudo con datos actualizados estadísticos, tales como los publicados por Patterson [6] en el trabajo mencionado, que proporcionen a los países de mediano a escaso desarrollo en exploración uranífera elementos de análisis y pronóstico de mayor exactitud.

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## DISCUSSION

P.D. TOENS: I would like to make a general comment about speculative resources. It is all very well speculating on undiscovered resources, but extreme caution must be exercised so that speculative resources are not confused with discovered resources, thus presenting an incorrect impression of global resources.

H. OLSEN: I agree with you, but we felt it necessary to describe the system used by the Argentine National Energy Commission for evaluating speculative resources in view of the desirability of achieving a worldwide consensus on the most appropriate method of estimating resources, which would ensure that all countries used a similar scheme.



# **SPECULATIVE URANIUM RESOURCES: THEIR LOCATION AND MAGNITUDE**

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## **Abstract**

### **SPECULATIVE URANIUM RESOURCES: THEIR LOCATION AND MAGNITUDE.**

During the first phase of the International Uranium Resources Evaluation Project (IUREP), data relating to the uranium potential of 185 countries were compiled and reviewed. Using these data, the NEA/IAEA Steering Group on Uranium Resources identified areas of the world thought to be favourable for the location of additional uranium resources and estimated the magnitude of these "Speculative Resources". A high percentage of the Speculative Resources is expected to be located within the major producing areas but the study has highlighted several other areas which, although they do not have at present known economic uranium deposits, could make a major contribution to uranium reserves in the future. In many countries, it will be necessary to perform exploration work beyond that at present being undertaken if a significant portion of the Speculative Resources is to be found. To encourage this exploration, a further IUREP evaluation, called the "Orientation Phase", has been initiated.

## **1. INTRODUCTION**

A report was published by OECD in December 1978 entitled 'World Uranium Potential: An International Evaluation'. This report, prepared by the Joint Steering Group on Uranium Resources of the OECD Nuclear Energy Agency and the International Atomic Energy Agency, contains the findings of a study that commenced in November 1976. At its first meeting, the Group undertook to 'review the present body of knowledge pertinent to the existence of uranium resources, to identify areas favourable for such resources, and to suggest new exploration efforts which might be carried out in promising areas in collaboration with the countries concerned'. The study was subsequently named the International Uranium Resources Evaluation Project - IUREP. This paper reviews the progress made by

## A.

Energy, Mines  
and Resources  
Canada

DEMONSTRATED		SURMISED		SPECULATIVE
MEASURED	INDICATED	INFERRED	PROGNOS- TICATED	SPECULATIVE

United States  
ERDA, 1976

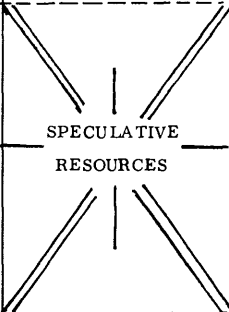
RESERVES	PROBABLE POTENTIAL RESOURCES	POSSIBLE AND SPECULATIVE POTEN- TIAL RESOURCES
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NEA/IAEA

REASONABLY ASSURED	ESTIMATED ADDITIONAL	SPECULATIVE RESOURCES
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The terms illustrated are not strictly comparable as the criteria used in the various systems are not identical. Grey zones in correlation are therefore unavoidable, particularly as the resources become less assured. Nonetheless, based on the principal criterion of geological assurance of existence, the chart presents a reasonably approximation of the comparability of terms.

## B.

SUBECONOMIC RESOURCES	Exploitable at costs from: \$ 130 to \$ X/kg U	REASONABLY ASSURED RESOURCES	ESTIMATED ADDITIONAL RESOURCES	SPECULATIVE RESOURCES
	\$ 80- \$ 130/kg U	REASONABLY ASSURED RESOURCES	ESTIMATED ADDITIONAL RESOURCES	 SPECULATIVE RESOURCES
ECONOMIC RESOURCES	Exploitable at costs Up to \$ 80kg/U	REASONABLY ASSURED RESOURCES <u>RESERVES</u>	ESTIMATED ADDITIONAL RESOURCES	

Decreasing Confidence in Estimates →

FIG.1. A: Approximate correlations of terms used in major resource classification systems.

B: Classification scheme for uranium resources. (Modified after Uranium Resources, Production and Demand, NEA (OECD)/IAEA (Dec. 1977).)

the project to date, highlights some of its findings and describes the activities to be undertaken in the future.

### 1.1 Methodology

Throughout 1977 and during the early months of 1978 members of the Steering Group and consultants engaged by the agencies compiled data on 185 countries. For each of these countries information was collected on:

- general geography
- geology, with particular reference to areas favourable for uranium deposits
- the extent of past uranium exploration
- known uranium occurrences and resources
- past production
- present status of uranium exploration

These data were used to identify areas believed to be favourable for the discovery of uranium resources in addition to those resources reported on in "Uranium Resources, Production and Demand, December 1977" (Figure 1). For each country, a judgement was made of the order of magnitude of this additional potential, based on the geological favourability for the existence of uranium deposits that could be exploited at costs less than \$130/kg U, which was ranked as low, moderate, moderate to high, high, high to very high or very high. For the purpose of obtaining an aggregate measure of continental and world resources, the Group applied ranges of tonnages to these rankings. These tonnage ranges are defined as Speculative Resources and refer to uranium, in addition to Estimated Additional Resources, that is thought to exist, mostly on the basis of indirect indications in deposits discoverable with existing exploration techniques.

To obtain the ranges shown for continental Speculative Resources in Table I, all the high

TABLE I. WORLD URANIUM RESOURCES

(1000 t U)

*Data available 1 February 1979*

<u>Continent</u>	<u>RAR<sup>a</sup></u>	<u>EAR<sup>b</sup></u>	<u>Speculative Resources<sup>c</sup></u>	
			<u>Low</u>	<u>High</u>
Africa	580.4	191.1	1,300	4,000
Asia	46.8	23.7	200	1,000
Australia and Oceania	296	49	2,000	3,000
Western Europe	412.8	105.6	300	1,300
North America	920.5	1,841	2,100	3,600
South and Central America	162.4	5.6	700	1,900
Totals	<u>2,419</u> =====	<u>2,216</u> =====	<u>6,600</u> =====	<u>14,800</u> =====

<u>Estimated Total Potential<sup>d</sup></u>		
	<u>High</u>	<u>Low</u>
Eastern Europe	300	1,300
USSR	2,000	4,000
Peoples Republic of China	1,000	2,000
Total	<u>3,300</u> =====	<u>7,300</u> =====

<sup>a</sup> Reasonably Assured Resources [OECD(NEA)/IAEA]

<sup>b</sup> Estimated Additional Resources [OECD(NEA)/IAEA]

<sup>c</sup> Refers to uranium in addition to Estimated Additional Resources, that is thought to exist mostly on the basis of indirect indications and geological extrapolations in deposits discoverable with existing exploration techniques. The location of deposits envisaged in this category could generally be specified only as being somewhere within a given region or geological trend. As the term implies, the existence and size of such resources are highly speculative.

<sup>d</sup> This includes an element for "Reasonably Assured Resources" and "Estimated Additional Resources" although these data were not available to the Steering Group.

values and all the low values of the ranges for each country were added. This technique was felt to be appropriate in view of the subjective nature of the exercise and the uncertainty associated with the ranking of each country.

## 1.2 Classification of Deposits

In addition to presenting the Steering Group's impression of the World's Speculative Resources, the report describes the areas of the world containing major occurrences and deposits and identifies areas favourable for the location of the Speculative Resources. For the purpose of this part of the report, it was found necessary to adopt a classification of uranium ore deposits. The major uranium resources of the world were assigned, on the basis of their geological setting, to six categories of ore types which are described in detail in the report and briefly described below:-

### 1.2.1 Quartz-pebble conglomerate deposits

Known quartz-pebble conglomerate ores are restricted to a specific period of geologic time. They occur in basal Lower Proterozoic beds unconformably situated above Archaean basement rocks composed of granitic and metamorphic strata. Commercial deposits are located in Canada and South Africa, and sub-economic occurrences are reported in Brazil.

### 1.2.2 Proterozoic unconformity-related deposits

Deposits of the Proterozoic unconformity-related type occur spatially close to major erosional unconformities developed during a generally world-wide orogenic period about 1,800-1,600 million years ago. They are represented by the ore bodies at Cluff Lake, Key Lake and Rabbit Lake in northern Saskatchewan, Canada, and those in the Alligator Rivers area in northern Australia.

### 1.2.3 Disseminated magmatic, pegmatitic and contact deposits in igneous and metamorphic rocks

The deposits included in this grouping are those associated with granites, migmatites, syenites, pegmatites, carbonatites and volcanic rocks. The largest known deposit in this grouping is Rössing, in Namibia, which is associated with pegmatitic granite and alaskite. Also included in this category are the Iberian deposits, for example those at Mina Fe, Spain, and Nisa, Portugal, and those in the Spokane area of the U.S.A.

### 1.2.4 Vein deposits

The vein deposits of uranium are those in which uranium minerals fill cavities such as cracks, fissures, pore spaces, breccias and stockworks. The dimensions of the openings have a wide range, from the massive veins of pitchblende at Jachymov Czechoslovakia, Shinkolobwe (Zaire) and Port Radium (Canada) to the narrow pitchblende-filled cracks, faults and fissures in some of the ore bodies in Europe, Canada and Australia.

### 1.2.5 Sandstone deposits

Most of the ore deposits of this type are contained in rocks deposited under fluvial or marginal marine conditions. The host rocks are commonly medium to coarse-grained, poorly sorted sandstones containing pyrite and organic matter. The Tertiary, Jurassic and Triassic sandstones of the Western Cordillera of the U.S.A. account for most of the uranium production in that country. Cretaceous and Permian sandstones are important host rocks in Argentina while Carboniferous sandstones are important in Brazil and Niger. Deposits of this type in Europe are usually found in Permian sediments.

### 1.2.6 Other types of deposits

Included in this grouping are deposits that cannot readily be classified with the ore types



already mentioned. Recently discovered uranium deposits associated with calcrete occur in Australia, Namibia and Somalia in semi-arid areas where water movement is chiefly subterranean. Other deposits occur in limestones and in limestone karst terrain as phosphatized fractions of the limestone. Uranium also occurs at low concentrations in marine phosphorite and shales but no estimates were made of Speculative Resources from these host rocks<sup>7</sup>.

1.3 Each continent is dealt with in the report in a separate chapter. Within each chapter, following a brief statement on the geology of the Continent, individual countries where the uranium potential was judged to be good are described in a standardized format as below.

- (a) Introduction - a description of the main physical features of the country
- (b) Uranium Occurrences - includes a description of known uranium resources
- (c) Areas of Uranium Potential - includes the ranking given by the Steering Group
- (d) Status of Exploration - a brief summary of past and present exploration activity.

At the end of each chapter the report also identifies several other countries where the potential for additional uranium resources was judged to be lower but still of some interest.

The last chapter of the report is concerned with other sources of uranium. These include sources from which uranium may be technically recovered at costs exceeding \$130/kg U or from which it may be produced as a by-product in the recovery of other minerals.

## 2. AREAS OF POTENTIAL

It would be a difficult task to summarise all data contained in the individual chapters of



FIG.2. Uranium deposits and occurrences in Africa.

the report and the summary produced would have little value. Therefore, we have highlighted several areas or rock units from each continent which, in our opinion, constitute some of the more interesting targets described in the report.

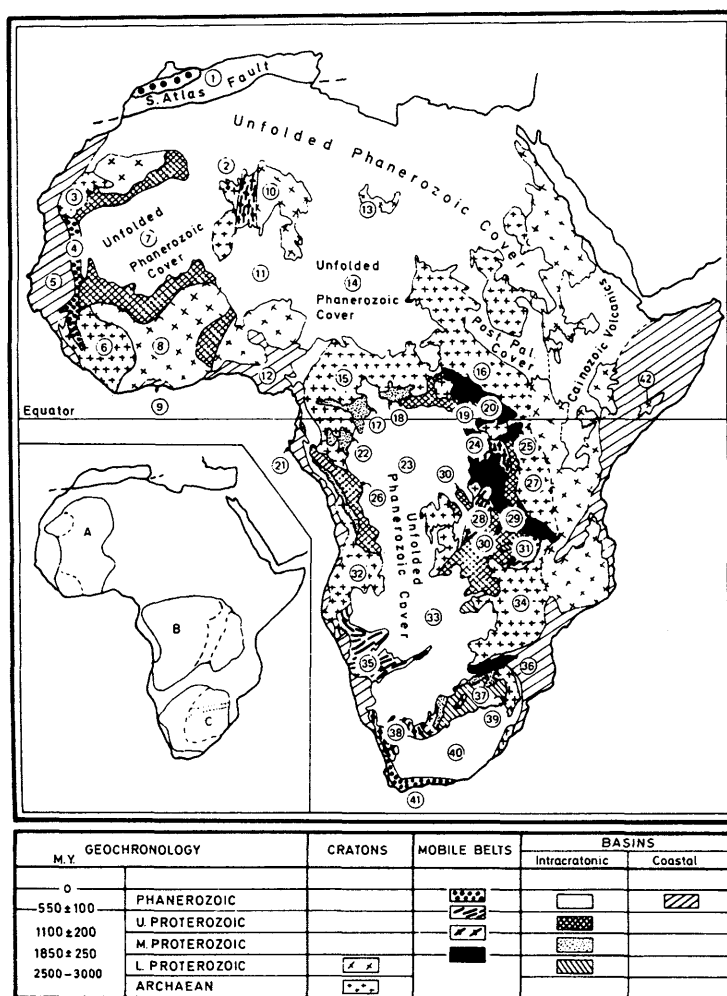
## 2.1 Africa

Twenty-eight of the African countries investigated have good potential for additional uranium resources. A major part of the Speculative

Resources of Africa are expected to occur in quartz-pebble conglomerate deposits in South Africa (Figure 2). Other areas which are already producing uranium should also make major contributions as deposits similar to Rössing could be found in Namibia and new ore bodies are to be expected in the Franceville Basin of Gabon and in the Agades Basin of Niger. The deposits in the Hoggar Massif may be indicative of other deposits that could be found in similar settings.

In general, the Precambrian of Africa is poorly known (Figure 3). This makes the prediction of favourable environments, outside of producing areas, very difficult, especially for those types of deposits that appear to be restricted in time. Keeping this in mind, there would appear to be potential for deposits of the unconformity-related type in the Lower Proterozoic rocks of Cameroon. The Lower Dja Series in the same country has been correlated with the Francevillian Series of Gabon and could be another favourable target. The Dahomeyan/Atacorian unconformity of Togo is very interesting, especially as uranium mineralization is known in shear zones in the Dahomeyan close to the unconformable boundary. In addition to the countries already mentioned, Zaire, Zambia, Central African Empire, Congo, Ethiopia, Botswana, Angola and Ghana, all have some potential for the discovery of this type of deposit. In several of these countries, the potential of the Katanga System is the critical factor.

There are several rock units in Africa which have the potential for sandstone deposits. Karoo sediments outcrop over large areas of Africa. They are known to contain sub-economic uranium mineralization in several countries and there are small deposits in Zambia. That this sequence of rocks could produce large quantities of uranium remains to be demonstrated. The Cretaceous "Continental Intercalaire" is also widespread and is mineralized at Azelik (Niger). The presence in the Agades Basin, which contains the Azelik deposit in addition to deposits in Carboniferous and Permian sediment, of volcanic ash and tuffaceous material, could be the governing factor in the formation of the deposits. The sandstones in



## EXPLANATION

- |                                    |   |                                    |
|------------------------------------|---|------------------------------------|
| 1. North Atlas Zone                | 15. Cameroon and Gabon Basement                         | 29. Ubendian-Rusizi Mobile Belt    |
| 2. In Ouzzal Massif                | 16. Central African Empire Basement                     | 30. Katangan & Bushimay Basin      |
| 3. Reguibat Massif                 | 17. Sembe-Ouessou Series                                | 31. Irumides Mobile Belt           |
| 4. Mauretanian Mobile Belt         | 18. M'Baiki & Liki-Bembien Series                       | 32. Angola Basement                |
| 5. Senegal Basin                   | 19. Ubangan, Fouroumbala, Lindien- and l'Iturien Series | 33. Kalahari Basin                 |
| 6. Sierra Leone-Ivory Coast Massif | 20. Kibali Mobile Belt                                  | 34. Rhodesian Basement             |
| 7. Taoudeni Basin                  | 21. Gabon-Cabinda Basin                                 | 35. Damaran Mobile Belt            |
| 8. Eburnean Block                  | 22. Franceville Basin                                   | 36. Limpopo Mobile Belt            |
| 9. Ivory Coast Basin               | 23. Congo Basin   | 37. Witwatersrand Basin            |
| 10. Hoggar (Ahaggar) Massif        | 24. Burundian Mobile Belt                               | 38. Namaqualand Basement           |
| 11. Iullemeden Basin               | 25. Buganda-Toro Mobile Belt                            | 39. Transvaal (Kapaavaal) Basement |
| 12. Cotonou Basin                  | 26. West Congo Basin                                    | 40. Karroo Basin                   |
| 13. Tibesti Basin                  | 27. Nyanzian-Dodoman Basement                           | 41. Cape Fold (Mobile) Belt        |
| 14. Chad Basin                     | 28. Kibarian Mobile Belt                                | 42. Bur Massif                     |

Cratons: A. Sierra Leone-Ivory Coast (Guinea-Eburnian); B. Congo; C. Kalahari.

FIG.3. Structural pattern of Africa. (Modified after Molina, P., Besombes, J.C., in *The Oklo Phenomenon* (Proc. Symp. Gabon, 1975), IAEA, Vienna (1975) 6-7.)

the Djado region of Niger, and Permian and Cretaceous continental sediments in Mali, could be interesting but available literature does not indicate if volcanic material is common in these rocks.

The folded Palaeozoic zone in the Atlas region of Morocco has been locally metamorphosed and granitized due to Caledonian and Hercynian orogenic activities. The latter granites could have potential as hosts for vein deposits and sources for uranium in post-Hercynian cover rocks. The recently discovered uranium veins in the "Younger Granites" of Chad and the high uranium contents reported in these granites in Nigeria could be worthy of further investigation.

There is the possibility of finding uranium in surficial and unconsolidated deposits in many countries. The Eocene to Recent Kalahari sands could be prospective in Angola, Botswana, Zaire and Zambia. Deposits similar to those in the Mudugh Province in Somalia could be found in Ethiopia. The main potential for surficial deposits occurs in the calcretes, in particular in Namibia and South Africa but also in Mali, southern Madagascar and possibly Sudan.

## 2.2 Asia

Asia and the Far East, including the Peoples Republic of China but excluding Asian USSR, covers an area of about 27,600,000 km<sup>2</sup>. The known resources of the continent are small when compared to its size. The relatively low Speculative Resources, especially if those of China are not included, are, we believe, a reflection of the relatively few environments which are favourable for the larger types of deposits. Environment for quartz-pebble conglomerate deposits and unconformity-related deposits occur only in China and, to a lesser extent, India (Figure 4). Uranium could occur in vein or disseminated deposits in several countries, in particular China and India but also in Indonesia, Malaysia, Saudi Arabia, Thailand and Turkey, but it is possible that a greater proportion of the resources will be contained in sandstone deposits.

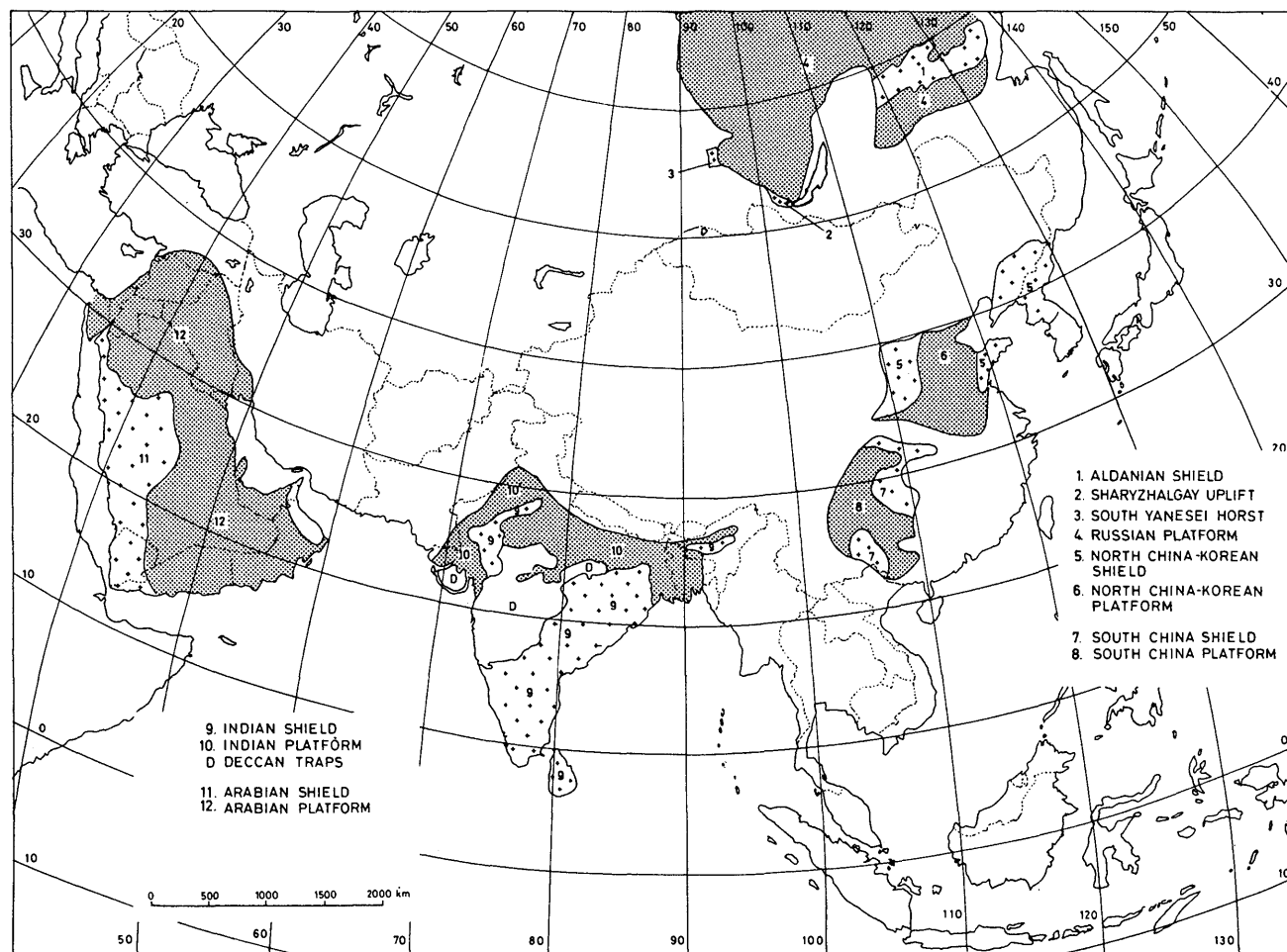


FIG.4. Map of Asia showing platforms and Precambrian shields.

The terrestrial Mesozoic sandstones of the Khorat Plateau (Thailand) are similar to the "red bed" sediments of the Colorado Plateau of the U.S.A. Both copper and uranium mineralization is known in the sediments and vanadium is widespread in the area. In addition, the source granites for these sediments are abnormally radioactive. The Tertiary Krabi Group of Thailand may also be mineralized. The Neogene Swalik System of India and Pakistan is known to be uraniferous at several locations and intensive exploration has been undertaken in some areas. This system may have significantly more potential than has been indicated to date. Both China and Mongolia contain large basins filled with Mesozoic and Tertiary continental sediments which often contain significant quantities of volcanic material. In Afghanistan, acid volcanic rocks are locally associated with Neogene red beds and some mineralization is known. The greatest potential for uranium deposits in Iran appears to be in the continental sediments, in particular, the Tertiary rocks of the Jaz-Murian Depression. The Neogene continental sediments in the interior basin of Turkey could be attractive targets and there is the possibility of finding uranium in sandstones in Afghanistan, Bangladesh, Burma, Iraq and Japan.

The possibility of finding uranium in calcretes in several countries in the Middle East, in particular in Iraq, the United Arab Republic, the Peoples Democratic Republic of Yemen and the Yemen Arab Republic, should not be overlooked.

## 2.3 Australia and Oceania

The great majority of the Speculative Resources of Australia and Oceania will be located in Australia (Figure 5), in particular in the Lower Proterozoic rocks of the Pine Creek Geosyncline in the Northern Territory. The Cahill Formation, the host for several of the world's largest deposits, extends over an area of 6,000 km<sup>2</sup> but less than 1% outcrops. It has been estimated that there is a potential of the order of five to ten times the known reserves. There is a potential for deposits of the unconformity-related type in all the blocks of the North Australian Orogenic

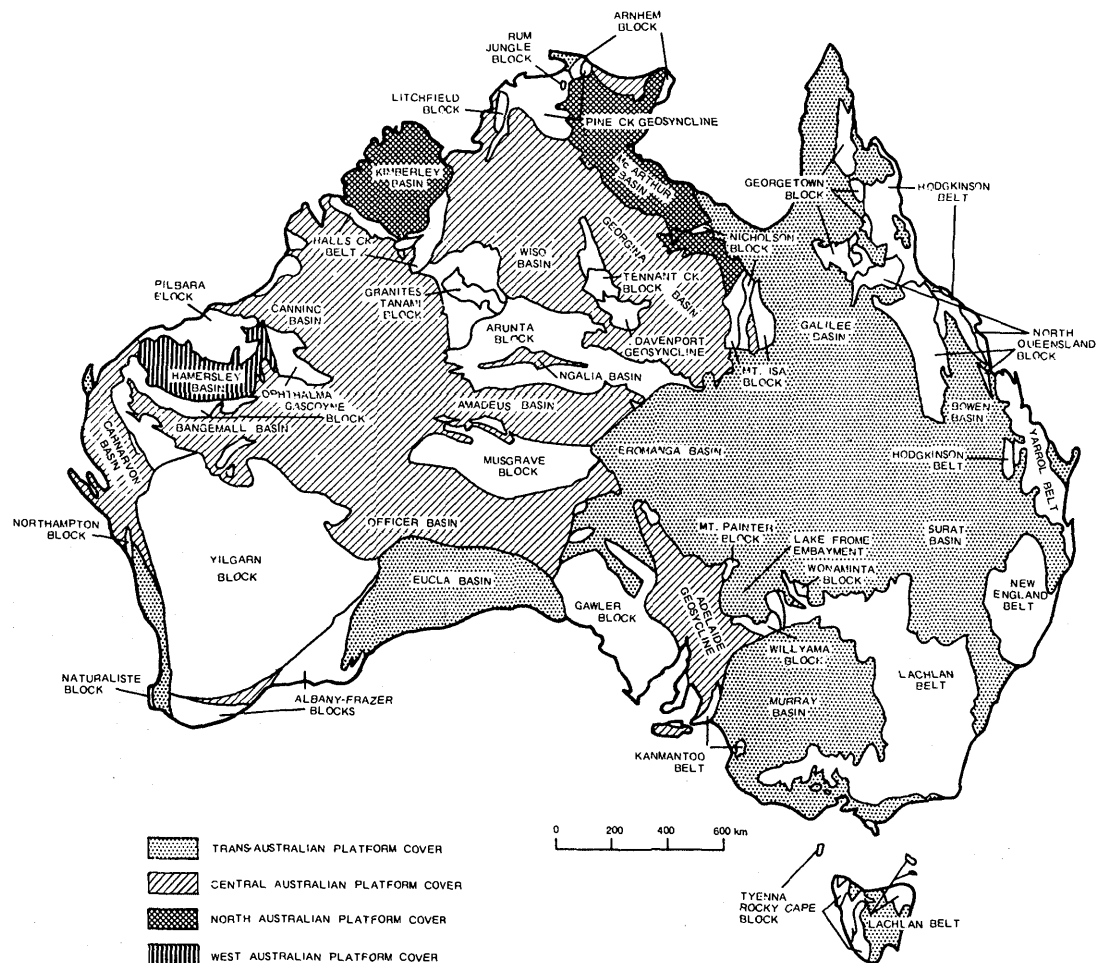


FIG.5. The major geological elements of Australia.



Province, in the Georgetown Block (Queensland) and in the Gawler Block (South Australia).

There is potential for sandstone deposits in sediments of the platform covers of Australia, in particular, in the vicinity of Precambrian orogenic blocks and especially where these blocks contain significant concentrations of uranium. Of particular interest here are the Tertiary sediments in the Lake Frome Embayment in which several deposits have already been discovered. The Miocene platform sediments of Papua New Guinea and the Lower Cretaceous sediments of New Zealand may have some potential for sandstone deposits.

There is the possibility of finding additional deposits in calcretes in, or adjacent to, the Yilgarn, Pilbara and other Precambrian orogenic belts in Australia. Uranium mineralization in calcrete is widespread and there has been insufficient exploration to adequately assess the numerous occurrences, though the extensive exploration around Yeelirrie has reduced the potential of finding new deposits in this part of the Yilgarn Block.

There is the possibility of finding disseminated- and vein-type uranium deposits in all four of the Australian orogenic provinces, in and around the major shear zones in New Caledonia, and in some areas of the Solomon Islands.

## 2.4 Europe

Twelve of the countries of Western Europe (Figure 6) have good potential for additional uranium resources. It is to be expected that a major portion of the Speculative Resources will be found in vein deposits or as disseminations in igneous and metamorphic rocks often spatially related to Hercynian granites.

A large part of Spain's uranium reserves are found as disseminations within contact metamorphic aureoles where Hercynian granites have

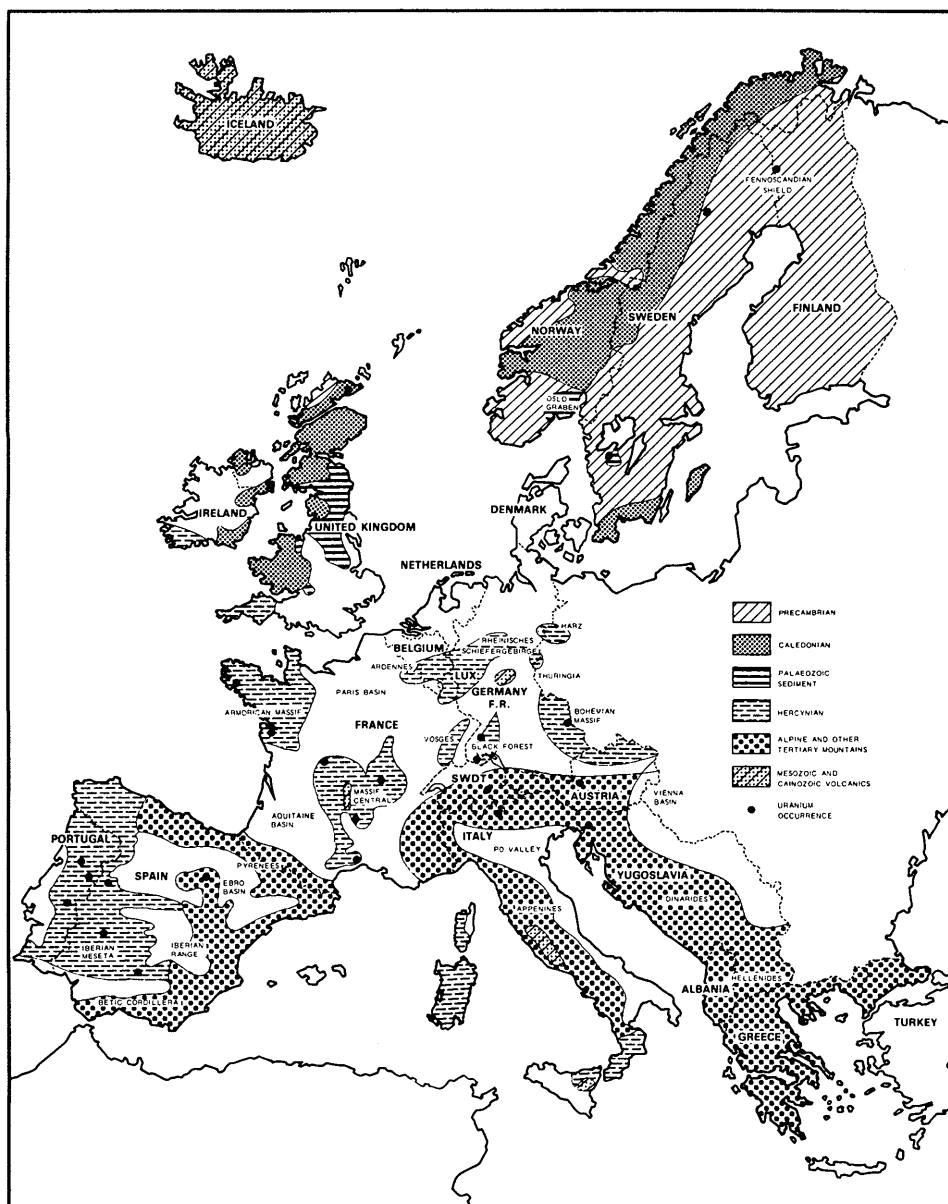


FIG.6. Map of Western Europe showing location of uranium deposits in relation to tectonic zones.

invaded pre-Devonian shales. This type of deposit is so common here that it has been called the "Iberian Type" by several geologists. It is possible that the full potential of deposits of this type has yet to be realised and many additional resources could be found in the Meseta in Spain and in Portugal and also in the southwest of England.

Vein deposits in France are almost entirely restricted to the Moldanubian Zone of the Hercynian Orogen. The deposit at Menzenschwand in the Federal Republic of Germany occurs as hydrothermal veins in a Hercynian granite also in the Moldanubian Zone. Granites within this zone of the Hercynian orogen are therefore priority targets for uranium exploration and other deposits both in and around these granites are to be expected in France, Germany and Austria.

In addition to acting as a good host for vein deposits, the Hercynian granites appear to have acted, in several instances, as good source rocks for sandstone deposits. Post-Hercynian continental sediments deposited close to or on the Hercynian Massifs are excellent exploration targets in Spain, France, Portugal, and the Federal Republic of Germany. Other sandstone deposits are expected in Permian sediments in Yugoslavia, Italy and Austria.

There are several uranium occurrences within the Precambrian rocks of the Baltic Shield. Many of these occurrences are either close to the edge of the Caledonides or are exposed through windows in these rocks. As the age of much of the mineralization predates the Caledonian orogeny, it appears that this relationship may not be a significant one and that palaeogeographic studies of the Precambrian rocks, similar to those performed on the Arjeplog-Arvidsjaur uranium province of Sweden, may be more rewarding.

The uranium bearing lujavrites of the Ilimaussaq intrusion in Greenland cover several tens of square kilometers and their resource potential could be very large. There is a good possibility that detailed prospecting, not just in but also outside this and other intrusions of the Gardar province, would significantly increase the known resources of this area.



FIG.7. Regional geological map of Mexico. (Modified after Torres et al., in *Exploration for Uranium Ore Deposits* (Proc. Symp. Vienna, 1976), IAEA, Vienna (1976) 604.)

## 2.5 North America

The continent of North America has the largest known resources and also environments favourable for all types of deposits. The national authorities of both Canada and the United States publish official estimates of their respective countries' Speculative Resources and there are many descriptions in the literature of the areas thought favourable to contain these resources. Briefly, the position is that the known uranium resources of Canada occur mainly in quartz-pebble conglomerates and in unconformity-related deposits and it is deposits of the latter type that could contain a major portion of the country's future resources. Similarly, the large majority of the known resources of the United States are in sandstone deposits and most of the Speculative Resources are expected to occur in deposits of this type, mainly in producing districts. We cannot stress too strongly, however, that large quantities of uranium are to be found in other types of deposits in both countries and that these resources will be needed to meet future demand.

Large areas of Mexico (Figure 7) are covered by volcanic rocks of acidic composition which could act as hosts for vein or disseminated mineralization or as sources of uranium for sandstone deposits. The Oligocene to Pliocene acid volcanics of the Sierra Madre Occidental, in particular along the mineralized eastern edge, are obvious hosts for disseminated deposits. Volcanics of the same age and composition also outcrop in the Sierra Madre Del Sur in the states of Guerrero and Oaxaca. Uranium deposits occur in the Frio Formation of the Coastal Plain of the Gulf of Mexico. Other deposits are to be expected in this area but other areas, in particular the northwestern part of the Baja California and the Parras, La Popa, Sabinas and Ojinaga Basins within the Sierra Madre Oriental, are favourable for additional deposits. Acid volcanics and continental sediments also occur in close association in many other parts of Mexico. In addition, there is the possibility of finding calcrete deposits in the restricted basins in arid regions of northern Mexico.

## 2.6 South and Central America

For its size, South America has relatively few known uranium resources, though this situation is slowly changing. Geologically, the continent contains environments favourable for all types of uranium deposits including, in the opinion of some geologists, the most favourable settings for new deposits in quartz-pebble conglomerates and deposits of the unconformity-related type outside of producing regions.

Precambrian conglomerates are widely developed around the Sao Francisco Massif in Brazil (Figure 8) and the possibility of finding economic deposits in these rocks appears promising. Further work will undoubtedly also be undertaken on the conglomerates of the Moeda Formation, and the geology of the Cavalcante-Tocantins mining district, 200 km north of Brasilia, also appears promising for deposits in quartz-pebble conglomerates. There appears to be a more limited possibility of discovering such deposits in the Proterozoic rocks of the Guyana Massif in Colombia, Guyana and Venezuela and Guaporé Massif of Bolivia.

The most favourable area for the location of Proterozoic unconformity-related deposits in South America is that underlain by the Guyana Massif and overlain by the Roraima Formation in parts of Brazil, Venezuela, Guyana and Surinam. Some of the sandstones present on the Guyana Massif, and the adjacent Llanos Basin, in Colombia may be correlative with the Roraima Formation and these areas could be prospective. The unconformity between the Lower Precambrian Caico Group and the Upper Precambrian Serido Group in the Serido district of Brazil may also have potential for this type of deposit.

The areas underlain by all the inter-massif fold belts appear to be favourable for vein and disseminated uranium deposits. This is particularly true of the areas underlain by the Sierra Pampeanas Belt of northwest Argentina, in which many occurrences and several deposits are known, and the Cariria-Sergipe Belt of Brazil which has many similarities to the area around Rössing and also to the area in Europe containing mineralization related

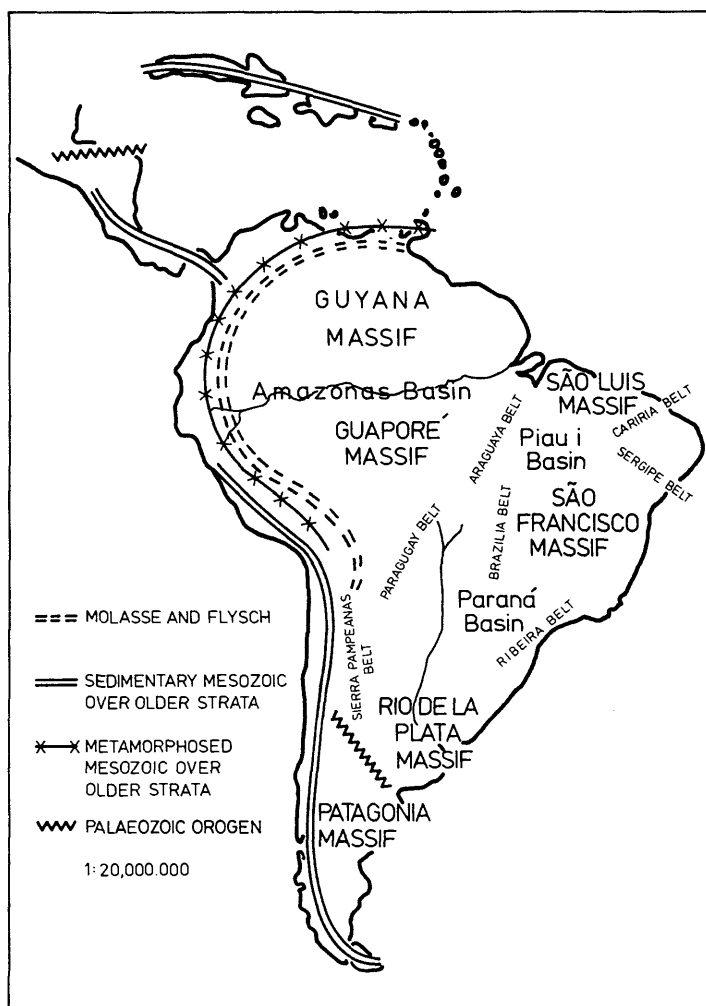


FIG.8. Main geological features of South and Central America.

to Hercynian granites. The widespread igneous activity throughout the Carboniferous and Permian in the Palaeozoic Orogen of Argentina gives this area a good potential for vein and sandstone deposits and also deposits of the "Iberian Type". The Cordilleras, especially the eastern parts, contain several environments favourable for vein and disseminated deposits especially where the country rock is of Precambrian or Palaeozoic age.

There is a good potential for sandstone deposits in South America. This is particularly true of the producing areas such as those of western Argentina and the areas around Figueira in the Parana Basin in Brazil. All three of the major sedimentary basins could contain uranium deposits in sandstones with the Parana Basin, in particular, containing several areas of interest especially the Cerro Partido Sub-basin and the Klabin coal area. Uranium occurrences are known in the smaller Mesozoic basins in the east and northeast of Brazil and there may also be potential for sandstone deposits where Palaeozoic, Cretaceous and Tertiary sediments overlap the Guyana and Guapore Massifs. In addition to the areas of Cretaceous sediments in Argentina, several parts of the Cordillera have potential for sandstone deposits. These include the Carboniferous and Permian sediments of Bolivia and Peru, the Mesozoic red beds of the Central Cordillera, and Tertiary sediments in many small basins scattered throughout the Cordillera and in the sub-Andean areas.

## 2.7 Eastern Europe and the USSR

Seven of the countries in Eastern Europe and the USSR have good potential for additional uranium deposits. The majority of the known and future resources in this area, excluding the USSR, occur within the Bohemian Massif. Although the mineralization of the vein and disseminated type is here spatially related to Hercynian granites, the majority is located outside these granites mainly in Precambrian or Early Palaeozoic metamorphic rocks. The sedimentary basins covering the massif also contain uranium and it is possible that these have not been fully explored. The presence of additional vein, disseminated and sandstone deposits is to be expected in Czechoslovakia, the German Democratic Republic and Poland.

The Permian and Triassic continental or shallow marine sediments of the West Carpathians, or their south-easterly extension, could contain significant quantities of uranium in Czechoslovakia, Poland and Romania, and the Permian sandstones of



the Mecsek Mountains of Hungary could contain additional deposits. In Bulgaria, uranium could be found in vein, disseminated and sandstone deposits in the Rhodope Massif and there is some potential for vein and sedimentary occurrences in the Balkan Mountain area.

The USSR is the largest country in the world and contains every possible geological environment for the location of uranium deposits with the two vast Precambrian platforms, Palaeozoic geosynclines which have been folded and intruded by granites of many ages, folded Mesozoic and Tertiary deposits also intruded by granites, large areas covered by acid volcanic rocks, and Cambrian to Recent continental and shallow marine sediments. While many areas are covered by great thicknesses of Recent sediments that are not prospective for uranium, vast areas remain. It is likely that many of these have not yet been fully explored.

### 3. THE "ORIENTATION PHASE"

As discussed earlier, the Steering Group on Uranium Resources identified a number of countries as having good potential for additional resources. In order to discover a significant portion of these resources, it will be necessary to perform exploration work beyond that presently being undertaken and the Steering Group recommended that exploration should be encouraged in all these countries. It was envisaged that a further assessment by IUREP, in co-operation with the countries concerned, could provide some of the necessary encouragement.

#### 3.1 The Objectives

This further evaluation, the "Orientation Phase", consists of fact-finding expert missions, usually two uranium geologists, being sent to the host countries for a period of approximately 2 months

to acquire, in consultation with the countries concerned, information which will lead to:-

- (a) a better evaluation of the countries' uranium potentials
  - (b) a better delineation of areas favourable for additional resources
  - (c) recommendations as to the best methods to evaluate the favourable areas
- and
- (d) a compilation of the logistical data required for possible future work.

The reports compiled will not only indicate which countries have good uranium potential and in which areas the potential lies but also indicate what will be involved in the search for the resources and the possibilities of producing them if they are found. By collectively undertaking the studies needed to compile this information, the participating countries will effectively speed up the process of evaluation and exploration.

### 3.2 Implementation

A meeting was held in Vienna in November 1978 to which representatives of a cross-section of those countries identified as having good potential were invited by the Steering Group. At this meeting, all the technical aspects of IUREP were discussed, including the work involved in the compilation of the Phase I report and the present and future assessment work. The representatives were informed that money was available to finance a number of missions as described above and that requests for such missions should be sent to the IAEA. Since that time, other countries with good potential, in addition to those invited to the meeting, have been informed of the possibility of receiving IUREP Orientation Phase missions and a total of thirty eight countries either have requested or are considering requesting such missions.

The information collected on each country during the Orientation Phase and the conclusions that may be drawn from them, will be immediately made available to that country. The timing of its subsequent wider dissemination will be decided on a case-by-case basis in consultation with the countries concerned. It should be stressed that the objective of the work will best be served by the wide and prompt distribution of all the data as the ultimate aim of IUREP is to provide with as much

accuracy as possible, an estimate of the world's uranium resources and to promote the discovery of additional uranium resources, especially as these resources will be required if nuclear power is to serve as a major source of energy.

#### 4. CONCLUSIONS

In their discussion of the Speculative Resource figures, the Steering Group wished to emphasize that "even if these Speculative Resources exist, there is no guarantee that the resources will be discovered or if discovered, that they can be made available...Serious constraints may well arise on many fronts ... Because of these constraints, and in view of the long lead times for exploration, mine development and production, it is likely that a major part of these Speculative Resources may not be discovered and brought into production during the first quarter of the 21st century".

There is, we believe, no doubt that some of the Speculative Resources exist and many of these will be located in the areas described in this paper. The major questions are:- (a) do enough of these resources exist and, (b) can they be found in the time available? We believe the answer is yes and rely on the uranium industry to prove us right.

#### ACKNOWLEDGEMENTS

A major portion of the data used in the compilation of this paper was gathered by the NEA/IAEA Steering Group on Uranium Resources as part of Phase I of the International Uranium Resources Evaluation Project.

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## DISCUSSION

A.J. N'DA (*Chairman*): It seems to me that there is a difference between speculative resources and hypothetical resources. Perhaps this is purely a question of terminology, but could you tell me where hypothetical resources would feature in a triangular diagram showing the extent of knowledge on one side, the degree of exploitability on the second and the ore content on the third? For me, speculative resources would come into the category of known low-grade resources with a small degree of exploitability. Do you agree? If so, could it perhaps be said that the information available on these resources is sufficient to be able to single out zones in which new deposits are likely to be discovered, especially in countries where geological information may be defective and systematic prospecting has only just begun?

D.M. TAYLOR: In general, speculative resources could be considered to have the same ranges of grade and, once discovered, of exploitability (upright) as resources into RAR + EAR at < US \$130/kg U. They are, however, not known at all but are falling into RAR + EAR at < US \$130/kg U. They are, however, not known at all but are thought to exist. We have more confidence in the existence of some speculative resources than in that of others — the location of some we believe we can predict quite accurately while that of others we cannot predict.

M. ROOHI: It is extremely difficult to estimate world uranium resources costing less than US \$130/kg U to exploit. Most of the information received by NEA/IAEA relates to uranium deposits with exploitation costs of around US \$80, whereas there are many deposits in the world with exploitation costs of more than US \$80, the information on which received by NEA/IAEA is insufficient. If defective information is used in conjunction with comparative geological studies for purposes of extrapolation, the results may be quite wrong. So more information is needed about the different countries involved before a figure for world resources of low-grade uranium can be announced.

D.M. TAYLOR: The more information we receive from whatever source — but directly from the country concerned in particular — the more confidence we can put in our speculative resource estimates. In fact we did not rely purely on information given by the countries concerned when compiling our IUREP data base.

A. HATTON: What was the scope of the geological work on which the original resource study was based?

D.M. TAYLOR: The extent of the geological information on which we based our speculative resource estimates varied tremendously from country to country. For countries whose geology is well known, and where a significant amount of exploration has been undertaken, our confidence in our estimates is greatest. For countries where the geology is not well known — in some instances simply extrapolated from adjacent areas — and where little or no exploration has been undertaken, we can have little confidence in our estimates.

J.J. SCHANZ: Could you give us some idea of the number of personnel available to you for making the estimates, so that we can appreciate the extent of the expertise available?

D.M. TAYLOR: Over a period of two years approximately 50 people were directly involved for varying periods of time in the production of the data base and in the preparation of the report on the first phase of IUREP. An unofficial estimate of the total effort would, I think, be somewhere in excess of five man-years.

R.G. WADLEY: As one who is involved in industry – in the exploration for and exploitation of uranium deposits – I must confess to being concerned about the number of papers presented at this Symposium that have dealt with global and national resource estimation, as compared with the number of those dealing with actual exploration for and evaluation of uranium occurrences. Whilst I appreciate the need for studies of the first-named type, I would suggest that there is a real risk of devoting too much effort to what is after all an exercise of limited (and often dubious) benefit.

In countries such as my own it is surely more important that national research be directed towards exploration for uranium deposits and their subsequent exploitation, than towards speculation as to their possible existence.

D.M. TAYLOR: I entirely agree with you that much more effort should be directed towards exploration for uranium deposits than towards speculation about their existence. However, it is up to the individual countries, developing or otherwise, to decide on the direction of their national research programmes, and we believe that the IUREP studies can assist countries in making this decision by suggesting the most promising areas for future investigation. We also believe that, by jointly undertaking these studies and disseminating the results widely, the participants in IUREP have been making it unnecessary for individual countries or organizations to spend funds on the same sort of work – thereby making more money available for exploration.

A.E. BELLUCO: I believe that there is a definite need for this type of evaluation of speculative resources – in order to establish whether there is enough uranium in the world for the implementation of nuclear power programmes and to find out what regions have the most favourable geological conditions for discovering uranium.

I would like to ask Mr. Taylor what system was used for estimating these speculative resources?

D.M. TAYLOR: The speculative resources of a given country were estimated by (a) collecting all the publicly available geological knowledge relating to uranium in the country, (b) reviewing the extent and nature of past uranium exploration and then (c) making comparisons between the geology of areas containing deposits. Each member of the NEA/IAEA Steering Group on Uranium Resources estimated his country's potential for additional deposits. These individual estimates were then discussed at a meeting of the Group, at which a consensus of opinion was

reached for the country. The composition of the Group was such that there were very few countries of the world in which not a single member of the Group had worked — even so, the Group preferred not to give resource estimates for individual countries but summed them by continent as explained in section 1.1 of the paper.

A.E. BELLUCO: During the Symposium different methods of estimating speculative or potential resources have been presented. I myself believe that it is desirable to combine meetings of working groups with symposia of the type we are now holding, whereby it would be possible to perform evaluations and to advise international organizations on the policies or standards which could be applied on a worldwide basis.

J.A. PATTERSON (*General Chairman*): I would like to comment on Mr. Belluco's call for the development of a uniform approach to the estimation of undiscovered resources. While such an approach would be very helpful, in the present state of development of this kind of work it may be too early to decide on the best method. In the United States of America we are still striving to develop an acceptable system. Our experience of ore reserve estimation suggests that more than one method will be needed to handle the varying situations requiring assessment. International efforts such as meetings like the present one can improve procedures for evaluating resources and can disseminate knowledge of ore deposit characteristics and of criteria for the determination of favourable areas.

D.M. TAYLOR: I am not sure that standardized methods for speculative resource estimation would be possible — or would increase our confidence in the results. The most important thing is to tell people exactly how you arrive at your numbers; they can then judge the implications of these numbers for themselves.

# THE INTERNATIONAL URANIUM GEOLOGY INFORMATION SYSTEM

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## Abstract

### THE INTERNATIONAL URANIUM GEOLOGY INFORMATION SYSTEM.

One of the primary efforts of the International Atomic Energy Agency is the dissemination of information. Within the broad framework of uranium resources, the Agency has published proceedings of many meetings organized by it, and has acquired a very large volume of geological data related to uranium resources, almost all of which are in different formats making a literature search for specific geological data elements a time-consuming, often unsuccessful endeavour. To put all data into a computer format with as much versatility as possible, the Agency engaged four special consultants who designed the International Uranium Geology Information System (INTURGEO). The system consists of four files with a fifth to be added later. These files are the Regional Reference File, Exploration Activity File, Deposit/Occurrence File, and International Summary File. The fifth will probably be the Ore Processing File. At present, the system is being serviced by the commercially developed ADABAS data management system, which may eventually be replaced by a smaller, more specific management system developed especially for INTURGEO to eliminate problems associated with broad usage of a commercial system, and to eliminate several awkward features of ADABAS. This would make easy transport and use of the entire system a simple matter. Development of INTURGEO is on schedule, but every effort is being made to bring it to full operation, including not only data storage and retrieval, but also processing of the data contained. At present the file contains slightly less than 500 records out of an anticipated 3000 and it is expected to be put to use almost immediately.

## INTRODUCTION

Despite the anti-nuclear campaigns, the long lead times and high capital costs, and the uncertainties connected with the nuclear industry, all projections of the need for energy show that, except for a very substantial but inadequate contribution from coal-fired plants, there is little hope of filling the demand for electricity for the next 50 years without large increases in the nuclear sector. The so-called "soft" technologies simply cannot be developed fast enough, nor to an extent large enough to supply the projected demand within an acceptable time.

With this knowledge, the Federal Republic of Germany continues on the nuclear energy path, and France is turning to nuclear power for electrical energy to conserve its premium fuels for transport, while the United Kingdom continues

to develop its nuclear capacity, even though it is approaching self-sufficiency in oil. The countries of CMEA have agreed to increase their nuclear electrical generation capacity to about 30% of the total by 1990. In the United States of America, a flood of enquiries launched in the wake of the accident at Three Mile Island near Harrisburg, Pennsylvania, will undoubtedly create additional delays, but nuclear energy continues to be vitally important to that country.

It is not likely that nuclear energy development will stop. It *is* likely that it will be slowed down for a while.

A reasonable view of the future shows that nuclear energy must play a significant role although the consensus view of experts making future forecasts is that any attempt to specify the most probable level of installed nuclear capacity for the long term is virtually impossible. The best that can be done is to try to establish the most probable high and low limits. In view of this uncertainty, estimates of experts for cumulative life-time requirements of reactors committed to the end of this century range from three to eight and a half million tonnes uranium. Estimates beyond that time vary even more widely. Projections of supplies of uranium that might be made available from undiscovered resources also vary widely, depending to a large extent upon whether the expert is from a "have" or a "have not" country. Projections of production capability from known resources are much more precise, showing that production rates will peak in the early 1990s and then decline into the next century while demand continues to rise, making it obvious that a large effort is necessary to develop new sources of supply.

In recognition of this, the Governments of the United States of America and Canada have begun programmes to appraise their uranium potential. The United States programme includes hydrogeochemical and gamma-ray spectrometric surveys of the entire country including Alaska, coupled with geological mapping and drilling and a large research and development programme. The Canadian programme, although temporarily slowed, was likewise a comprehensive effort to evaluate its uranium resources.

South Africa has been carrying out airborne radiometric surveys supported by geological mapping and geochemical surveys; and a national resources evaluation programme is in progress in close co-operation with the uranium mining industry. The Australian effort is largely done by private companies and has slowed considerably for a number of well publicized reasons. European activity is increasing and the Commission of European Communities is supporting to varying degrees several programmes. In Latin America virtually every country has a uranium exploration programme and some, like those of Brazil, Venezuela and Mexico, are fairly large, while in Africa interest has developed in many countries other than South Africa and Namibia. Asian and Middle East programmes are led by India, Pakistan, and Turkey, and the efforts in centrally planned economy countries are undoubtedly similar in scope to the programmes in the rest of the world.



In addition, in recognition of the need for an evaluation of the speculative potential of the world, a group of experts under the direction of the joint NEA/IAEA Steering Group on Uranium Resources has completed the first phase of the International Uranium Resources Evaluation Project, and a new phase – the Orientation Phase – was recently started. This project has so far collected a very large amount of information on many subjects relating to uranium resources in 185 countries of the world. An abridged version of the Phase One report, entitled “World Uranium Potential”, has been published, and in all likelihood, the full text will be published in the not-too-distant future.

These programmes are developing vast amounts of data, which will be interpreted, and uranium resource reports written, read, exploration programmes initiated, and uranium resources subsequently affected, and more reports written. In many countries, the information in these reports may not be rapidly or readily retrievable in a standard systematic manner on an international scale.

In recognition of this likelihood, and in view of very large quantities of uranium geology related information in the IAEA files, and publications in other places, a group of experts was called to Vienna in the spring of 1978, and in five days of intensive discussions they produced the design for our computer file on uranium resources which we have named INTURGEO – International Uranium Geology Information System. Since its design only a few minor changes have been made. The nature of the information included allows logical grouping into five subsets which are readily adaptable to the multiple file concept of the data management system at present used at the IAEA:

- (1) *Regional Reference File (RRF)*: contains geological production and resource information on a broad regional basis
- (2) *Exploration Activity File (EAF)*: contains exploration information for areas that have been explored to some extent
- (3) *Uranium Deposit and Occurrence File (DOF)*: contains geological, production and resource information of specific deposits and occurrences
- (4) *International Uranium Summary File (ISF)*: contains resource, production and demand information compiled at national levels (data from Red Book and IUREP Phase I report)
- (5) *Ore Processing File (OPF)*: will contain a complete technical description of ore-processing plants around the world and more specific information on costs of production

The data for the OPF file are at present being compiled by the Joint NEA/IAEA Working Group on Uranium Extraction and details on its design will be completed at a later date when data collection is complete.

Still another file, at present inactive, will be reactivated to provide historical and current information on uranium related activities country by country throughout the world. This file is not now considered for inclusion in the INTURGEO system, but some consideration will be given to its inclusion if desirable. It will, in any case, be reactivated.

Our intent with the INTURGEO is at present two-fold — first, to provide a convenient source of information in an easily retrievable form; and, second, as the system grows, to use it in uranium resource evaluation.

## 1. INTURGEO AS A SOURCE OF INFORMATION

The development of INTURGEO is being done more or less with a note of urgency, and for this reason we decided not to develop a large soft-ware system especially to service it, but rather to use the commercially developed ADABAS data management system already in use at the Agency.

Although the ADABAS system is very versatile, it has some features that have proved awkward to use, requiring special symbols, or not altogether desirable changes in the data base design; thus, tentative plans for the system include the possibility of either adapting an existing data management system to accommodate the special features of INTURGEO or to develop an entirely new one. The first option is the more practical, but the second has some distinct advantages which will be discussed later in this paper.

The INTURGEO Information System design was completed in April 1978, work commenced in mid-June and the system was finally implemented in February 1979 after 38 person-weeks of work including 15 of data acquisition. The work load comprised the following tasks:

Task	Work load (person-weeks)
1. Data definition	2
2. Preliminary coding and testing	3
3. Preliminary demonstration	1
4. Final data definition	3
5. Form preparation	3
6. Software preparation	9
7. Data acquisition	15
8. System demonstration	2
	<hr/> 38 weeks

Data acquisition and entry and system demonstration are ongoing tasks. We thought it desirable to enter and update data with our own software for special

editing and display of entry date, and updated information. ADABAS is capable of this but does not provide for the special features we consider essential. The file building and updating processes list exactly what is being put into the file, without running extra routines to see it. As the system grows and we become more experienced with it, minor changes involving software, data definition and form preparation will probably be added either to meet additional needs, or to make the system more versatile.

### 1.1. Description of the files

The reference areas of the files are on a geographical basis, i.e. each record describes a particular location, or reference area. The reference areas covered by the RRF can be political, geological, physiographic or any other appropriate selection. In the EAF, the reference area is the area covered by a particular exploration programme. This may or may not coincide with an area in the RRF. The geographical reference of the DOF may be the name of the deposit or occurrence, or a commonly used name with broad acceptance. The reference area of the ISF is defined by national boundaries.

The co-ordinate reference for all areas in the five files is the central or nearest practical reference point in degrees and fractions of degrees of latitude and longitude. Although the reference points in the Ore Processing File will be the actual location of the plant, some consideration will be given to defining the area tributary to the plant — or the production unit. UTM co-ordinates were considered desirable but we decided that latitude and longitude would be more satisfactory in the long run and more universally available. Some conversion routines are considered for the future. Some consideration will be given, after we have developed our own system software, to storing a series of co-ordinate pairs — perhaps up to 1000 or so to provide a page-sized, stylized but recognizable plot of the areas described by the records in the file. The plot would be a standard A4 page size, but would be definable as well. An example of this is shown in Fig.1.

In addition to the name and location, three other data elements common to all files are:

- (1) Record identification number
- (2) References with the titles and author and year of publication
- (3) Name of recorder or updater including the date and any comments the person wishes to make on the data.

A schematic comparison of all files in the system is shown in Fig.2.

*1.1.1. Regional Reference File.* This file will systematically store geological information pertaining to a region. It was decided that the best way to do this was to determine what percentage of the region is covered by each unit shown on the legend. A more usable system is sought, however. It is always possible that

DRAWN FOR URANIUM RESOURCES  
A FUTURE CAPABILITY OF INTURGED INFORMATION SYSTEM

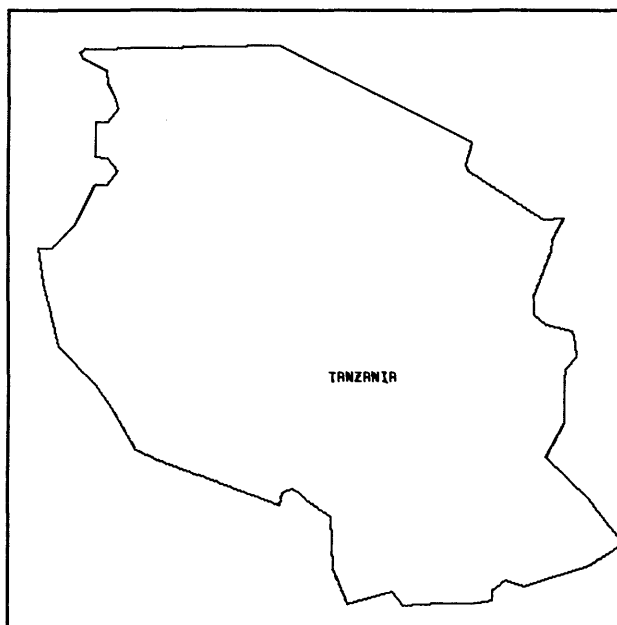


FIG.1. Stylized map of the country of Tanzania.

the desirable geological features of an area may not be expressed at the surface; thus, may not even appear in the file. An evaluator using a geological map will probably know this, but it may not be known to other INTURGED file users who may be thus misled. It is, therefore, vital that some additional data be stored. The very least should contain a geological column, and possibly the best would be cubic kilometres of a geological unit to a predetermined depth, expressed as a percentage. This is considered one of the key problems with the system now.

The main elements or groups in this file are:

- (1) Identification (name and location)
- (2) Region type
- (3) Region size
- (4) Geological environment
- (5) Uranium statistics
- (6) Level of exploration
- (7) References
- (8) Recorder and comments

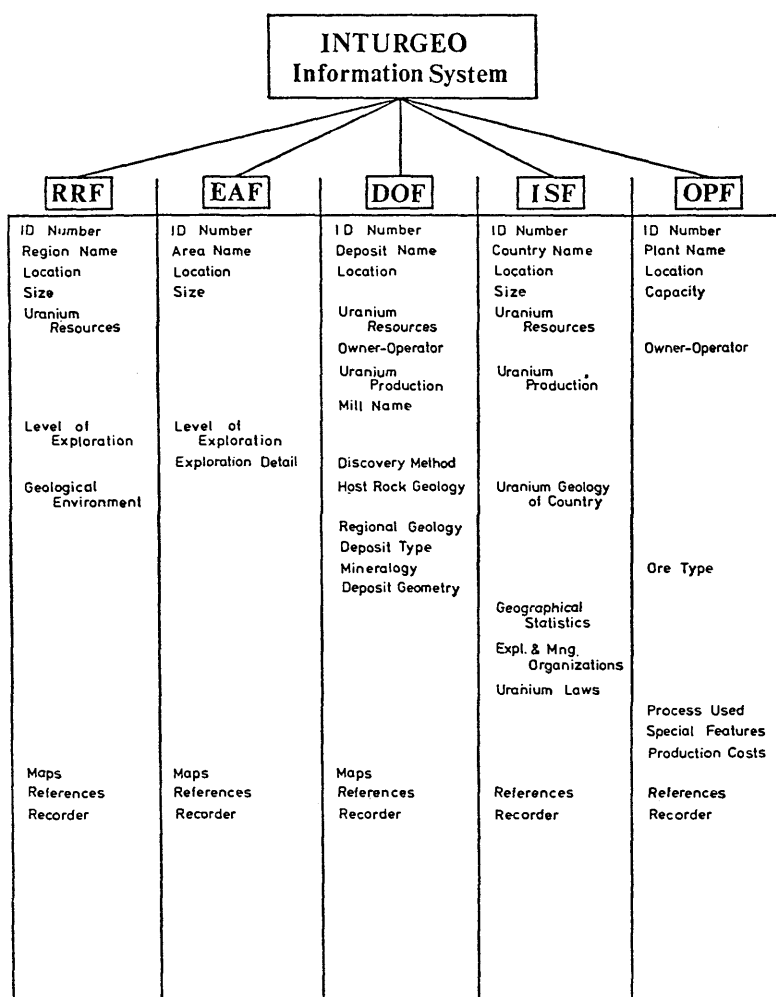


FIG.2. Schematic comparison of the files in the INTURGEO system.

The region is identified by its name, including former or alternate names, and a central point. It can be geological, tectonic, political, or something else that is reasonably definable. Area is in square kilometres. The geological environment is a group of data items defining each unit, the per cent of the area covered by the unit and its geological age (coded by number) are included. The file structure provides for lithological type with colour and modifier, depositional environment, structure, tectonic setting, igneous activity and any elements of interest that are enriched in the unit.

Uranium statistics of the region are characterized by three groups of data. The first group contains the type of uranium deposit and number of deposits and occurrences of that type found in the region. The second group contains production data. The third contains uranium resource data as reported in the Red Book, i.e. category of resource, tonnes U, cost of production category and year of estimate. An example of an entry in this field would be 1000 tonnes of Reasonably Assured U Resources recoverable at US \$80–130/kg U in 1977.

The exploration of a region is simply classified by level of activity with encompassing dates, past exploration expenditure with dates and present level of activity.

Three of the four files include information on the type of maps available and their scales. References and recorder with comments are included in each record as appropriate.

*1.1.2. Exploration Activity File.* This file contains a summary of the uranium exploration that has been done in a specific area, so that both the uranium favourability of the region and the need for further exploration can be properly assessed. The main data elements are:

- (1) Region identification
- (2) Size
- (3) Level of exploration
- (4) Details of exploration performed in the region
- (5) References
- (6) Recorder and comments

Five of these data elements appeared in the Regional Reference File and have previously been discussed. The main part of the record describes the types of exploration and results. The exploration entity, joint partners or members of the consortium are identified, if possible with the date of the exploration, the size of the area explored and its central point. The exploration platform is identified (satellite, airplane, helicopter, surface or subsurface) along with a variety of methods, number of samples, and sample medium used with the platform indicated. An example of a surface exploration programme would be "Geochemical, 1000 stream sediment samples". The record also contains comments on the technique, size of the sampling grid or line spacing of an air survey, etc. The success of the programme is indicated by the Number of Targets Identified and Summary Results. Costs (US \$  $\times 10^3$ ) are also included. The record repeats this detail for each type of exploration performed in the area, and, as well, for each platform used.

*1.1.3. Deposit/Occurrence File.* The Deposit/Occurrence File is designed to characterize a uranium ore deposit or occurrence. This file will undoubtedly

be the most interesting from the standpoint of use on an international basis, and it is the content of this file that will be most often referenced in the anticipated future uses of INTURGEO which are described later. Fortunately, data are readily available from a large variety of sources and conversion to INTURGEO a relatively inexpensive task.

Essentially, the file comprises the following:

- (1) Identification and location
- (2) Owner or operator
- (3) Mining status
- (4) Resource statistics
- (5) Discovery
- (6) Deposit type
- (7) Geological description of host and related rocks
- (8) Regional geology
- (9) Mineralogy
- (10) Deposit geometry
- (11) References
- (12) Recorder

The data items for identification, location, references, and recorder are the same as those previously discussed in the other files.

The name of the owner or operator is essential information from a legal standpoint. Mining status is indicated by whether the uranium deposit is an anomaly, occurrence, prospect, potential producer, producer or past producer. If the deposit is being or was mined, the mining method and processing plant are specified. The processing plant name will provide a link with an ore processing file.

Resource statistics comprise five data groups. The first lists all commodities present, with their status (Ex.: Au, by-product). Uranium production includes tonnes uranium produced with the dates, and production capacity in tonnes uranium per year with a date. Grade of the deposit, cut-off, maximum and average per cent uranium are listed with a date. Data on the uranium resources of the deposit are handled in much the same way as the uranium resources of a region from the first file, i.e. "Reasonably Assured" tonnes uranium, cost of recovery and date of estimate.

The method of discovery of the deposit is described in the same manner used for the survey detail in the Exploration Activity File.

Deposit type and sub-type are chosen from a classification system devised by the IUREP Steering Committee. In addition, other classification systems will be considered to allow for international differences, and to provide wider flexibility of use of the system.

Descriptions of host and related rocks include the geological unit name and alternate names, and volume per cent of host. The file will also accommodate

multiple hosts. If the geological unit being described is not the host rock, its relationship to the host is defined. The geological age, or range of ages is coded by number for easy sorting of the file by age as well as for data retrieval.

Lithology, descriptor and colour are included. The organic content is specified as rich, moderate or poor and depositional environment, structure and tectonic environment during deposition are described as well as post-depositional events important to mineralization, and their ages, alteration and igneous activity. Also included are descriptions of related rocks not already described. Major structures and tectonic environment describe regional geology.

Mineralogy of the deposit is described by age of mineralization (in millions of years), minerals and abundance (in vol.%), gangue minerals and qualitative abundance, and elements associated with mineralization and their abundance.

Deposit geometry is described by depth to top, maximum length, maximum width, strike and dip of the body, if available. The last items in the file, references and recorder, have previously been discussed.

*1.1.4. International Summary File.* The International Summary File summarizes uranium information on a country-by-country basis. Information elements are basically from the bi-annual publication Uranium Resources, Production and Demand (the "Red Book") and the recently published World Uranium Potential, An International Evaluation, which is the result of the first phase of the International Uranium Resource Evaluation Project (IUREP). Both of these are jointly prepared by the NEA and the IAEA. Other information not available in these documents is obtained from standard reference books and other sources considered appropriate for the information.

The main elements of the file are:

- (1) Location
- (2) Geographical statistics (including population, capital, transportation)
- (3) Verbal summary of geograpy
- (4) Verbal description of geology
- (5) Uranium favourability
- (6) Uranium resources and production
- (7) Exploration and/or mining organizations active in the country
- (8) Uranium law
- (9) References
- (10) Recorder

The location of the centre, size of the country in square kilometres, population and capital city along with data on transport, including total kilometres of road, railroad and navigable river length, form an important part of the geographical statistics of the file. Average and maximum elevation, rainfall and temperature are also reported on regional basis. While this information may be meaningless



or even cumbersome for a developed nation, it is of considerable interest in uranium exploration in many of the developing countries. Verbal descriptions of geography and geology expand on the purely statistical information.

Uranium favourability determined during the IUREP exercise is classified from nil to very high and is included with an expanded verbal description of regions considered particularly favourable for uranium mineralization as reported in World Uranium Potential.

Uranium resources of the country are recorded by category, tonnes U, cost of production and date of estimate and are further classified by geological type of deposit. Annual uranium production in tonnes U is also included. The names of government organizations or private companies mining or exploring for uranium in the country and their addresses provide additional useful information.

Excerpts from uranium exploration and mining laws tell whether foreign investment is allowed (yes or no with a date), and who may conduct exploration (the state only, the state jointly with private companies, the state and private companies independently or just private companies) with the date of this information. Additional uranium law information is then briefly summarized in verbal comments. The last items in the file are references and recorder.

*1.1.5. Ore Processing File (in planning stage).* A joint NEA/IAEA Working Group is preparing a report on all uranium ore processing plants in the world for which it is possible to obtain the data. The manner in which the information is being gathered and the technical expertise of the members of the Working Group are expected to ensure considerable accuracy of the information in file.

Although the exact details of the file have not been decided it is reasonable to assume that the following items will be included:

- (1) Name of plant
- (2) Location
- (3) Operation
- (4) Capacity
- (5) Geological types of ore processed
- (6) Process used
- (7) Special features
- (8) Cost of uranium production
- (9) References
- (10) Recorder

The first three items in the above list are self-explanatory. Capacity will be in metric tons of ore per day and tonnes U per year. Types of ore processed provide a link with the Deposit/Occurrence File, and also determine the type of process used. If special features are a part of the plant process and are known they will be described.

The joint Working Party is also attempting to develop a method for estimating uranium production costs related to cost indices and amount of work involved in production as a part of this exercise. This will also form a feature of this file.

## 1.2. Use of the files

At first we anticipated that the files would ultimately contain in the neighbourhood of 1000 records, which is not unreasonable for storage of information of the type described. It will, in fact, provide a very good source of considerable data on the world's uranium resources. Many of the results of the first phase of IUREP have been already installed in the system and information from a very large USGS open-file report, entitled "Principal Uranium Deposits of the World", is now in the system. Coding forms have been developed in draft form for use in future IUREP activities and to use for data conversion from other sources.

The most immediate use of INTURGEO is simply as a place to keep information in an easily retrievable form in a systematic manner. From this we will be able to provide various compilations of information either to satisfy queries, or to publish.

We especially expect to assist additional IUREP activities, first, by providing information to the investigators and, second, by providing a systematic guide for collecting specific information and a place to store it for easy retrieval. The retrieval of reports of areas similar to the ones being considered during the IUREP exercise will also provide additional guidance to the investigators.

## 2. INTURGEO AS A COMPUTER PROCESSIBLE FILE

Regarding this conception of the INTURGEO Information System it is necessary to point out that there are several quite sophisticated uses for such a set of files. Considerable effort in several places in the world by different groups and organizations indicates the feasibility of using the information contained in the INTURGEO files as data for further computer analysis, and eventually to develop uranium resource evaluation capability within the IAEA to appraise the speculative resources of any area for which proper geological information can be obtained. It should be emphasized that resources so estimated would perhaps be even more speculative than the term implies at first, but it could serve as an exceptionally good tool for developing priorities for subsequent activities, by locating the most favourable areas for further work. This would be an invaluable aid to IUREP activities if it could be initiated soon.

Several resource appraisal methods using computer analysis have been developed to estimate undiscovered mineral resources, using as many different approaches. Some of these are discussed here as they may relate to INTURGEO.

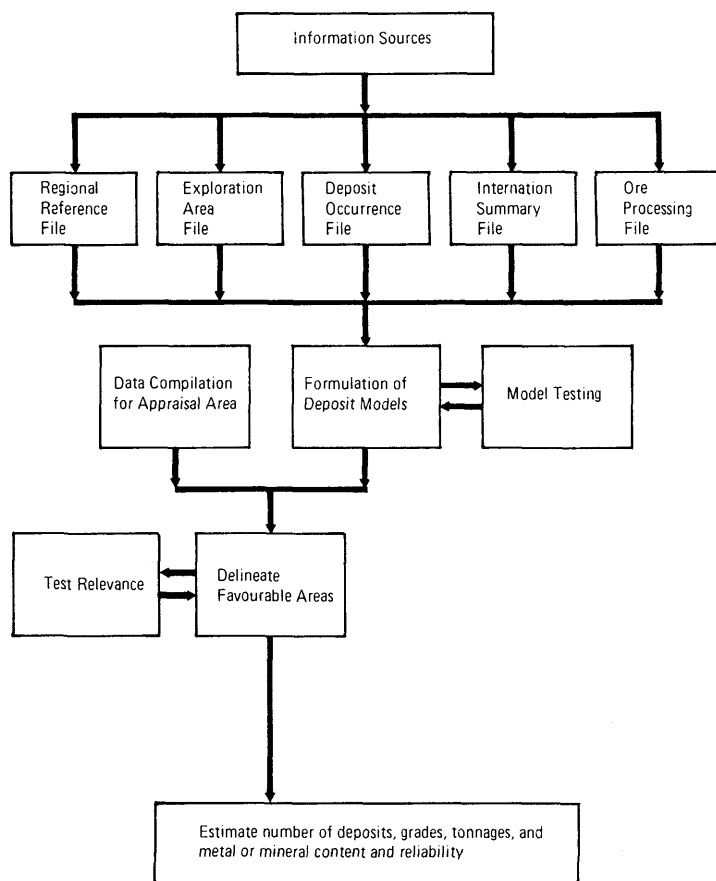


FIG.3. Flow diagram for deposit modelling.

## 2.1. Deposit modelling

The most natural method of mineral resource analysis is to compare the features of a well-known area with those of an area being studied, then on the basis of similarities to estimate the quantities of resources that might be in the area under question. This method is especially applicable to estimates of undiscovered uranium resources, and has been used by many organizations. The main problem with the method is that it is almost purely subjective, and so the experience or lack of experience of the appraiser plays an inordinately large part in the appraisal. Some of this subjectivity can be removed by computer processing

of information, and routines to do this have been and are being developed. We feel that Deposit Modelling is the most reasonable method that could be employed by the IAEA using INTURGEO. It would be expected, however, that results of such an analysis would be substantiated through other means until we have had considerable experience with it. Figure 3 shows schematically the part played by INTURGEO in Deposit Modelling.

## **2.2. Delphi estimation**

This is an interesting methodology based on iterated opinions of experts. The very highly subjective nature of this method is tempered by re-circulating the opinions of all experts to all other experts and asking for a second opinion. This is probably not, at this time, of any substantial value to the Agency, but could be considered in the future for special purposes. INTURGEO could provide considerable background information to participating experts. (Experts could also be asked for their opinion of the results of an exercise employing any other appraisal method.) Results of the Delphi method are statistically analysed to obtain final estimates of ranges of quantities of resources.

## **2.3. Unit regional value estimation**

This is quite complicated requiring a large quantity of data concerning all mineral resources and so is not considered of significant interest to the Agency's INTURGEO system at present, although it is conceivable that the INTURGEO Information System could provide data to such an estimation method being performed by another organization.

## **2.4. Abundance estimation**

This relates the average content of a mineral in the earth's crust to the quantity of the resources of the same mineral. Several software routines have been developed for estimating uranium resources and, although the accuracy is unproved, it provides an interesting technique to determine the possible range of uranium resources that might be in an area.

## **2.5. Volumetric estimation methods**

These extrapolate from a representative mean concentration of a unit volume to a volume being appraised. A unit volume of the earth's crust contains a mean content of mineral based on past production experience. On the basis of this bit of information, an estimate of resources of a similar unit volume of the earth's crust can be made. The method requires little data and so is not considered

particularly useful except for conceptual information, but as more data are used the method becomes more and more similar to Deposit Modelling. It is conceivable that this method may employ information from INTURGEO for studies of rather limited usefulness.

## 2.6. Integrated synthesis

This combines all the above methods and, although considered quite accurate, may not in practice be feasible for uranium resource estimates made from information in INTURGEO. It is, however, very likely that some of the methods may be employed for a semi-integrated synthesis.

Other methods are now under development, and more will be developed in future years. Most of these require the most basic data (drill-hole logs, physical measurements, etc.) of a type either not possible to collect in significant quantities, or not practical to include in the INTURGEO Information System, and so will be of no value in the system. Others that can be applied will be considered for inclusion as they are developed.

## 3. THE FUTURE OF INTURGEO

INTURGEO incorporates virtually all types of uranium resource related information into one or another of its main five files. Its role as a source of information will always remain its primary one, and its role as a data base or a computer processible file will be important but secondary. As far as we know, its design incorporates the most comprehensive collection of information on a single metal anywhere in the world, and when fully implemented can serve as model for others.

As an information file, we expect that the contents of the files may be published at appropriate intervals for world-wide dissemination of the most current information on uranium geology, reserves, production and exploration. In addition, we anticipate easy time-sharing access to all countries. To do this, the INTURGEO Information System may be made portable so that a copy can reside at several central locations in the world. To facilitate this, we have considered development of a simple new system for query, manipulation and display to reside with the data base. The ADABAS data management system now in use was developed commercially and its application in this manner would probably not be justified. As different elements of software are developed and added to the system, they would be included in the periodic updates of the files.

The applications are expected to be as diverse as the users. The inexperienced geologist may consult the data base on regional geology, deposit descriptions, etc. thus alleviating his lack of experience to some extent. The resources analyst may

use sophisticated techniques to make resource appraisals and researchers will be able to examine complex geological relationships and thereby test current hypotheses on the formation of uranium deposits.

We have so far slightly more than 400 records in the system, and it appears that one original estimate of a total of 1000 may have been far too conservative, except as a beginning target. Of special significance to this system is the Regional Reference File. The addition of records to this file requires regional geological interpretations which in turn require some interpretation in data conversion, thus increasing the cost of each record. Its principal use is likely to be in regional uranium resource evaluation which at present is being carried out under the International Uranium Resources Evaluation Project previously mentioned.

The first phase of this project has been completed and subsequent more detailed appraisals will employ consultants from various countries with varying amounts of experience, who may never even see each other. Therefore, one of our most basic problems is simply to be able to add up the quantities reported by the consultants — that is, to provide a degree of standardization of methodology and interpretation that prevents adding “pears and potatoes”. We expect to achieve this by requesting specific data from the consultant, in a specially designed questionnaire, processing the information, and return to the consultant some guidelines to use his own estimate. To do this, we would employ a somewhat modified deposit modelling technique, drawing heavily from the Regional Reference File and the Deposit/Occurrence File for basic information.

There is a certain urgency in our effort. The consultants engaged to carry out the Orientation Phase of the International Uranium Resource Evaluation Project, which will start again soon, will both provide data which we are prepared to enter into the system, and require information from it which we are at present only partially able to provide. However, we hope to move forward sufficiently rapidly to overcome this shortcoming of the system in time to provide information for this effort before many more evaluations have been made.

#### 4. CONCLUSION

In summary, INTURGEO serves three main functions:

- (1) Worldwide dissemination of uranium resource information
- (2) The provision of rapid, reliable answers to a variety of queries
- (3) The provision of a data base to assist in uranium resource appraisals

At a time, which has not yet been decided, the contents of INTURGEO will be prepared in a suitable format for publication. As the system grows, we would also expect that its users would increase, thus augmenting the variety of uses to

which the system is put, and as an aid to global uranium resource evaluations several appraisal techniques will be installed and used when considered desirable.

INTURGEO is still an infant and in terms of effort a monumental task lies before us. We have gained considerable support in both financial and in human resources. Several organizations have been approached for assistance at many levels, and more will be approached in the future. Our main problem today is data conversion — the system has been designed by experts; all files except one are installed and contain some information, albeit in small amounts; we can retrieve information from all files; and standard forms to facilitate systematic collection of information have been printed.

It is our hope that through the involvement of several organizations at an international level we can promote interest in INTURGEO, thus perpetuating its existence, keeping it up to date and, most important of all, offering a source of accurate, up-to-date information to all nations undertaking a search for uranium.

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### DISCUSSION

B.L. DICKSON: Will the deposit/occurrence file (DOF) contain information on the state and variability of radiometric disequilibrium in the deposits?

M.V. HANSEN: At present there is no provision for information about disequilibrium in the deposit/occurrence file. However, your question suggests that we should look into this.

B.L. DICKSON: Will the ore-processing file contain information on the mining and grade control methods for the individual deposits?

M.V. HANSEN: The ore-processing file has not been completely designed yet. But it might in any case be more appropriate to include information on grade control methods in the mining section of the DOF.



## GENERAL DISCUSSION

### ON URANIUM RESOURCES, DEMAND AND AVAILABILITY

D.M. TAYLOR: At this Symposium we have discussed resource evaluation and mining techniques, and the impression may have been gained that all that is required to meet uranium demand is to mine the resources, and also that all the resources could in fact be mined. This is not only a gross oversimplification but is also untrue.

What do we mean by “resources”? First, according to the NEA/IAEA classification scheme, there are what are called “reasonably assured resources” (RAR). This term relates to the uranium in known mineral deposits, for which estimates of tonnage and grade are based on specific sample data. We have more confidence in the existence of these resources than in that of any other type but we are not certain that they all exist. The lower-cost category of RAR (up to US \$80/kg U) is known as the reserves for the purposes of the “Red Book”.

Secondly, there are “estimated additional resources” (EAR). Strictly speaking, these are resources in little-explored deposits but, by extension, the term also includes some resources in well-explored or even in undiscovered deposits. EAR are resources which are believed to exist, but we have less confidence in figures for them than in those for RAR. EAR need exploration, in some cases a great deal of exploration, to reclassify them in the RAR category, and it may not be possible to reclassify them at all. Many of the resources that are reclassified will move into the higher-cost (sub-economic) category, which are exploitable at over US \$80/kg U. RAR and EAR are often collectively called “known” resources.

Thirdly, there are “speculative resources”, which are *thought* to be located within deposits discoverable by existing techniques and exploitable at costs of more than US \$130/kg U. We do not know of their existence. Some are thought to exist in areas for which geological information is insufficient or totally lacking and in which there are no known occurrences. A lot of exploration will be needed for us to reclassify a significant quantity of speculative resources in the RAR category — we do not know whether this exploration will be undertaken, or, if undertaken whether the discoveries will actually be made.

It should be remembered that a reclassification of resources from speculative resources to EAR and from EAR to RAR, which is the result of exploration, will only redistribute the resources and will not change the absolute quantity, whatever that may be. In other words, we must not expect new speculative resources to replace mined reserves. I should also emphasize that the figures obtained for the different classes of resource should not be added together, because different levels of confidence are applicable to each of them and speculative resources are given not as a single value but as a range.

Let us turn now to demand. Of course, demand forecasts are always open to criticism. You have heard at this Symposium that our "Red Book" estimates of demand have shown a decrease over the past four years and you may be wondering whether this trend will continue. Mr. Ziegler believes it will not and he gave a hypothetical example showing that demand could, under certain circumstances, increase more rapidly than predicted<sup>1</sup>. Mr. Patterson, on the other hand, stated that the contribution of nuclear energy to future energy supply "will be much less than had been expected just a few years ago"<sup>2</sup>.

In any case, although this is admittedly just a forecast, a conservative estimate of the cumulative uranium commitment to the year 2025 could be around 7.5 to 8 million tonnes of uranium.

So let us now consider the question of availability. It is obvious that, in spite of the demand forecast I have just mentioned, we cannot compare uranium resources directly with uranium demand. Uranium in the ground cannot be used to fuel nuclear reactors or to feed a uranium mill! What is important is the availability of the uranium.

Let us look once again at the different resource categories. Within the RAR category there are many deposits which may never be produced. Some of these resources, especially the higher-cost ones, will be left behind and irretrievably lost when higher-grade resources are mined. Some are sub-economic now and may never become economic (although this may not prevent them from being mined). Some may never be mined because of political or environmental constraints (examples of this are the Randstad shales in Sweden and the Kvanefjeld deposit in Greenland). Another factor is time. Production from certain known deposits may not be possible in the near future, or they may, regardless of their size, be exploited only at a very slow rate, or exploitability may be constrained by their physical characteristics, location, co- or by-products, etc.

As far as EAR are concerned, once we have reclassified these resources, or some portion of them, in the RAR category — which in some instances means that they will first need to be discovered — the same problems with production arise as with RAR, but the time scale is now longer.

Speculative resources must first be discovered. For various reasons, including lack of access to prospective areas, lack of outcrop in prospective areas, terrain which is difficult to explore and lack of exploration funds (this last may be caused by cutbacks in exploration expenditure owing to falling prices or, alternatively, the risk may be thought too high in certain areas), many of them may never be discovered, or if discovered, production from them will be subject to the constraints already described for RAR — but once again the time scale is longer.

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<sup>1</sup> IAEA-SM-239/12, these Proceedings.

<sup>2</sup> IAEA-SM-239/21, these Proceedings.

However these are not the only constraints. As Mr. Ziegler pointed out<sup>3</sup>, even after the uranium has been produced we cannot be certain that in all instances it will be made available to the consumer who needs it. Such a constraint is very difficult to quantify. Forecasts of production capabilities — as contained in the “Red Book” — are a first step on the way to matching resources to demand. These estimates of maximum production capabilities will prove correct only under ideal conditions and slippage caused by less than ideal conditions, even if they last for only a short time, may take several years to catch up on. In the real world these production capabilities cannot be equated with availability. This is also true, of course, for the uranium production capabilities with the different types of resource (including speculative) projected by the United States Department of Energy and presented in Mr. Patterson’s paper<sup>4</sup>.

In conclusion, I should like to say that sufficient uranium is available to meet demand in the short term but it appears likely that we will need uranium from undiscovered, including speculative, resources before the end of this century. With the long (and increasing) lead times from the beginning of exploration to production, statements being made at this early stage that undiscovered uranium will be required may give some cause for concern. We must, however, take into account the fact that we are already exploring and that today’s speculative resources could be tomorrow’s deposits. We should be careful not to overestimate exploration and production because by doing so we may cause or aggravate a situation of excess supply, which would be as damaging to the industry as a shortage. However, we should continue, with one eye on the clock, to encourage both exploration and production.

There are many constraints that can affect exploration, discovery, production and availability. These constraints will need to be evaluated before we can view the uranium supply position in the longer term with any degree of confidence. And in order to evaluate these constraints it will be necessary to discover, produce and make available more uranium.

C.D. MASTERS: You made a particular point of noting that the EAR category includes some deposits that are in fact undiscovered or unknown. Do you think that this is right, since you then have to add together the resources that represent the growth potential of existing reserves and those that are unknown or undiscovered, although the probability of discovery varies greatly between the two components?

D.M. TAYLOR: I pointed out that EAR include an element of undiscovered resources simply in order to indicate the extent of our confidence, or rather lack of confidence, in their existence. I do not believe that they should be called “known” resources if undiscovered resources are included. One can, however, be

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<sup>3</sup> See footnote 1.

<sup>4</sup> See footnote 2.

quite confident in some instances that these undiscovered resources do exist, and it is the degree of confidence that determines whether they are to be placed in the EAR or speculative resources category.

C.D. MASTERS: What proportion of EAR represents growth potential and what proportion is in fact undiscovered?

D.M. TAYLOR: My personal feeling is that a relatively small proportion of EAR is undiscovered (none in the case of Australia), but this depends on the definition of the term "discovered".

P.D. TOENS: With regard to the definition of estimated additional resources, it should be noted that in Australia and South Africa deposits that are merely *thought* to exist are not included in the EAR category. This will be pointed out in the next edition of the "Red Book".

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