

New developments in uranium exploration, resources, production and demand

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FOREWORD

Despite the worldwide lack of public acceptance of nuclear power, uranium continues to be an important energy resource.

In 1990, there were 423 nuclear power plants in operation with a combined electricity generating capacity of 326 GW(e). Over 1900 TW.h electricity were generated, equivalent to 16.6 per cent of the total. To achieve this, 54 000 tonnes U were required as nuclear fuel. The market value of the raw material alone is estimated at over US \$3000 million.

In view of the economic importance, the International Atomic Energy Agency and the Nuclear Energy Agency of the OECD have had a long standing interest in uranium exploration, resources, production and demand. This is reflected in numerous publications covering different aspects in this field. One of the more important ones is the periodic publication of both Agencies "Uranium Resources, Production and Demand", the thirteenth edition of which was published in early 1990.

It was the objective of this Technical Committee Meeting, whose proceedings are presented in this volume, to bring together specialists in the field and to collect information on new developments, especially from countries which in the past considered uranium a strategic commodity and the related information as confidential or even secret.

This meeting showed that this attitude of a number of important resource countries has changed to a more open information policy, in line with the liberal exchange of information practiced since many years by the larger majority of countries in the world.

In view of this, the participation and contributions made at this meeting are gratefully acknowledged. Thanks are also due to the session chairmen: W. I. Finch (United States Geological Survey, United States of America), J.M. Matos Dias (Empresa Nacional de Uranio, S.A., Portugal), S.D. Simov (Uranium Company "Rare Metals", Bulgaria), and F. H. Barthel (Bundesanstalt für Geologie und Rohstoffe, Germany).

The IAEA staff member responsible for the organization and implementation of the meeting was E. Müller-Kahle, Division of Nuclear Fuel Cycle and Waste Management.

EDITORIAL NOTE

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This text was compiled before the recent changes in the former Union of Soviet Socialist Republics.

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SUMMARY OF THE TECHNICAL COMMITTEE MEETING

Since about 1980, the international uranium market has been experiencing a decline, mainly as a consequence of the slower than expected growth in nuclear power development in the world.

This decline has affected all areas of the uranium industry, ranging from exploration to production, and has impacted also on the employment and the financial standing of the mining companies.

Nevertheless, in order to remain viable and to maintain the capability to compete on the market, companies have streamlined their activities, where possible and where needed. Exploration has concentrated on a few countries, which offer a potential for low cost, high grade uranium deposits in a stable socio-economic environment. Production is adjusting to lower prices by closing those mines which produce at costs not covered by sales prices, and concentrating on mines which can still operate profitably under these conditions.

In the last ten years, the uranium exploration expenditures, expressed in current terms, have declined in WOCA from about US \$750 million in 1980, to an estimated US \$120 million in 1990. While in 1980, there were ten countries (Australia, Canada, France, the Federal Republic of Germany, Italy, Mali, Niger, South Africa, Spain, the USA) which reported exploration expenditures of more than US \$10 million in current terms, in 1990, this level was reached in only five countries (Australia, Canada, France, India, the USA). Despite this decline, important discoveries were reported in some countries.

This shows the high level of efficiency uranium exploration techniques have reached. Contributing to this are the advanced economic assessment methods of exploration results as well as technical progress based on a better understanding of uranium deposit models and the exploration methods applied.

As uranium exploration in its regional approach is concerned with the investigation of geological media such as rocks, soils, waters and stream sediments, the acquired data are increasingly being used for environmental purposes. A good example is an IAEA supported project in Sudan, which detected a potentially hazardous drinking water contamination due to an anomalous radon content. In addition, the natural radioactivity measured in airborne surveys provides useful background data, against which later measurements can be correlated, which may include later contaminations, as for example contamination originating from nuclear fallout.

Despite the decrease in exploration activities, the level of aggregated uranium resources in WOCA has remained fairly stable during the period 1980 - 1990 despite the depletion due to the cumulative production of nearly 380 000 tonnes U. The Reasonably Assured Resources (RAR), recoverable at costs below US \$80/kg U, moved in the narrow range between 1.75 and 1.50 million tonnes U.

This small decline reflects the reassessment of resources in a number of countries, both downwards and upwards. For example, the USA reduced its resources of this category from over 360 000 tonnes U as of 1981 to about 106 000 tonnes U as of 1991, or by 70 per cent. France also reported decreases over this time period from over 59 000 tonnes to less than 24 000 tonnes U. Australia, however, reported significant increases in its resource levels. The RAR of the low cost category increased from 294 000 tonnes U to nearly 470 000 tonnes U, or by nearly 60 per cent.

The dynamic behaviour of this resource estimate is due to economic as well as technical reasons. Economic criteria include the reclassification of the recovery costs due to cost changes prompting the transfer into a higher cost category or even complete elimination from the data base as consequences of mine closures. The technical reasons include depletion due to production as well as addition due to new discoveries. Canada is a good example for such a development, as its resource stock, which was divided between the quartz-pebble conglomerate deposits in Ontario, eastern Canada, and the unconformity-related deposits in Saskatchewan, western Canada, is shifting to a larger degree to Saskatchewan, due to the eminent closure of the Elliot Lake mining district and the new discoveries in Saskatchewan.

New information on the uranium resources in Bulgaria, China, Czechoslovakia, Hungary, Romania, the USSR and Viet Nam are forthcoming to an increasing degree. While the resource classification systems used in these countries can be correlated with those systems used for example in Australia, Canada, Niger, South Africa and the USA, there are difficulties for the resource specialists in those countries which are not familiar with the relevant concepts, to determine the cost categories expressed in US \$/kg U. Nevertheless, these resource estimates will be incorporated into the relevant data base published periodically in the report Uranium Resources, Production and Demand, also referred to as the "Red Book". This involves a learning and teaching process on the part of the national resource estimators and the IAEA secretariat and shows to what level of sophistication the Red Book exercise, as regards terminology and information sought, has developed over the years.

The aggregated known resources (equivalent to RAR + EAR-I) recoverable at costs below US \$130/kg U, of Bulgaria, China, Czechoslovakia, Hungary, Romania and the USSR are nearly 900 000 tonnes U. The bulk of these resources are in the USSR with over 70 per cent of the total, and in Czechoslovakia with about 17 per cent. It can be expected that this resource stock will also have to be subjected to adjustments and modifications in view of the changing economic conditions. This development has started already in Czechoslovakia, where mine closures, for example in Pribram, may remove resources from the stock.

It is of geological interest that a large portion of these resources is hosted in sandstone and vein type deposits, as well as in deposit types referred to as unconventional, where uranium occurs in association with other metals, such as copper, molybdenum, phosphate, and rare earth elements. In many cases, the economics of uranium is determined by the economics of the main revenue earning metal. These predominantly occurring deposit types

allow the conclusion that unconformity-related deposits are not yet known in these countries. It can be assumed that explorationists, as for example in China, are giving high priority to the search for this deposit type, which is known for its high grade and consequently low cost resources.

The uranium production between 1980 and 1990 - unfortunately only information from WOCA countries is available - shows a similar development as described for the uranium exploration expenditures. The aggregate production decreased from over 44 000 tonnes U in 1980 to less than 30 000 tonnes U in 1990, with a further decreasing trend. The development in individual producer countries was not uniform, reflecting the different internal and external conditions such as the deposit characteristics, and cost levels. For example, the uranium production in the USA decreased from 16 800 tonnes U in 1980, equivalent to 38 per cent of the WOCA total, to just over 3800 tonnes U in 1990, or about 13 per cent. Also South Africa, which had produced over 6100 tonnes in 1980, reduced its production to less than 2500 tonnes in 1990. Canada produced 7150 tonnes in 1980, increased the production to a peak of 12 400 tonnes U in 1987, and, in adjusting to prevailing market conditions, reduced the production to 11 300 tonnes in 1989 and to about 8700 tonnes U in 1990. While these countries experienced the decline of their uranium production, Australia was able to secure a stable market position for its uranium; in 1980 the production was still about 1600 tonnes U, reached a peak of nearly 4500 in 1982 and zigzagged to a level between 3600 - 3800 tonnes U.

In the course of this production adjustment, numerous producer companies left the uranium business, including for example many oil companies. This led to a concentration of the uranium production in the hands of very few companies, which control the largest share of the WOCA production. Interesting is also an insight into the changes in the ownership structure of the uranium production between 1980 and 1990; the ownership structure distinguishes between domestic and foreign, as well as between private and government. The comparison of the productions of 1980 and 1990 by ownership shows a strong decline of the private domestic ownership from over 63 per cent in 1980 to just below 34 per cent in 1990. However, the domestic governments owned nearly 12 per cent of the 1980 production and over 28 per cent in 1990. A similar development is evident among private foreign entities, which controlled 24 per cent of the 1980 production and over 36 per cent in 1990.

In general, it seems that mining companies which exclusively produce uranium are very vulnerable in the current open market. On the other hand, companies which are integrated vertically in all nuclear fuel cycle activities, and companies which produce a number of other commodities in addition to uranium, have a good chance for survival. The trend towards these two concepts is evident in a number of cases, such as General Atomics investing both in uranium mining ventures and fuel cycle activities and Cameco exploring both for gold and diamonds and getting further involved in fuel cycle related activities with its interest in laser enrichment technologies.

The above information is still limited to WOCA as mentioned at the outset. This is due to a lack of uranium mining statistics and technical information on, for example, production centres,

production methods, etc. This is contrary to the situation in the area of uranium resources, where the new openness revealed interesting information.

The uranium market in WOCA, the underlying cause of the developments described, changed drastically in the time period discussed. While in 1980 the average contract price was estimated at about US \$90/kg U and the spot price was about US \$62/kg U, these two indicators declined to about US \$50 and 25/kg U respectively in 1990. In a parallel development, the uranium volumes traded on the spot or near term market approximately tripled from about 4000 tonnes in 1980 to 13 000 tonnes in 1990. While this number may include amounts which were traded several times thus increasing the total volume, it shows the increased activities of traders, brokers and intermediaries as new participants in the market.

A review of the WOCA supply and demand situations in 1980 and 1990 reveal significant differences. While in 1980 reactor related requirements were approximately 25 000 tonnes U, they increased to nearly 44 000 tonnes in 1990, equivalent to an increase of over 75 per cent. In the same time span, the supply from the mining industry declined from 44 300 tonnes U to about 30 000 tonnes U. This development marks the transition from a period of oversupply, which in 1980 amounted to nearly 25 000 tonnes U, to one of production gaps, for example over 14 000 tonnes U in 1990. This points to the basic problem of the uranium industry, the imbalance between supply and demand. Through about 1984-1985 this imbalance consisted of an oversupply, which led to a production gap. During the times of oversupply, which practically started in the 1960s, a large volume of uranium was stockpiled for various reasons. The civil portion of the inventories in the hands of all market participants is estimated to be about 150 000 tonnes U. Despite the production gap of the last 5 years, the stockpile does not seem to shrink due to changing perception of uranium supply assurance, but rather increase as governments and other organizations unload their stocks. In addition, there are believed to exist large though exactly unknown uranium stocks in China and the USSR, from which material in different forms (natural uranium, uranium hexafluoride, enriched uranium) is being exported.

For the supply-demand projections, which are still restricted to WOCA due to the lack of reliable data, three projecting periods have been considered: through 2005, 2020 and 2035. As the approaches and the data bases used by different authors are different, different features were emphasized.

Through 2005, the supply-demand projection shows a high degree of certainty as the uranium requirements are based on reactors currently in operation or under construction. Based on this, the requirements are expected to increase from 43 800 tonnes U in 1990 to about 52 600 tonnes U in 2005. The corresponding supply, based on the expected WOCA mining production and expected imports from non-WOCA countries, mainly China and the USSR, is projected to increase from 33 000 tonnes in 1990 to 36 000 tonnes in 2000 and decline then to 33 000 tonnes U in 2005. The resulting production gap over this period is about 230 000 tonnes U. This gap can be filled by the WOCA inventories, whose available portion is estimated to amount to 110 000 tonnes U plus non-WOCA supplies,

mainly from the USSR inventory, which are assumed to easily fill the remaining deficit.

Through 2020, a low and a high demand scenario were considered for the reactor related uranium requirements. They are projected to increase from 41 500 tonnes U in 1990 to 62 100 tonnes for the low case and to 77 000 tonnes U for the high case in the year 2020. Based on the supply from "firm projects", there is also a production gap projected for the time after available inventories are exhausted, which through 2020 accumulates to 585 000 tonnes and nearly 800 000 tonnes U for the low and high demand scenarios respectively. In the case of "all projects", i.e. firm plus potential projects, the low case demand can be filled under the assumption of high production increases. The high demand case cannot be filled through the entire period, as known resources sustaining the production would be exhausted and presently undiscovered resources would have to be discovered and developed by about 2015. In both cases, the strain on the resources and production capabilities is projected to be such that full production costs are assumed to increase significantly.

Through 2035, also two demand scenarios have been chosen: a low one, increasing from 41 500 tonnes U in 1990 to 65 000 tonnes U in the year 2035, and a high one, requiring 97 000 tonnes U in the same year. The supply modelled both by producer country and resource category shows the following picture: after the exhaustion of the available inventories, nearly all producer countries are assumed to increase their production and two producer countries, Namibia and Niger, to deplete their resources and stop production. The consequences are an even stronger concentration among a few producer countries than is currently the case. However, the demand is being filled in both cases. The resources which sustain this supply are currently known resources through 2010, which, by then, are expected to need a growing contribution from presently undiscovered resources.

In conclusion, the present problems of the international uranium industry include the current low prices with related cutbacks in production, as many producers are producing at costs higher than current market prices, and the unknown level of worldwide available uranium inventories. This imbalance is creating a perception of assurance of supply, which cannot be confirmed by the projections of the future supply and demand situation. While there is currently a strong buyers market with related favorable supply possibilities for the consumers, this may change in the mid-term to an equally strong sellers market with favourable conditions for the suppliers.

URANIUM IN GRANITOIDS: RECOGNITION CRITERIA OF URANIUM PROVINCES IN BRAZIL

C.C.G. TASSINARI

Instituto de Geociências,
Universidade de São Paulo,
São Paulo

P.M.C. BARRETTO*

Comissão Nacional de Energia Nuclear,
Rio de Janeiro

Brazil

Abstract

Uranium distribution in granitoids, including gneisses and migmatites were investigated for the rocks of entire Brazilian territory. Uranium concentrations were determined in 1800 Rb-Sr whole rock isotopic analysed samples with known geological setting. The samples were plotted in the Brazil's geotectonic outline, and those with high U contents (above 12 ppm) provide the characterization of probable uraniumiferous provinces. In addition the results were interpreted in relation to age, tectonic environment, lithologies, Rb contents and $^{87}\text{Sr}/^{86}\text{Sr}$ initial ratios. With respect to the lithology, the granites with alkaline compositions showed higher U contents than their host rocks which consist of gneissic-migmatitic terranes, and others granitoids. In general the Uranium-enriched rocks are mainly related to the Mid-Proterozoic time. In terms of isotopic geochemistry, the higher $^{87}\text{Sr}/^{86}\text{Sr}$ ratios rocks (high Rb contents) showed a uranium enrichment trend when compared with those less differentiated material. On the other hand rocks with low Sr initial ratio may present uranium concentrations similar to those with high initial ratios, which suggest the primary uranium enrichment process during the mantle-derived rock-forming process.

1 INTRODUCTION

This research deals with the recognition criteria for identification of Uranium Provinces in Brazil, the related study on petrological, geochemical and geochronological characteristics of the U - enriched granitoids and their correlation with tectonic environments.

For the development of this study a large number (1,800) of powder samples of granitic nature have been selected, including granitoids, migmatites, gneisses and acid volcanic rocks, all of them isotopically analysed for age dating in the Geochronological Research Center (CPGeo) of the University of São Paulo, Brazil. This selection was not only representative of the main granitic lithologies but also has a reasonable geographic distribution within the entire Brazilian territory which is about 8,5 million Km^2 .

Afterwards it has been done a search through the scientific publications and CPGeo files for analytical and geological informations like rock types, Rb and Sr contents, geochemical characteristics, tectonic setting, age, and Sr initial ratios.

The samples have been selected from known geologic units and analysed for determination of total Uranium by activation analysis (DNA) and soluble Uranium by fluorimetry.

Finally the data assemblage has been treated in special computer geostatistical program (factor analysis) for correlation of different results, such as U content versus age, Rb and Sr contents, tectonic setting and so on, in order to represent the trends, graphically.

This research has been done with financial support provided by the International Atomic Energy Agency, FAPESP (Brazilian Research Agency) and Geoscience Institute of University of São Paulo, Brazil.

The Uranium analysis has been done under an agreement between the CNEN (National Commission of Nuclear Energy) and the IPEN (Research Institute of Nuclear Energy) in São Paulo, Brazil.

2 DISTRIBUTION OF URANIUM ENRICHED GRANITOIDS

The Uranium enriched granitoids occur in several geological settings in the South American platform, and present a wide range of lithological types and ages. In general the U average contents in Brazilian granitoids is about 3 to 7 ppm, but for clear characterization of the U-anomalous granites, it has been considered 12 ppm as a limit among U-enriched and normal granites. The distribution of the U-anomalous granitoids will be discussed on geotectonic and regional geology aspects.

The South American Platform can be divided in four cratonic units, separated by Late-Proterozoic mobile belts. These units are covered by extensive Phanerozoic sedimentary rocks as showed in Figure 1. Our research is limited to the Precambrian granitoids.

The largest cratonic units in South America are the Amazonian and the São Francisco cratons. Smaller cratonic fragments also occur as the Luiz Alves, Rio de La Plata and São Luiz Cratons (Figure 2).

The Amazonian Craton is a very large unit limited by the Late Proterozoic Paraguay-Araguaia belt. This

* Present address: International Atomic Energy Agency, Vienna

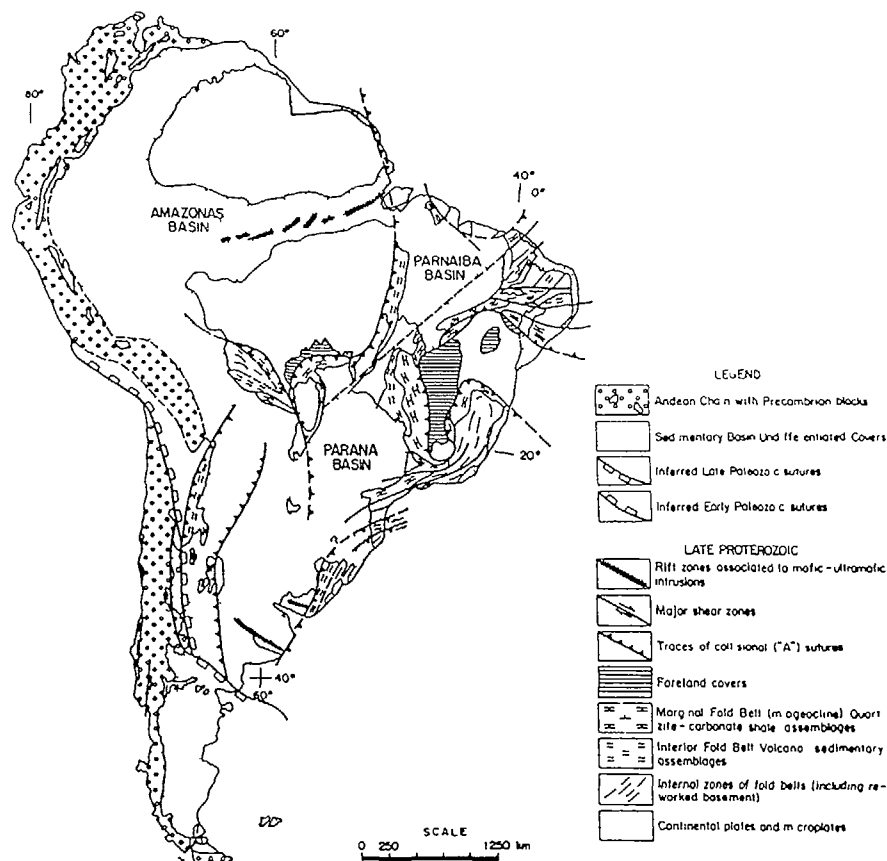


Fig 1-MAIN TECTONIC FEATURES OF SOUTH AMERICA
(Source Brito Neves e Cordani, in press)

unit can be described as an Archean core named Central Amazonian Province, surrounded by Early to Mid-Proterozoic mobile belts. These belts are the Maroni-Itacaiunas (2,25-1,9 Ga) characterized by ensialic and simatic evolution, Rio Negro-Juruena (1,75-1,55 Ga) with a magmatic arc evolution, Rondonian (1,45-1,3 Ga) with ensialic evolution and Sunsas (1,25-1,0 Ga) developed by reworking of older crustal material (Cordani et al., 1979 and Cordani and Brito Neves, 1982).

The São Francisco Craton (Almeida, 1977) is surrounded by the Late-Proterozoic Brasília, Rio Preto Sergipano and Ribeira folded belts. Most of this craton is

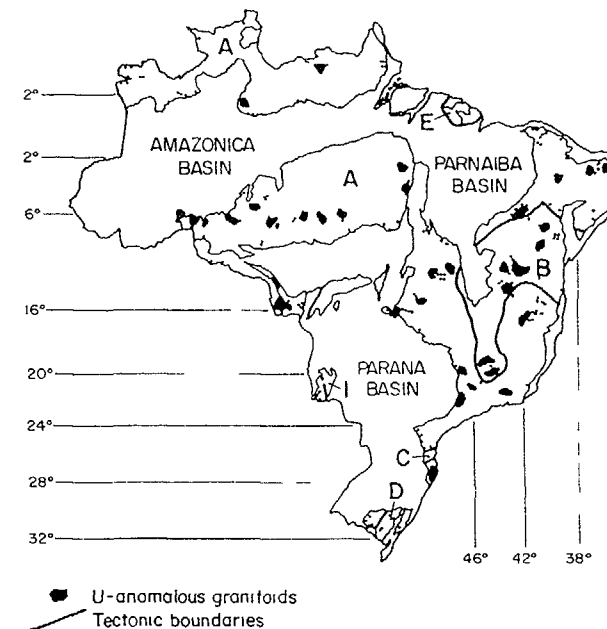


Fig 2- GEOGRAPHIC DISTRIBUTION OF U-ENRICHED GRANITOIDS IN BRAZIL: A-AMAZONIAN CRATON, B-SÃO FRANCISCO CRATON, C-LUIZ ALVES CRATON, D-RIO DE LA PLATA CRATON, E- SÃO LUIZ CRATON

covered by Late-Proterozoic chemical and clastic sedimentary rocks, which became slightly metamorphosed at the borders of the Craton. Mid-Proterozoic metasedimentary sequences belonging to the Espinhaço Supergroup also affected the Craton, which comprises clastic sediments and volcanics submitted to low metamorphic grade.

The Archean and Lower Proterozoic rocks of the São Francisco Craton are well exposed in central and eastern and southern parts of the craton and it's possible to separate three types of geologic terranes: Archean granite-greenstone terranes, Lower-Proterozoic supracrustals belts and medium to high grade metamorphic terranes which include Archean cratonic fragments of granulitic compositions (Cordani and Brito Neves, 1982).

Regarding the small cratonic fragments, we can consider three geotectonic units which are the Rio de La Plata, Luiz Alves and São Luiz Cratons.

The Rio de La Plata unit (Almeida et al., 1973) occur in Argentina, Uruguay and south Brazil, and have been

described as gneissic-migmatitic terranes with granite-greenstone characteristics. The western part of this craton is covered by sediments of the Paraná Basin. The radiometric pattern established for the lithological assemblages of the cratonic unit, which also includes granulites and basic to ultrabasic rocks, indicated ages around 2 000 Ma, related to the Transamazonian Orogeny.

The Luis Alves cratonic area (Kaul, 1980) occurs in the southern Brazil and is composed by high-grade metamorphic terranes which includes gneisses, migmatites, meta-ultrabasic rocks with intercalations of quartzites and banded iron formations metamorphosed within the amphibolite to granulite facies. U-Pb zircon ages and Rb-Sr whole rock isochrons suggests ages around 2,6 Ga for the granulitic metamorphism and K-Ar ages in between 2,000 - 1,800 Ma time interval indicated the cratonization period.

The São Luiz Craton (Hurley et al., 1967) constitute a basement fragments along the north-northeastern Brazilian coast. The sparse basement outcrops are composed mainly by granite-gneissic migmatitic terranes with shists subordinate. The whole rock Rb-Sr and K-Ar isotopic data indicated ages within 2,200-1,700 Ma time interval.

The cratonic units in South America Continent are geotectonically separated by late-Proterozoic mobile belts, which may include ancient terranes as median massifs, like the Central Massive of Goiás, and Borborema Gneissic-migmatitic massive (Cordani and Brito Neves, 1982).

In general these massifs are located in between the different folded belts, and exhibit lithological association which comprises mainly granitic-gneissic-migmatic terranes. In some areas occur strong late-Proterozoic granitizations events overprint. In others regions occur granite-greenstone terranes, and basic to ultrabasic rocks with high grade metamorphism. Archean and lower Proterozoic ages are founded within these median massifs.

During the late Proterozoic time the South American platform was conditioned by the tectonic evolution of the following mobile belts. In the central-western Brazil by the Paraguay-Araguaia and Brasília belts, in the northeastern part by the Seridó, Sergipe, Jaguaribe and Riacho do Pontal belts, and in the south-southeastern region by the Dom Feliciano and Ribeira folded belts.

In general these fold belts exhibit linear structures and are composed by gneisses and migmatites as infraestructure and metasedimentary and acid to basic metavolcanic rocks as supra crustal sequences.

Figure 2 is a map showing the U distribution in granulitoids in Brazil with zones of U-enriched granites and boundaries of the main geotectonic features. Can be observed that the granulitoids with anomalous Uranium contents occur in a variety of tectonic environments.

Within the entire Brazilian territory (about 8 000 000 km²) were established 28 zones with Uranium anomalous contents, which can be grouped within the following geotectonic units:

2.1 The Amazonian Craton

In the Amazonian Craton the U-enriched granulitoids are composed by gneissic-migmatitic rocks, anorogenic intrusive granulitoids and alkaline ring complex.

The gneissic-migmatitic rocks constitute part of the Central Amazonian and Rondonian Provinces. The first unit is composed by gneisses, migmatites and granulites with tonalitic-thronjemitic and granitic compositions, metamorphosed within the amphibolite and granulite facies. These rocks in general, have an age about 2,000 Ma, and in some places were affected by strong granitization during the Mid-Proterozoic time. The gneisses from this unit present 13 to 21 ppm of U.

The gneissic-migmatitic terranes of the Rondonian Province occur in the southwestern part of the Amazonian craton and comprises granites, gneisses, acid-granulites, amphibolites and blastomylonite-gneisses. In general these rocks presents ages about 1,450 Ma with Sr initial ratios around 0,705 (Teixeira and Tassinari, 1984). The average U contents in gneiss and granulitoids range within 13 to 30 ppm interval.

The cratogenic granitic magmatism with U anomalous contents is related to the Rondonian, Teles Pires and Jamon granulitoids.

The Tin-bearing Rondonian Granulitoids are composed by cratogenic bodies, with granite to granodioritic compositions, in general comprising pink coarsed biotite-granulitoid, grey porphyritic biotite granulitoid and microgranitic ring-dykes. The ages of the Rondonian granulitoids range 1,300 to 900 Ma time interval and in general have very high Sr initial ratios around 0,720 (Leal et al., 1978). The Rondonian granulitoids present average Uranium contents from 16 to 22 ppm.

The Teles Pires Granulitoids (Silva et al., 1978) are related to the acid volcanic rocks associated with the Rio Negro-Juruena Mobile Belt evolution and it's comprises phorpyro-granulitoids, microgranite, granites, rapakivi granites, and granophyre. The main mineralogical assemblages includes quartz, orthoclases with perthites, plagioclases subordinates, biotites, titanites and fluorspars.

These granulitoids yielded an age of 1550 Ma with Sr initial ratio of 0,707. The average uranium contents of the Teles Pires Granulitoids are about 14 ppm.

The Jamon Massif is an anorogenic A-type granite intrusive in the granite-greenstone terranes of the Serra dos Carajas region. Rb-Sr isochronic age is 1600 Ma with Sr initial ratio of 0,712 (Dall'Agnol et al., 1984). The rock is a equigranular biotite granite, with microgranite and alaskitic granite associated. The U content in this granulitoid range from 12 to 14 ppm.

The alkaline ring complex of Canamã (Silva et al., 1978) is located in the southern of Rio Negro-Juvena Province, and it's composed mainly by syenites, quartz-syenites and microsyenites. It's constituted mainly by alkali-feldspars, amphiboles, pyroxenes, apatites, fluorspars, zircons and

sphene. The age of the alkaline magmatism is 1,200 Ma and have a Sr initial ratio of 0,704. In general these rocks have been characterized as alkaline and peralkaline rocks. The U contents in the Canaã alkaline ring complex is very high about 27 ppm.

2.2 The São Francisco Craton

The Uranium anomalous rocks within the São Francisco Craton comprises gneissic-migmatitic complex, granite-greenstone terranes, intrusives granitoids and alkaline rocks.

The gneissic-migmatitic complex is represented by the Caraíba-Paramirim Complex (Lima et al., 1981) which occur in the northwestern part of the São Francisco Craton and consist the basement of the metasediments of the Espinhaço Supergroup. It's comprises with gneisses with granodioritic to tonalitic compositions, migmatites, granitoids and cataclastic rocks.

The mineralogical assemblages of the gneisses is plagioclases, quartz, perthitic microcline, biotite, titanite, epidote, hornblende and diopside. The granitoids are composed by perthitic microcline, quartz, plagioclases, biotite, muscovite and sometimes hornblende and titanite also occur. These rocks were affected by amphibolite facies metamorphism.

In general the metamorphic terranes of the Caraíba-Paramirim complex are related to the archaean times, but the U-enriched granitoids associated gave an Rb-Sr isochronic age of 1700 Ma with Sr initial ratio of 0,710 (Fernandes et al., 1982). This parameter suggest that the U-enriched granitoids were derivated by partial melting process of older continental crust.

The U-contents in these rocks range from 16 to 27 ppm.

The Cansação-Tanquinho granitoids (Gava et al., 1983) constitute a granite-greenstone terranes of the Serrinha greenstone belt, which is part of the Itapicuru Complex. The contacts between the granitoids and the volcanosedimentary sequences are tectonic.

These granitoids includes biotite-granites, two-micas granites, hornblende-biotite granitoid, adamellites and granodiorites. Sometimes also occur porphyroides facies with gneissic aspects. In general these rocks presents medium coarse and are composed mainly by quartz, alkali-feldspars, plagioclases and subordinated hornblende and biotite.

The Rb-Sr whole-rock isochronic age for these granitoids is 2100 Ma with Sr initial ratio of 0,704. The K-Ar results on biotites from the granitoid yield ages around 2000 Ma, which reflect the cooling period of the batholith. The low initial ratio value is evidence that the granites were not generate through crustal anatexis.

The average U-contents in these rocks is about 15 ppm.

The U-enriched igneous granitoids comprises the Lagoa Real, Carnaíba, Pará de Minas and Porto Mendes bodies and

the Lagoa Real Massif is the most important of them, because is related to the Uranium mineralizations.

The Lagoa Real Granitoid is enclosed in the Archean-Lower Proterozoic metamorphic terranes of the North of the São Francisco Craton and is limited to the west by the Mid-Proterozoic Espinhaço fold belt (Lobato et al., 1983).

This granitoid mainly consist of orthogneisses resulting from the deformation and metamorphism of granites. Metasomatic alteration leading to the formation of albitites also occur.

The Lagoa Real granitoid Complex include five lithological units, such as undeformed granites, deformed granites and orthogneisses, quartz-albitites, pyroxene + garnet albitites and amphibolites. The pyroxene + garnet albitites are host of uranium mineralizations.

The granites, deformed granites and orthogneisses present a composition which corresponds to the most evolved terms of the Fe-rich subalkaline association and have geochemical and mineralogical characteristics intermediate between calcalkaline and alkaline associations but closer to the last one.

The Lagoa Real Granitoids have been emplaced at 1,725±5 Ma, which is showed by the zircons U-Pb ages, and present a very high Sr initial ratio about 0,718, which strong suggest a crustal source for them (Maruejol et al., 1987).

The samples with U anomalous contents are mainly with syenitic compositions and pegmatoids and the results range from 16 to 37 ppm.

The Pará de Minas Granitoid constitute part of the archaean gneissic-migmatitic terranes which occur in the southern part of the São Francisco craton. It's composed by rocks with granodioritic to granitic compositions and are related to the rocks with ages around 2,7 Ga (Teixeira, 1985). There are no ages for this granitoids.

The U contents measured in the samples from the Pará de Minas granitoids are within the 15-24 ppm interval.

The Porto Mendes granitoid is intrusive in the archaean gneissic-migmatitic terranes of the southern part of the São Francisco Craton and it is related to the Minas diastrophism which is responsible by the generation of strong granitic plutonism in this region.

This granitoid have a Rb-Sr isochronic age of 2200 Ma with Sr initial ratio of 0,7012 (Teixeira, 1975). The average U contents of the Porto Mendes granitoid is 20 ppm. These data shown U-enriched rocks with very low initial ratio, which could suggest an example of primary Uranium enrichment from the upper mantle.

The Carnaíba granitoid (Gava et al., 1983) with batholithic sizes is intrusive within the metasediments of the Serra da Jacobina metamorphic belt and presents tectonic contacts. It's composed by alkaline-granites, micaceous granodiorites, adamellites and subordinated quartz-monzonites, pegmatites and two micas granites. The main mineralogical assemblages is constituted by quartz, microcline, oligoclase, muscovite and biotite, and the granite presents emerald mineralizations. No chemical analyses are available.

Samples from the Carnaíba granitoids yielded a Rb-Sr isochronic ages of 1960 ± 16 Ma with Sr initial ratio of 0,708. This value suggests a upper crust material contributions in the rock-forming process of this granitoid.

This granite presents U-contents of 13 ppm.

The Alkaline Guanambi Complex (Barbosa and Moutinho da Costa, 1973) is composed by granitoids which includes mainly syenites and also granites, granodiorites, tonalites and monzonites subordinated. A large K-metassomatism can be observed in this complex, which have a intrusive characteristics.

The mineralogical assemblages of syenites is composed by porphyroblasts of perthitic microcline, with plagioclases and few quartz, biotite, hornblende and augite.

The Guanambi complex were affected by amphibolite facies metamorphism which was responsible by the mineral paragenesis composed of perthitic microcline + clinopyroxenes + hornblende + biotite and plagioclase + quartz + K-feldspars + biotite.

Samples from the Guanambi Complex, mainly syenites and granites yielded a Rb-Sr isochronic age of 2100 Ma with Sr initial ratio of 0,706, and U contents of 17 ppm.

2.3 Central Goiás Massive

Within the Central Goiás Massive were found U-enriched contents in samples from the gneissic-migmatitic terranes and tin-bearing intrusive granitoid.

The Gneissic-Migmatitic Terranes of Central Goiás Massive, includes granite-greenstone terranes with tonalitic-trondhjemitic plutonic suites of Archean ages (Danni et al., 1982) and gneisses and migmatites with ages related to the early and late Proterozoic times.

The U-enriched granitoid in this domain is related to the early Proterozoic belt, which is composed by gneisses with granodioritic compositions. The Rb-Sr whole rock isochronic age of these rocks is 2,000 Ma with Sr initial ratio of 0,708. The U average contents in this unit is 12 ppm.

The Tin-bearing Serra da Mesa Granitoid presents subvolcanic characteristics with fluorspars, blue-quartz and cassiterite. In geochemical point view is characterized by high Rb, K and F and low Sr (Drago et al., 1981). The emplacement of this body with granitic composition was about 1,650 Ma (Reis Neto, 1983) and the average of Uranium contents is 22 ppm.

2.4 Borborema Province

The granitic rocks of Borborema Province with high U-content are related to the Nordeste and Presidente Juscelino Complexes.

The Nordeste complex is characterized by migmatites, gneisses, granitoids, amphibolites, quartzites,

schists, limestones and calc-silicated rocks. The gneisses and granitoids with 700 Ma has been analysed for Uranium and the results have an average about 25 ppm.

The Presidente Juscelino Complex according Gava et al., (1983) is composed by gneisses, migmatites, granitoids and ultrabasic rocks. The granites have an age varying from 570 Ma to 515 Ma and Sr initial ratio within 0,720-0,707 interval. The Uranium contents for these granites range from 16 to 38 ppm the higher value is related to the granite with higher Sr initial ratio about 0,720.

2.5 Socorro-Guaxupé Domain

The Socorro-Guaxupé Domain is located in the southern marginal zone of the São Francisco Craton and comprises granulite-granite-gneissic-migmatitic terranes overlying metavolcanosedimentary sequences which characterize a regional nappe structures (Campos Neto, 1985).

The Varginha Complex is located at Guaxupé Massif and includes high to medium grade metamorphic terranes composed by acid to basic granulites, migmatites, charnockitic gneisses and granitoids. The U-enriched gneiss comprises garnet, cyanite, cordierite and sillimanite. There are no ages very well defined for these rocks, but the best age estimated is around 1,300 Ma.

The U-content measured for these gneisses is 14,2 ppm.

The Morungaba granitoid complex is intrusive in the Socorro-Guaxupé Nappe and it's composed mainly by biotite-granitoids and diorites. This massive can be separated in the pink granitoid suite, made up by equigranular quartz monzonites and 3b-3a granitoids, and the grey suite which is composed by 3b granites and granodiorites, the Porphyritic granitoid suite, which comprises porphyritic quartz monzonites and 3a and 3b granites (Vlach, 1985).

The ages of these different suites range from 500 to 620 Ma and the Sr initial ratio are in general around 0,706 (Vlach op cit). The U contents of the Morungaba granitoid is about 13 ppm.

2.6 The Late-Proterozoic Folded Belts

Several granitic bodies related to the tectonic evolution of the late-Proterozoic Mobile Belts in Brazil, presents high contents of U. They are associated to the Ribeira, Paraguai-Araguaia and Dom Feliciano Belts.

2.6.1 Ribeira Fold Belt

The Itinga, Itaobim and Novo Cruzeiro Granitoids bodies are related to the terminal stages of the Ribeira Fold Belt evolution during the late-Proterozoic times. These granitoids are represented by biotite-granite and two-micas granodiorites with associated pegmatites. The main

mineralogical assemblages identified for these granitoids is composed by microcline, plagioclase, quartz, tourmalines, apatites, zircons and opaques. The geochemical characterization of the granitoids indicated a calc-alkaline to alkaline character.

The geochronological studies on samples from the Itaobim, Novo Cruzeiro and Itinga bodies suggested respectively the following ages and Sr initial ratios: 626 ± 26 Ma and 0,708, 505 ± 35 Ma and 0,715, 540 ± 10 Ma and 0,712 (Siga Junior, 1986). The high values of the Sr initial ratios clearly indicate a crustal source for these granitoids.

The U contents measured for the granitoids range from 13 to 18 ppm.

2.6.2 Dom Feliciano Belt

The Pedras Grandes Granitic suite occurs in southern Brazil and is characterized by late to post tectonic magmatism of "I" Caledonian type. This body has an age of 540 Ma and Sr initial ratio of 0,710 (Basei, 1985). The U contents of the massive is 13 ppm.

2.6.3 Paraguai-Araguaia Belt

The evolution of the Serra Negra granitoid is related to the final stages of the development of the Paraguai-Araguaia belt during the late-Proterozoic time in the Central portion of Brazil. This body is composed by granodiorites with microcline, plagioclase, quartz, biotite, chlorite, muscovite, clay-minerals and epidote. The age of this intrusion is 520 Ma and Sr initial ratio of 0,705 (Pimentel et al., 1985). The Uranium contents in this granitoid and in the host-rocks near the contact are 15 ppm.

3. RELATIONSHIP AMONG U-ENRICHED GRANITOIDS AND THEIR GEOLOGICAL CHARACTERISTICS

The results of U concentrations analyses of the U-enriched granitoids were interpreted in relation to age, Sr initial ratios, Rb contents, tectonic environment, lithologies and geochemical characteristics.

The correlation of the U-enriched granitoids and geological-time-bound has been investigated through diagrams for U-content related to the rock-formation ages which are shown in Figures 3 and 4. These ages are supported mainly by whole-rock Rb-Sr isochrons and subordinated zircons U-Pb concordies and whole rock Pb-Pb isochrons.

The histogram of Figure 3 shows that the majority of the U-enriched granitoids are related to the 2,200-1,800 Ma time interval, but this tendency is produced because most of the analysed samples belongs to this epoch. On the other hand, in Figure 4, it is possible to see the frequency of the U distribution within each time-bound.

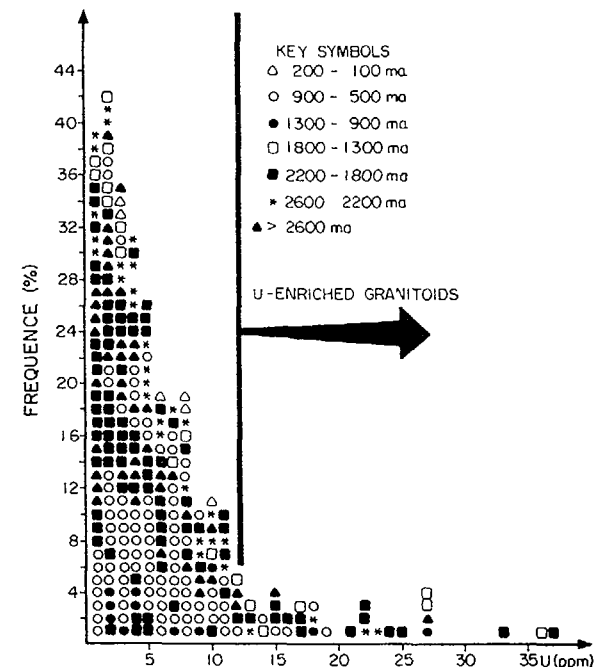


Fig 3-HISTOGRAM OF URANIUM CONTENTS WITH ASSOCIATED ROCK-FORMATION AGES

considered. It can be observed that the frequency of U-anomalous granitoids is more important in the 1,800-1,300 Ma time interval.

This time-interval delineated here coincides with the ages of the world's largest Uranium deposits, like the Cluff Lake, Rabbit Lake and Key Lake deposits of Saskatchewan in Canada, the Ranger, Jacobiluka and Nabarlek deposits of Australia and the large Lagoa Real deposit in São Francisco Craton, Brazil, which yielded ages around 1,700-1,500 Ma, similarly.

Figures 5 and 6 show the U-contents distribution related to different granitic lithological groups. The histogram of Figure 5 shows that the U-enriched granitoids are mainly composed of granites 'sensu strictu' and alkaline granites. Figure 6 shows the proportions of U-enriched granitoids related with each lithological group. It is also clearly indicated that the granitoids with granitic compositions and the alkaline granites presents higher U concentrations. In general the U-enriched alkaline granitoids

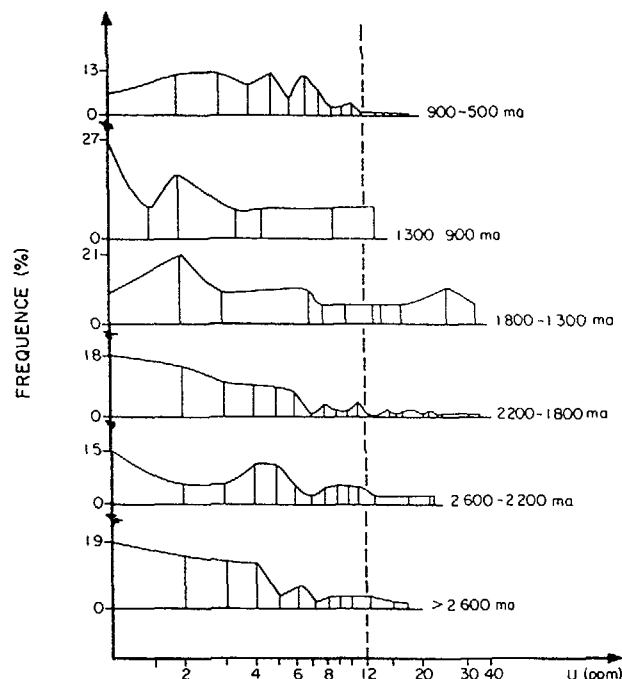


Fig 4-THE TIME - BOUND URANIUM DISTRIBUTION

are mainly composed syenites and quartz-syenites and the granites are constituted by biotite-hornblende granites

In the diagram UxRb of Figure 7 is observed that the U concentrations increase with Rb concentrations in general, which is normal considering that the uranium and Rb have the similar geochemical cycle and are concentrated by geological processes in the crust

According to Wilson and Akerblom (1980) there are two opposing hypotheses about the genesis of U-enriched granites. The first one is that U-enriched granites are generated by anatexis process in upper crust (Beckinsale et al, 1979), and the other is that these granitoids have a much deeper origin (Simpson et al, 1979). These two types of granitoids can be related to the S and I-type granites of Chappell and White (1974). As we know, the S-type granites are derived from sedimentary sources and are considered as having been generated by partial melting processes of the thickened crust and are characterized by limited compositional range, depletion in Na and Ca and high Sr initial ratios. The I-type granites are derived from igneous sources and in general

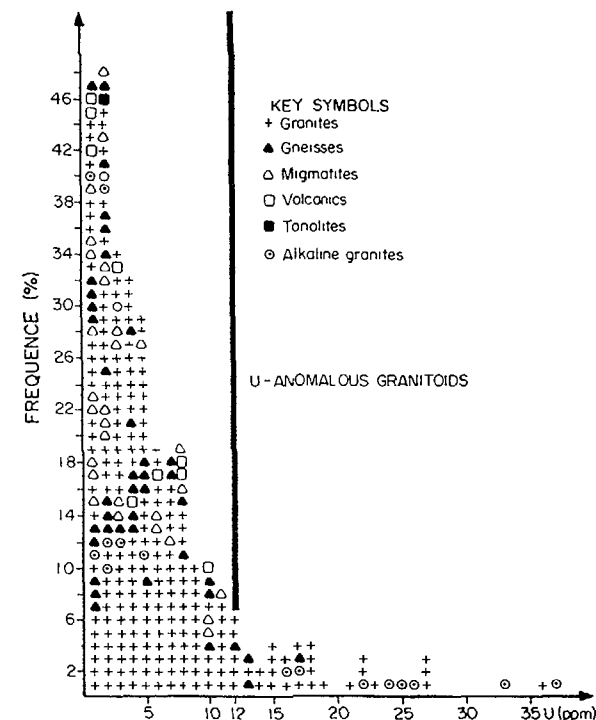


Fig 5-HISTOGRAM OF URANIUM CONCENTRATIONS WITH ASSOCIATED LITHOLOGIES

correspond to the calc-alkaline granitoids, and have a wide compositional range and low $^{87}\text{Sr}/^{86}\text{Sr}$ initial ratios

The U-enriched granitoids from Brazil show a wide range of Sr initial ratios values from 0,700 to 0,727 as is observed in Figure 8. Thus, the uranium concentrations on these granitoids are not clearly correlated with Sr initial ratios, and so clearly indicate that the primary Uranium enrichment process are not related with the magma sources. In other words, the U-enrichment process is possible to be correlated with both the I and S-types granitoids.

On the other hand, the U-enriched granitoids when plotted Sr initial ratios against their rock-formation ages, indicate that the generation of granitoids with initial $^{87}\text{Sr}/^{86}\text{Sr}$ ratios lower than 0,704 is mainly related with period older than 2,0 Ga. The period younger, than 2,0 Ga, are mainly represented by the granitoids with the higher Sr initial ratio.

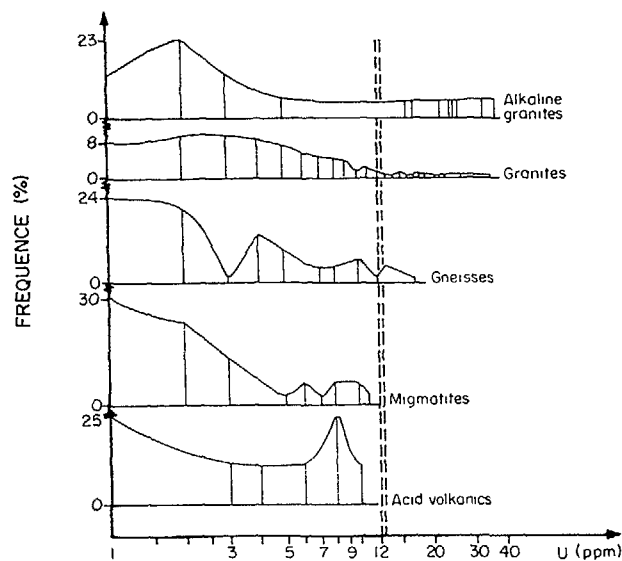


Fig. 6 - THE LITHOLOGY AND URANIUM DISTRIBUTION

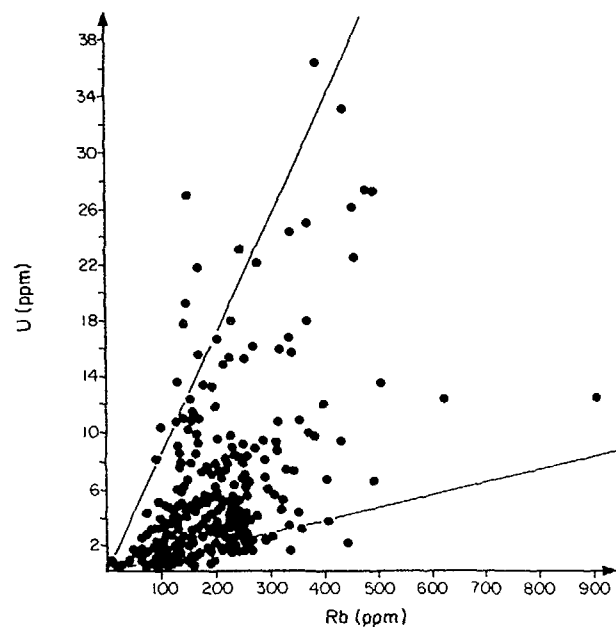


Fig. 7 - U (ppm) VERSUS Rb (ppm) DIAGRAM

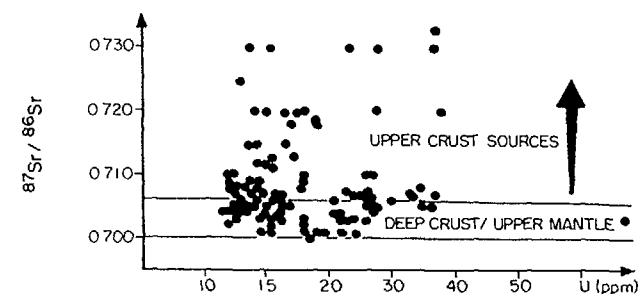


Fig 8-SR INITIAL RATIOS VERSUS U-CONTENTS DIAGRAM FOR THE U-ENRICHED GRANITOIDS

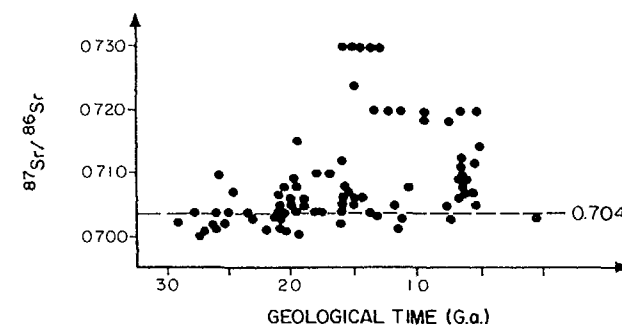


Fig.9-Sr INITIAL RATIOS VERSUS GEOLOGICAL TIME DIAGRAM FOR THE U-ENRICHED GRANITOIDS

From the above considerations it is suggested that most of the U-enrichment process in granitoids generated during the Early Proterozoic and Late Archean. May be mainly related with mantle-derived material or partial melting of deep crust. In turn most of the U-enriched granitoids of the Middle and Late Proterozoic may be derived by anatexis process, when the continental crust has attained a large thickness.

4. CONCLUSIONS

This work characterizes and describes the U-enriched granitoids in Brazil. They occur in a variety of tectonic environments, are represented by a variety of granite types and distinct ages. In general they have been generated by partial melting process of continental crust, but some of them, with tonalitic compositions, were generated through important contribution of mantle-derived material, which may suggest primary Uranium enrichment from the upper mantle.

In addition the research provides the most common characteristics of the Brazilian uranium-enriched granitoids, which are very useful for identification of Uranium Provinces and perhaps Uranium-mineralizations, although the Uranium-mineralizations are not directly related to the primary uranium enrichment process.

In general the U-anomalous granitoids are composed of granites with alkaline composition and granite "sensu strictu" which comprises mainly syenites, quartz-syenites and biotite-hornblende-granites, with ages between 1,800-1,300 Ma. The U-anomalous granitoids belonging to this period present high Sr initial ratios values, above 0,706, and high Rb contents.

Most of the U-enriched granitoids occur within ancient cratonic areas, or within Early to Mid-Proterozoic mobile belts, but after their cratonization. Generally these granitoids are related to the border zones of the mobile belts or deep crustal discontinuity.

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BASELINE URANIUM EXPLORATION SURVEYS AND THEIR USE IN THE EARLY DETECTION OF POTENTIAL NATURAL HAZARDS: A CASE HISTORY FROM SUDAN

G.R. PARSLow
Department of Geology,
Regina University,
Regina, Saskatchewan,
Canada

B. KHALIL, A.-R.K. HASSAN
Department of Geology,
Khartoum University,
Khartoum, Sudan

Abstract

In 1969, a small dam was erected on Khor Berdab about 20 Km NW of Kadugli, Southern Kordofan Province, Sudan. At the time the plan was to build a pipeline from the reservoir to the town of Kadugli, to alleviate the water shortage there. Also, the reservoir was expected to replenish shallow aquifers in the region and UNICEF planned to drill some 400 wells (20-80 m deep) capped with the "UNICEF hand pump" in Southern Kordofan; of these some 200 were sited close to Lake Miri (the reservoir) and Kadugli. In 1982, one of the authors (B.K) reported on field work (using hand-held scintillometers and collection of soil samples) that established that Lake Miri straddled a zone of pitchblende(?) mineralization (vein-type) and constituted a potential health hazard. In 1985, a further visit by one of the authors (B.K.) and W. Uchdorf revealed radon values of about 3000 cpm in springs behind the dam and in UNICEF well water at the village of Hayat el Nar, near Kadugli. It was also noted that the well water contained 380 ppb U. In 1986, as part of IAEA TC project SUD/3/003-1, the authors revisited the area and ran two profiles with a GAD-6, four channel, gamma ray spectrometer (one south and one north of the lake) to better define the extent and magnitude of the radioactive zone. Typical total count channel values averaged 400 cps, with peak values of 1200+ cps; typical stripped U and Th channel counts were similar and averaged 8 cps, with peak values of 60+ cps. In 1987, one of the authors (A.-R.K.H.) revisited the area to extend the scintillometric survey and collect more samples (which were analyzed for major and trace elements in the latter half of 1987 and the first half of 1988). In 1988, a final visit was undertaken by two of the authors (G.R.P and B.K.) to assess radon values in some village wells and in soils around wells and near the lake. Virtually all ten locations visited returned anomalous values; the highest Rn value for water exceeded 8000 cpm. This case history illustrates that the

dissemination and integration of "early" uranium exploration survey data into the projects of other agencies (e.g. WHO, UNICEF, NGO, and Gov't) is imperative and would help to avoid scenarios such as the present unfortunate situation around Lake Miri. The villagers now have an adequate supply of water in this arid region but much of it is contaminated.

1. INTRODUCTION

This case history refers to the Lake Miri and Kadugli areas of Southern Kordofan Province, Sudan (Fig.1). Much of Southern Kordofan is known also as the Nuba Mountains, a region bounded by coordinates $028^{\circ}00' - 032^{\circ}30'E$ and $009^{\circ}35' - 013^{\circ}30'N$. Kadugli, the administrative capital of the region, is linked by a seasonal track to Kosti, which in turn is linked by a paved road to Khartoum, the country's capital. The region has a subtropical climate with a rainy season lasting from April to October (average annual rainfall is 700 mm); it is typical savanna. The area is one of domal uplift with radial drainage and scattered low massifs (called jebels) rising some 800 m above the surrounding plains. All valleys, at present, host intermittent streams and are referred to locally as khors (synonymous with wadi). The region is inhabited predominantly by nomadic Nuba tribes who breed cattle and sheep, cultivate fruit in wadis, and grow cereals on the plains (Hassan [1]).

2. GEOLOGY

2.1 Regional Geology

The Nuba Mountains are not well documented, in a geological sense, and most of the available data come from two regional surveys: the Nuba Mountains Project (a joint project of the Department of Geology, University of Khartoum and the Faculty of Engineering, Peoples Friendship University, Moscow) covering the period 1977-1982 and the Sudanese-German Project covering the period 1981-1985.

In summary, the Nuba Mountains represent an uplifted block of Precambrian "Basement Complex" (consisting of gneisses, migmatites, and metasedimentary/metavolcanic sequences, intruded by syn- to late orogenic "granites") which has been intruded by numerous, younger, undeformed (i.e. post-orogenic) igneous bodies along regional fracture zones. These later intrusives are alkaline to peralkaline syenites and granites, with minor basic phases, having the form of ring dykes, bosses, stocks, and batholiths. In the northern part of the Nuba Mountains nepheline syenites and carbonatites have been documented. Three episodes of deformation have been recognized in the Nuba Mountains and the later brittle fracturing events are essentially N-S in orientation, typical of much of the African continent.

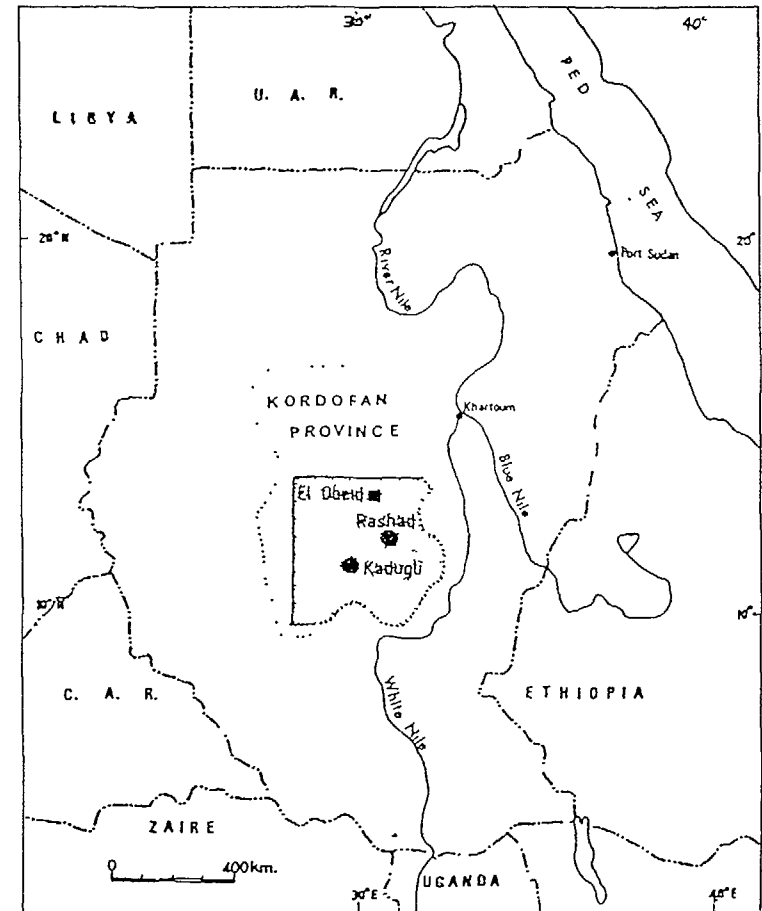


FIGURE 1 : Location map of Nuba Mountains region, Sudan

2.2 Local Geology

The area around Kadugli comprises "Basement Complex" intruded by syenitic intrusions (Fig.2). The basement is predominantly a quartzo-feldspathic gneiss and minor schist package with a regional strike of 340° and essentially vertical dips. The (later) intrusives are alkali syenites, subordinate quartz syenites and granites, and rare gabbroic and marginal "mixed" rock phases.

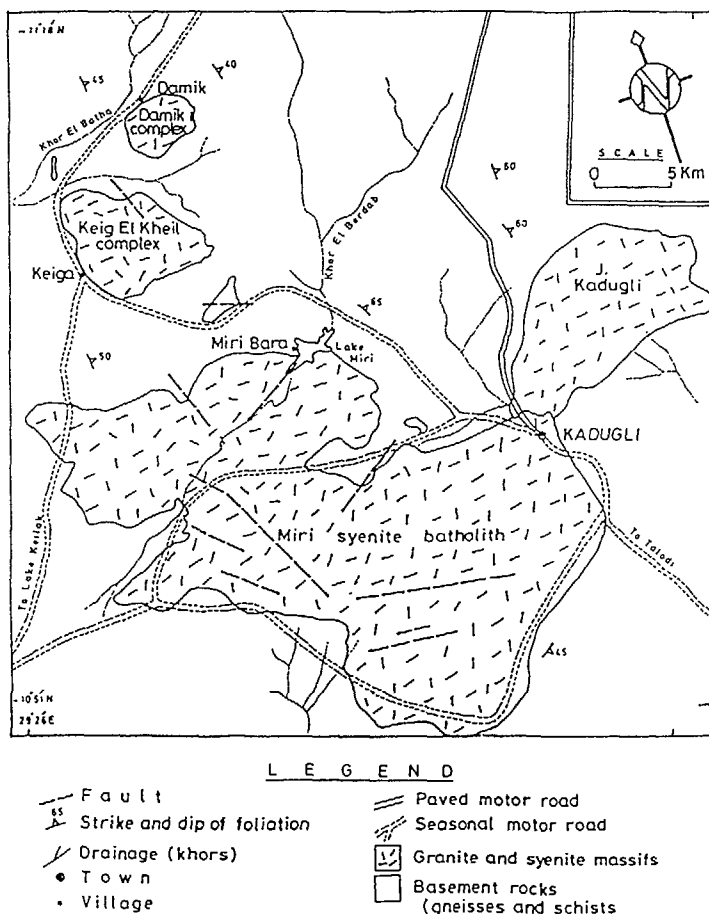


Figure 2 : Geological map of Lake Miri region, Numba Mountains

3. THE CASE HISTORY CHRONOLOGY

3.1 1969

Where Khor el Berdab cuts through the Miri syenitic complex a narrowing and steepening of the river valley allowed the construction of a small dam to create an artificial lake (Lake Miri, named after the small village of Miri Barra just north of the resultant lake). Completion of the dam produced a reservoir some 2 km long, about 0.5 km wide, 7 m deep at the dam, and averaging 3 m deep. This reservoir served two

purposes: 1) the continual recharge of shallow aquifers (mainly fracture zones) between Lake Miri and Kadugli and 2) the potential to supply water for a planned surface pipeline to be built between Lake Miri and Kadugli.

With UNICEF pursuing a vigorous, shallow well, drilling program (with depths ranging from 20 to 80 m) in the Southern Kordofan area, the eventual number of hand pump wells rose to about 400 in the subsequent 20 years; of these, about 200 wells are in the vicinity of Kadugli and Lake Miri and some 20 are along a trend linking Lake Miri to Kadugli.

3.2 1977-1982

During the course of the Nuba Mountains Project, which involved reconnaissance geological mapping and more detailed mapping in areas thought to have mineral potential, four areas of uranium enrichment were identified, one of which was the Lake Miri area. Detection of these zones was based upon readings from a small, hand held, total count scintillometer and analyses of soil samples. The definition of an anomalous region was quite simple - total counts (as cps) above 100 and/or soil uranium values above 20 ppm.

3.3 1985

One of the authors (B.K.) revisited the area with Dr. W. Uchdorf from the Department of Economic Mineral Deposits, University of (West) Berlin in order to verify the initial data on the radioactive veins and, with the use of a radon meter, to assess the "leakage" of radioelements into the reservoir and local groundwater.

The radon meter gave values of 5-7 cpm (over a normal background of virtually 0 cpm) for water samples taken from within the reservoir. Such values could not be considered anomalous; however, given the shallow nature of the reservoir, degassing of the water by wind (i.e. wave) action was probable and thus the results may be misleading. This concern was corroborated by the fact that small leakage springs behind the dam (i.e. downstream) returned values of up to 3000 cpm total alpha activity. A further 100 m downstream of the dam the activity rate dropped to 200 cpm; nevertheless, still a significant count rate. Furthermore, a UNICEF well in the village of Hagar el Nar, on the outskirts of Kadugli, gave values of between 2500 and 3000 cpm for total alpha activity. Even the public water supply in Kadugli had an activity of 50 to 60 cpm.

The radon meter used (Gewerkschaft Brunhilde), as with most radon meters, can only differentiate between ^{220}Rn (i.e. thoron) and ^{222}Rn in soil samples and not in water samples. This is because of the short half life of ^{220}Rn , which is 56 secs. compared to 3.8 days for ^{222}Rn . This means that a soil gas analysis count integrated for 1 minute and then repeated a few minutes later for a further 1 minute integration will allow an estimation of the two isotopes (by simple subtraction of the two readings). On the other hand, a degassing of water, which takes a few minutes, will reduce the ^{220}Rn , by decay, to very

low levels and thus water analyses are effectively a measure of ²²²Rn only. Given these facts, the calibrated radon meter was registering a high of some 200 Bq/l of radon 222 activity alone. In order to put this anomalous value in perspective, the reader should note that WHO recommends, for a daily consumption of 2 l/day, water should contain no more than 0.1 Bq/l of total alpha activity; in the Kadugli region the average consumption is about 7 l/day. Finally, a water analysis from the same well gave a uranium content of 380 ppb.

The results of this brief survey indicated that : 1) the anomalous gamma results from the previous survey were substantiated, 2) the radon meter data indicated a significant "leakage" of radioelements into the local groundwater, and 3) further follow up work was desirable.

3.4 1986 to 1987

As part of IAEA TC project SUD/3/003-1 the authors revisited the area to further evaluate the zone of radioactive veins in the vicinity of Lake Miri. Using a four channel spectrometer (Scintrex GAD6 with a GSP4 sensor provided by IAEA for SUD/3/003 TC project) two traverses were run from west to east, starting at the dam and with readings every 100 m to the east, on the north and south shores of the lake. The results are shown in Table I. One of the authors (A-R.K.H.) further refined these data, using a grid to delineate the radioactive zones (not illustrated); his data corroborated the basic results obtained on the profile. Since there are no calibration pads in the Sudan, the data are given in cps (stripped) values; however, based upon experience with the same equipment in Canada the cps to ppm or % conversion can be made as follows: $eTh = 0.117cps/ppm$, $eU = 0.33cps/ppm$, and $K = 2.83cps/\%$ (i.e. 10 ppm of Th, U, and 10 % K = 1.17cps, 3.33cps, and 28.3cps, respectively). The peak cps values of eTh and eU represent about 580 and 195 ppm, respectively, assuming the decay series is in equilibrium. The K values average 5%, typical of granitic rocks, and have peak values of 10%, typical of late stage "pegmatitic" veins or anatectic pods. Thus, the mineralized veins identified by earlier work are Th dominated but do contain anomalous amounts of U. In what mineral form these elements occur was not determined.

Gamma ray measurements were also made in the immediate vicinity of the UNICEF well that gave anomalous radon values for the water in the previous survey. The average results (TC=116 cps, K=8.7 cps [3%], eU=4.0 cps [12 ppm], eTh=5.8 cps [50 ppm]) suggest that the rocks through which the well was drilled are not anomalous in terms of radioelement content and thus the radon in the water must be derived from another source, possibly groundwater flow from the Lake Miri area, with its radioactive veins.

At the time of the survey, local people contributed information (rumours ?) that 1) fruit trees in the area around Lake Miri tended to die after a number of years, 2) that workers involved in the dam construction had developed skin pigmentation problems, and 3) the area had, historically, been

TABLE I
Stripped gamma counts around Lake Miri
(values in cps)

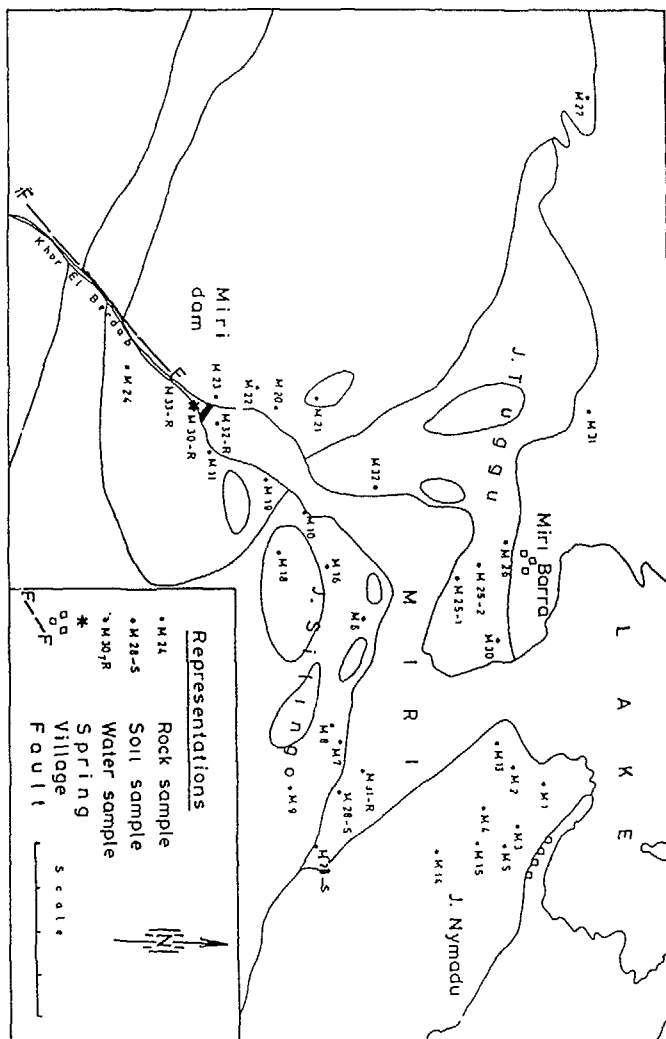
	south side of lake				north side of lake			
	TC	K	U	Th	TC	K	U	Th
dam(0mE)	109	16	3.0	3.6	88	15	2.0	2.3
100mE	115	11	4.0	4.3	242	14	6.8	12.8
200mE	123	13	4.5	3.7	206	17	5.4	9.5
300mE	389	17	12.8	21.8	188	11	6.3	9.2
400mE	179	14	4.6	7.9	151	10	4.8	7.1
500mE	255	17	15.1	9.1	189	15	5.9	8.1
600mE	227	12	7.4	12.9	214	14	6.5	10.5
700mE	426	14	11.4	26.8	119	17	4.7	3.5
800mE	228	14	6.8	18.5	114	12	4.8	3.5
900mE	346	15	5.4	22.1	223	15	9.0	9.3
1000mE	223	11	3.4	15.4	392	15	20.0	16.3
1100mE	139	14	2.7	5.4	420	10	11.7	27.3
1200mE	227	15	7.4	10.6	287	13	9.9	15.0
1300mE	157	17	5.7	6.6	170	11	8.3	6.2
1400mE	239	13	7.4	12.8	159	13	5.0	7.1
1500mE	536	16	15.9	31.7	895	16	31.3	51.4
1600mE	590	16	22.0	34.4	880	18	45.2	41.7
1700mE	316	18	10.6	16.5	117	17	3.0	3.1
1800mE	97	17	1.5	3.8	738	14	21.7	48.2
1900mE	225	12	7.8	13.4	1104	10	41.9	68.2
2000mE	191	13	6.0	9.1	183	15	7.8	6.8
2100mE	194	16	8.9	6.3	239	16	8.8	10.5
2200mE					512	18	19.0	28.8
2300mE					1130	15	65.2	52.0
2400mE					1210	16	63.8	59.9
2500mE					567	9	36.0	21.0
2600mE					560	30	19.1	10.2
2700mE					203	29	12.4	8.5
2800mE					82	10	6.9	3.2

Typical "background" values in area are 100 15 3.0 3.0

avoided by local tribes as a place of death. These "stories" may have no substance in a scientific sense but medical workers in the region did say that the incidence of cancer in the area was high, relative to the national average.

As a follow-up of this field work, one of the authors (A-R.K.H.) collected some samples of rock and soil for analysis. The location of these samples is shown in Figure 3 and the analytical results are summarized in Tables II and III (complete analytical data are available from the authors, on request). The data in Table II, representing the main alkali syenites, later pegmatitic phases, and the youngest quartz stockworks, illustrate that most rock types are enriched in uranium, of which about 16% is readily soluble and thus capable of entering the groundwater. Table III illustrates that the trace element composition of the rocks is very typical of high level alkaline rocks intruded into stable cratonic basement. It is noted that the radioactive component is Th dominant.

Figure 3 : Location map of samples - Miri area

TABLE II
Total and labile uranium values (ppm) in selected Miri samples

#	Total U ¹	Labile U ²	%LU/TU
M1	174	24.0	13.8
M3	30	4.5	15.8
M5	82	13.3	16.2
M7	750	149.0	19.9
M8	238	40.7	17.1
M10	61	13.1	21.5
M11	163	20.2	12.4
M13	186	32.1	17.3
M20	33	6.3	19.1
M25.1	141	27.0	19.1
M25.2	210	30.0	14.3
M30	132	12.9	9.8
M31	25	4.6	18.4
M32	296	71.2	24.1

1=XRF analysis, 2=fluorimetry, with 4N HNO₃ digestionTABLE III
Basic statistics of selected trace elements in late stage veins
(19 samples), Miri area
(values in ppm)

	Mean	Minimum	Maximum
U	122.3	13	300
Th	183.0	5	769
Zr	11381.5	310	58709
Nb	4017.2	275	9544
Ta	214.8	16	1200
Pb	267.8	13	982
Sn	158.1	9	479
Zn	194.1	15	798
Ni	213.5	8	2611
Ba	555.3	100	2000
Be	208.0	3	3000
Sr	541.1	9	1811
Y	1160.7	156	2226
Ce	5168.4	100	50000
La	1092.1	50	3000
Rb	259.4	36	477

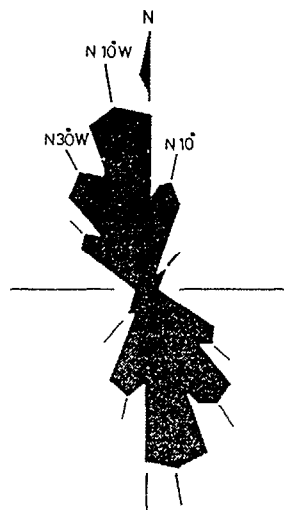


FIGURE 4 : Rose diagram of azimuths of radioactive veins, Miri area, southern Kordofan.

At the same time as collecting the samples a compilation of vein azimuths was made and is shown as Figure 4. With a width of at least 2 km and azimuths predominantly between N10°W and N30°W, the mineralized stockwork zone trends in a southerly direction directly towards the town of Kadugli. Since the quartz stockworks, when exposed, are vuggy they must represent a very porous and permeable zone that tends to control the circulation of shallow groundwater. Although the veins are not exposed in the vicinity of Kadugli, due to the presence of recent unconsolidated deposits, if they still exist at depth it is probable that many of the UNICEF wells in the area are tapping aquifers in the mineralized quartz stockworks (fed in large part by Lake Miri).

3.5 1988

A brief visit to the area, by two of the authors (G.R.P. & B.K.), was undertaken to illustrate the use of, and test, a radon detector (EDA 200 with degassing unit) provided by IAEA as part of the SUD/3/003 TC project.

Unfortunately, the degassing regulator valve proved defective, however, two manual degassing procedures were devised that gave similar degassing times to those recommended by the manufacturer. The procedures are described here because it may help readers with the same equipment to overcome breakdowns in the field. Basically, the EDA 200 degassing unit is a vacuum system linking the Ag activated zinc sulphide

scintillation cell to the water sample, held in a glass cylinder with a fritted glass disk base (to disperse the incoming air into fine bubbles) below which is a pinch clamp on latex tubing. After the system has been evacuated, release of the pinch clamp allows air to bubble through, and degas, the water sample regulated to three minutes by the regulator valve. If the regulator valve proves to be faulty, or difficult to adjust, judicious application of the thumb and forefinger to the latex tubing, after release of the pinch clamp, allows the degassing time to be limited to within 2.5 to 3.5 minutes. The other method requires the availability of soluble antacid tablets. In this procedure a small wire harness holds the tablet above the water during the evacuation of the system, a sharp tap on the glass cylinder will cause the tablet to fall into the water and start dissolving, with the release of numerous CO₂ bubbles, and degas the water. Although the radon meter was not calibrated for water degassing using an appropriate liquid standard such as ²²⁶Ra, counts per minute can be converted reasonably accurately to pCi/l or Bq/l on the EDA 200 system as follows: cpm*0.616=pCi/l and cpm*0.0231=Bq/l.

The soil gas probe extends to a depth of about 30 cm and three one minute readings were taken in sequence in order to assess the thoron contribution to the total alpha activity. If all three readings are similar then thoron is virtually absent, but if the second and third readings are significantly smaller then thoron is present, as recorded in the high initial reading.

Having developed the manual degassing procedures, a series of soil gas (Rns) and water gas (Rnw) measurements were made in the Lake Miri - Kadugli area.

Two small leakage springs behind (i.e. downstream) the dam were sampled and gave Rnw values of 1282 and 1881 cpm, respectively. These high counts corroborate the results obtained in 1985. Surface water in front of the dam gave a Rnw value of 71 cpm, much lower than the springs but still higher than average surface waters (<10 cpm).

A series of measurements were taken in and around the village of Miri Barra on the north side of the lake. The main well in the village gave a Rnw value of 4347 cpm and adjacent soil gave sequential Rns values of 2727, 1356, and 822 cpm. Clearly, the wellwater has strongly anomalous alpha activity; however, the soil gas values are of interest not only for their magnitude but for the fact that they indicate a significant thoron component within the total alpha activity recorded. Thus, assuming the anomalous soil gas values are derived from the same mineralized quartz stockworks that contribute radioactive isotopes to the local groundwater, the Rnw values observed after 3 minutes degassing must represent only about half of the actual alpha activity present the moment the water reaches the surface. In addition, it must be assumed that daughter elements to the thoron (i.e. ²²²Rn), as well as those from ²²²Rn, must be in the water. Wellwater from a local school about 2 km NE of the village gave 4095 cpm but the sequential Rns values close to the well were very low : 7, 7, and 3 cpm.

The lack of exposures in the area suggest the overburden is much thicker which may explain the low values in the soil. Surface water from a small bay on Lake Miri just south of the village gave a value of 24 cpm; this relatively low value is probably a function of wind degassing the very shallow waters of the bay (<1 m). On the other hand, two Rns sites at the edge of the lake gave sequential values of 334, 278, and 285 cpm and 982, 696, and 575 cpm, respectively. These values indicate high alpha activity in the soils with, as previously noted, a significant thoron component. It is to be noted that all high soil gas values occur in areas close to exposed quartz stockworks that gave the highest gamma readings in previous surveys.

A further four wells, three recent UNICEF wells W of Kadugli and one old hand-dug well in S Kadugli, were sampled for Rnw determinations. The three recent wells gave counts of 3154 at Tafari village, 8325 at Hagar Elnar village, and 4030 cpm at east Hagar Elnar village. Also, a sequential Rns determination at Tafari returned values of 41, 19, and 22 cpm. Although the soil gas values are small, probably due to the thicker overburden in the region, they correlate with other soil gas values and indicate a substantial thoron component in the alpha activity. Undoubtedly, the high total alpha activity in the wells is a measure of the residence time of the groundwater in the mineralized quartz stockwork aquifers. The old (and disused) hand-dug well gave a Rnw value of 118 cpm, low but still somewhat anomalous.

No further determinations could be carried out because of extremely high background counts in the scintillation cells. These high counts arose because Ag activated zinc sulphide cells accumulate a "memory" of previous determinations which takes a number of days to decay; the very high counts encountered on the survey, within a short period of time, raised the background counts to unacceptable levels (>2000 cpm).

The main conclusion from this last survey, where many wells have an alpha activity, after the decay of ^{220}Rn , of about 2500 pCi/l (the highest value recorded was 5130 pCi/l), is that the whole area around Lake Miri and extending southwards into the town of Kadugli is underlain by groundwater with disturbingly high levels of radioactive gases (and presumably their daughters). In addition, the soil gas results indicate anomalous alpha activity (on average 600 pCi/l) in areas of thin overburden close to exposed radioactive quartz stockworks. Although atmospheric dilution of soil gas at the surface probably means that health hazards due to inhalation are slight, any cultivated crops in the affected areas must uptake some of the radioactive daughter products from both the soil and the groundwater. To the authors' knowledge no systematic study of vegetation in the region has been undertaken.

Another factor that requires clarification is whether or not the radon gas is an orphan or a daughter; that is to say, is the soluble gas selectively extracted by the groundwater or is the water also extracting elements from higher up the decay series, in particular long lived isotopes such as ^{226}Ra ? In the

former case (i.e. the orphan) potential health hazards due to ingestion of water could be minimized by degassing the water (by agitation) prior to use. In the latter case (i.e. the daughter) the problem is more serious because ingested water will continue producing radioactive daughters from the contained radium.

4. CONCLUSIONS

The above text clearly demonstrates the presence of certain radioactive elements at anomalously high levels in groundwater in the Lake Miri - Kadugli area; certainly, further more sophisticated studies are warranted.

However, the main purpose of the text is to illustrate the unfortunate fact that data collected under the designation "uranium exploration" tend to remain just that. The data are rarely circulated to other areas or jurisdictions, even within the same organization! Thus, the end result is a mass of geological, structural, geochemical, etc. data (not just in uranium, but in exploration geology in general) that accumulates around the world but never gets utilized by other involved parties, such as, environmental scientists, health scientists, engineers, agriculturists, etc.

The present case study illustrates the above statements admirably: the earliest evidence of radioactive mineralization in the area of the dam was available in the late 1970's; evidence of groundwater contamination by radioelements was apparent by the mid 1980's and was substantiated in the late 1980's. Yet throughout this time UNICEF had a vigorous, and successful, campaign to drill some 200 wells in the area to serve some 40 000 people. In consequence, the local population now has an adequate water supply in this arid region, but much of it is contaminated (to what extent is not fully understood at this time).

Much of the blame for such case histories must lie at the feet of exploration geologists who have never impressed upon the various authorities that exploration data, often obtained "early" in the development of a specific region, are also an EXCELLENT BASELINE DATA SET that can be used for an initial assessment of environmental conditions in a given region. It is time for exploration geologists, in uranium or other commodities, to actively lobby their superiors as to the importance of geological work vis-a-vis the environment. After all, the "environment" is what the geologist studies and takes samples of; (s)he is the ultimate environmentalist.

As a final comment, the authors would suggest that the Division of Nuclear Fuel Cycle and Waste Management convene, at some appropriate time, a technical committee meeting to discuss guidelines for enhanced liaison between (ostensibly) uranium exploration projects and other interested parties further along the Fuel Cycle (e.g. mining, environmental, and waste management) and outside it (e.g. WHO, FAO, UNICEF).

REFERENCE

- [1] HASSAN, A-R. K. Prospecting methods for uranium and associated mineralization in the Nuba Mountains, Sudan. Unpublished MSc thesis, University of Khartoum, 266pp.

STUDY OF AIRBORNE GAMMA RAY SPECTROMETRIC DATA IN PREDICTION OF URANIFEROUS MINERALIZATIONS IN THE KUUSAMO AREA, NORTHEASTERN FINLAND

H. ARKIMAA

Geological Survey of Finland,
Espoo, Finland

Abstract

The Early Proterozoic Kuusamo Schist Belt in northeastern Finland contains many uraniferous mineralizations. Both bare U-mineralizations and Au-Co-U-bearing sulphide mineralizations have been found. At the moment exploration is carried out mainly for gold. Low altitude airborne magnetic, electromagnetic and gamma-ray measurements have been flown at an altitude of 30-50m with a 200m line spacing. Many of the gold bearing formations, so far discovered, are clearly indicated by the airborne magnetic and electromagnetic data. Some are indicated by gamma-ray data. Since only three percent of the bedrock in Finland is exposed, factors such as thickness of overburden and moisture content must be considered when interpreting gamma measurements. The ratios between the different energy channels proved to be the best variables to characterize the largest known uraniferous mineralizations under various overburden layers. Digital image processing and statistical operations were used to study how the known U- and Au-Co-U-bearing sulphide mineralizations are indicated in low altitude airborne gamma-ray data and how these measurements could be used more effectively in ore prospecting in the area.

1. INTRODUCTION

The study area in Kuusamo is a part of a greenstone belt extending from the Norwegian Sea to Lake Onega (Fig. 1). In the Early Proterozoic volcanic and sedimentary bedrock of Kuusamo, sericite quartzite formations have been actively explored in recent years (Fig. 2). Both bare U-mineralizations and Au-Co-U-bearing sulphide mineralizations have been found. Current exploration activity is directed towards gold in the area. Some thirty sulphide occurrences are known in the Kuusamo Schist Belt, of which about twenty are gold bearing. Intensive hydrothermal alteration is always associated with these gold deposits. Faults within antiformal structures in the middle part of the belt, appear to be the most important control of mineralization [1].

2. URANIFEROUS MINERALIZATIONS

The Geological Survey of Finland started exploration for uranium in the Kuusamo Schist Belt in 1979. As a result of these investigations, a narrow stratabound uranium mineralization, over three kilometer long, in an albitized sericitic quartzite was found.

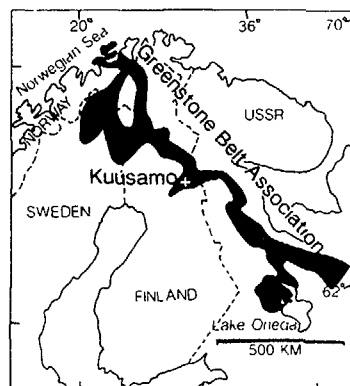


Fig. 1. Location of the Kuusamo area and the greenstone belt association, modified according to Pankka and Vanhanen [1].

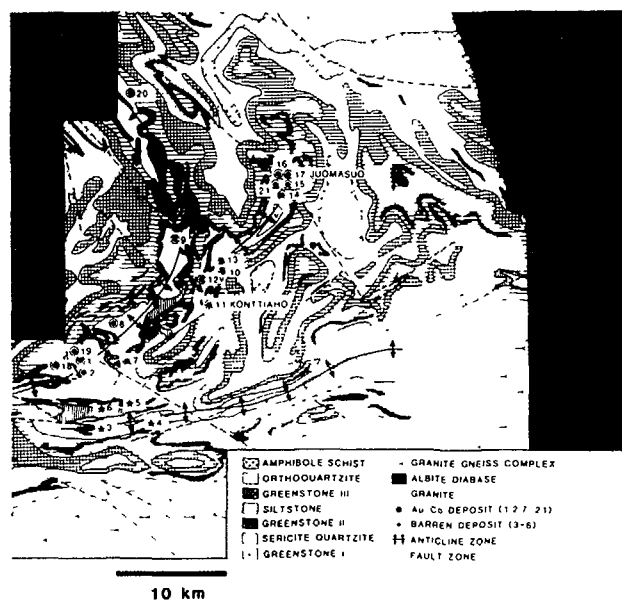


Fig. 2. Geological map of the study area in the Kuusamo volcano-sedimentary belt, modified according to Pankka, Puustinen and Vanhanen [9]. Names of the remarkable deposits are indicated.

This mineralization represents the ore group in the Kuusamo Schist Belt, where uranium is the only enriched component and only little or no sulphides occur [2].

Since 1982, the Geological Survey has explored for gold in the Kuusamo Schist Belt. Within Au-Co-U-bearing mineralizations uranium is, in some places, clearly enriched (as is gold). Uranium is, in general, only a trace element [2]. However, the positive correlation between uranium and gold gives a chance to use radiometric airborne and ground data in gold exploration.

3. GLACIAL GEOLOGY

Morphological streamline features are abundant in the study area due to an active ice lobe present during the last glaciations. Indicating the distribution of drumlins and fluting formations the streamline features are best developed in areas where the strongest movement of the ice coincides with the general tectonic direction [3]. The influence of the glacial overburden and its transportation must be considered when interpreting airborne gamma measurements for exploration of uriferous mineralizations.

4. AIRBORNE GEOPHYSICAL DATA

Low-altitude airborne geophysical measurements have been undertaken by the Department of Geophysics of the Geological Survey. At a nominal flying altitude of 40m and a line spacing of 200m, magnetic total intensity, electromagnetic in-phase and out-of-phase components and gamma radiation were registered simultaneously [4]. Six sodium-iodide detectors, with a total volume of 25 liters, were measuring gamma radiation in 120 channels once per second. Recorded digital raw data were combined in four windows: K, U, Th and total count. The energy ranges are 1.37-1.57, 1.66-1.86, 2.41-2.81 MeV and 0.41-2.81, respectively (Fig. 3). After corrections for background, scatter, air density and

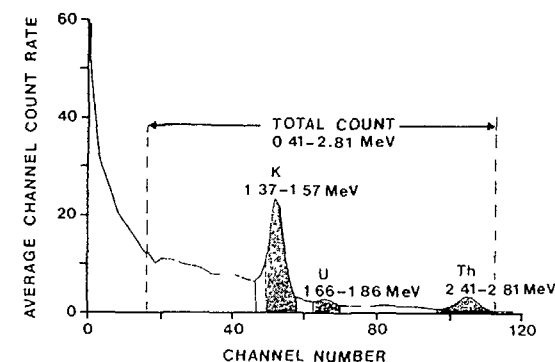


Fig. 3. An airborne gamma-ray spectrum showing the four windows: K, U, Th and total count and their energy ranges.

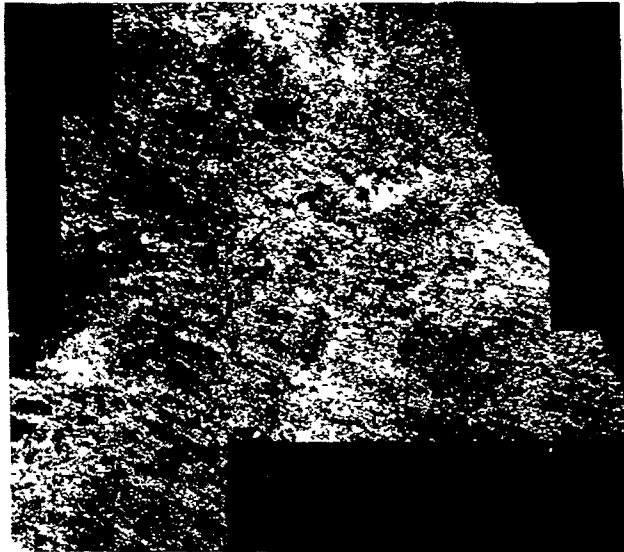


Fig. 4. Greytone coded map of airborne uranium measurements.

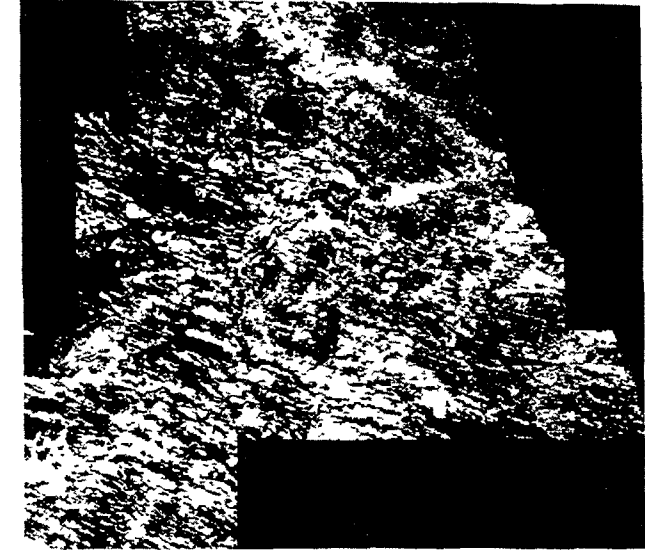


Fig. 5. Greytone coded map of airborne thorium measurements.

altitude the recorded count values were converted in equivalent concentrations (ppm for U and Th, % for K and Ur for total count). Operative corrections are not always able to eradicate the background level differences in data between flight lines. Therefore in this study, median type filtering of the data was carried out [5]. Finally the data were transformed into an uniform array with 50m*50m pixel size.

5. METHODS OF INTERPRETATION

5.1 Digital image processed data

Many image processing operations were made to the gamma-ray data in order to study general morphological, lithological and structural features of the area. In this paper only contrast stretched and greytone coded maps of uranium, thorium and potassium radiation are shown (Figs. 4-6).

The most conspicuous features in the gamma-ray data are the NW-SE trending streamline patterns. The data reflects distribution of drumlins, fluting formations, peat bogs and tectonic structures, in the area. Some features interpreted as tectonic structures are also indicated by other geophysical data, such as magnetic data.

Eskers in the study area are characterized by high potassium radiation. High potassium seems to indicate some lithological units, such as quartzite formations. Thorium radiation seems to increase in river valleys which are often formed by fractures.

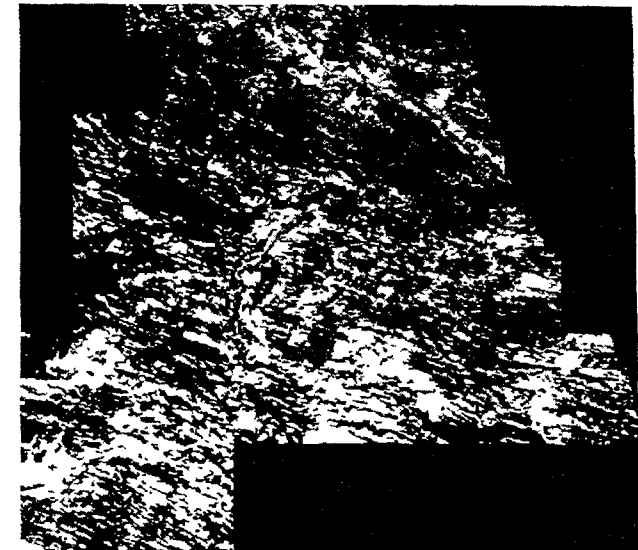
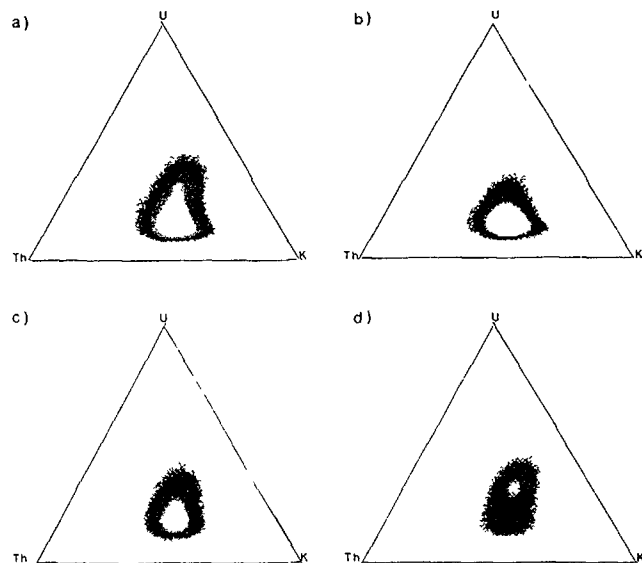


Fig 6 Greytone coded map of airborne potassium measurements.



7. Triangular diagrams showing the ratios between U, Th and K values
a) for the whole study area b) for dry land c) for bogs and d) for water-lain areas.

The high uranium radiation peaks shown in airborne measurements indicate surficial uranium enrichments in peat bogs. The Kouvertvaara hill area, with its uranium mineralization and uranium rich boulders, is characterized by high uranium anomalies.

5.2 Masks for water-lain areas and peat bogs

Water and moisture content of the soil are effective attenuators of gamma radiation. Because the water-lain areas and bogs cover about 38% of the study area, the data from inside and outside them could be studied separately with the help of masks. The mask for the water-lain areas was prepared from Landsat data. The mask for the bogs was prepared using digitized bog data.

5.3 Feature spaces and classification

In order to minimize the effect of overburden and water content, the ratios of the radioelements have been found to be more useful than the original radioelements [6, 7].

In this study the ratios between uranium, thorium and potassium radiation values were visualized using a triangular diagram. Using masks for water-lain areas and bogs it was possible to study the feature space more accurately dividing it into different parts (Fig. 7).

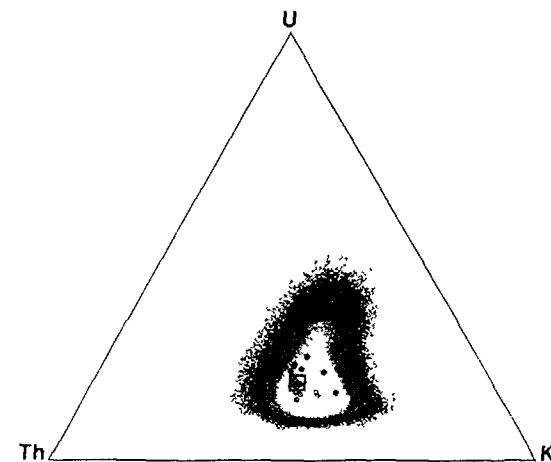


Fig. 8. The ratio values between U, Th and K and the known uranium mineralizations indicated by circles. The training area used in the classification is indicated by the box.

The gamma anomalies indicating the known uranium mineralizations were chosen as models, taking glacial transportation into account. The most indicative pixel for each mineralization was plotted on the triangular diagram (Fig. 8). The pixels coinciding with the known uranium mineralizations were concentrated in a specific part of the diagram. This subspace of the diagram was used as a training area for classification. Due to the statistical nature of the gamma radiation and the different attenuation in each energy window, the ratios calculated using low radiation values are not reliable. Therefore, the targets in low radiation areas were rejected.

5.4 Integration with other results

This work is part of a larger ore prediction study in the Kuusamo schist belt. The Landsat satellite data and airborne geophysical magnetic and electromagnetic data were interpreted separately [8]. The targets derived from the three different data sets were superimposed, and targets occurring close to each other were picked out for ground follow-up survey. The survey is in progress and has already resulted in discovery of one gold-bearing mineralization and two barren occurrences of altered rocks.

6. CONCLUSION

The interpretation of gamma-ray data in glaciated and poorly exposed area, as in this study, has many limitations. However digital image processing, integration of

different data sets and visualization of the feature spaces proved to be efficient in studying the data from different sources within the area and in selecting training areas for classification in order to predict favourable targets for mineral exploration

ACKNOWLEDGEMENTS

The writer would like to thank the Geological Survey of Finland for the possibility to publish this paper. Prof. Jouko Talvitie and Phil lic Viljo Kuosmanen are thanked for critical reading of the text. I am also indebted to Mr. A.G. Meadows for correcting the English.

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SOME ASPECTS OF THE URANIUM SITUATION IN ROMANIA

C. BEJENARU

Regia Autonoma Pentru Metale Rare,
Bucharest

M. BOBE

Regia Autonoma Pentru Metale Rare,
Feldioara

Romania

Abstract

Uranium exploration in Romania started in 1950 and in 1951 a high grade tabular sandstone deposit in Permian sediments had been discovered in the Apusani mountains of the West Carpathians. Subsequent work led to the discovery of additional deposits in this area and established a uranium province in the Western Carpathians. Further uranium provinces were outlined in the Banat mountains as well as in the Eastern Carpathians.

The Reasonably Assured Resources of Romania, unassigned to any cost category are estimated at 18 000 tonnes U.

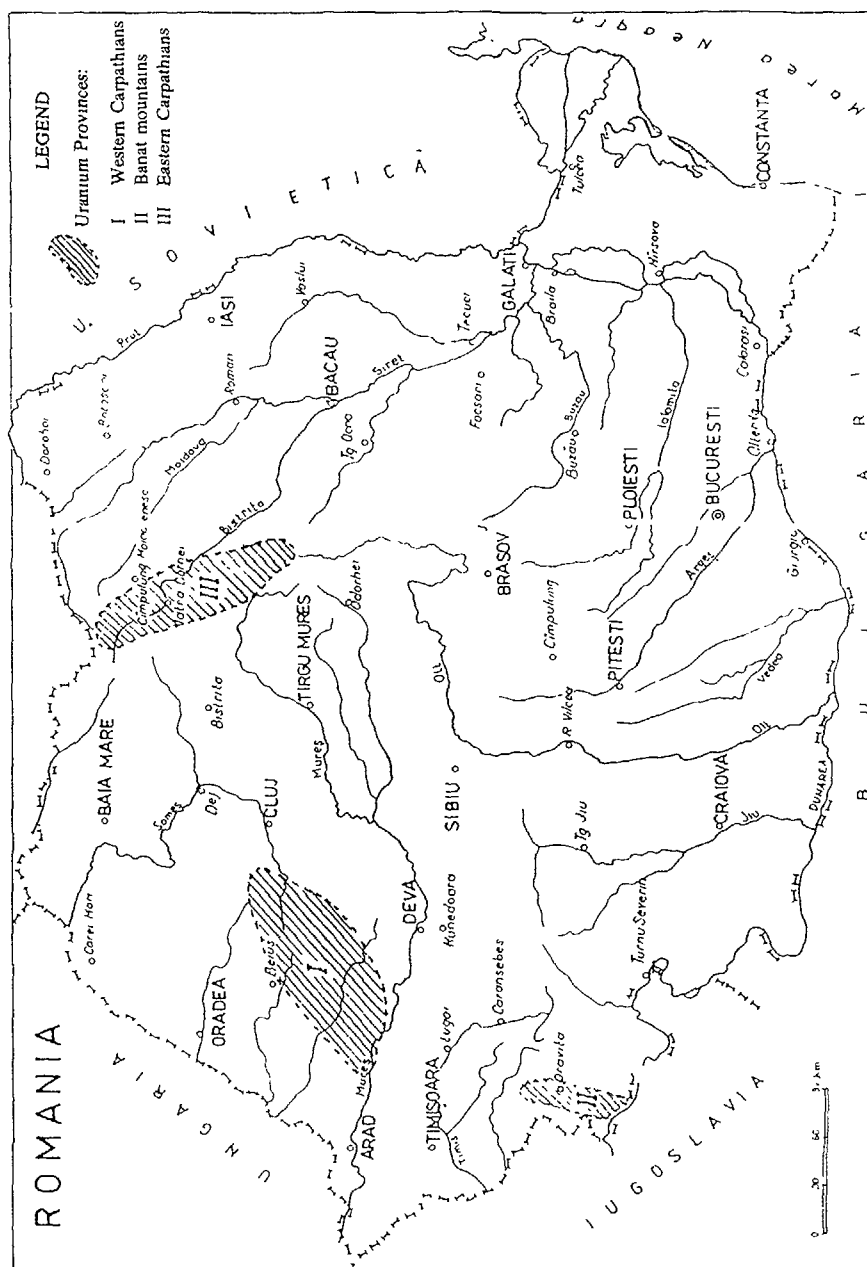
The current uranium production is about 200 000 tonnes of ore mined by underground methods. The production centres are located in the three uranium provinces. Due to the unfavorable geological conditions the uranium production costs are considered high.

Uranium exploration started in 1950 and in 1951 an exceptionally high grade deposit was discovered in Permian sandstones in the Apusani mountains of the Western Carpathians.

Subsequent exploration led to the discovery of a uranium district in the Apusani mountains of the Western Carpathians (see map), where the following deposit types had been found:

- vein type deposits in metamorphic rocks, with uranium associated with cobalt, nickel, copper, lead and zinc;
- stratiform deposits related to metarhyolites, where uranium occurs with molybdenum and occasionally, with lead and zinc;
- tabular sandstone deposits associated with low grade copper deposits.

The individual deposits of these types contain typically 1500 to 2000 tonnes ore with average grades ranging from 0.1 - 0.2 per cent U.



Further work in Romania discovered a second uranium province in the Banat mountains in the southwestern part of the country (see map). Here, uranium occurs in sandstone, in general associated with anthroxolite. The resources of this province range between 700 and 3000 tonnes U.

In the period 1960 - 1961 a third uranium province was discovered. It is associated with the metamorphics of the Eastern Carpathians (see map). The special characteristics for this province are the vein type deposits associated with organic matter. The veins are parallel to deep seated faults or to the schistosity of the host rock. The orebodies are surrounded by a clay-carbonate alteration halo.

Currently, exploration continues within the known provinces in order to increase known resources, and outside these provinces, to find new resources. In addition, all exploration activities for commodities are radiometrically checked. The resulting information are stored electronically in order to integrate all available information.

Resource estimates are done using in general classical methods. At present, the Reasonably Assured Resources unassigned to any cost category, amount to about 18 000 tonnes U.

Uranium mining is being done in all three uranium provinces by underground methods. The annual production is about 200 000 tonnes ore. Although the mining systems used have to be adjusted to the individual deposit, the basic most frequently used methods is a type of upwards leading sublevel stoping. Due to the frequent changes in the ore body characteristics, the ratio of development vs mining is unfavorable, leading to high production costs.

ECONOMIC TARGET MODELLING IN EXPLORATION: METHODOLOGY AND CASE STUDY OF THE WESTERN PART OF THE ATHABASCA BASIN, CANADA

R. GATZWEILER, B. VELS, R. BRAUN
Uranerzbergbau GmbH,
Wesseling, Germany

Abstract

Economic evaluations using the methods of Discounted Cash Flow (DCF) - analysis are standard practice for advanced projects in the mineral industry. At the feasibility stage of a project investment decisions have to be taken of major financial and strategic importance. The project data on which technical and economic evaluations are based upon is accurately calculated at this stage with limits of error which should not exceed 10 per cent.

Exploration can be regarded in a broader economic sense as an investment activity aiming at the definition of profitable investment possibilities for a mining company. Success in exploration is dependant amongst other factors on a continuous availability of risk capital which is always in short supply. Though - as well known - failures in exploration exceed by far successes it is generally expected that well planned longer-term exploration efforts pay off, i.e. the accumulated profits of a mining company exceed by far the accumulated exploration expenditures. Success in exploration and potential profitability of a project should be defined economically, i.e. by projected cash flow estimations or the Net Present Value (NPV) or Internal Rate of Return (IRR).

Economic orientation studies based on DCF calculations are a valuable evaluation tool also in the early stages of projects. They are suitable to define the economic target of a project, i.e. the potential profitability and its dependance on project parameters which are partly known and partly still unknown at this stage. They also allow target modelling and thereby can provide guidance for exploration. They also fulfill an educational obligation of the exploration group within a company by more clearly describing the essential factors involved in exploration, in particular time, money and risks - but also potential profits.

Since at an early stage of an exploration project many of the variable project parameters and in particular the deposit parameters are still unknown, a model deposit is established as the most likely case according to the best knowledge available at that time. Then the costs of a mine and mill based on this hypothetical deposit are estimated. Subsequently the cost estimates are used together with the various fixed parameters as e.g. taxes and royalties in DCF-calculations.

The methodology is further explained by a case study which evaluates a modelled project at the western rim of the Athabasca Basin. Comparisons are drawn with advanced projects at the eastern rim of the Basin.

One aim of this paper is to show that an economic evaluation is a powerful tool at all project stages beginning at the armchair stage, guiding the exploration phase until the final feasibility and investment decision.

1 INTRODUCTION

Economic evaluations using the methods of Discounted Cash Flow (DCF) - analysis, also referred to as dynamic methods since they take the time value of money into account, are standard practice for advanced projects in the mineral industry. At the feasibility stage of a project investment decisions have to be taken of major financial and strategic importance. The project data on which technical and economic evaluations are based upon is accurately calculated at this stage (detailed engineering studies) with error limits not exceeding 10 per cent.

This expected degree of accuracy can not be reached at the prefeasibility stage when an ore deposit has been discovered but reserves and other deposit parameters have not been defined by detailed drilling and other investigations. At this stage the results are correct within limits of 25% [1]. The cost estimation of prefeasibility studies generally is based on preliminary cost estimation and parameter definition.

Orientation studies are performed at the early project stages - sometimes even prior to the start of field activities. Their general aim is to provide a target model and a rough and preliminary analysis of the economic realization chances of this particular project. It is expected that an orientation study indicates in terms of costs and target models the order of magnitude. The cost evaluation is based on comparison with similar projects in this area and/or experience from past projects. Information from public statistics, resource reports and evaluation papers prepared by professional institutions be of use as well.

2 OBJECTIVES OF AN ORIENTATION STUDY

An ordinary economic analysis is based on given input parameters, and yields certain output parameters like IRR and NPV characterizing the economic viability of a project. In contrast the emphasis of an orientation study is put on the input parameters, while the output parameters are predetermined based on economic expectations.

The major objectives of an orientation study can be summarized as follows:

- the definition of the economic significance of a hypothetical model deposit,
- the definition of the minimum expectation for the individual parameters in order to meet certain economic expectations,
- the comparison between different conceptual models,
- to analyse the influence of different internal and external parameters on the overall economics of the project,
- the influence of individual parameter variation on the economic viability (sensitivity analysis),
- to determine the probability at which the project is yielding an IRR above or beyond an expected value (risk analysis).

3. ORIENTATION STUDY

The variable project parameters like for instance many of the deposit parameters are still unknown at this stage. Therefore a model deposit is established as the most likely case according to the best geological knowledge at the time.

Then the costs of a mine and mill based on this hypothetical deposit are estimated. Subsequently these cost estimates are computed together with the existing financial parameters of the project as e.g. private and provincial royalties, compensations, taxes etc., see Fig. 1.

At this early stage of a project, it is advisable to calculate with "constant monetary values", i.e. without considering any escalation [2]. This method implies that prices and costs develop at the same rate in the future. For project evaluations with predictable price and cost developments it is more favourable to consider an escalation, i.e. to calculate with "current monetary values".

A correct evaluation of the influence of taxation on the projected cash flow can be achieved only if the influence by an escalated calculation. A constant calculation will overestimate the benefit of the constant depreciation pool. An escalated calculation will take the declining value of depreciation into consideration.

Orientation studies are of particular importance when considering activities in new areas. Then they are part of the fact-finding stage of a project during which all information which could possibly influence the economics of a mineral project, is gathered and analysed. Especially in developing countries without an established mining industry and mining related legislation and taxation, orientation studies are part of the negotiating process to define the tax and royalty regime applicable for the successful development of an ore deposit.

Orientation studies should be carried out by an interdisciplinary group comprising geologists, mining engineers, metallurgists and economists. The same group, ideally, should later in case of a successful development of the project, be involved in the preparation of the prefeasibility and feasibility studies. This early familiarisation with the project parameters, scope and target of the project concerned will in practice result in a more objective judgement on project related matters and a broader support for the project.

4. INPUT DATA

Conventionally the input parameters are subdivided into "internal" and "external" or "natural" and "economic" project parameters [3]. Krige [4] introduced the classification into "decision parameter" and "risk parameter". In the following a classification has been attempted combining both methods.

4.1. Main internal geological/technical parameters

Internal risk parameters

- ore grade) ore reserves, metal content
- ore tonnage)

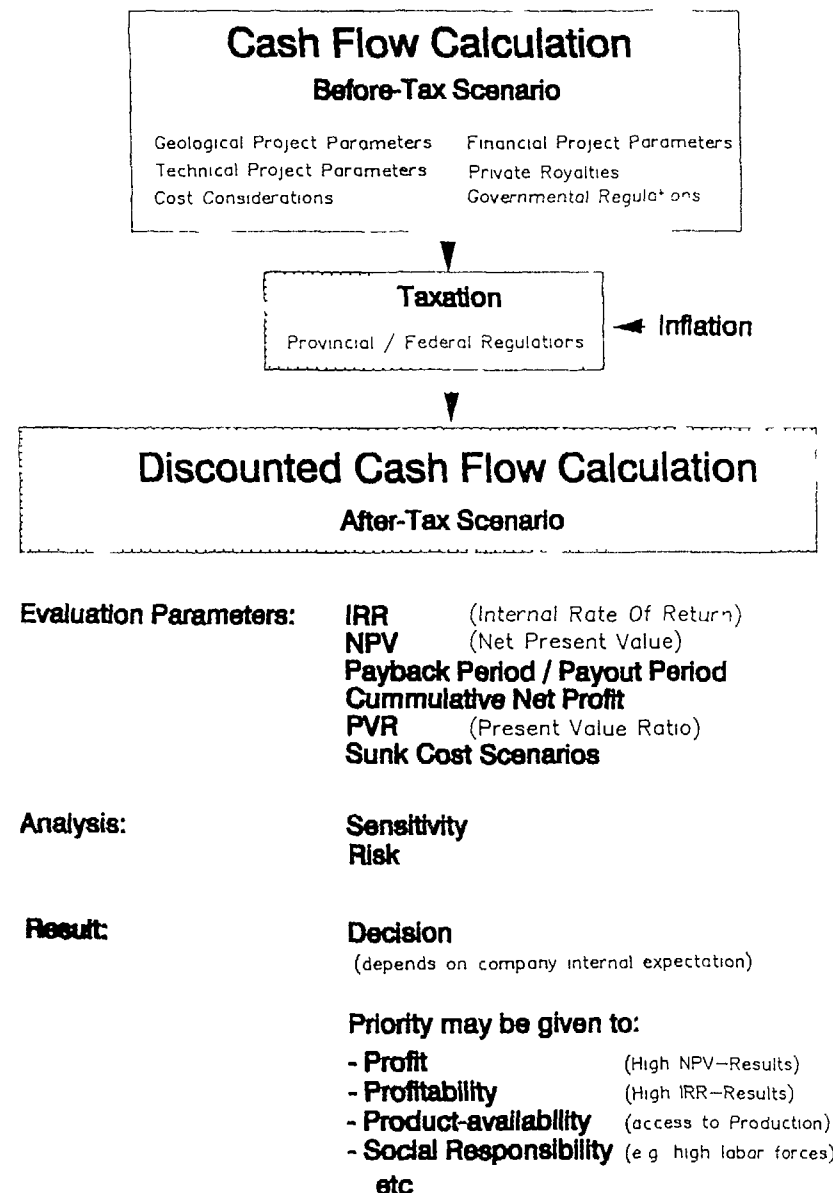


FIG. 1 Discounted cash flow calculation, input and evaluation possibilities

- ore paragenesis
- ore distribution, ore depth
- geotechnical conditions
- hydrogeological conditions

External risk parameters

- infrastructure
- environmental restrictions
- availability of trained labor
- availability of toll milling facilities
- political acceptance

Internal decision parameters

- production capacity
- mining method
mining recovery
- extraction method
metallurgical recovery
- quality of end product

The main geological parameters, ore grade and ore tonnage determine the metal content, which - after correction for mining and recovery losses - is the saleable product of a mine. After definition of the mining method and the suitable size of the selective mining unit, grade and tonnage of a deposit depend on the applicable cut-off grade. This is the back-calculated minimum metal concentration required to yield a minimum profit based on cost and commodity price. The cut-off grade thus classifies the rock into ore and waste. The dependence of grade and tonnage on the cut-off grade is illustrated in grade/tonnage curves like the one shown in Fig. 2. In dependence of the commodity price ore grade and tonnage thus determine a mine's revenue.

The main technical parameters, mining and milling, are the major cost items. Mining options generally range from low unit cost bulk mining to high unit cost selective mining methods. The achievable selectivity and mining recovery play a major role in deciding on a mining method, and in turn influence the mineable metal content. In choosing a mining method from the technically possible options, the problems arising from dilution in bulk mining have to be carefully balanced against the higher unit costs of avoiding these problems with more selective mining methods.

For ore milling the options usually vary between high tech - high investment - high recovery mills and low tech - low investment - low recovery processing options. Apart from the stand-alone processing option there may be opportunities for toll milling in developed mining districts.

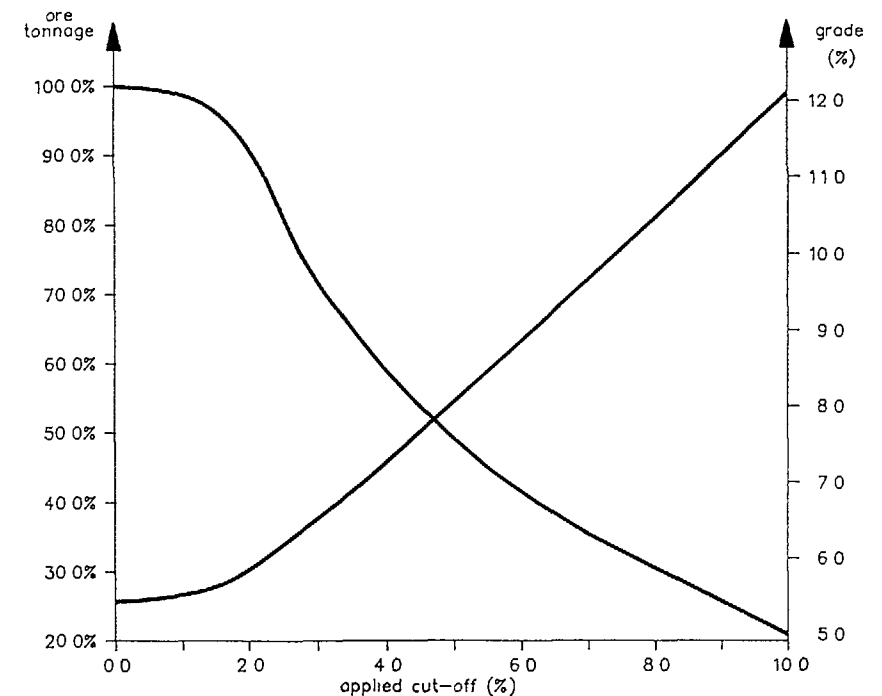


FIG. 2 Grade-tonnage curve illustrating dependency of ore grade and ore tonnage on applied cut-off grade

The varying influences of the geological and technical parameters on the economic evaluation can only be fully considered if reserve estimation, preliminary mine design and economic evaluation are carried out by an interdisciplinary team.

4.2. Internal financial parameters, directly derived from the geological/technical parameters

Internal financial risk parameters

- exploration costs
- acquisition costs
- operating costs
- capital costs
initial investment
sustained investment

- working capital
- restoration/rehabilitation costs
- salvage value

The internal financial parameters and cost estimates are quite often based on first or second hand experience in similar projects with comparable conditions

The transition from the geological/technical parameters to the internal financial parameters is the most important interface of the entire chain of information

The means of meter -, cubicmeter -, per cent -, pound - and kilogramfigures will be transferred into monetary terms. After this transformation dollars and dollars per year or dollars over time are the economic measure

4.3 External financial parameters, independent from any special geological/technical scenario

External financial risk parameters

- taxes (provincial and state/federal)
- royalties (private and provincial)
- commodity price
- exchange rate

External financial decision parameters

- project financing (equity or debt financing)
- economic expectation (depending on the financial strategy of the company)

External financial parameters are more or less fixed and independent from any special geological/technical input. The influence of the owners and the operator on these parameters is rather restricted, though royalty and tax schemes can substantially influence the layout and management of an operation. Sensitivity analyses should define the influence of varying tax-, royalty- and commodity price-scenarios on the project economics.

The ratio between equity and debt financing can have a considerable influence on the overall project economics. An investigation of varying financing scenarios including their implications on taxes and royalties is advisable.

5 CASH FLOW METHODOLOGY

The economic outcome of an investment opportunity is determined by the time distribution of cash flow, the difference between cash inflow (or benefits) and cash outflow (or costs) over its projected future life [5].

Fig. 1 explains in general terms the major input items of a cash flow calculation and some evaluation possibilities. It still remains to be mentioned that a model project (a cash flow calculation is the monetary expression of a model project) always should represent a most likely case scenario, wishful thinking can result in a financial disaster.

"Non-cash" items such as depreciation, amortization, depletion, and deferred taxes, which may be a part of financial statements or allowable deductions for the assessment of tax payments, are not included in the cash flow calculation for economic evaluation purposes.

The main cash inflows are the revenues from the sale of product.

The main cash outflows are exploration expenditures, capital cost, operating costs, royalties, tax payments, transportation, insurance and further processing charges. Normally, the annual cash flows are negative during the exploration and investment phase whereas they are positive during the production phase.

The Internal Rate of Return (IRR) measures the average annual percentage return on capital investment that the particular project is expected to yield over the total project life [5]. The IRR is defined as the discount rate which equates the present value of the positive cash flows with the present value of the negative cash flows [5], in other words, the IRR is the discount rate that results in a net present value of zero.

This again means that the investment alternative with the highest IRR is preferred. The lowest acceptable limit is an IRR equal to the company internal interest rate (determined by the interest rate paid on debt capital and by the interest rate expected on equity capital) at which the funds can be made available (cost of capital). The IRR measures "profitability", the relationship between profit and investment.

The Net Present Value (NPV) measures the monetary value of the economic worth of a particular project. This method converts the anticipated time distribution of cash flow for an investment alternative into an equivalent value at the present point in time [5]. The NPV indicates e.g. to a potential investor the maximum amount that could be offered to acquire this project without suffering an economic loss. The NPV is directly related to the selected discount rate and thus grossly depends on the correct estimation of this discount rate over the life of the project.

The NPV measures "profit", the difference between the discounted annual cash flows. A NPV result is the consequence of both project size and profitability.

characteristics. A larger investment alternative with a higher NPV value does not necessarily indicate a more profitable opportunity.

The payback period is most commonly defined as the number of years required to recover (or pay back) the project related investment costs out of the positive annual cash flows, measured from the start of production. The payout period takes additionally the timing of cash flows and all other cash disturbances (taxes, front-end payments, royalties etc.) into account, considering the economic advantage e.g. by providing higher cash flows in early years.

Payback period measures the return of the investment itself - as a technical measure. Investment opportunities with a short payback period (short in relationship to the total mine life) are preferred. The payback or payout period are indicators of the project risk.

Cumulative Net Profit is the total anticipated financial profit at the final year of a project lifetime, no discounting and no methods of converting this figure into present (as present point in time) values are applied.

The Present Value Ratio (PVR) method measures the net present value (NPV) per unit of investment [5]. The PVR assists with the ranking of investment alternatives.

Sunk cost considerations are applied, when the objective is to identify an investment opportunity with the most favourable future cash flow. Because only future consequences of investment opportunities can be affected by current decisions, an important principle in economic evaluation is to disregard costs that have been incurred in the past [5].

6 WEST RIM ATHABASCA BASIN MODEL PROJECT

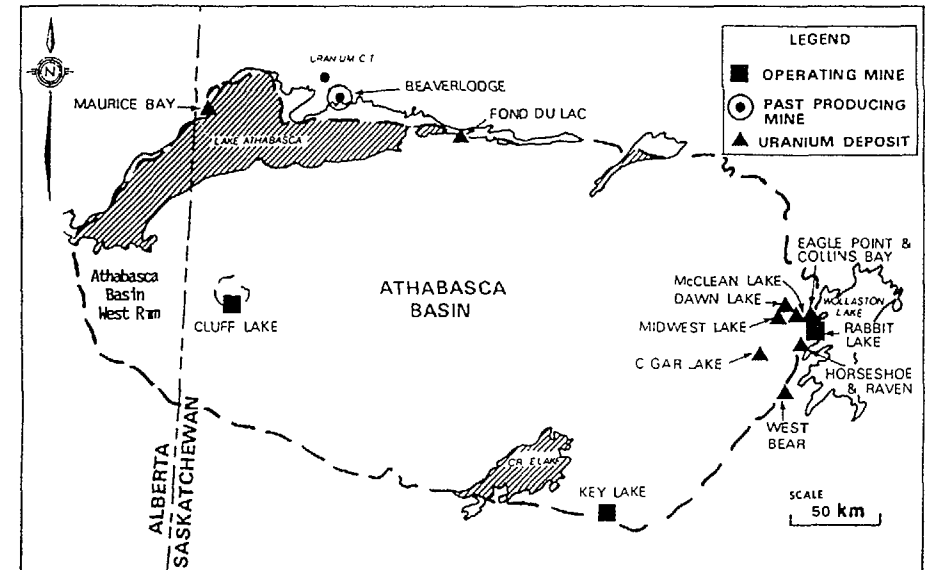
The location of the west rim model project is shown on Fig. 3. The project area is located in the Province of Alberta where, contrary to Saskatchewan, neither historically uranium mining has been carried out nor is it conducted presently.

The infrastructure of the project area is nearly as poorly developed as at the eastern rim of the basin. The nearest supply town with a railroad link is Fort McMurray. The Athabasca River is navigable by barges during summer. Some road construction would be necessary for a mine development in the area.

There are no permanent settlements in the area.

Trained labor would need to be attracted in communities to the south and would work on a fly-in/fly-out schedule as is common at the mines at the east rim. Therefore only temporary camps have to be considered for accommodating and boarding a work force.

The topography is quite similar to other parts of the Athabasca Basin. About 40% of the surface area is covered by lakes and swamps.



Source: Ref. [6]

FIG. 3 Principal uranium deposits in Saskatchewan

Major environmental restrictions only exist in an area with large sand dunes to the west which is protected by a conservation zone and exempt from exploration and mining.

In summary, the external risk parameters of the model project at the west rim of the Basin are in many respects comparable to those of the existing projects at the eastern rim. A certain negative influence, though difficult to quantify in terms of costs, relates to the fact that so far the Province of Alberta has not seen any uranium mining.

The internal risk parameters of the model project concern shape, depth, size and host rock of the orebody and grade and nature of the ore. Furthermore, the geotechnical and hydrogeological conditions have to be considered.

From reconnaissance surveys performed within the project area, the regional geology and in particular stratigraphy and structure were sufficiently well known to speculate on the occurrence of unconformity-related deposits. Therefore, based on the best available information at the time of the study, a geological model, a geotechnical model and a hydrogeological model of the hypothetical orebody were established.

The generic geological model comprised the following elements

- stratigraphy, major factors are thickness and type of Quaternary overburden, petrology of the prevailing Athabasca Group formations, average depth to unconformity, presence of regolith and basement stratigraphy
- structure, the emphasis is on potentially ore controlling basement structures and in which way they effect the overlying sediments
- orebody, geometry, tonnage, grade, nature of mineralisation including acid and oxidant consuming constituents
- alteration, of sandstone and basement

The generic geotechnical and hydrogeological models naturally are closely related to the above parameters of the geological model. Since quantified data on geotechnical and hydrogeological parameters in the early project stages are commonly still lacking the models were largely based on average parameters prevailing at eastern rim projects with adjustments to local conditions where significant differences were known.

Once the generic ore deposit model has been established total exploration costs and expenditure per year have to be estimated up to and including a Feasibility Study, an Environmental Impact Statement (EIS) and the procurement of licences and permits. For the purpose of economic orientation studies for projects at an early stage all of these costs should be included in the DCF calculations since the main objective is the evaluation of the economic target of the project which can be substantially influenced by an expensive and long-lasting exploration phase. At the prefeasibility and feasibility stage of a project more commonly sunk cost considerations are applied.

Due to the likely depth, size and geometry of an orebody the conceptual mine model considered only underground mining. At the time little specific reference data was available for underground mining of high grade uranium ore from projects at the east rim. Historical and recent data from underground mining at Beaverlodge and Cluff Lake and planning data from the Midwest and Eagle Point projects were considered for estimates of capital and operating costs for the hypothetical mine.

The milling concept was based on a high-grade polymetallic ore. Costs estimates for milling therefore were derived from the Key Lake operation.

7 DISCUSSION OF RESULTS OF DISCOUNTED CASH FLOW CALCULATIONS

The main objective of the orientation study for a conceptual project at the west rim of the Athabasca Basin was to provide guidance for exploration with respect to the following questions:

- how does the east rim project regime compare to the west rim and
- what is the minimum required grade and tonnage for an economic ore deposit at the west rim in case of a stand alone project and in case of toll milling

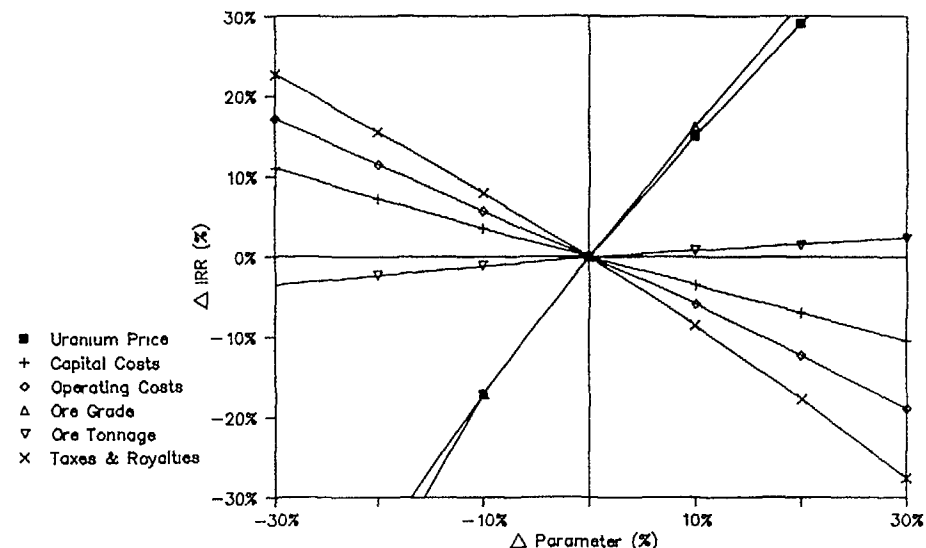


FIG 4 Average producing uranium mine, East Rim Athabasca Basin, stand-alone alternative, results of sensitivity analysis

Fig 4 summarizes the sensitivity analysis of the six most sensitive parameters for an average producing uranium mine on the east rim of the Athabasca Basin. Information of currently producing mines and of economic deposits are likely to be developed as mines has been integrated in this diagram.

The major conclusions for the east rim deposits are

- Ore grade and uranium price are the most sensitive parameters (as usual) followed by taxes and royalties, operating costs, capital costs, and ore tonnage.
- The ore grade and the uranium price curve are close together but not parallel, due to the variable influence of different ore bodies.
- Ore tonnage is the most stable parameter with only very minor influence on the project rentability. The east rim deposits are known for their outstanding sizes (e.g. Key Lake, Eagle Point, McArthur, Cigar Lake), which makes the project economics insensitive against total tonnage variations.
- Changes of the existing high tax- and royalty-regime (e.g. the Graduated Royalty in Saskatchewan can reach 50%, calculation base is the profit) can have a major impact on the project economics.

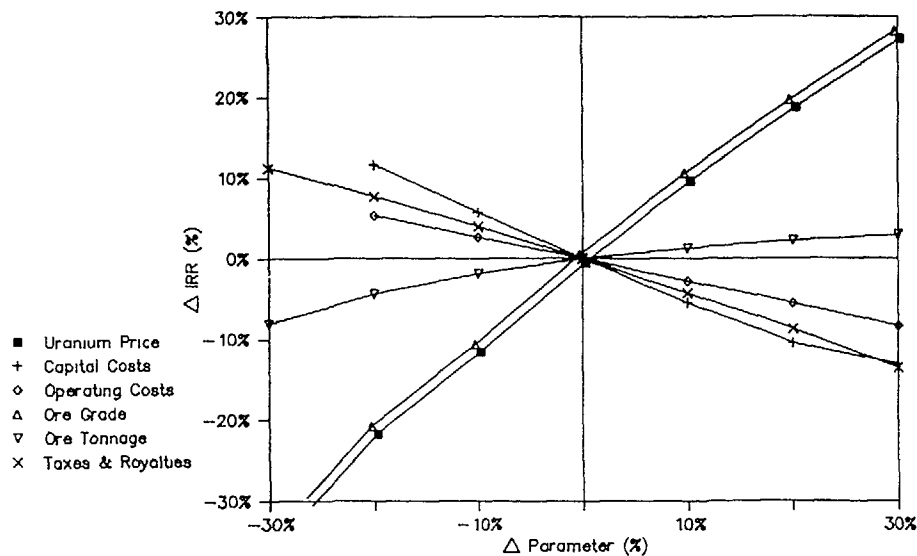


FIG 5 Model uranium deposit West Rim Athabasca Basin, stand alone alternative, results of sensitivity analysis

The model uranium deposit on the west rim of the Athabasca Basin shows sensitivity curves (Fig 5) which are quite different to the east rim

- The ranking of the sensitive parameters is ore grade and uranium price followed by capital costs, taxes and royalties, ore tonnage, and operating costs
- Ore grade and uranium price are again the most sensitive parameters but less sensitive (at least for the observed range) as for the average east rim deposit. This seems to be an anomaly, but can be explained: the modeled west rim deposit showed under base case conditions a higher IRR-result as the average east rim deposit

Capital costs are for smaller deposits commonly an important factor to keep under control since the depreciation base is smaller

Ore tonnage becomes a somewhat more sensitive parameter. High ore tonnage sensitivity can have a severe influence on the sampling strategy

Taxes and royalties are less sensitive (The Alberta Mining Tax for the post-payment period does not exceed 12%)

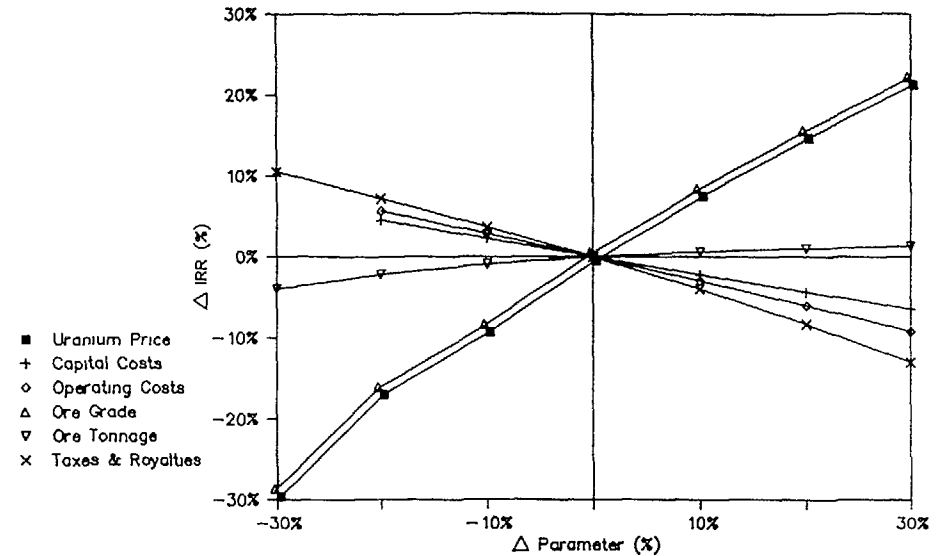


FIG 6 Model uranium deposit West Rim Athabasca Basin, Toll-Milling Key Lake, results of sensitivity analysis

A very important part of economic project evaluation is the search for conceptual alternatives. For the model deposit at the west rim a conceptual alternative could be a toll-milling scenario. Fig 6 shows the sensitivity analysis of a toll-milling scenario at Key Lake, considering road construction costs and including allowances for mill depreciation.

The ranking of the sensitivity parameters for the model deposit under toll-milling condition is ore grade and uranium price followed by taxes and royalties, operating costs, and ore tonnage.

Noticeable is the less sensitive influence of the capital costs (no mill on site).

Operating costs are negligibly more sensitive (transportation, toll-milling fee). Ore tonnage is less sensitive as well.

Previously it was indicated that the tax and royalty regimes, a major factor of the external financial parameters, vary between countries and provinces.

Fig 7 shows the relative annual cash flow of the model deposit under the Alberta tax and royalty regime and for comparison purposes under the Saskatchewan tax and royalty regime. The more favourable influence of the Alberta tax and royalty regime is obvious.

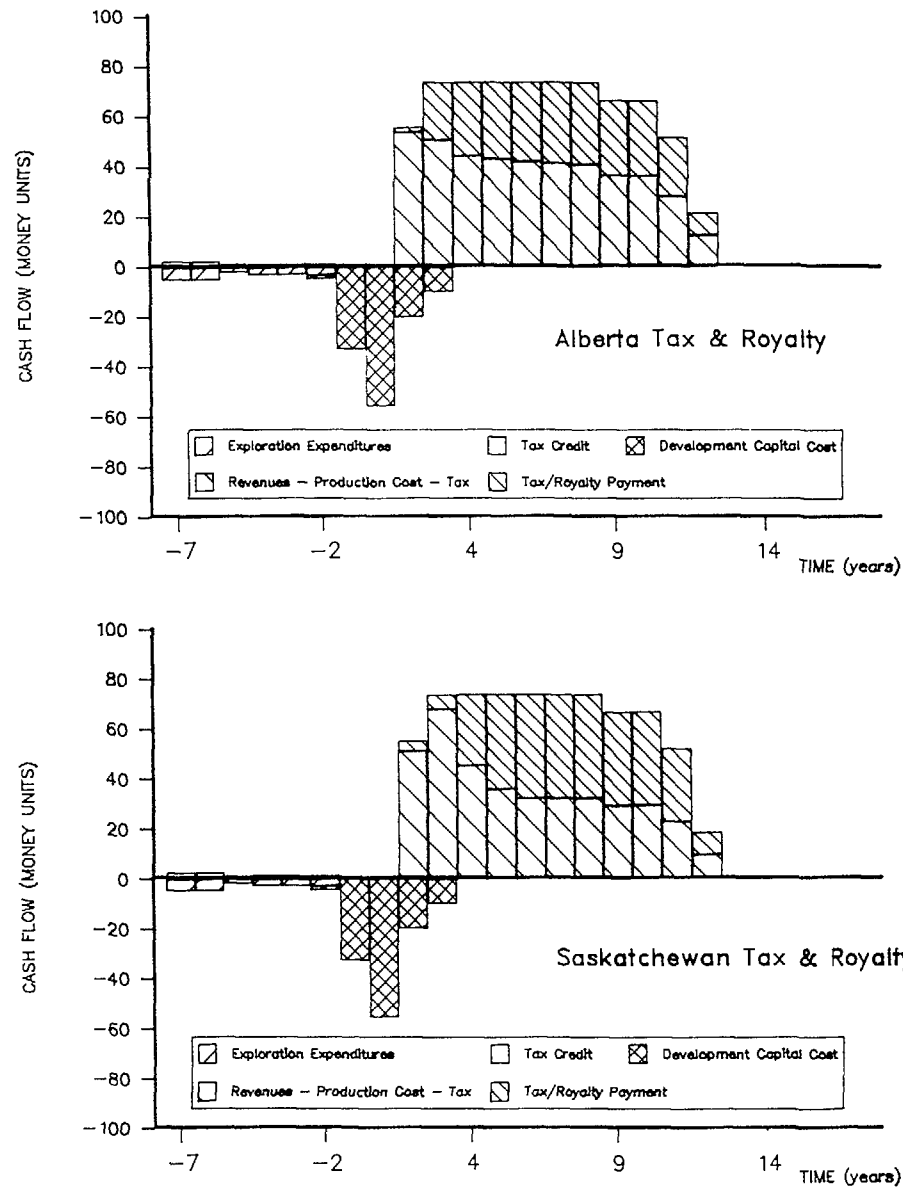


FIG 7 Influence of different tax and royalty regimes on the model uranium project West Rim Athabasca Basin

8. CONCLUSION

A project evaluation is often misunderstood as a static, one time reporting effort. One aim of this rather general paper is to indicate that a continuous project evaluation is a powerful tool which should accompany a project during all phases from target definition during the very early stages of exploration to prefeasibility and feasibility. After start of production it is extensively used to evaluate the impact of varying internal and external project parameters.

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HOW FINITE ARE THE URANIUM RESOURCES IN SOUTH AFRICA?

B B HAMBLETON-JONES, L C AINSLIE,
M A G ANDREOLI
Atomic Energy Corporation of South Africa Ltd,
Pretoria, South Africa

Abstract

In South Africa, uranium is in all instances mined as a by product, the major portion being produced from the gold mines of the Witwatersrand basin. The stagnation of the gold price over the past three years and the likelihood of it continuing for the next three years, engenders limited optimism in the industry. Uranium in particular, because of its low price, is already suffering in the rationalisation process taking place in the gold mines. Gold production is falling steadily making uranium the inevitable casualty.

As an alternative source of uranium, attention is being refocussed on the Karoo formations which are the only other potentially viable sources of uranium in South Africa outside the Witwatersrand Basin. Factors being re-examined are influences of stratigraphy, source areas, tectonics and volcanism on the distribution of uranium.

Lesser attention is being given to uranium mineralization in the Namaqualand Metamorphic Complex which has associated monazitic ores.

URANIUM RESOURCES AND PRODUCTION

In terms of the tonnage of uranium contained in the various deposit types in South Africa, it is effectively placed second after Australia. As is well known, all of the annual uranium production in South Africa, with the exception of about 140 tons U from Phalaborwa, comes as a by product to gold from the Witwatersrand basin conglomerates. Table 1 gives the resources for 1987 and 1989. It is pertinent to observe that there was a significant change to lower tonnages in 1989 as compared to 1987. This was due to escalating working costs and the consequent shift to higher RAR + EAR cost categories. The trend of increasing working costs is continuing. To place it into perspective, Fig 1 illustrates the cost of uranium production against the RAR + EAR resources. The available resources at the spot market price of \$25/kg U are about 80 000 tons U whereas at prices for long term contracts at \$50/kg U are about 140 000 tons U which are very much lower than the resources given in Table 1.

In the Witwatersrand, there are currently four producing uranium mines, namely Buffelsfontein, Hartebeesfontein, Vaal Reefs and Western Areas and five that are no longer producers, namely, Stilfontein, Randfontein, Harmony, ERGO and Freegold. The latter two recovered uranium from slimes dumps. At the non-producing mines, uraniferous slimes are being dumped in tailings dams which are now being used in some instances as back filling underground thereby reducing the potential resources.

TABLE 1 SOUTH AFRICAN URANIUM RESOURCES (t U)

	RAR <\$80/kg U	RAR + EAR I <\$130/kg U	RAR + EAR I <\$260/kg U
1987	324 800	536 500	679 300
1989	253 100	432 500	561 100
DIFFERENCE %	-28.33	-24.05	-21.07

- NOTES
- 1 The drop in resources is a function of a drop in the uranium and gold prices and an increase in working costs on South African mines. These factors resulted in an upward shift of cost of exploitation of the uranium resources.
 - 2 A stagnant gold price for the last two years, and increasing working costs will result in the resources as at 1 January 1991 being still lower.

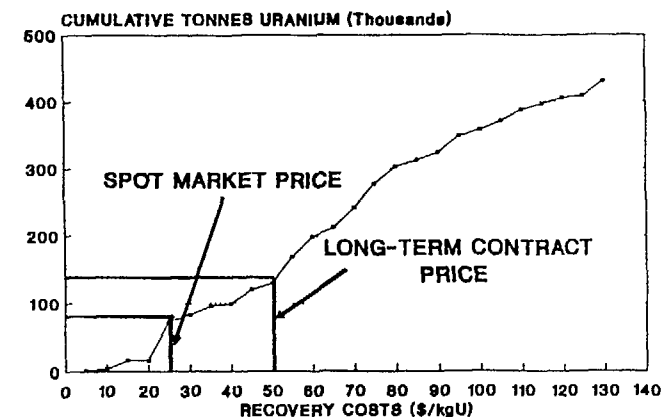


FIG 1 South African uranium resources, 1989 (RAR and EAR I <\$130/kg U)

Historical, current and future production trends are given in Fig 2. There has been a steady decline to 2 487 tons U in 1990 which is predicted to fall further to level out at 1 000 tons U by 1995.

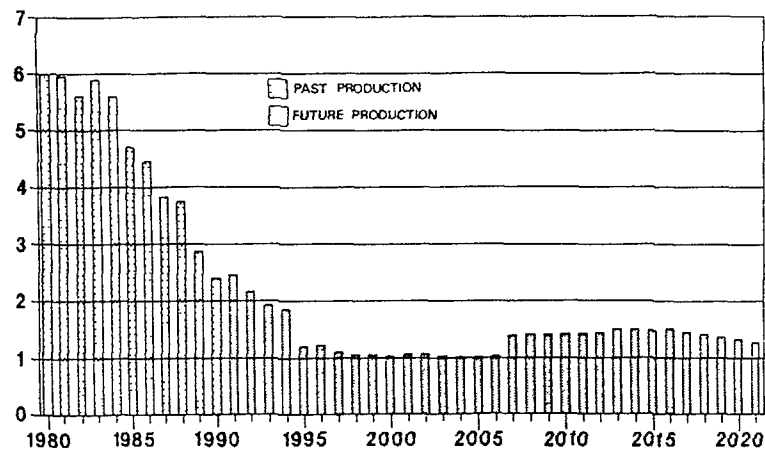


FIG 2. Uranium metal production (thousand tons).

URANIUM RESOURCES IN TERMS OF THE GOLD PRICE

There are two main influencing factors controlling the viability of uranium production in South Africa. Firstly, the price of uranium to a large extent is governed by a simple supply and demand scenario and currently the demand is low. Secondly, it is the gold price. In the past, gold formed the corner-stone of the world's economies, the value of which responded rapidly to conflict crisis, high inflation, high fuel prices or debt crises but now sentiments on gold have changed and other factors are now the major players. Gillan (1990) and Murray *et. al.* (1991) have highlighted some of these influencing factors.

In real terms, the Rand gold price has fallen for the third year in a row and coupled with somewhat volatile and chaotic trading, investor sentiment has been sluggish. Combined with forward selling, gold has been forced to trade in a narrow range between \$350 - 420 per oz. Even the Gulf Crisis in early 1991 or the temporary deposing of Mikhail Gorbachev in August 1991 had only transitory influences on the gold price. Other factors included the US inflation rate, real rates of interest in the US, banking crises in the US and Japan and the disposals by the National Commercial Bank in Jeddah depressed the market.

The effect of this low gold price is having serious consequences for the South African mining industry (Gillan, 1990). In 1989 at R1 008/oz and before capex four mines were operating at a loss. At the beginning of 1990, 11 mines operated at a loss constituting 15% of the production.

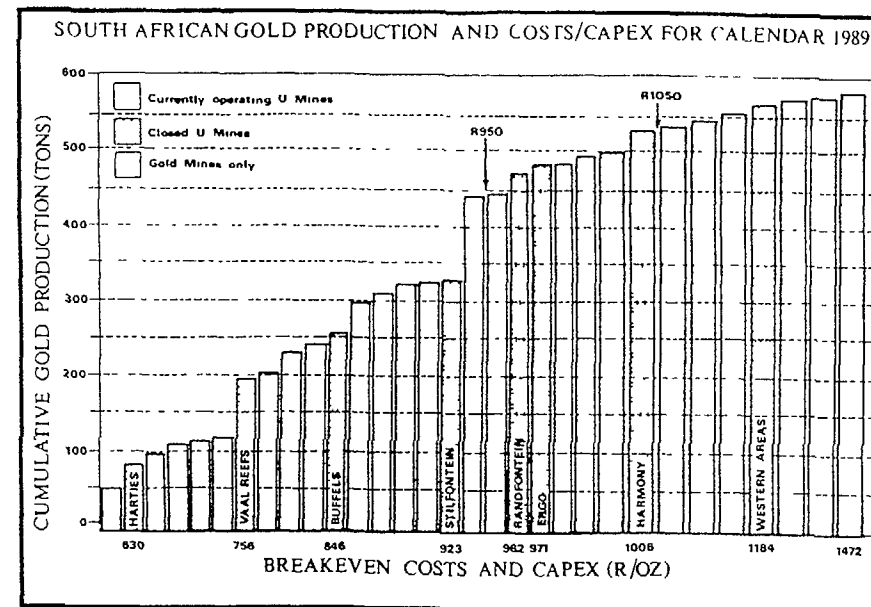


FIG 3

This scenario is exacerbated when capex is added (Fig. 3 and 4). It is noted that depending on the breakdown costs accepted, at R1 050/oz there were 7 and 9 mines for 1989 and 1990 respectively that were operating at a loss but for R950/oz there are 14 to 20 mines in jeopardy.

Quoting Gillan (1990), "if the gold price were to remain for the next three years at the levels achieved over the past three years (\pm R1 000/oz) all South African gold mines would be running at a loss".

Using a slightly different scenario Graham-Parker and McDermott (1991) rated the mines into four categories using a declining real rand gold price of R950/oz over the next three years. They found that of the 47 gold producers considered (which include the uranium mines);

- 6 would remain profitable,
- 12 would have to take substantial steps to improve yields or reduce costs including capex,
- 14 would have great difficulty in remaining profitable,
- 15 would probably close

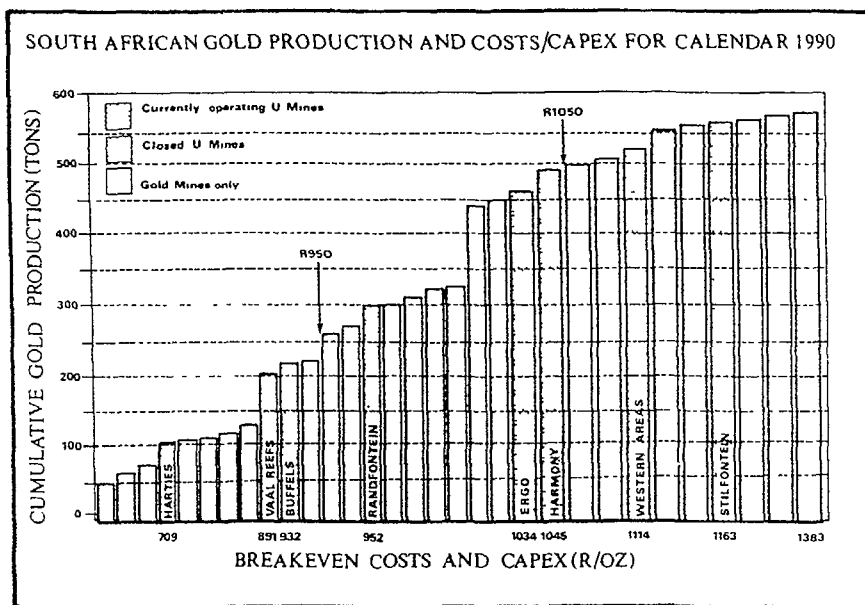


FIG. 4.

The four categories A to D highlighting the past and present uranium producers are shown in Table 2.

In Category A, Ergo is no longer a uranium producer. Operations have been rationalized resulting in the closure of the uranium plant, which was unprofitable. The future status of the uranium plant is uncertain, but it is likely that uranium production could be resumed at relatively short notice. Average annual production amounted to less than 5% of South Africa's total thus its influence on the market is not large.

In B Category, Harties is the only uranium producer. This is one of the lowest-cost gold producers in the world and its long-term future seems assured but could become a marginal producer within the R950/oz scenario. The mine operates a reverse leach process, and the uranium extraction process liberates more gold for recovery in the gold plant. Uranium is produced at a loss, (possibly in excess of \$130/kg U) but the extra profit generated by the enhanced gold recovery more than covers these losses, and uranium will continue to be produced.

Randfontein was a substantial participant in South Africa's uranium industry prior to the closure of its uranium plant. The operations were profitable, and the rapid conversion of the plant to gold extraction suggest that the main rationale behind its closure was that

TABLE 2. VIABILITY RATING OF PRESENT AND PAST PRODUCERS OF URANIUM IN THE WITWATERSRAND

<p>A. RATED MINES which would be profitable in year 3 at a gold price of R950/oz without making any drastic changes to current operations:</p> <p style="text-align: center;">ERGO</p>	<p>B. RATED MINES which would only remain profitable over the 3 year period if they take steps to improve yields or costs or to reduce capex:</p> <p style="text-align: center;">• HARTIES RANDFONTEIN STILFONTEIN</p>
<p>C. RATED MINES which will find it difficult to remain profitable in the 3 year term:</p> <p style="text-align: center;">• VAAL REEFS • WESTERN AREAS</p>	<p>D. RATED MINES which will have little hope of remaining profitable and will probably face closure within the 3 year term:</p> <p style="text-align: center;">• BUFFELS HARMONY</p>

• PRESENT PRODUCERS OF U

it could be operated more profitably for gold than uranium. The extension to the mine necessitated more gold plant capacity and the cheapest way of achieving this was by converting the uranium plant. Uranium production could be recommenced, but at substantial cost and time delay.

Stilfontein mine operated the Chemwes tailings reclamation plant for some years and produced substantial amounts of uranium. Operations came to an abrupt halt when their major client terminated their uranium contract. The plant was converted for gold extraction. The future of Stilfontein's gold operations is very short as its ore reserves are virtually depleted.

Category C has Vaal Reefs and Western Areas as producers. Vaal Reefs produces about 55% of South Africa's uranium and will continue to dominate in future. Production will however be reduced because one of the three uranium plants has been temporarily shut down and a second is operating at half capacity possibly settling down at 1 000 tons U/year. At present full production can be re-established at short notice. Construction of a new shaft system has commenced and the mine's long-term future is assured only if the gold price rises above the Graham-Parker & McDermott scenario.

Western Areas is a very high-cost mine, and has only recently regained profitability as a result of drastic re-organization, including the closure of one out of two shafts for gold production. The closed shaft is the uranium producer and limited operations are being maintained for uranium production, which is profitable. The future of this mine is uncertain, but is enhanced by its large share-holding in the adjacent South Deep project and the forward selling of its gold.

Finally in Category D, Buffels is the only uranium producer. It is a consistent gold producer, but the uranium operations are only marginally profitable, and often operate at a loss. Its gold operations are also only marginally profitable as a result of recent drastic rationalization of its operations. A serious problem here is a shortage of mineable gold reserves which cast doubt on the long-term survival of this mine.

At one time Harmony mine operated three uranium plants, but low grades and low uranium prices resulted in the closure of all three plants over a period of years. The plants are nominally moth-balled, but a large rise in the uranium price would be necessary before these plant would be re-opened. In addition, the mine is in serious difficulties at the current gold price, and unless the gold price rises significantly, its future seems very bleak.

With respect to both the Gillan and Graham-Parker/McDermott scenarios the closing of the mines could remove up to 500 tons gold from the world markets which could influence the gold price. It would therefore be unlikely that the mines would close but the most likely scenario would be cuts in production to achieve an equilibrium with the gold price.

GOLD IN TERMS OF INFLATION IN SOUTH AFRICA

Returns on investment are an indication of the state of health of the market. Since 1986 the Johannesburg Stock Exchange gold index fell for the first time since 1970 below the inflation rate which is reported to be between 12 - 16 % per annum. Graphically this is shown in Fig. 5. In August 1991 the gold index is almost at an all-time low (Steven-

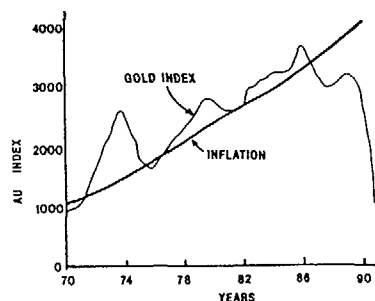


FIG. 5. Gold index versus inflation.

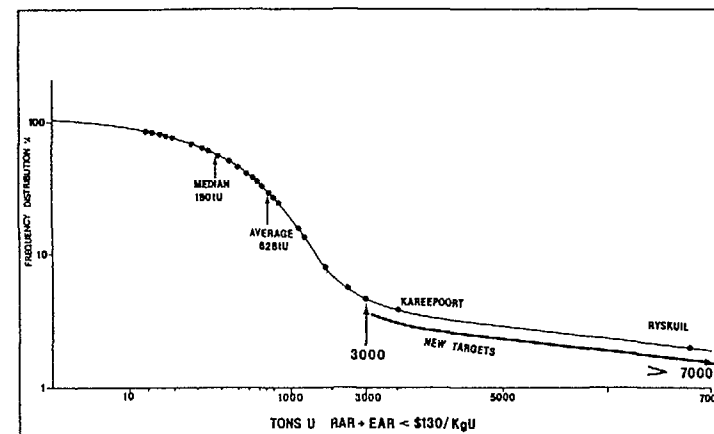


FIG. 6.

Jennings, 1991). This does not bode well for the South African gold mining industry which will back-lash significantly on the uranium production capability in the longer term.

RE-EXAMINATION OF ALTERNATIVE URANIUM RESOURCES

In the light of the above it may be prudent to re-examine the lower cost category Karoo uranium deposits which are the only known significant alternative uranium resources outside the Witwatersrand Basin. Evidence of uranium mineralization in the rocks of the Namaqualand Metamorphic Complex is coming to light but are of lesser significance.

Uranium in the Karoo

During the 1970's considerable effort was put into uranium exploration in the main Karoo basin by up to ten companies, both local and international. For example there were a total of 1 575 km of drilling completed for a total resource of about 100 000 tons U including the high cost categories. However, it is considered that a new initiative is required in the light of yet unanswered questions.

Fig. 6 gives the frequency distribution of the uranium deposits in terms of the grade. Most of the deposits are small having a median of 180 tons U and an average of 628 tons U. There are only two Kareepoort and Rystkuil that are in excess of 3 000 tons U. Therefore the strategy would be to locate uranium deposits that have tonnages greater than 3 000 tons U.

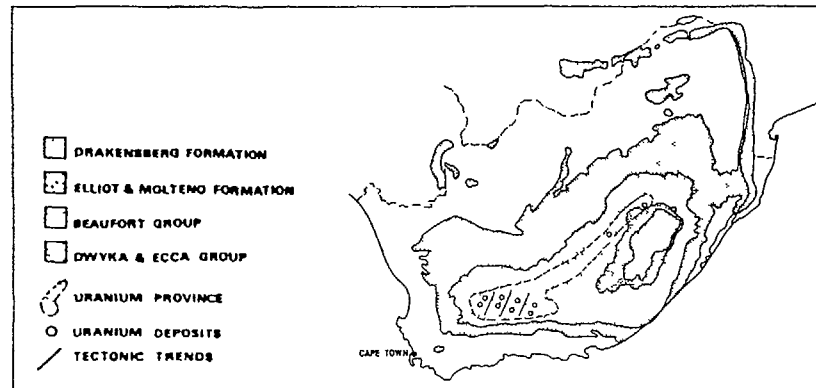


FIG 7 Uranium in the Karoo sequence

The questions that still require answers are:

- What is the influence of stratigraphy?
- What is the influence of tectonics?
- What is the role of vulcanism or thermal events?
- What are the alternative hypotheses or models for the formation of the U deposits?

Stratigraphy

Originally the largest Karoo uranium deposits were considered to be located only in the Poortjie sandstone unit of the Beaufort Group. Subsequently, it was found that uranium deposits occur throughout the Karoo stratigraphy with the exception of the Dwyka tillite and the Drakensberg lava formations. On the basis of this, it would appear that tectonics and source areas probably had a significant role to play in their formation. Figure 7 shows the distribution of the Karoo basins in South Africa as well as the outline of the main uranium province with localities of some of the more important uranium deposits.

Tectonic Trends

Recent studies of satellite imagery, the analysis of regional national seismicity and local micro-seismic events, topographic analysis using landforms, marine bathymetry and SEASAT imagery are being used to determine tectonic trends in Southern Africa. Figure 8 is a preliminary map showing some of the trends that have been located. It is important to note that these trends seem to be transcontinental in magnitude having both north easterly and north westerly trends. The distribution of kimberlites lies directly on a north westerly trend. Tectonic trends in the southern Karoo basin (Fig. 9) appear to coincide with the distribution of the uranium occurrences containing known resources.

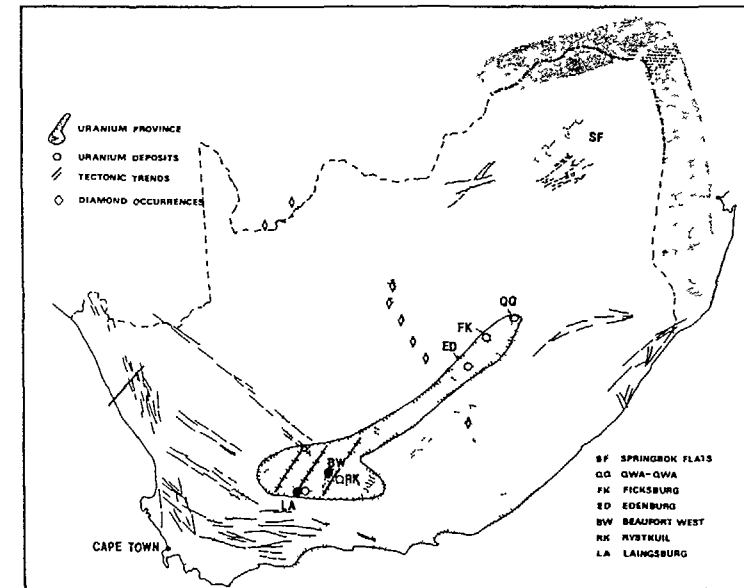


FIG 8 Tectonic trends in South Africa.

The Rystkuil trend parallels the Rystkuil palaeochannel containing the largest uranium deposit. Similar but less well defined are the Merweville and Fraserburg trends.

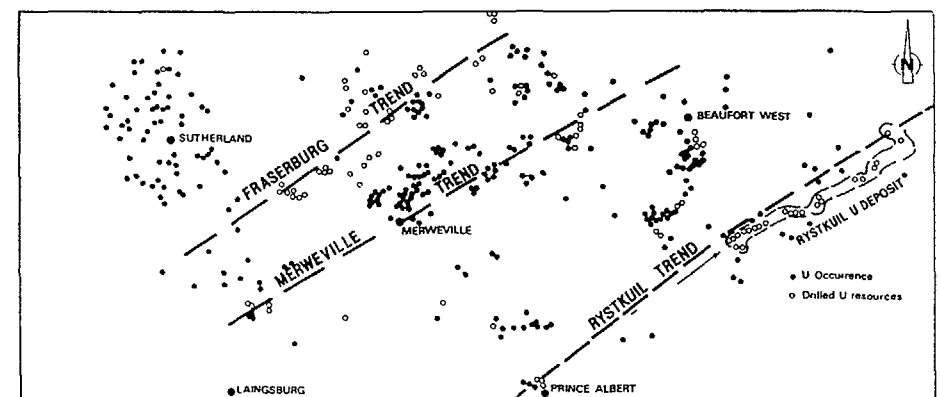


FIG 9 Tectonic trends in the southern Karoo uranium deposit

The hypothesis is as follows. Macro- and micro-seismic analyses coupled with the modelling of magnetic and gravity data suggests that tectonic movements along the trends commenced during the Proterozoic and have continued through into the younger overlying formations such as the Permo-Triassic Karoo sequence and even into the younger Pleistocene to Recent rocks. These trends are mega-shears with associated faulting. Analyses of river palaeochannels suggests that they are largely controlled by faulting which form weaker zones that are susceptible to erosion. Movement along these shears during the time of the deposition of the Karoo may have controlled the distribution of the fluvial river systems in which the uranium was deposited. The task is therefore to identify the trends and model the magnetic and gravity data to determine whether tilting of crustal blocks occurred and the influence on river systems.

Vulcanism or Thermal Events

The Karoo formations have been intruded by dolerite dyke swarms and in places, volcanic events. In addition the Karoo rocks were influenced by the Cape gold belt orogenic episode. Both these phenomena have left thermal imprints and their influences on the uranium mineralisation are not understood.

Uranium in the Namaqualand Metamorphic Complex

Multidisciplinary studies conducted in the Namaqualand Metamorphic Complex ($\pm 1,1$ Ga age) have indicated the existence of a new type of uranium/thorium/rare earth mineralisation.

Deposits of this type are largely restricted to granulite facies metamorphic mobile belts and in their type area are constituted by massive monazite veins with associated base metals sulphides.

The Steenkampskraal monazite deposit provides the best example of this type of mineralization, previously considered to be of hydrothermal nature. The monazite ore is associated and linked to the intrusion of quartz-anorthosite. Both rocks were derived from the fractional crystallization of a KREEP-type high Fe, Al basalt of enriched/chondritic mantle source.

Deposits of this type are also found in Mozambique and in southern Madagascar.

CONCLUSIONS

The trend in the production costs in South African gold mines is increasing ahead of the gold price thereby placing many mines in jeopardy of severe rationalisation and possible closure. This does not auger well for uranium production in the longer term. Currently there does not seem to be any likelihood of any increases in the uranium price which could offset the high gold costs.

Based on these factors the lower cost category Karoo uranium deposits are being examined in the light of new stratigraphic, tectonic and vulcanogenic information.

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UNCONFORMITY-RELATED URANIUM-GOLD DEPOSITS OF NORTHERN AUSTRALIA: RESOURCES, GENESIS AND EXPLORATION

A R WILDE

Exploration Department,
BHP Minerals,
Hawthorn, Victoria,
Australia

Abstract

Australia has a uranium resource of 370 kt U_3O_8 (313 kt U) contained in unconformity related deposits. Over 85% of this resource is located in the Alligator Rivers Province of the Northern Territory. In addition to uranium several of the deposits here contain economic gold (38t have so far been delineated) and two contain major amounts of platinum group metals.

Uranium gold deposits of the Alligator Rivers Provinces are in many ways similar to deposits of the Athabasca Basin in Saskatchewan (Canada). They are located at the unconformity between Mid Proterozoic clastic rocks and older metamorphic basement. Ore is associated with fault zones which in some cases transgress and offset the unconformity. Ore is enveloped by intense clay sized phyllosilicate alteration haloes. Differences (i.e. lower grade, no ore in sandstone, no Ni Co As ore) can largely be attributed to the effects of extensive erosion and weathering which has exposed the Alligator Rivers deposits, and overall lack of exploration in areas of thick sandstone cover.

All the known deposits in the Alligator Rivers Province were discovered by relatively simple airborne or ground radiometric surveys. Modern, geophysical methods as are routinely employed in Saskatchewan (such as airborne TEM), have not been tried in the Alligator Rivers Province. There has been a paucity of exploration in the Alligator Rivers Province in the past 15 years, as a result of creation of a national park over the most prospective area.

1 INTRODUCTION

1.1 SIGNIFICANCE OF UNCONFORMITY-RELATED DEPOSITS

Unconformity-related uranium deposits are the most important primary source of low-cost uranium in the western world. In 1989, production from such deposits was almost one-third of the western world total. Furthermore, these deposits contain a large share of the low cost resource and this share is increasing as more deposits are discovered. An additional economic benefit of this type of deposit is the possibility of substantial gold and even platinoid mineralization with uranium. Furthermore, the Coronation Hill deposit

demonstrates that economic unconformity-related gold-platinum group element (PGE) deposits exist with only minor uranium.

It is the aim of this paper to present a review of unconformity related mineralization in Australia with emphasis on the Alligator Rivers Province. This province contains 85% of unconformity-related uranium resources in Australia and is the only area in which an example of the deposit type is being exploited (Ranger Mine).

1.2 AUSTRALIAN UNCONFORMITY-RELATED DEPOSITS

Australia contains at least 14 significant examples of unconformity-related uranium mineralization (Tables 1 & 2, Fig 1). These are found in five areas or provinces. While the Rum Jungle Province saw minor production during the immediate post second world war period, only the Alligator Rivers Province has seen significant production (35,600t U_3O_8 to 1990) from the Nabarlek and Ranger Mines. Nabarlek is now exhausted although its mill remains on "care and maintenance". Further mining operations are prevented by the Australian Labour government's "Three Mine Policy".

Reference to Table 1 shows that the average grade of Australia's unconformity-related uranium deposits is actually quite low (ca 0.3% U_3O_8). This is a marked contrast to equivalent deposits in Saskatchewan Canada where grades are typically an order of magnitude higher. It is not clear whether this difference is real, and reflects intrinsic geological parameters, or whether it reflects a greater amount of exploration in Saskatchewan.

2 REGIONAL SETTING OF THE ALLIGATOR RIVERS DEPOSITS

2.1 McARTHUR BASIN COVER SEQUENCE

Uranium and gold deposits of the Alligator Rivers Province are located at the present erosional margin of the Mid Proterozoic McArthur Basin (Fig 2). This is a large, intracratonic sequence with basal coarse clastics and mafic flows (Kombolgie Formation) and upper carbonates and evaporites [1]. Ojakangas [2,3] interprets the Kombolgie Formation as a series of alluvial fans sourced to the west and north-west. A maximum thickness of 2 km is preserved, but over much of the region it is less than 600m thick [4].

Page et al [5] have provided a minimum age for deposition of the Kombolgie Formation, by dating interbedded basaltic rocks (Nunbargarni Volcanics). Five total rock samples gave an Rb-Sr isochron age of 1641 ± 219 myrs. This spread is probably due to varying degrees of chlorite/mica and hematite alteration. The favoured age of 1648 ± 28 myrs was based on a Rb-Sr isochron of mineral separates from a single sample, even though these separates included

**TABLE 1. URANIUM RESOURCES OF AUSTRALIA
CONTAINED IN UNCONFORMITY-RELATED DEPOSITS**

DEPOSIT NAME	PROVINCE	RESOURCE (t ₃ O ₈)	GRADE (% U ₃ O ₈)	MAIN OWNER
Jabiluka 2	Alligator Rivers	204,000	0.39	North Broken Hill
Ranger 1/3		71,300	0.26	North Broken Hill
Ranger 1/1		26,200	0.26	North Broken Hill
Koongarra 1		13,300	0.27	Denison Mines
Jabiluka 1		3,250	0.25	North Broken Hill
Koongarra 2		2,300	NA	Denison Mines
Nabarlek 2		2,100	NA	Pioneer International
Hades Flat		726	NA	North Broken Hill
Kintyre	Paterson	36,000	0.28	CRA
Jack	Westmoreland	1,750	0.24	Pioneer International
Garee		2,470	0.27	Pioneer International
Langi		130	0.19	Pioneer International
Junnagunna		3,480	0.29	Pioneer International
Redtree		2,500	0.35	Pioneer International
Sue		490	0.24	Pioneer International
Outcamp		400	0.19	Pioneer International
Redtree		1,630	0.12	MIM & CRA
Mt Fitch	Rum Jungle	1,400	0.04	AOG Minerals
Turee Creek	Turee Creek	250	0.05	Noranda Pacific
Angelo River		797	0.12	North Broken Hill
TOTAL RESOURCE		374,473		

**TABLE 2. UNCONFORMITY-RELATED GOLD RESOURCES IN THE
ALLIGATOR RIVERS PROVINCE. Jabiluka contains significant but sub-economic
Pd ore, while Coronation Hill has an indicated resource of
4.9 mT ore containing 0.19 g/t Pt and 0.65 g/t Pd (using a 0.5 g/t Au cut off) [16]**

	Total Ore (million tonnes)	Gold Content (Kg Au)	Grade (g/t)
Coronation Hill	4.9	21,000	4.3
Jabiluka #2	1.1	11,800	10.7
Koongarra #1	1.0	3,000	3.0
Ranger #1	2.0	2,500	1.3

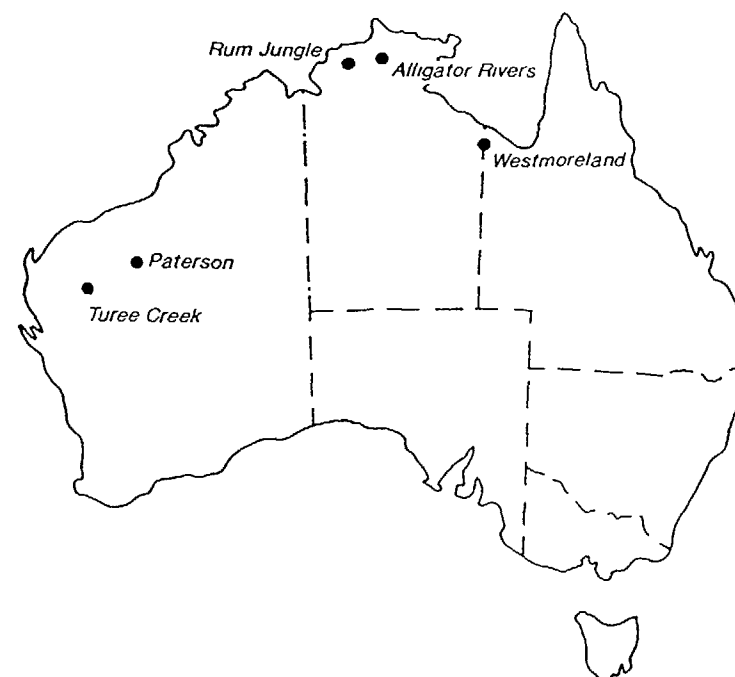


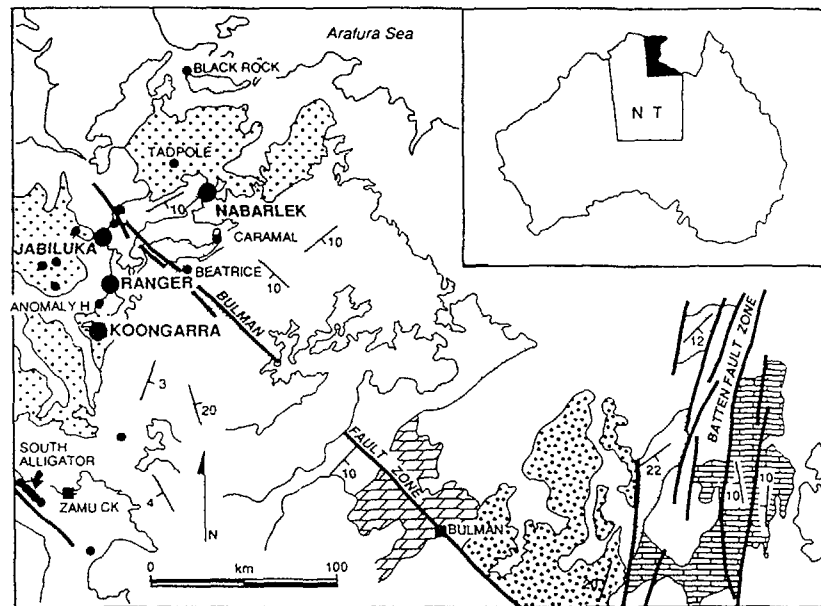
FIG 1 Location of unconformity-related uranium provinces in Australia

secondary chlorite/mica. Furthermore U-Pb ages of 1690 ± 25 myrs from the equivalent or younger Barney Creek Formation [6] suggest that the Kombolgie Formation is at least 50 myr older than the 1648 myr age

2.2 METAMORPHIC BASEMENT ("PINE CREEK GEOSYNCLINE")

The McArthur Basin overlies with angular unconformity, metamorphic basement rocks of the so-called "Pine Creek Geosyncline" [4]. Basement consists of fault-bounded Archaean granitoid inliers and Lower Proterozoic meta-sediments and mafic igneous rocks [7]. Metamorphic grade increases from greenschist facies in the west to granulite in the east. All the larger uranium occurrences are hosted by amphibolite facies rocks assigned to the Cahill Formation [8] or the possibly equivalent Myra Falls Metamorphics [4].

These stratigraphic units are dominated by semi-pelitic schist, but graphitic schist, amphibolite, dolomitic or magnesite marble and magnetite-green biotite-schist also occur.



LITHOLOGIES:

CAINOZOIC

Colluvial / alluvial sediments

MID PROTEROZOIC

Roper Group ferruginous arenites

Mt Rigg Group dolomites and evaporites

Katherine River Group alluvial sandstones

McArthur Group dolomites and evaporites

Parsons Range Group alluvial? sandstones and conglomerate

LOWER PROTEROZOIC / ARCHAEOAN

Metamorphic basement

● Uranium deposit >10,000tonnes

● Uranium deposit <10,000tonnes

■ Pb - Zn deposit

— Fault

— Strike and dip of bedding

FIG 2 Geology of the McArthur Basin, Northern Australia. Modified after [1]. The Coronation Hill deposit is located in the South Alligator Valley.

2.3 SUB-KOMBOLGIE FORMATION REGOLITH (?)

The McArthur Basin is probably partly equivalent to the Athabasca Basin of Saskatchewan (Canada). A clay-hematite rock is often encountered beneath the basal Athabasca Group, and has been interpreted as a fossil "regolith" [9]. An alternative explanation, and that preferred here, is that it represents post-Athabasca alteration [10].

Few drillholes penetrate the sub-Kombolgie unconformity in the Alligator Rivers Province, and there is little exposure of it at outcrop. At Jabiluka 2 some drillholes have intersected a clay-hematite zone beneath the Kombolgie [11]. Other drillholes show that the unconformity is enveloped by chlorite, white-mica and apatite alteration which involves massive desilicification of the basal sandstone and underlying metamorphic rocks [11].

2.4 PHANEROZOIC

The period following deposition of the McArthur Basin was one of remarkable quiescence. Substantial erosion of the Kombolgie Formation had occurred prior to the Tertiary when a lateritic profile developed. Remnants of this once widespread lateritic surface are preserved over much of the lowland Alligator Rivers area.

3. NATURE OF THE MINERALIZATION AND ALTERATION

3.1 RELATIONSHIP TO MID PROTEROZOIC UNCONFORMITY

As the classification suggests uranium-gold mineralization is spatially related to the Mid Proterozoic unconformity beneath the Kombolgie Formation (Fig. 3). Only at the Jabiluka can the relationship be demonstrated, in the other deposits it must be inferred.

3.2 STRUCTURAL CONTROL

Ore is localised by fault breccias and fractured host rocks adjacent to the faults. In only a few cases can the faults be shown to offset the unconformity (Koongarra 1 and 2, Fig. 4). The breccias are typically matrix-supported, with angular clasts of host rock showing evidence of transport beyond their source and re-orientation of pre-existing metamorphic fabrics [7].

This structural control has resulted in a bimodal ore grade distribution. At Jabiluka vein and breccia ore account for 66% total U metal at a grade of 1.15% U_3O_8 , but only 19% of the total ore. Conversely 81% of ore is made up of disseminated ore (ie mineralization in fractured wall-rocks) which comprises 33% of total U at 0.14% [13]. At Nabarlek vein ore averaged over 5% U_3O_8 ,

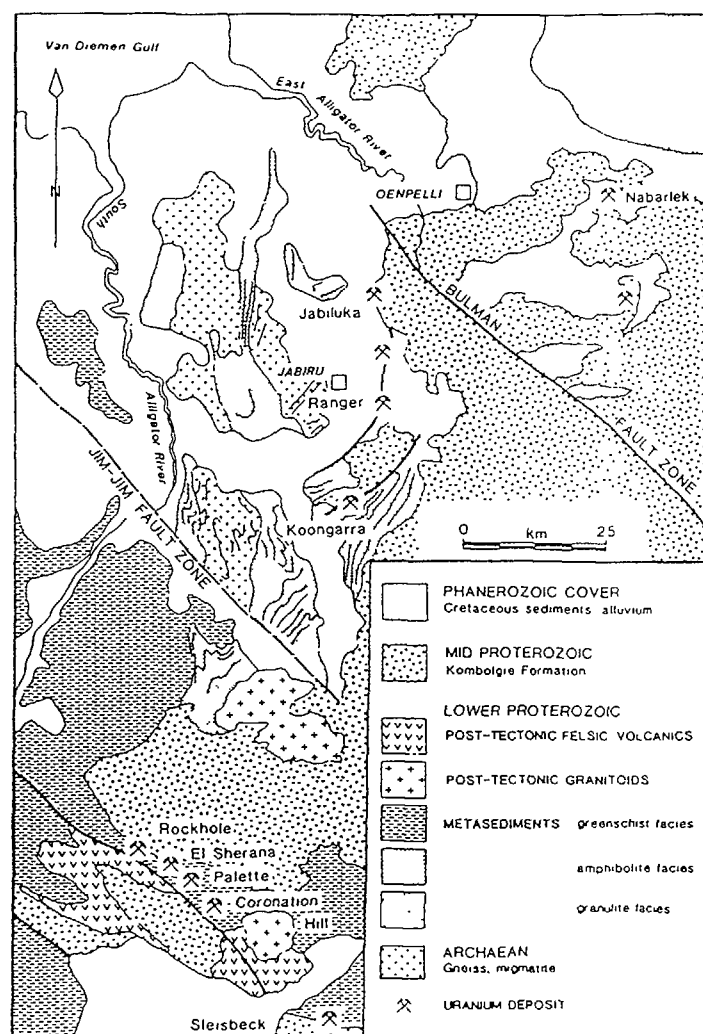


FIG. 3 : Setting of the major unconformity-related deposits of the Alligator Rivers Province. (Modified after [7, 8]).

while disseminated ore averaged less than 0.5%. The orebody as a whole averaged 1.84% [14].

3.3 HOST ROCKS

Host-rocks range from semi-pelitic schist, which include biotite-feldspar-quartz varieties and lesser garnetiferous varieties, amphibolite, silicified marble and rarely dolerite (Oenpelli Dolerite at Nabarlek). Virtually no ore is hosted by the Mid Proterozoic sandstone. In fact, absence of mineralization in the sandstone is often cited as a significant difference between deposits of the Alligator Rivers Province and in Saskatchewan. Furthermore, it has lead some authors to propose a pre-Kombolgie origin for the mineralization [15]. Since the sandstone cover has been eroded from most of the Australian samples, however, it is impossible to be certain that no mineralization existed within it! Furthermore, restrictions on exploration in areas of thick Kombolgie Formation cover (Arnhemland Aboriginal Reserve) has meant that the possibility of ore in the sandstone has never been adequately tested.

Gold and PGE mineralization at Coronation Hill (Fig. 3) differs somewhat from the predominantly uranium deposits to the north-east in that it is hosted by intrusive quartz feldspar porphyry and quartz diorite as well as coarse clastic rocks [16]. These clastic rocks are thought to be somewhat older than the Kombolgie Formation [17].

3.4 ORE MINERALOGY AND HYDROTHERMAL ALTERATION

Hydrothermal alteration in basement rocks associated with mineralization is extensive and can be recognized over 1 km from ore [11, 18, 19, 20] (Fig. 4). A distinct outer zone is manifested as pseudomorphous replacement of metamorphic phases in host schists and amphibolite by iron-rich chlorite and white mica, involving hydration and loss of Ca, Na and Sr due to incongruent dissolution of amphibolite facies anorthitic plagioclase [18, 20]. Quartz, pyrite, and/or hematite were deposited in pre-ore veins and breccias.

Primary ore, however, is associated with a more restricted pervasive alteration (inner zone) in the basement, and along and above the sub-Kombolgie unconformity involving development of white mica, magnesian, and/or aluminous chlorite (amesite), anatase, and locally, minor tourmaline, hematite, and apatite. Alteration associated with mineralization at Jabiluka clearly postdates deposition of the Kombolgie Formation [11, 19]. Most pre-mineralization phases were removed during alteration, including metamorphic and detrital quartz. Desilicification in altered schists around the orebody at Nabarlek resulted in up to 40 percent SiO_2 loss [19]. Magnesian chlorite and white mica-dominant alteration overprinted the outer zone alteration which is preserved as relics of iron-rich chlorite (after biotite) in a pervasive magnesian chlorite matrix [19]. Metamorphic muscovite-rich mica is another residual phase.

FIG. 4 : Cross-section showing inferred distribution of alteration at Koongarra from [11].

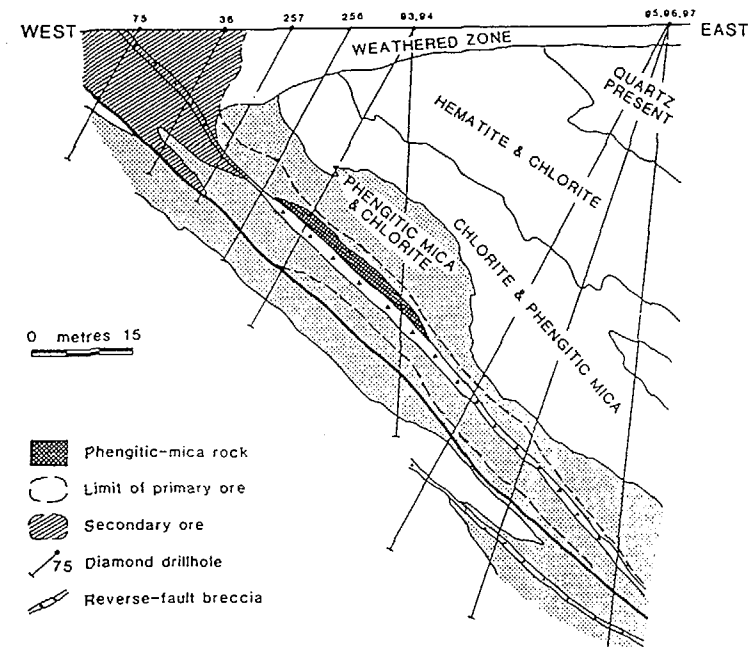
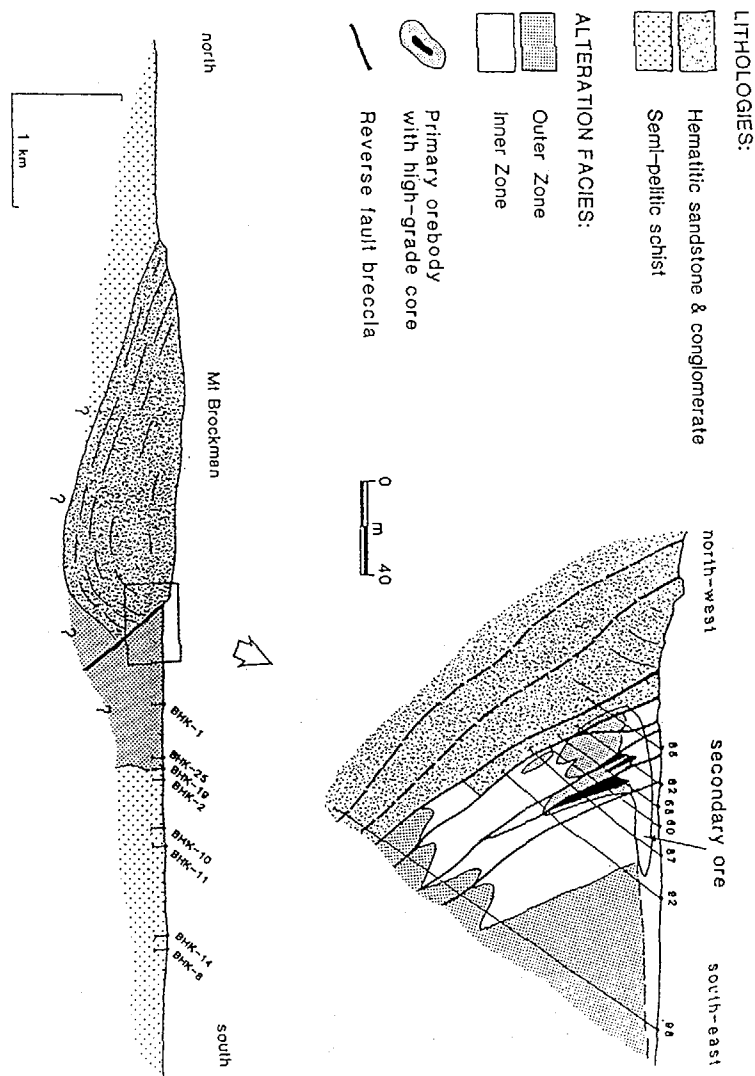


FIG. 5 : Cross-section showing distribution of alteration in the Nabarlek pit [20].

The proportion of chlorite to alteration white mica varies spatially with respect to mineralization. High-grade ore at Nabarlek is associated with monominerallic white mica rock, and similar rocks (described as greisen or griesenlike rocks) have been identified at Jabiluka [18] (Fig. 5). Koongarra's primary alteration is, in contrast, mica poor.

Owing to the fact that only vestiges of Kombolgie Formation remain in the vicinity of the main deposits there have been no studies aimed at establishing the nature and extent of interstitial clay mineralogy as has been done in the Athabasca Basin [10].

Primary uraninite is spatially and paragenetically associated with magnesian chlorite (amesite). Gold at Jabiluka is also closely associated with uranium (Fig. 6) forms inclusions or veins in uraninite, in association with lead and nickel tellurides. At Koongarra and Nabarlek, fine gold grains (10-100 μ m) are included in uraninite. It is inferred that gold deposition overlapped that of uraninite. The mineralogy and paragenesis of palladium at Jabiluka are

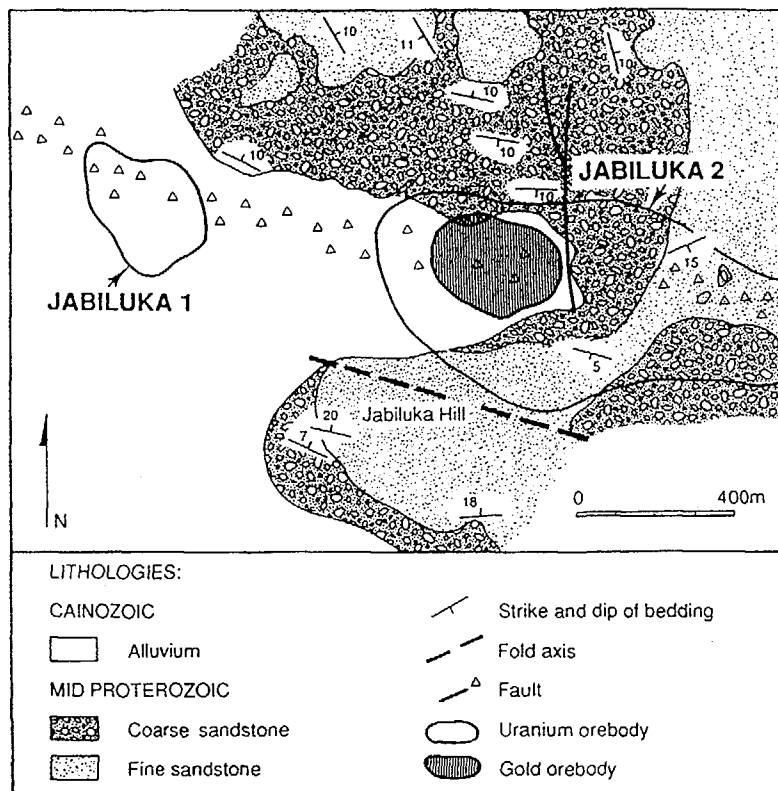


FIG. 6 : Distribution of uranium and gold mineralization at the Jabiluka deposit. The distribution of Pd mineralization is very similar to that of gold.

unknown, but the distribution of anomalous palladium is very similar to that of gold, implying that palladium deposition was contemporaneous with that of gold and uranium. Minor chalcopryrite and galena, but no gold, occur with dolomite and quartz in post-uraninite veinlets.

At the Coronation Hill Au-PGE-U deposit two types of gold mineralization have been recognized [16]:

- (i) A gold-PGE-selenide association is represented by gold in both pure and silver-bearing varieties, clausthalite (PbSe) and stibiopalladinite (Pd_5Sb_2), together with rarer precious metal phases. Sulphides are generally absent.

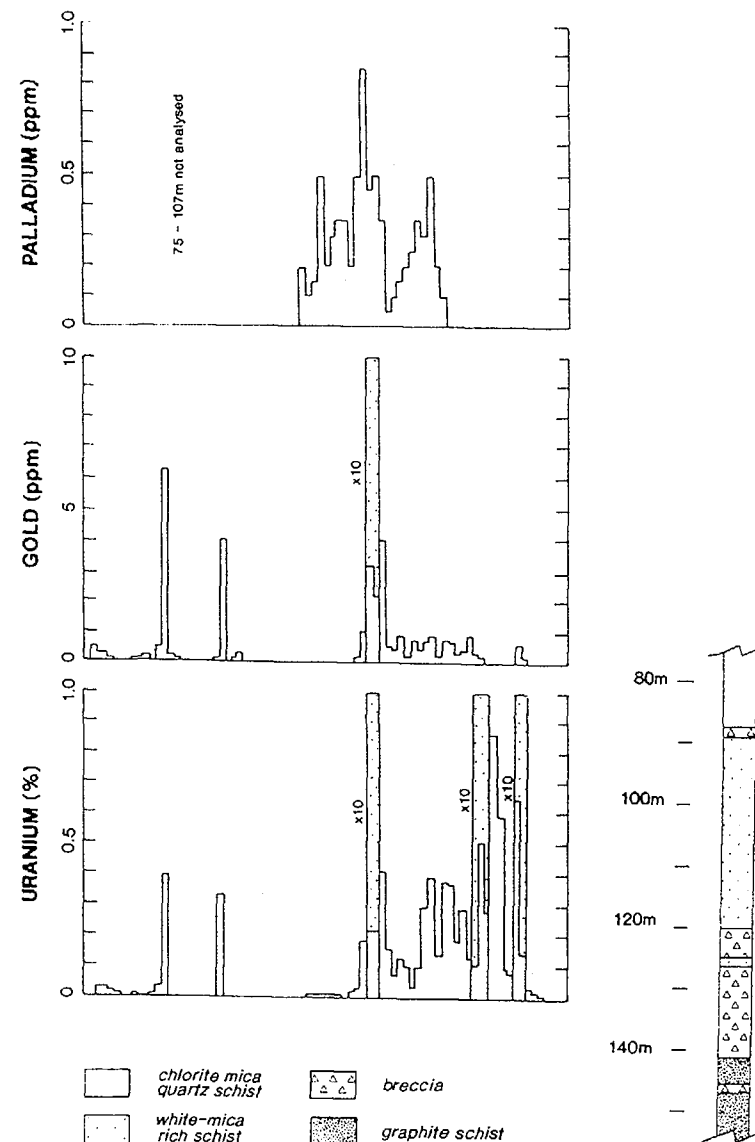


FIG. 7 : Section of drillhole W141V2, Jabiluka deposit, showing the distribution of U, Au and Pd and relationship to brecciated schist.

- (ii) A gold-PGE-selenide-sulphide association is represented by the mineral phases listed above but in association with replacive pyrite in altered igneous rocks.

Gold mineralization occurs along microfractures, microveinlets, quartz-carbonate hematite veinlets, and as disseminations within the alteration matrix in altered intrusive rocks. Trace to minor pyrite occurs as disseminations, fracture fillings and alteration haloes in all rock types but there is no relationship between pyrite and gold abundance. Traces of marcasite, pyrrhotite, sphalerite, chalcopyrite and galena occur.

Erosive stripping of the Kombolgie Formation from above several of the Alligator Rivers uranium deposits has resulted in the exposure of primary ore to weathering processes, and consequently development of yellow and green secondary minerals over the exposed orebodies. This event seems to have resulted in only minor redistribution of uranium [26]. It was important however, as exposure of the primary ore has facilitated the discovery of the orebodies by radiometric techniques.

3.5 AGE OF PRIMARY MINERALIZATION

Perhaps the most reliable dating of the primary mineralization has been carried out using the Sm-Nd method [27]. This method applied to a sample suite carefully selected to minimise the influence of later alteration suggests ore formation at Jabiluka, Nabarlek and Koongarra at close to 1640 myrs, somewhat after deposition of the Kombolgie Formation. Both Sm-Nd and U-Pb dating suggests that the Ranger deposit may be 100 myrs older [27, 28]. In view of uncertainty regarding the age of the Kombolgie Formation, this does not necessarily imply a pre-Kombolgie origin for Ranger [11].

4. GENESIS OF THE DEPOSITS

Some aspects of the formation of these deposits remain speculative, however, some important constraints on genesis can be established. Clearly permeability for ore-forming fluids was provided by faults [7]. There is limited evidence that these faults were active during ore formation [7]. Reverse movement can be demonstrated in some cases, (e.g. Koongarra), but displacement in others (e.g. Ranger, Nabarlek and Jabiluka) appears to be negligible. It has been speculated [11] that the key to the development of this permeability after deposition of the Kombolgie Formation is reactivation of older structures during strike-slip faulting of which there are several major examples in the region.

Several lines of evidence point to the ore-forming solutions being hot (>200°C) oxidising acidic chloride-rich brines [23, 25]. Fission-track dating of zircon from the deposits suggests that temperatures in the Lower Proterozoic host rocks

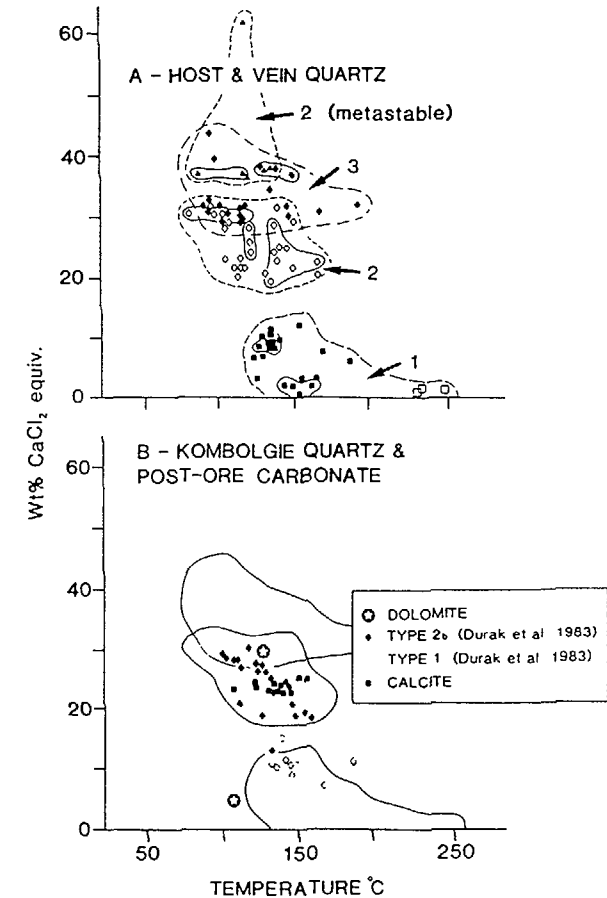


FIG. 8 : Fluid inclusion data from the Nabarlek, Koongarra and Jabiluka deposits [23]. (1) Pre-ore inclusions. (2) Inclusions related to alteration and ore-formation. The "metastable" group refers to inclusions which nucleated a solid in freezing which persisted to as much as 100°C on heating [11]. (3) Inclusions in post-ore dolomite and in altered host rocks.

exceeded 175°C (the annealing temperature given a 10⁹-Ma cooling history) prior to 1,420 Ma. Preservation of peak metamorphic ages in residual metamorphic muscovite suggests that 300°C is a maximum temperature during mineralization and alteration [5]. Secondary fluid inclusions in host-rock quartz and in early (preore) quartz veins are thought to sample fluid related to mineralization and alteration [23]. This is supported by the presence of

daughter minerals which also occur in altered host rocks and the absence of such inclusions remote from mineralization. Vapor disappearance temperatures typically fall in the range $150^{\circ} \pm 50^{\circ}\text{C}$. This range is identical to that observed in inclusions in silicified, chloritized, and boron-metasomatized sandstone of the Kombolgie Formation above Jabiluka [24].

Several features of the alteration assemblages point to the ingress of a relatively oxidized fluid. Hematite and anatase are common in the inner alteration zones at the deposits, and at Nabarlek hematite forms a shell above the mineralization [20]. Anhydrite has been found associated with magnesian chlorite at Jabiluka [18] and anhydrite occurs in fluid inclusions from Nabarlek [11]. The magnesian composition of the chlorites spatially associated with uraninite is thought to indicate that the coexisting fluid had a high oxidation state [25] since iron would be present in the ferric state, which is not readily accommodated in the chlorite structure.

These chemical characteristics favour an origin within the McArthur Basin sequence, a derivation also suggested by Nd isotope data [27]. Absence of any significant intrusive suggests that magmatic fluid could not be involved [11, 21]. Metamorphic rocks were dehydrated prior to mineralization and so metamorphic devolatilization is also an unlikely source of fluid [21].

Massive removal of SiO_2 during ore formation suggests that large volumes of fluid were involved in ore formation, and that the fluid was most likely undergoing heating and pressure increase [20]. Given evidence for high fluid to rock ratio specific uranium, gold and PGE enriched source rock or rocks are thought to be less important than a mechanism of concentrating large volumes of fluid within the ore zone. One can only speculate on the nature of this process.

Graphite is implicated in the formation of ore at Jabiluka and Koongarra, either by direct reduction or by an intermediate CH_4 -rich phase [21]. Paucity of graphite at Nabarlek and Ranger suggests another mechanism was important. It has been suggested that ferrous iron in amphibolites (as at Nabarlek) could have acted as a reductant [20, 25].

In summary, ore formation is thought to be a consequence of the development of oxidising brines in sediments of the McArthur Basin. Fault zones in the basement rocks facilitated access for the fluids to basement rocks immediately beneath the unconformity. Thus the fluids became reduced where appropriate graphitic or iron-rich lithologies were encountered. In other words, the unconformity represents a boundary between rocks of contrasting redox state and hydrologic regimes.

5 EXPLORATION

Exploration activity in the Alligator Rivers Province was most intense at the end of the sixties and early seventies and culminated in the discovery of the deposits listed in Table 1. The establishment of the Kakadu National Park and its subsequent world heritage listing has prevented exploration in a considerable portion of the prospective terrain. The remainder of the region lies within the Arnhemland Aboriginal reserve. Aboriginal groups have vetoed exploration in many parts of the reserve.

The principal exploration tool used in the discovery of the Alligator Rivers deposits was radiometrics. Ranger, Koongarra and Nabarlek were all found by airborne surveys [29, 30] while Jabiluka was discovered by a ground survey [31]. Other geophysical and geochemical techniques were not of great importance in regional exploration. Little attention was paid during this early exploration phase to identifying mineralization beneath substantial Kombolgie Formation cover. This was partly due to the fact that early genetic models regarded the presence of the Kombolgie Formation as incidental. Secondly a considerable expanse of potential host rocks was exposed, or only thinly covered by Quaternary and Tertiary sediments, and presented a far more attractive target than mineralization buried beneath several hundred metres of sandstone.

None of the sophisticated geophysical techniques now used routinely in Saskatchewan (e.g. TEM) have been used extensively in the Alligator Rivers Province. Indeed there are no published data on whether the host-rocks at each deposit are anomalously conductive, although the graphitic rocks at Jabiluka and Koongarra are likely to be so.

In conclusion, the Alligator Rivers Province remains highly prospective for unconformity-related U-Au-PGE mineralization, having seen only minor exploration activity in the past 15 years.

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URANIUM POTENTIAL OF THE YOUNGER GRANITES OF EGYPT

H.A.M. HUSSEIN, T.A. SAYYAH
Nuclear Materials Authority,
Cairo, Egypt

Abstract

The post-tectonic Pan-African younger granites of Egypt are characterized by abnormal radioactivity. Several plutons of these granites in the Eastern Desert, host a variety of rare metal mineralizations including uranium. Two younger granite plutons, namely El Missikat and El Erediya, in the central part of the Eastern Desert, host siliceous vein-type uranium mineralization, which is structurally controlled by faults and their feather joints associated with NE and NNE trending shear zones. The Gattar granite pluton, at the northern part of the Eastern Desert, hosts vein-type uranium mineralization associated with molybdenite. At the southern part of the Eastern Desert, Um Ara granite pluton hosts uranium as disseminated unconformity contact type.

Pitchblende is the primary mineral, while secondary uranium minerals are mainly: uranophane, β -uranophane, soddyite and renardite. Small amounts of pyrite, chalcopyrite, galena, sphalerite, molybdenite and violet fluorite are present in association with the uranium mineralization in the mineralized zones.

Present data suggest an origin by hydrothermal fluids derived from the granitic magma. Redistribution by circulating meteoric water may have taken place as evidenced by widespread alteration, particularly silicification. Exploratory mining resulted in clarifying the possibility of the extension of the uranium mineralization zone laterally and in depth. The estimation of the uranium potentiality of the four younger granite plutons is 14000 tons uranium as speculative resources.

1. INTRODUCTION:

The granitic rocks in Egypt are classified into two main groups: older syn to late tectonic granites referred to broadly as older granites, and post tectonic granites referred to as younger granites (El Ramly and Akaad[1], Akaad and El Ramly[2], El Ramly [3], Sabet[4], El Gaby[5], Akaad and Noweir[6]. Hussein, Ali and El Ramly[7] added a third group of alkaline granites which was previously identified with the younger granites.

The younger granitoids are widely distributed in the Egyptian shield and they constitute approximately 30% of its plutonic assemblage. Their relative abundance to the older granites increases from 1 to 4 in the south of the Eastern Desert to approximately 1 to 1 in the north (Stern [8]), and 12 to 1 in Sinai (Bentor [9]).

The younger granitoids in the Eastern Desert are represented by: red and pink granites, white granites and some granodiorites and hornblende granites.

Akaad and Noweir[6] classified the younger granites into three phases according to the mode and relative time of emplacement:

Phase I : granites of calc-alkaline, hornblende biotite granodioritic composition.

Phase II : granites of alkaline character and adamellitic composition.

Phase III: plutons of leucogranite with thick chilled margins of microgranite.

Greenberg [10] classified the Egyptian younger granites into three petrologic geochemical groups:

Group I : Granites that have been extensively albitized and silicified, resulting in the modification of original hypersolvous quartz, feldspar fabrics. They are very siliceous (up to 80% SiO₂), and relatively rich in Rb, Y, Nb and U, and like group II they are depleted in MgO, TiO₂ and Sr relative to "calc-alkaline granites".

Group II : Granites are less albitized and have higher mafic contents than group I and have transsolvous texture.

Group III: Granites have larger proportion of plagioclase and mafic minerals, and they have high MgO, total Fe, CaO, TiO₂, Al₂O₃ and Sr content.

Hussein, Ali and El Ramly[7] proposed a new classification of the Egyptian granites based on the ultimate source of the silicic-felsic magma, whether of crustal, subcrustal or mantle origin. They included the younger granites in Group II which have been formed by the partial melting of the lower crust, due to collision (suturing) at plate boundaries, probably with some additions from the mantle.

The post-tectonic younger granites represent the magmatic activity marking the end of the cratonization process of the Pan-African Orogeny. Several of these plutons are hosts of rare metal mineralization in the Eastern Desert of Egypt. All the uranium mineralizations are closely connected with some younger granite plutons.

The present paper aims at describing the common features of the uranium bearing younger granites and their uranium potential.

2. CHARACTERISTIC FEATURES OF THE YOUNGER GRANITES:

The term younger granitoids was introduced by El Ramly and Akaad[1] to include granites previously known as pink, red, Gattarian and some granodiorites and adamellites.

The main characteristic features of the younger granites are:- (El Ramly and Akaad[1], Akaad and El Ramly[2], El Ramly[3], Sabet[4], Sabet, Bessonenko and Bykov[11]. Akaad and Noweir[6], Greenberg[10], Hussein, Ali and El Ramly[7], and Stern, Gottfried and Hedge[12].

- 1- They form the most prominent topographic features in the Eastern Desert and Sinai with oval rounded and circular outlines.
- 2- They were intruded generally in old metamorphosed basic rocks which suffer thermal metamorphism.
- 3- They are dissected by some post granite dykes and veins of pegmatites, aplites, porphyries, jasperoid, quartz, as well as basaltic dykes.
- 4- The contacts of the younger granitic masses are mostly sharp, sometimes gradational.
- 5- Granitization effect of these granites on the metamorphosed volcanic rocks is obvious.
- 6- They are medium to coarse grained, pink to reddish pink caused by the prevalence of potash feldspars invariably impregnated by hematite dust, speckled with some milky white plagioclases and smoky quartz.
- 7- Photogeologically, the younger granites are characterized by lighter tone, angular to rectangular drainage pattern.
- 8- Within individual plutons, younger granitoids show uniformity in mineralogical and chemical composition, but between plutons, notable variations occur in texture, grain size and modal mineralogy.
- 9- Their chemical composition is characterized by 70-75% SiO₂, 4-5% K₂O and K₂O/Na₂O ~ 1.
- 10- They are affected by post magmatic deuteric and hydrothermal alterations associated in many plutons by rare metal mineralizations.

- 11- Their emplacement was most probably influenced by major tectonic lineaments and fractures.
- 12- They are more radioactive than their country rocks (5 to 15 times) due to their relative high uranium and thorium contents. Their common accessory minerals are: zircon, xenotime, uranothorite, allanite, sphene, apatite.
- 13- They belong to the Pan-African plutonism, and they are post-tectonic.

The younger granites can be considered as products of low-temperature eutectic partial-melting process that took place in depleted upper mantle or lower crustal material. The source of the granite melt was probably "cratonized" oceanic, continental margin or intra-oceanic arc and basic material (Greenberg [10]).

It is believed that not all the younger granitic masses are of the same age, but they were intruded over a lengthy span of time, and the majority of them are believed to be formed at the end of Late Precambrian.

Akaad and Noweir[6] stated that the younger granites are those that cut, thermally metamorphose or even laterally displace the sediments of the Hammamat group.

Hashad [13] compiled the radiometric dates of the younger granitoids. The compiled ages lie within 620 to 530 Ma range. During this 90 Ma time span, magmatic activity in the Eastern Desert and Sinai reached its peak. The final cratonization and silicification of the Egyptian shield was achieved in this interval (Hassan and Hashad [14]).

3. URANIUM MINERALIZATIONS IN YOUNGER GRANITES:

Aeroradiometric survey covering vast areas in the Eastern Desert discovered the most significant anomalies in various younger granite plutons. Ground follow-up of these anomalies indicated the presence of four plutons with uranium mineralizations. Three of these plutons namely El Erediya, El Missikat and Gattar are related to uranium vein type deposits, and the fourth, Um Ara pluton, belongs to the disseminated type (Fig.1).

3.1. El Missikat

From both surface and subsurface studies, it was found that silica filling and occupying the rejuvenation tension fractured shear zones, represents three generations namely: light coloured silica, black silica and jasperoid silica.

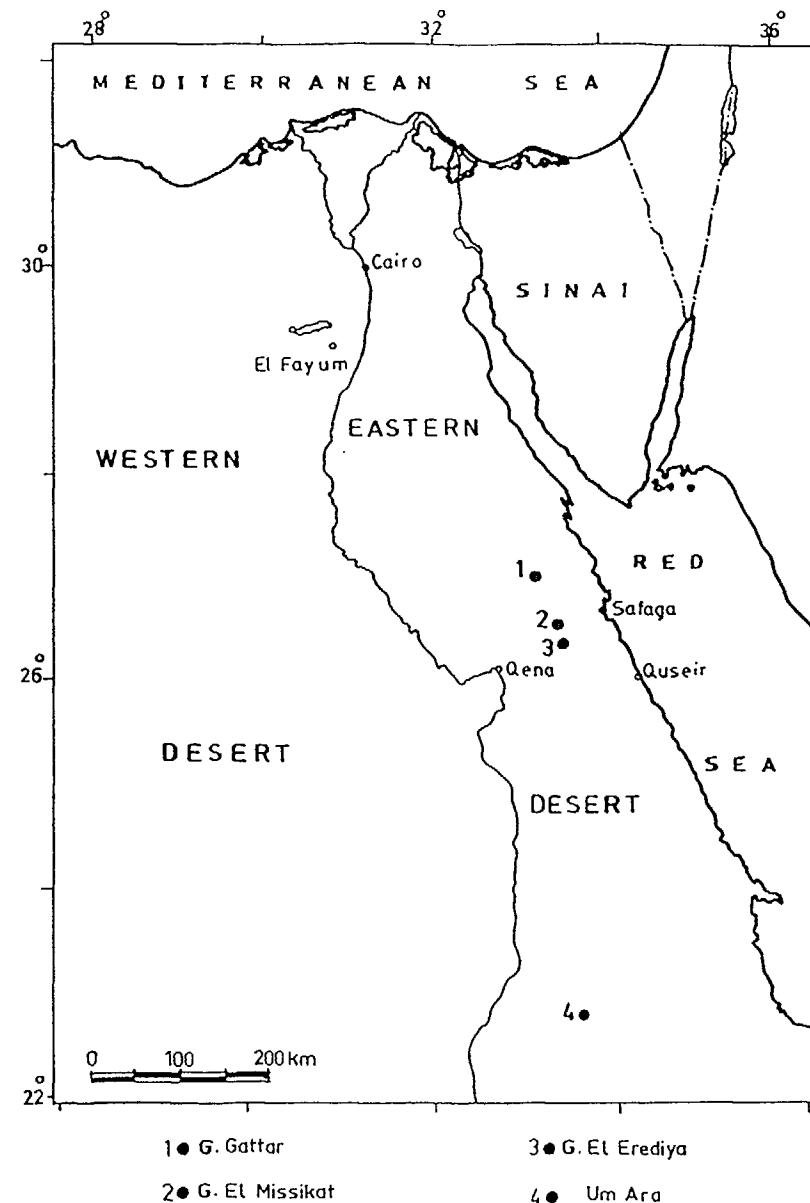


FIG. 1. Uranium occurrences in younger granites.

The light coloured silica is the oldest and it seems to represent more than one phase, with different colours and composition, these phases of silica are non-mineralized by uranium. Black silica is cryptocrystalline with smoky, brownish to true black colour, highly radioactive, including the main episode of uranium mineralization. The jasperoid silica is also cryptocrystalline with red colour, and it represents the youngest generation, including only some uranium mineralization. These three main types of silica are usually present in association with each other (Abou Deif [15]).

The uranium minerals are mainly: uranophane and soddyite with finely disseminated sooty pitchblende. They are accompanied by some sulphide and gangue minerals. Sulphides are mainly: pyrite, chalcopyrite, galena, sphalerite and molybdenite. The gangues are mainly iron and manganese oxides and fluorite.

The uranium mineralization of El Missikat is associated with silicification, sericitization and kaolinization. These three types of alteration mainly occur in a consistent zonal arrangement, where silicification occurs in the innermost zone followed successively by sericitization and then by kaolinization.

The uranium bearing structures of El Missikat were developed at the borders of the post-tectonic granitic pluton, characterized by uranium enrichment. These structures were subjected to successive sequence of rejuvenation. The introduction of silica took place in successive phases, and the sequence of events were:-

- Fracturing the granite along ENE directions.
- Invasion of these fractures by light coloured silica in more than one phase.
- Further opening of ENE fractures accompanied by brecciation of light coloured silica and introduction of the black silica carrying uranium minerals.
- In the last stage of rejuvenation, brecciation of earlier silica took place accompanied by the deposition of the jasperoid silica with some uranium minerals.
- Finally NW faulting leading to displacements and causing off set of the ENE shear zones.

The uranium mineralization of El Missikat is lithologically and structurally controlled, connected to black and jasperoid siliceous materials representing the two younger generations of silica filling and occupying the rejuvenated tension fracture shear zones. The mineralized shear zones are intensively oxidized from the surface till an undefined distance depth, and the spotty nature of mineralization seems to be a result of oxidation and leaching.

The uranium mineralization of El Missikat was introduced into the shear zones by the percolation of hydrothermal fluids,

probably mixed with meteoric water. These fluids seem to be originated from the younger granite magma at its late phase of evolution and were released in successive pulses.

3.2. El Erediya

The uranium mineralization in El Erediya is structurally controlled, it belongs to the vein type uranium deposits. This is supported by direct association of the uranium mineralization with faults and fractures filled by jasperoid veins in a typical shear zones. These faults and their branches strike in 4 directions: N, NNE, NE and ENE and dip 60°-85° SE. The shear zones are characterized by alteration features that have generally a hydrothermal nature (El Tahir [16]).

The jasperoid veins acted as traps capturing and protecting the uranium minerals. These uranium minerals, however, were protected either between the jasperoid veins and the hard silicified and mylonitized granite, or completely surrounded by jasperoid veins from all sides in silicified and mylonitized granite.

Uraninite is the primary mineral in subsurface of El Erediya, beside secondary uranium minerals: uranophane, β -uranophane, soddyite and renardite.

The jasperoid veins occur in more than one generation and the uranium mineralization is associated with the latest one. However, the association of mineralized faults with the brecciated jasperoid veins may suggest that the mineralized faults were reactivated more than once and the silica was introduced along them in many pulses. The first phase of reactivation was accompanied by the introduction of the first pulse of silica which filled these faults. The second phase of reactivation caused the brecciation of the silica and reopening these faults, and this was followed by the introduction of uranium. The third phase of reactivation, by which the silica and uranium were brecciated, occurred forming the brecciated lenses with uranium mineralization. The parts which were surrounded completely by silica were protected from the oxidation and leaching processes, while other parts suffered oxidation and leaching causing either complete leaching of the primary mineral leaving only small indications on the surface, or only oxidized to secondary uranium minerals. A massive pitchblende mass about (40x40x400 cm) jacketed by jasper, was found in a shear zone. The mineralizing fluids seem to be the agent responsible for carrying and depositing the uranium. The hydrothermal activity may take place in pulses associating the episodes of the reactivation of faults, and the uranium minerals were associated with the latest episode of reactivation, and consequently with the latest phase of hydrothermal activity.

The hydrothermal uranium bearing fluids may have been derived from the magma that formed the granite itself, most probably at a late stage of magmatic activity. The concept of the circulation of meteoric water in connection cells which could leach out the uranium in descending current and deposit it in the roof of the peripheral parts of the granite plutons could not be excluded.

Uranium may be released from the granite by dissolution of accessory uranium bearing minerals, and then deposited in shear zones. It may be directly released from the mantle into the granitic magma and concentrated in late-stage fluids, or released from the mantle into volatile phases and mix the hydrothermal fluids.

3.3. Gattar

The Gattar younger granite pluton is dissected by several fractures and faults belonging to various trends: NE-SW, NNE-SSW, NW-SE and N-S directions. Uranium mineralization occurs following two main tectonic trends, NNE-SSW and NW-SE, and the mineralization increases at the zones of intersection of these two directions. The uranium mineralization is structurally controlled, which is a characteristic feature of all uranium mineralization in younger granitic rocks.

The uranium minerals are mainly uranophane, carnotite, clarkite, umohite, kasolite, zippite and soddyite (Sayyah and Attawiya [17]). These minerals are accompanied by deep violet fluorite. The presence of fluorite accompanying the complex type of deposits of polymetallic type (Rich, Holland and Peterson [18]) is clearly seen in Gattar where the uranium mineralization is a typical vein type hydrothermal mineralization (Abdel Monem and Salman [19]).

The hydrothermal fluids are epithermal or mesothermal in character and introduced the uranium mineralization in the open spaces resulted from shearing and structural deformation. The wall rock alterations are hematitization, kaolinization, silicification and partial carbonitization, which are the major wall rock alteration types associated with vein type deposits as cited by Boyle [20].

3.4 Um Ara

The younger granites of Um Ara are hosts of uranium mineralization of the disseminated type. Hussein, Kamel and Mansour [21] stated that the distribution of radioactivity and the uranium mineralization of Um Ara are structurally controlled, following WNW-ESE and NW-SE trends (Hussein, Hassan, El Tahir and Abou Deif [22]).

The uranium minerals are uranophane and curite, with some specks of uraninite. These minerals are associated with deep violet fluorite. The wall rock alteration comprises silicification, microclinization, albitization, hematitization.

Table I shows a comparative study between the characteristics of the four younger granite plutons, El Missikat, El Erediya, Gattar and Um Ara.

4. URANIUM POTENTIALITY OF THE YOUNGER GRANITES

The high radiometric anomalies, and the uranium mineralization of the younger granites in many plutons from the far south to the far north in the Eastern Desert indicate the great potential of these granites as host for uranium deposits.

The younger granites are considered as fertile granites as they are hosts for uranium mineralizations and as source rocks for exogenetic uranium deposits.

According to Greenberg [10], Hussein, Hassan, El Tahir and Abou Deif [22], Stern, Gottfried, and Hedge [12] El Erediya and El Missikat granites originated from partial melting of subcrustal material of oceanic character with possible additions from the mantle.

Greenberg [10] showed that El-Erediya and El-Missikat plutons have been affected by widespread deuteric and hydrothermal alterations. This soaking is a common phenomenon in the younger granites of Egypt (Bugrov, Abou El Gadel and Soliman [23] Hussein [24]). The uranium bearing fluids could have been derived from the same magma that formed the granite itself, most probably at a late stage of the magmatic activity.

Two possible sources can be considered for uranium. It can be released from the granite itself by dissolution of accessory uranium bearing minerals and then redeposited in shear zones by percolating fluids (Attawiya [25]). It is possible that uranium may be released from the mantle not only into magmas but directly into volatile phases. A possible source is leaching of uranium from metasediments and acidic metavolcanics of the country and roof rocks.

The alteration of pluvial and arid periods in past geologic time must have caused strong and effective oxidation and leaching of uranium from the shear zones. The water table during the latest pluvial period in the Pleistocene must have been higher than at present, and was lowered after the onset of the present arid conditions. Any uranium deposit which might have been present between these two levels of water table would be only partly leached out due to the short time

Table 1 CHARACTERISTICS OF URANIUM OCCURRENCES IN YOUNGER GRANITES

Features	Area	El-Missikat	El-Erediya	Gattar	Um Ara
Country rock		Metamorphosed volcano-sedimentary rocks.	Basic metavolcanics	Basic metavolcanics	Metamorphosed volcano-sedimentary rocks.
Acid Magmatism		Orthoclase-granite	Orthoclase granite	Orthoclase-granite	Orthoclase granite
Primary minerals		Uraninite Pitchblende?	Uraninite Pitchblende?	Uraninite	Uraninite thorite
Secondary minerals		Uranophane	Uranophane, Sordyite, renardite	Uranophane, Carnotite, clarkite, umoholite, kasolite, zippeite, sordyite.	Uranophane curite.
Wall rock alteration		Silicification Brecciation hematitization fluorite	Silicification Brecciation hematitization fluorite	Silicification Argillic alteration hematitization fluorite	Silicification microclinalization Albitization hematitization fluorite.
Associated sulfides		molybdenite galena, chalcocopyrite, sphalerite, pyrite	molybdenite galena, chalcocopyrite, sphalerite, pyrite	molybdenite	
Structural elements		ENE trending shear zones	N to NE trending shear zone	NE trending fissure zone	NE trending fissure in the north, uranium dissemination due south.

they have been subjected to the process. The uranium ore bodies should be expected slightly above, but not far above, the present water table, which is at least several tens of meters deep (Hussein, Hassan, El Tahir and Abou Delf [22]).

Surface and subsurface data, in addition to structural control, origin of uranium mineralization and other evidences assure the great potential of the Egyptian younger granites as host for uranium deposits.

The uranium tonnage of the four investigated granitic plutons, as speculative resources, is as follows:-

El Erediya	3000	tonnes	U
El Missikat	3000	"	U
Gattar	4000	"	U
Um Ara	4000	"	U
Total	14000	Tonnes	U

5. CONCLUSIONS:

El Erediya, El Missikat, Gattar and Um Ara uranium occurrences occur in four younger granite plutons in the Eastern Desert of Egypt. These younger granites are produced by magmatic activity marking the end of the Pan African Orogeny and the beginning of an orogenic activity in the Egyptian basement. El Erediya, El Missikat and Gattar uranium occurrences are vein-type deposits, while Um Ara occurrence is a disseminated type. The deposition of uranium is structurally controlled along shear zones and fractures. The great potential of the younger granite as hosts for uranium deposits is estimated to reach 14000 tonnes of uranium metal as speculative resources.

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URANIUM OCCURRENCES IN SHABA, ZAIRE

A.P. FRANÇOIS

Brussels, Belgium

Abstract

Besides the famous Shinkolobwe Mine, many other uranium ore occurrences are known in Shaba. Nearly all of them are closely linked to the two Cu-Co stratabound ore-formations located at the lower part of the R.2 Group. This Group belongs to the Roan Supergroup, i.e. to the lower part of the Katanga System, dated late Proterozoic.

The R.2 Group occurs as huge "fragments" of dolomitic-psammitic rocks, included in a silty brecciated matrix. The whole forms a kind of megabreccia, possibly formed by creeping and dissolution of evaporitic thick beds, during important tectonic movements.

A stratiform poor proto-ore, probably syngenetic, is observed at the bottom of the lower Cu-Co ore-formation, linked to a marked redox contact, the first observed in the Katanga System. Locally, in some R.2 fragments, this proto-ore was remobilized in veinlets of uraninite. Superficial weathering could result in hexavalent uranium minerals, scattered in the vicinity of the orebody. Ores of economic grade (appr. 0.3% U3O8) and tonnage (500 to 3,000 tons U3O8) may result from these changes.

Shinkolobwe is an exception, considering the high U3O8 tonnage. It is conjectured that the ore came from uranium previously hosted in several deep seated R.2 fragments, leached by hot brines.

Shinkolobwe is practically mined out. Nevertheless, a significant amount of U3O8 (100 to 200 tons ?) has been and is still extracted yearly from other mines with the Cu-Co ores. It has been and is still lost, due to mining and metallurgical problems. It could perhaps be recovered in the future. Meanwhile, it should be interesting to carefully pile the extracted uranium ores and the uranium-bearing iron precipitate. Some new exploration attempts should be undertaken in the future, with the aim of finding new uranium deposits.

1. HISTORICAL INTRODUCTION

1.1. The first discoveries

In August 1945, two atomic bombs put an end to world war II. More than 120,000 beings were lost, but the cessation of hostilities saved millions of others. Both bombs were made with uranium from the Shinkolobwe Mine, located in Katanga, a

province of the former Belgian Congo (presently the Shaba Province of Zaire), and mined by UMHK (Union Minière du Haut-Katanga)

Shaba uranium has a long history. In 1913, H. Buttgenbach discovered the metal on yellow and orange specimens taken off in the Luiswishi deposit by an unknown prospector (1) (see location maps figures 1 and 2). On April 12, 1915, Major R. Sharp was settling the boundaries of a claim over the Shinkolobwe copper showing, known since 1902. As he was erecting a cairn over a small hill named Kasolo, he found, in holey siliceous rocks, a yellow mineral similar to the one observed in Luiswishi. He dug a trench and cleared a thick uraninite vein. This deposit was named Kasolo, and eventually Shinkolobwe.

1.2. The Shinkolobwe working

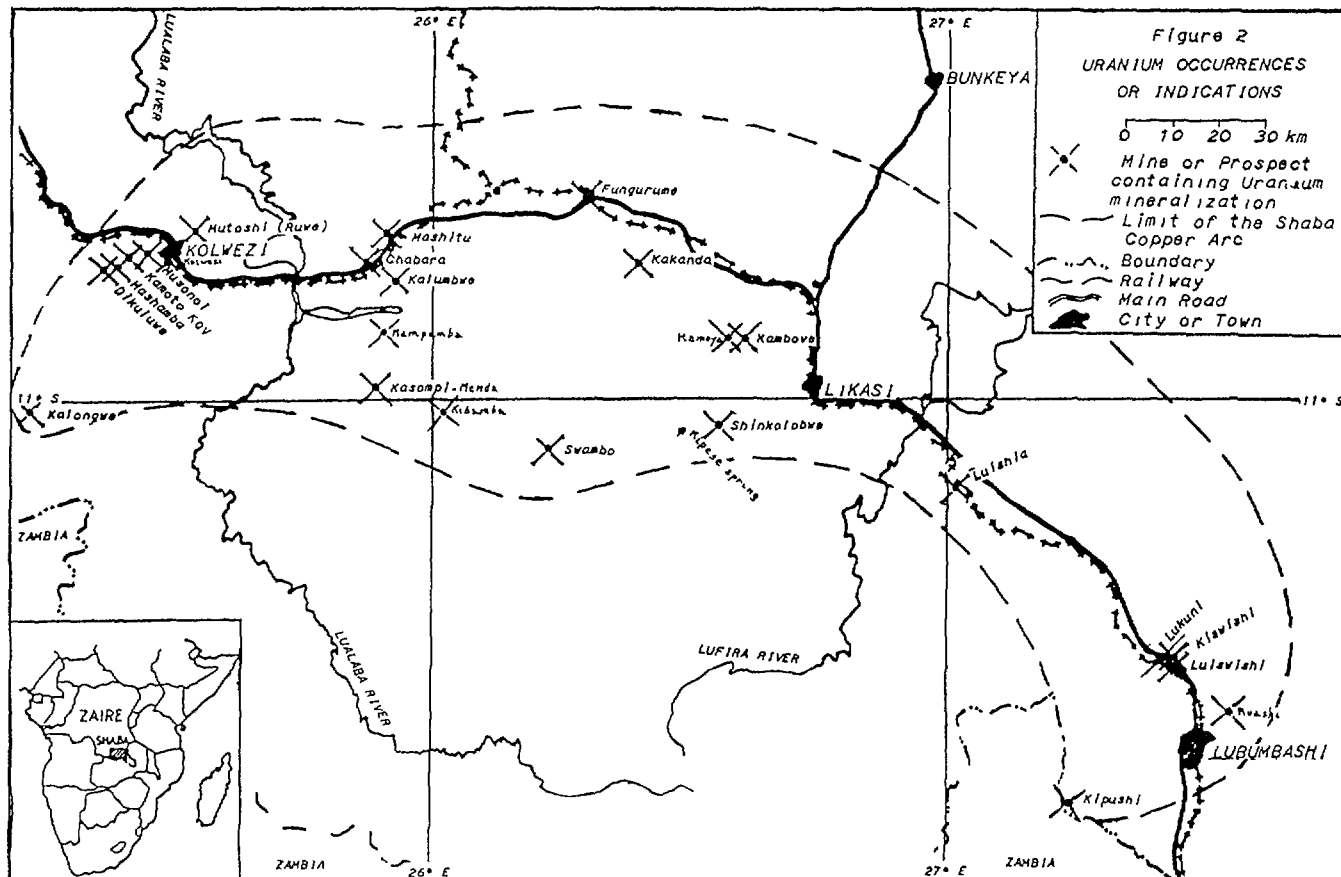
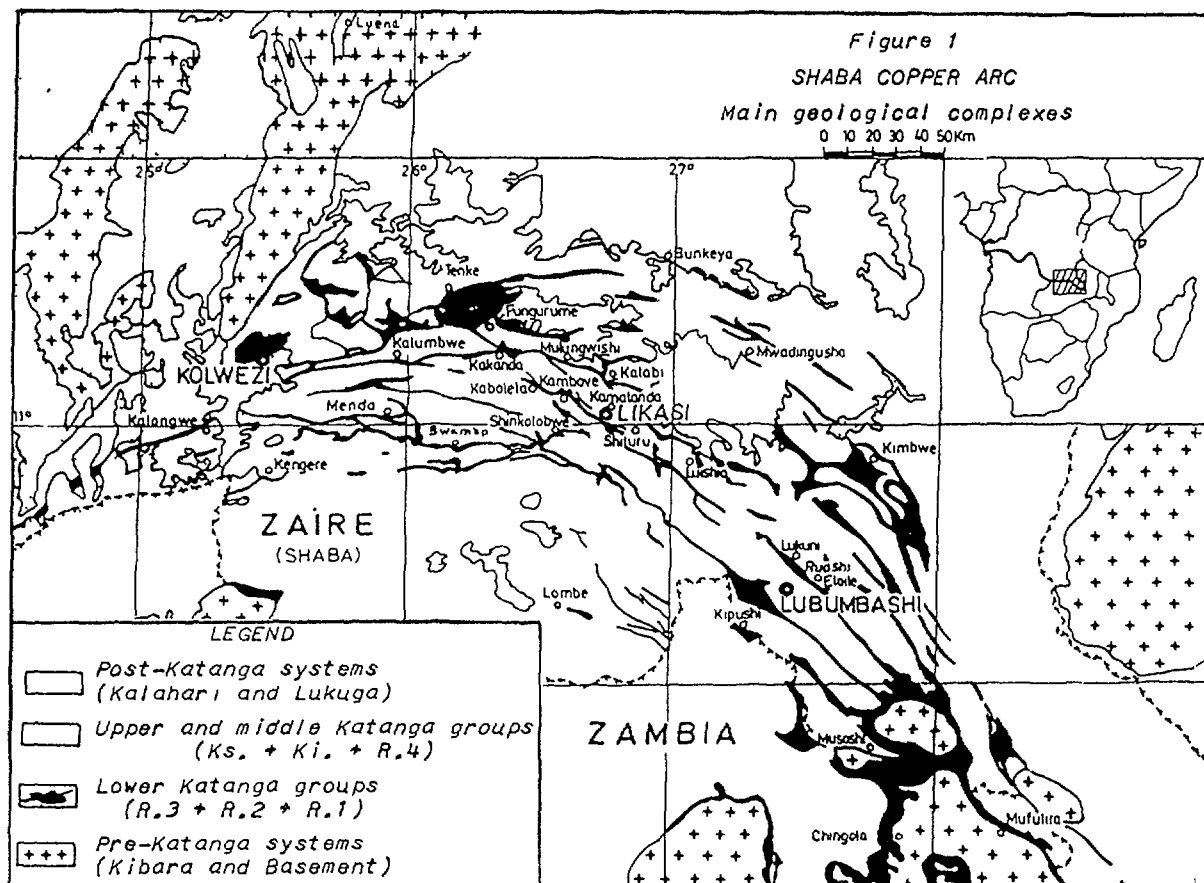
Working started slowly in 1921, as the aim was only to provide small amounts of radium to hospitals (at most 69 gr per year). The ore was mined out by open-pit. The level 45 (m below the Kasolo hill top) was reached in 1930. The richest ore was treated in the Oolen plant (Belgium).

Then the world crisis spread. Exploration drifts dug in 1932, below the water table, found high grades of precious metals: locally hundreds grammes/t of gold and platinoids (1). Whence an intensive underground exploration of these ores was planned. The level 79 was reached in 1936. Then the working stopped. Meanwhile, about 100,000 t of uranium-bearing rocks had been stockpiled.

Unexpectedly, in the height of war II, the USA needed uranium in order to start a nuclear programme. The tailings of Oolen, transferred in a safe place (New York) by the extraordinary clear-sighted UMHK manager E. Sengier, were at first used. From 1943 on, the tailings produced by gold workings were sent to the USA, whilst reopening of the Shinkolobwe mine was under way. Production resumed in 1945, first by open-pit down to the 79 level, thereafter underground. Uranium supply reached up to 2,500 metric tons of U3O8 per year till 1957. During 36 years Shinkolobwe remained the most important radium and uranium mine in the world.

Afterwards, the mine production decreased step by step, and the concentrator was fed more and more from low grade stocks piled up since 1922. The mine was closed on June 30, 1960, and the concentrator one year later.

The termination of mining happened casually just when the Belgian Congo became the Congo Republic. Actually, the main deposit was very poor below the 220 level, and practically barren on the 255 one. An extension had been found by drilling



close to the east, but the exploration by drifts stopped because of tremendous hydrogeological obstacles. Moreover, the uranium price had fallen 50%, after the discovery and mining of many other deposits in Canada, USA, South Africa and Australia. Therefore, exploration and dewatering of the east deposit halted, and the manpower was transferred to other UMHK mines.

1.3. Other Southern Shaba uranium occurrences

For a long time Shinkolobwe provided all the radioactive material needed in the world. Moreover, radium and uranium sales represented only a small part of the UMHK total turnover (at most 20%), and suffered the burden of high operating cost. Therefore, till 1955, no systematic exploration for uranium was sustained, excepted in the Shinkolobwe and Menda areas.

From 1955 up to 1958, some exploration was carried out: an airborne and carborn radiometry, some field radiometry on foot, drifting in Swambo, Menda and Kalongwe, testing of drillholes samples by scintillometry. Some uranium occurrences were found. However it seems that Shinkolobwe, with its tonnage of high grade ore, prevented additional exploration for uranium in Shaba. After 1960, this province has undergone many mishaps, that have cramped more prospection. Nevertheless, some other U showings were found, often by chance, in drillhole cores and in the mined copper deposits. Others ones were probably missed by exploration, or were not reached by mine workings.

1.4. Aims of this paper

Finally, you will see that numerous uranium occurrences are known. They are listed in the tables given hereafter. Some of them contain important amounts of uranium. In other respects, most of them, like Shinkolobwe, are closely linked to the beds hosting the stratabound copper deposits in the so-called Shaba Copper Arc. Where did this uranium come from? Could these occurrences produce important amounts of uranium in the past? Could they produce important amounts in the future? How to find other interesting U deposits in the Shaba Copper Arc? I will try to answer these questions hereafter.

2. REGIONAL GEOLOGIC ENVIRONMENT

2.1. Stratigraphy

2.1.1. Introduction

In addition to the recent alluvions, five main geological complexes are known in Southern Shaba:

- The Cenozoic Kalahari System (sand, sandstone).

- The Mesozoic and Paleozoic Karoo System (tillite, sandstone, siltstone).
- The Late-Proterozoic Katanga System (tillite, dolomitic sandstone and siltstone, dolostone and limestone).
- The Middle-Proterozoic Kibara System (micaschists, quartzite, granite).
- The Archean (?) Basement (micaschist, gneiss, granite).

As practically all the uranium deposits were found in the lower part of the Katanga System, I will deal only with this one.

2.1.2. Stratigraphy of the Katanga System

The Katanga System could be divided into 3 units. There are, from top to bottom (2), (see fig. 3) :

- The Upper Kundelungu Supergroup Ks.: around 3,000 m of marine detritic sediments, hardly without limestone or dolostone, lying on a thin paraconglomerate, called Petit Conglomérat.
- The Lower Kundelungu Supergroup K1.: 1,000 to 3,000 m of marine detritic sediments, partially replaced in the middle part by limestone or dolostone beds, lying on a thick or thin paraconglomerate (marine tillite?), called Grand Conglomérat.
- The Rban Supergroup R.: more than 1,500 m of lagunal units, alternately impure dolostones, dolomitic-psammitic rocks and chloritic siltstone.

2.1.3. Stratigraphy of the Roan Supergroup

Several gaps marked by silty breccias are observed in this unit, always on the same stratigraphic levels (fig. 4). Whence it is impossible to work out a complete and continuous sequence. These hiatuses could correspond to evaporite beds which have disappeared by flow and dissolution. I used them to subdivide the Roan into 4 groups. They are, from top to bottom:

- R.4 (Mwashya Group), including a detritic upper part and a dolomitic lower part, bounded downwards by an ubiquitous gap. 50 to 350 m.
- R.3 (Dipeta Group), an alternance of silty or dolomitic formations, bounded downwards by a gap. More than 1,000 m.
- R.2 (Mines Group), including 3 formations grey or dark coloured: an upper dolomitic one (R.2.3, or CMN), an intermediate detritic one (R.2.2, or SD) and a lower dolomitic one (R.2.1). This group is bounded downwards by an ubiquitous gap. 100 to 200 m.
- R.1 (RAT lilac Group): chloritic and dolomitic siltstone, specularite bearing, beige-pink and massive in the upper part, purplish red and banded in the lower part. An ubiquitous gap is observed at the bottom. More than 230 m.

As rocks older than the Katanga System do not crop out in the Shaba Copper Arc, the lower part of this unit is unknown.

TABLE 1a. KNOWN URANIUM OCCURRENCES IN SHABA

N.B. -- Showing = occurrence with a little U minerals. -- Small deposit = less than 100 t. U308. -- Deposit = 100 to 1,000 t. U308. -- Interesting deposit = 1,000 to 5,000 t. U308. -- Important deposit = more than 5,000 t. U308.

Discovery year	Occurrence name	Occurrence kind	Discovery method	Mineralized formation	Discoverer	Remarks
1913	Lulswishi	Small deposit	Observation in trenches	R.2 bottom	Buttgenbach	
1915	Shinkolobwe	Important deposit	Observation on outcrops	Lower R.2 Fault breccia	Sharp	With nickel
Towards 1925	Kambove Principal (1)	Showing	Observation in trenches	R.2 bottom	?	Reported by Schulling in 1946
Towards 1930	Ruashi	Showing	Observation in drill holes	R.2 bottom	?	
1931	Kalongwe	Deposit	Observation on outcrops	Fault breccia and R.2 bottom	Jamotte	
1934	Musoshi	Small deposit	Observation in drifts	Zambia Lower Roan	?	Confirmed by drilling in 1936
1936	Musonol Junction	Showing	Observation in drifts	R.2 bottom	?	Mentioned in a report wrote in 1968
1939	Kasompi East	Interesting deposit	Observation in trenches	R.2 bottom	?	With some nickel
1939	Kiswishi	Deposit	Observation in little pits	R.2,3	?	Explored by drilling in 1955
Towards 1945	Musonol Extension (1)	Showing	Observation in a drill hole	R.2 bottom	Hirt	

TABLE 1b. KNOWN URANIUM OCCURRENCES IN SHABA (cont.)

Discovery year	Occurrence name	Occurrence kind	Discovery method	Mineralized formation	Discoverer	Remarks
1946	Kambove West (1)	Showing	Observation in a drill hole ?	R.2 bottom	?	Mentioned in a report wrote in 1968
1946	Lulshia	Deposit	Observation in open pit	Fault breccia R.2 bottom	Gonze	
Towards 1954	Shinkolobwe East	Interesting deposit	Observation in drill holes	Fault breccia Lower R.2	Derricks	With nickel
1955	Kolwezi	Deposit ?	Observation in a drill hole	R.2 bottom	François	
1955	Mutushi (former Ruwe)	Small deposit	Observation in drill holes	R.2 bottom	?	
1955	Swambo	Deposit	Carborn exploration	Fault breccia and Lower R.2	Giroult	With some nickel
1955	Chabara	Anomaly without visible mineral	Airborn exploration	Lower R.2	Falrohlid	
1955	Kamoto Principal (2)	Interesting deposit	Observation in drill holes	R.2 bottom	François	
Towards 1955	Kasompi West	Anomaly without visible mineral	Ground radiometry	Lower R.2	?	
Towards 1955	Henda	Anomaly without visible mineral	Ground radiometry	Lower R.2	?	
Towards 1955	Kipese spring	Anomaly without visible mineral	Ground radiometry	R.3 ?	?	Radon is given off the spring
1956	Dikuluwe	Anomaly without visible mineral	Radiometry on drill hole cores	R.2 bottom	François	

TABLE 1c. KNOWN URANIUM OCCURRENCES IN SHABA (cont.)

Discovery year	Occurrence name	Occurrence kind	Discovery method	Mineralized formation	Discoverer	Remarks
1957	Ant. Virgule-FSSR (KOV)	Deposit	Observation in drill hole	R.2 bottom	François	
1957	Fungurume (1)	Anomaly without visible mineral	Radiometry on drill hole cores	R.2 bottom	François	
1958	Mashitu	Showing	Observation in trenches	R.2 bottom	Coda	
Towards 1958	Kalumbwe-Myunga	Anomaly without visible mineral	Ground radiometry	R.2 bottom	Bora	
1959	Musonol S.A. Extension	Interesting deposit	Observation in open pit	R.2 bottom	Houreau ?	
1959 ?	Kipushi	Showing (autunite)	Observation in open pit ?	Fault breccia	?	Reported by Schulling in 1968
Towards 1962	Kakanda South	Showing	Observation in a trench	R.2 bottom	Coda	
1965	Kampamba	Anomaly without visible mineral	Ground radiometry	Lower R.2	François	
1966	Kambove West (2)	Interesting deposit	Observation in underground working	R.2 bottom	Dessart	
1969	Mashamba	Small deposit ?	Observation in a drill hole	R.2 bottom	?	Some ore was found later in open pit
1970	Kaluku (Luena)	Deposit ?	Helicopterborn exploration	Karoo coal	Mortilla	Some ore was found later in the open pit
Towards 1970	Kibamba	Anomaly without visible mineral	Ground radiometry	Lower R.2	Lefebvre	

TABLE 1d. KNOWN URANIUM OCCURRENCES IN SHABA (cont.)

Discovery year	Occurrence name	Occurrence kind	Discovery method	Mineralized formation	Discoverer	Remarks
Towards 1970	Kamoya South	Anomaly without visible mineral	Ground radiometry	R.2 bottom	François	
1972	SE of Luena	Anomaly	Helicopterborn radiometry	Kibara	Hunting	No follow up
1976	Luena extension	Interesting deposit ?	Drilling	Karoo	Chevallier	
1977	Kamoto North	Deposit	Observation in open pit	R.2 bottom	?	
1977	Virgule	Deposit	Observation in open pit	R.2 bottom	?	

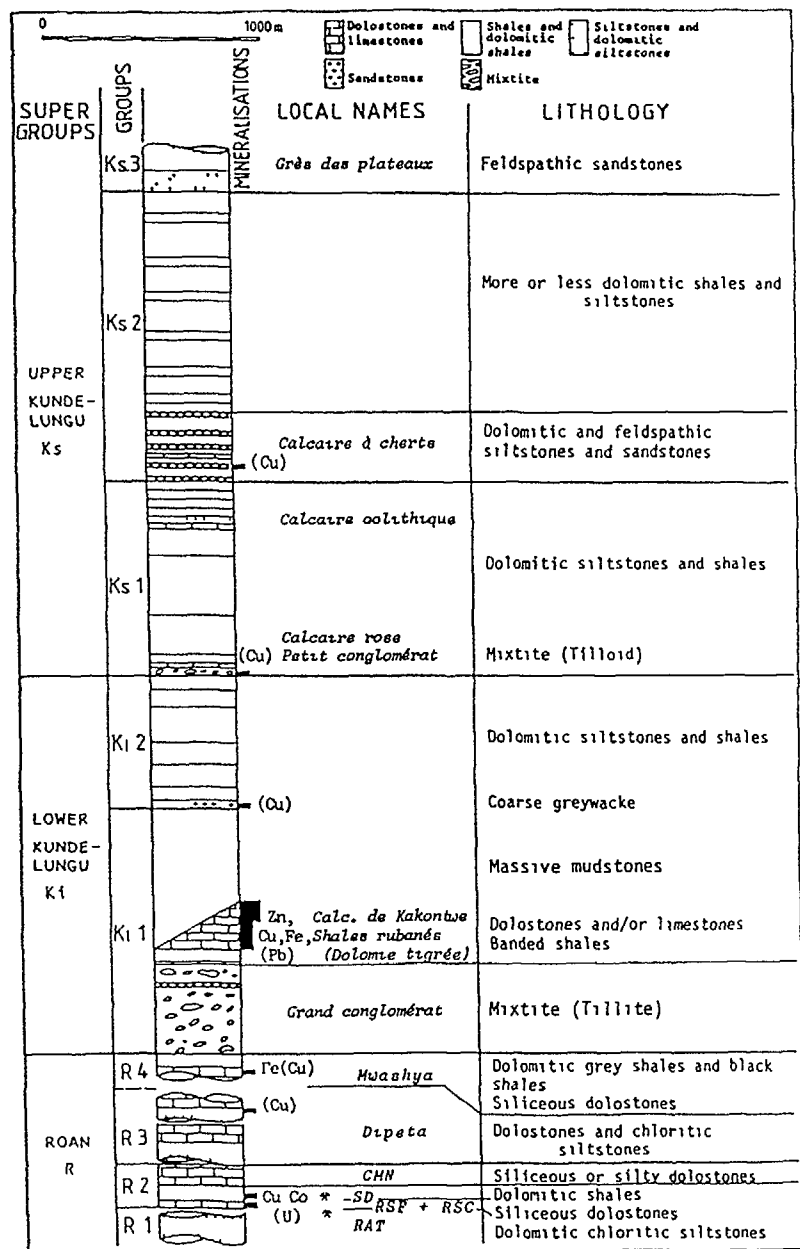


Figure 3 General stratigraphy of the Katanga system

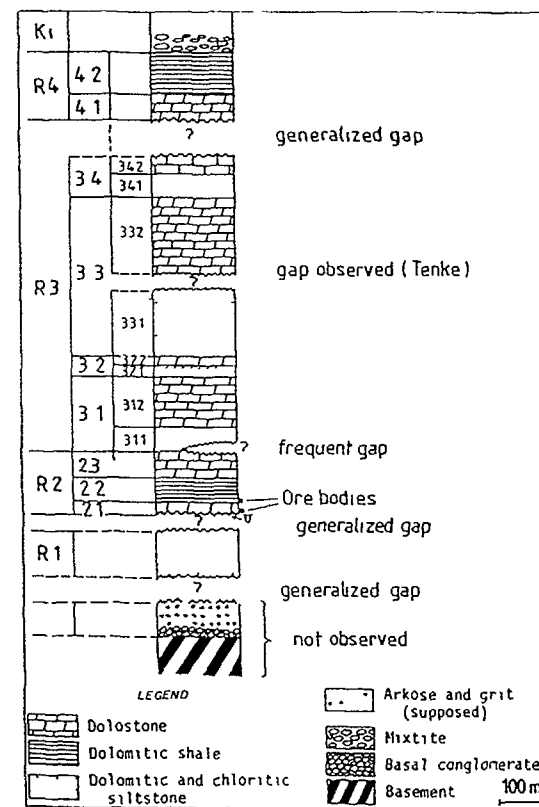


Figure 4 Stratigraphy of the Shaban Roan Supergroup

Like in the neighbouring Zambia, it consists probably of sandstones and grits, overlying a basal conglomerate.

Almost all the uranium-bearing rocks of the Katanga System, as well as the stratabound Cu-Co deposits, are located in the lower part of the Mines Group R.2, composed as follows (see figures 4 and 7):

- R.2.2.3 and R.2.2.2 Members or SDS (shale dolomitique supérieurs). 3 black banded shale horizons, separated by 2 grey-green dolomitic siltstones. Locally, dolostone or sandstone beds.

- R.2.2.1 Member or SDB (shale dolomitique de base): 5 to 10 m of grey dolomitic siltstone, locally capped by a dolostone bed. This unit, called also SD de base, constitutes the upper Cu-Co stratabound orebody

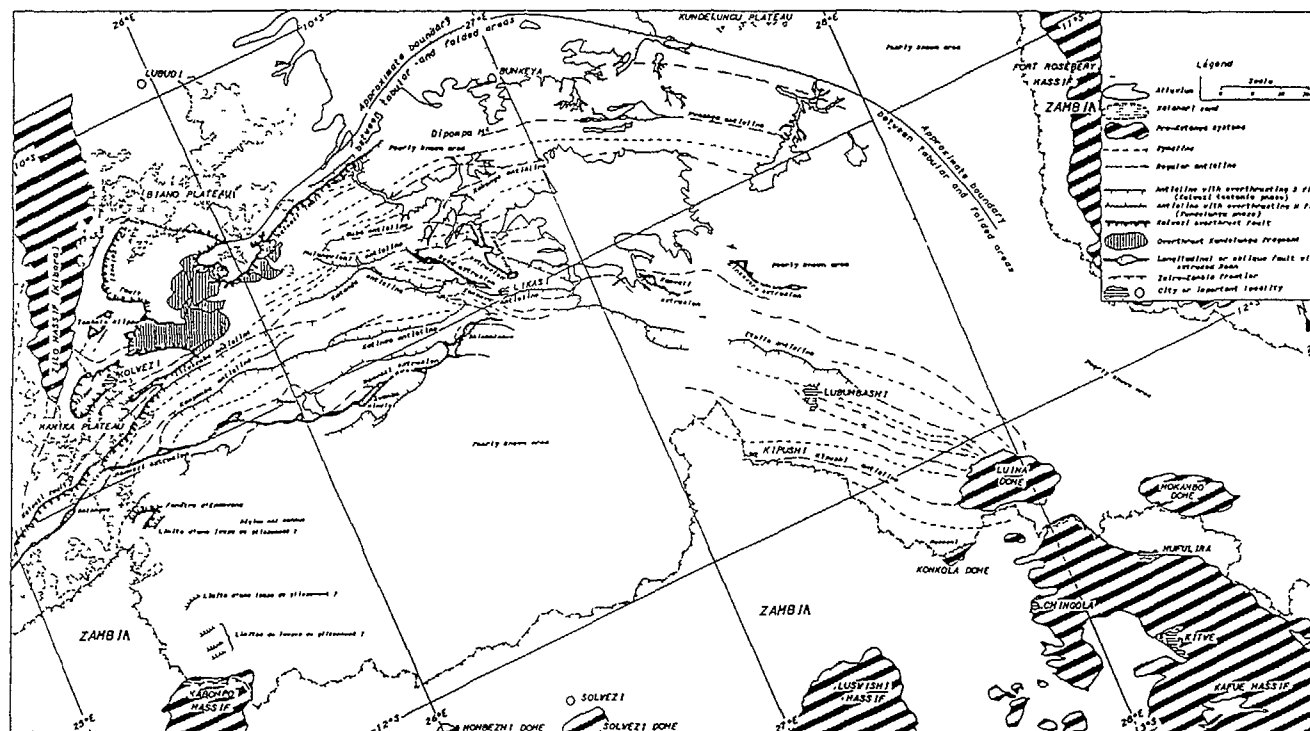


Figure 5. Main tectonic structures.

- R.2.1.3 Member or RSC (holey siliceous rock): clear massive siliceous dolostone, locally algal and lenticular. 0 to 20 m.
- R.2.1.2 Member: grey laminar siliceous dolostone, with silty layers, called also "RSF" and D.strat.". 7 to 10 m.
- R.2.1.1 Member: Grey massive chloritic and dolomitic siltstone, called also "RAF grises". 1 to 4 m. It constitutes with the R.2.1.2 the lower Cu-Co stratabound orebody.

2.2. Metamorphism in the Katanga System

Metamorphism is weak in the Shaba Copper Arc: primitive argillaceous minerals are transformed in sericite and chlorite (epizone grade). Biotite and amphibole zones occur to the south, covering also the Zambian Copperbelt. It is a regional metamorphism, possibly caused by an increasing sedimentary load.

2.3. Tectonic in the Katanga System

2.3.1. The tectonic structures

The Katanga System remains flat to the north of the Copper Arc. It is strongly folded along the Arc and south of it (2), (see fig. 5).

No early orogenic event occurred in the Copper Arc

During a first major tectonic phase (called "of Kolwezi"), the late proterozoic sediments seem to have slid northwards, producing folds overturned toward the north. This event occurs in the middle of the Upper Kundelungu period. The southern flanks of some anticlines are thrust over the northern flanks and overlap them. The horizontal component of

the displacement can reach 10 km. It reaches about 70 km for an anticline located in the western part of the Copper Arc, forming the Kolwezı nappe. This nappe settled over horizontal autochthon rocks, along the Kolwezı fault.

A third kind of structures are important faults which cut the Kundelungu cover. Along them, irregular masses of Roan are observed, dipping steeply between two walls of younger Kundelungu rocks, forming what I called extrusions. The most important one follows a sinuous course over a distance of 170 km, from west of Kalongwe to east of Shinkolobwe (fig. 5). It is the Monwezi extrusion. It cuts across folds at a low angle, with a dextral horizontal displacement of about 4 km. Therefore it seems to be subsequent to the last folding (Monwezi phase).

The results of these two (or three) tectonic phases are obvious in the cross section fig. 6, drawn along the Kambove-Shinkolobwe meridian.

Locally, anticline flanks are breached along steep abrasion surfaces and the resulting void is filled by Roan rocks, coming from the axis of the fold. This kind of gnawing occurs probably where transversal faults weakened the resistance of the Kundelungu cover.

Whatever, the more obvious feature of the Copper Arc is a general dislocation of the R.1, R.2 and R.3 Groups in a kind of megabreccia, with fragments of dolomitic rocks of every size, up to several km, scattered in a silty brecciated matrix.

(see fig. 9 and 10). This megabreccia fills as well the anticline axis, the extrusions and the gnawed flanks of folds. It is also found in the lower part of the overthrust anticlinal flanks and in the Kolwezi nappe.

This generalized megabrecciation, as well as the extrusives and gnawing structures, are solid arguments in favour of the existence of important evaporitic formations in the Roan of Shaba.

2.4. Base metals and uranium mineralizations in the Katanga System

2.4.1. Generalities

Base metals occurrences of different kinds are numerous in or near the Shaba Copper Arc. Most of them are stratiform and bear Cu. They are bound to redox contacts between red and grey coloured rocks.

The first sequence of this type is found at the base of the R.2 Group, just above the R.1. It is the most contrasted one, and also the most important economically. Indeed it holds all the Cu-Co stratiform deposits mined in Shaba. As the uranium showings are generally linked to it, I will deal only with this subject.

2.4.2. Main characteristics of the Cu-Co and U stratiform ore of the base of the R.2 Group

a.- There are two Cu ore horizons (fig. 7):

- The upper orebody: 5 to 10 m of grey dolomitic banded siltstone, called "SD de base".
- The lower orebody: 8 to 10 m of siliceous banded grey dolostone, called "RSF" and "D.strat", and 0.5 to 2 m of massive grey dolomitic and chloritic siltstone, called "R.A.T grises".

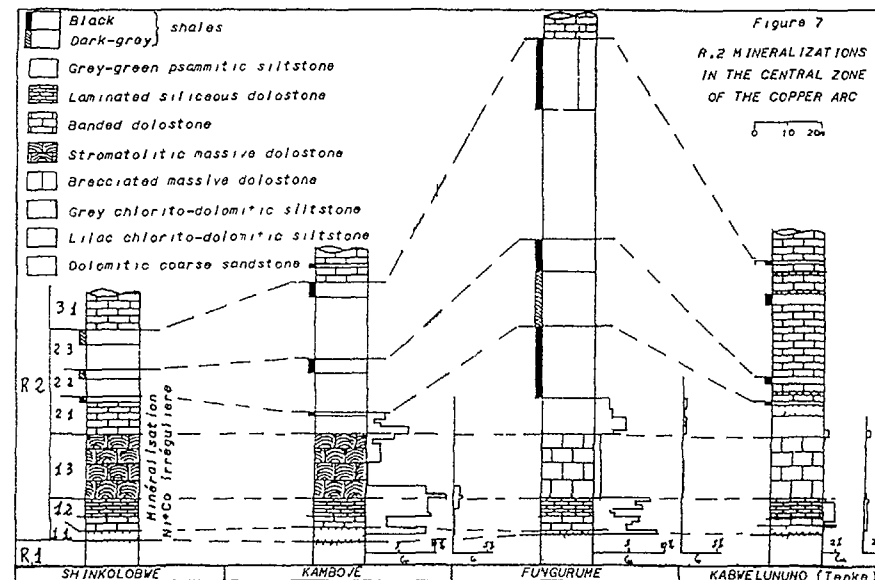
They are separated by 0 to 20 m of clear siliceous massive dolostone, called "RSC".

b.- Normally, these ore horizons contain disseminated pyrite, with some chalcopryrite. Locally, the pyrite is replaced by chalcocite, with minor bornite and carrollite, finely disseminated or gathered in small aggregates in cracks or nodules. Then they reach Cu and possibly Co high grades (some 4.5% Cu and 0.3% Co).

c.- These Cu-Co mineralizations seem unrelated to the nature of the host-rock and to the tectonic structure in which they lie. They are abruptly interrupted by the transversal faults and the silty breccias, always barren, that delimit the R.2 fragments.

d.- The Cu mineralizations cover larger surfaces in the lower orebody than in the upper one.

e.- Radiometric measurements, systematically carried out in many Cu deposits of the Kolwezi, Fungurume and Kambove areas,



reveal a probably uninterrupted anomaly, several decimeters thick, located at the base of the lower Cu-Co ore horizon (the "R.A.T. grises" or R.2.1.1). The U308 grade is between 60 to 400 ppm (3).

2.4.3. Cu-Co ore and U anomaly genesis

It is assumed that the pyrite is syngenetic or early diagenetic. Iron and traces of copper and uranium in solution in the sulfate brine of the lagoon were precipitated when the sedimentary environment, originally oxidizing, became reducing. From which the uranium anomaly above-mentioned.

Subsequently meteoric waters from the continent, as well as connate water from the sediments, percolated through the base of the Roan, becoming hot and saline in contact with evaporites. These hot brines leached the metals from the R.1 and the basement. They came up through the lower part of the R.2 Group, in some particular places, whence the lenticular distribution of rich Cu-Co ore, and its large extension in the lower orebody.

2.4.4. Subsequent uranium evolution

In some R.2 fragments, uraninite of typical vein aspect is observed in narrow cracks or brecciated zones which cross

the Cu lower orebody (4 and 5). Close to the surface, uraninite is leached and precipitated as secondary minerals, in decreasing amount away from the lower Cu orebody. These occurrences have a clear stratiform trend. The U308 grade could exceed 5% (see example fig. 8) and the total U308 content could reach up to 2,000 tons. These occurrences could result from the removal of the proto-ore by diagenetic, tectonic or metamorphic processes.

The same kinds of ore are known in the Zambia Copperbelt, as well as around the Basement domes, about 100 km south of the Copper Arc (6).

In 3 cases (Kalongwe, Swambo and Shinkolobwe), the uranium ores form vein-shaped bodies along faults. They lie also close to the R.2 Cu orebodies. Therefore, they result probably also from remobilization of the proto-ore. However, the case of Shinkolobwe is unclear, considering its very large amount of uranium.

3. SHORT DESCRIPTION OF THE SHABA URANIUM OCCURRENCES

The Shaba uranium occurrences may be either simple radioactive anomalies, without reported U minerals, either uneconomic uranium showings, or uranium economic deposits. They may be divided into 3 types:

- Stratiform type, generally bound to the R.2.1.1, first member of the R.2 Group.
- Vein type, located in faults, generally near the R.2 base.
- Occurrences located outside the R.2 Group.

3.1 Stratiform uranium occurrences in the R.2 Group

The known age determinations have given relatively late figures, that indicate a metamorphic or a thermal event, subsequent to a tectonic motion (4, 5 and 7):

- 582 ± 15 m.y. (Kamoto Principal)
- 555 ± 10 m.y. (Kambove West)
- 520 ± 20 m.y. (Kamoto Principal, Kolwezi).

There are, from west to east (see fig. 2):

3.1.1 The Kolwezi klippe occurrences

The Kolwezi klippe is a 23 x 10 km Roan pile, isolated above a Upper Kundelungu basin, with a maximum thickness of 1.2 km. It holds more than 60 R.2 fragments, where 10 uranium occurrences were found (8), (see fig. 9).

- Dikuluwe (anomaly, with Cu, mined out). To my knowledge, strong radioactivity was observed in the R.2.1.1 of some drillholes.
- Mashamba West (showing or deposit?, with Cu, mined out). A hole intersected 2.6 m of ore grading 0.16% U308. Afterwards,

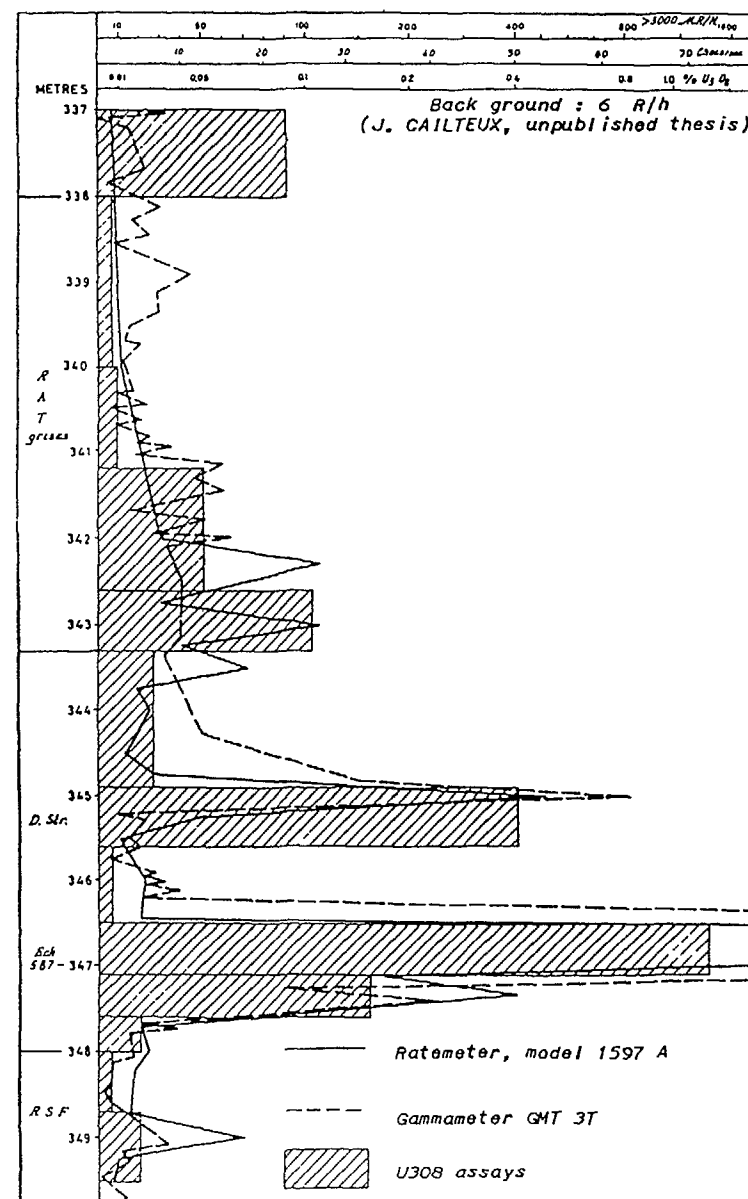
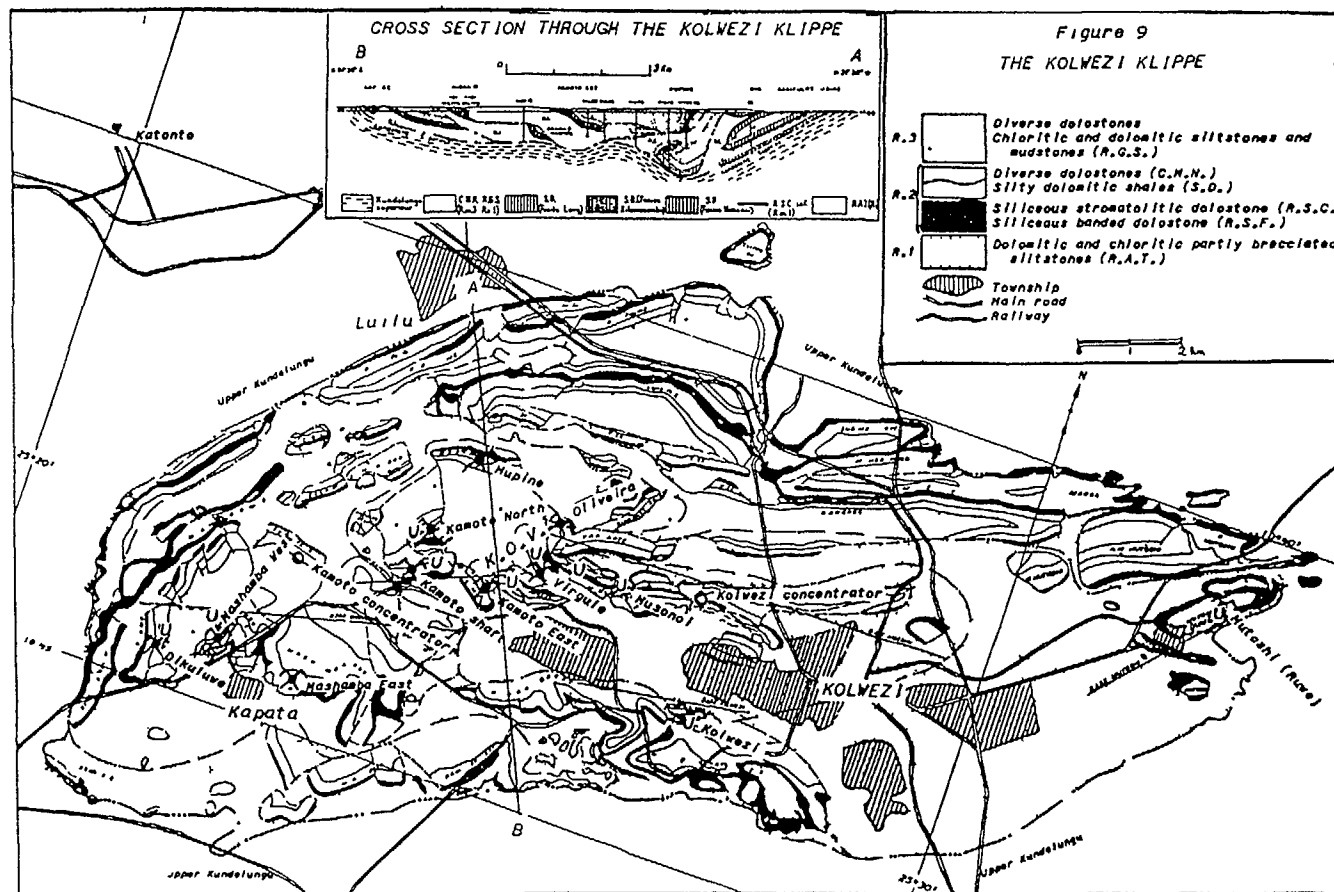


Figure 8. Example of uranium ore body in R 2 (Kambove W. drill hole 207)



beautiful U minerals were collected. Ore tonnage and grade are unknown.

- Kamoto North (deposit, with Cu-Co, mined out). It was discovered through an excess of U in plant solutions. 80,000 t of ore were stockpiled, and eventually fed the Cu concentrator. I don't know the grade of this ore.

- Kamoto Principal (deposit, with Cu-Co, partly mined out). U minerals were observed in trenches, drift and drillhole intersections of R.2.1.1. The orebody is thin (25 to 60 cm) but very rich (up to 11% U₃O₈). The total U₃O₈ content was not exactly estimated. It certainly exceeds 2,000 tons.

- Kamoto East (showing or deposit?, with Cu and Co, partly mined out). Some uranium minerals were mined in the open-pit. Ore grade and tonnage are unknown.

- Virgule (deposit, with Cu and Co, partly mined out). At first uraninite was observed in R.2.1.1 intersected by a drillhole. Subsequently, open-pit mining discovered low grade ore. The U₃O₈ tonnage reached approximately 100 tons.

- Musono Extension (deposit, with Cu and Co, mined out). Uranium ore was detected at first in a hole, and later mined by open-pit. This occurrence furnished beautiful mineral specimens, such as metatorbernite and cuprosklodowskite, and more than 3,000 tons U₃O₈, in ore grading more than 0.5% U₃O₈. A part of this ore was stockpiled, but later fed the concentrator.

- Musono Junction (showing, with Cu, Co, Au, mined out). Some uranium minerals were observed long ago in a small gold working.

- Kolwezi Mine (deposit or showing, with Cu, untouched). One drillhole intersected probably at the base of the R.2.1.1 Member, a 30 cm core grading 56.6% U3O8, very rich in gold. Nothing more is known about this occurrence.
- Mutoshi (showing, with Cu, mined out). Three drillholes intersected a large amount of poor ore in the lower orebody (possibly 750.000 tons of ore, grading 0.05% U3O8 ?).

3.1.2. The Chabara district occurrences

This district is crossed by 3 anticlines, where many R.2 fragments were mapped, as well as some uranium indications (8).

- Chabara (anomaly, with Cu and Co, untouched). Airborne prospection found a notable radiometric anomaly.
- Mashitu (showing, with Cu and Co ?, untouched). Some torbernite was observed in the R.2.1.1 of a 200 m R.2 fragment.
- Kalumbwe-Myunga (anomaly, with Cu and Co, untouched). Two 500 m R.2 fragments are covered by a large but weak anomaly.
- Kampamba (anomaly, with Cu and Co ?, untouched). A weak anomaly was observed above a 850 m R.2 fragment.

3.1.3. The Menda district occurrences

The Menda anticline holds 15 R.2 fragments, including 4 uranium occurrences (8):

- Kasompi East (deposit, with some Cu, Co and Ni, untouched). At first, uranium minerals were observed in a trench, with some Mo and Au. Later, a drillhole intersected a uraninite veinlet at 100 m depth. Three levels of drifts and 15 drillholes gave 838 t U3O8 in an ore grading 0.07 % U3O8 (including 446 t in a 0.28 % U3O8 ore), as well as 80,000 t of ore grading 1.2 %Cu and 1.8% Co.
- Kasompi West (showing, poor in Cu and Co, untouched). Rare uranium minerals were observed in trenches. Drifts and drillholes did not give interesting results.
- Menda (anomaly, poor in Cu and Co, untouched). No interesting result was found in drifts and drillholes.
- Kibamba (anomaly, without any Cu and Co, untouched). A weak anomaly was reported over a 100 m R.2 fragment, without any uranium mineral.

3.1.4. Fungurume (showings or deposits, with Cu and Co, untouched). Between Tenke and Fungurume, about 70 R.2 fragments belonging to an overthrust Roan crop. Radiometric anomalies were reported in the R.2.1.1, in many drill holes intersecting the Fungurume deposits. In one of them, 2.8 m of ore grading 2.11% U3O8 were intersected. The volume of this uranium orebody cannot be estimated.

3.1.5. Kakanda South (showings, with Cu and Co, mined out). This deposit belongs to the northern flank of an anticline, overthrust southwards. A metatorbernite coating was

locally observed in the lower orebody. The unknown grade and tonnage of this orebody are certainly very low.

3.1.6. Kambove district

The Kambove Roan belongs to the southern flank of an anticline, overthrust northwards. It holds about twenty R.2 fragments. Among them:

- Kamoya (anomaly, with Co, untouched). This anomaly was observed on a very small R.2 fragment. Black cobalt oxyde grades up to 0.1% U3O8.
- Kambove West (deposit, with Cu and Co, mined out). Uranium minerals were discovered a long time ago (1946) in this 300 m R.2 fragment. Uranium ore was observed later in drifts, drillholes and working faces. It consists of uraninite, generally hosted in the R.2.1.1 horizon, where the Cu-Co mineralization appears, finely spread out or in small veinlets. This ore is generally thin, but the grade could reach up to 10% U3O8 (see example fig. 8). The total amount of U3O8 contained in this orebody could reach up to 1,500 tons.
- Kambove Principal (showings, with Cu, mined out). Small amounts of uranium minerals were observed in the past in this huge deposit.

3.1.7. Luishia Principal (deposit or showings, with Cu, partially mined out). This mine is located 55 km SE of Kambove, along the same anticline. More than 15 R.2 fragments crop out in the vicinity. Luishia Principal is a 1,400 m R.2 fragment, dipping 10° to 45° to N 55° E.

Uranium was observed on two locations:

- A vein shaped uraninite and a yellow minerals occurrence along the fault that limits the fragment to the NW.
- A stratabound occurrence of yellow minerals along the lower orebody.

No estimate was done. Age determination gave a figure similar to these found in the vein-shaped deposits (620 ± 10 My). However this uranium deposit seems rather stratabound.

3.1.8. The Etoile district

The Etoile anticline runs about 7 km north of Lubumbashi. His southern flank overthrusts the northern one. Along it, 6 R.2 fragments are cropping out. Uranium indications were found in the following ones:

- Kiswishi (deposit, with Cu, untouched). Exceptionally, uranium is not located in the base of the R.2, but in the top (CMN), probably as a result of per descensum removal. The host rock is a weathered dolostone, surrounded on all sides by brecciated siltstones. The reserve found by drilling reaches up to 107 U3O8 tons, in a 0.13% U3O8 ore.
- Luiswishi (deposit, with Cu, partially mined out). As mentioned before, Shaban uranium ore was discovered for the first time in this deposit, which consists of a 1,500 m R.2 fragment, cut at 20 to 50 m depth by a subhorizontal fault

Uranium is located either in CMN, as in Kiswishi, or in the R.2.1.1 Member. After mining, about 20 t U₃O₈ remain, in a 0.1% U₃O₈ orebody.

- Ruashi (showings or deposit, with Cu, partially mined out). Logs of some old drillholes mention yellow minerals of uranium in the lower orebody. Not any assay is reported.

3.2. Vein-type deposits in the R.2 Group

Three deposits of this type are known. All are located along the southern limit of the Copper Arc, in the Monwezi extrusion. Relatively early ages could indicate deposition or removal of ore during one or several tectonic phases of the Katangan orogeny (4, 5 and 7):

- 706 m.y.	Shinkolobwe	Kolwezi phase ?
- 670 +/- 20 m.y.	Shinkolobwe	Kundelungu phase ?
	Swambo	
- 620 +/- 10 m.y.	Shinkolobwe	Monwezi phase ?
	Kalongwe	
	(Luishia)	

3.2.1. Kalongwe (deposit, with Cu and Co, untouched).

In this district, located 40 km SW of Kolwezi, the Monwezi extrusion holds about 10 R.2 fragments. The deposit, described elsewhere (9), is located in a 200 m fragment of unknown depth (possibly deeply rooted). The ore stretches 30 m along the R.2.1.1 Member, and also along a small transverse fault. It vanishes at 70 m depth.

Exploration by trenches, drifts and drillholes indicates reserve of 500 U₃O₈ tons in ore grading 0.2% U₃O₈.

The disappearance of the orebody downwards could indicate.

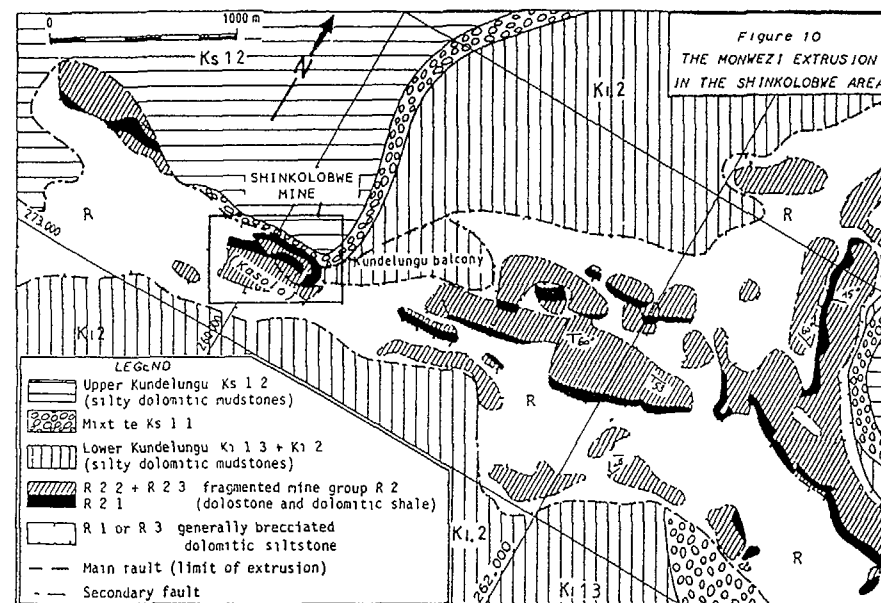
- either a per descensum origin of the ore. The metal could be derived from the proto-ore held in the R.2.1.1 Member in the eroded part of the fragment.

- or a per ascensum origin of the ore. The metal could be derived from proto-ore held by the deep seated part of the fragment, leached by hot brines and deposited far above, as a result of modifications in the temperature and the salinity of the mineralized solutions.

The age of the ore (620 My) supports the second thesis.

3.2.2. Swambo (deposit, with Co, Ni and some Cu, untouched).

Swambo was discovered by carborne radiometry along the Monwezi extrusion. It is described elsewhere (9). It is located in a fault that marks the western bound of a 2,000 m R.2 fragment, of unknown depth (probably deeply rooted). The ore is in a subvertical pipe along the fault. It encroaches partly the SD and the RSC in the upper levels. Below, it is fully enclosed in the fault breccia.



The mineralized area with more than 0.1% U₃O₈ decreases strongly downwards, but the grade increases:

- Level 1537 (20 m below surface): 1070 m² of ore
- Level 1517: 840 m² of ore.
- Level 1497: 170 m² of ore.
- Level 1435: 20 m² of ore, grading 1% U₃O₈

The reserve reaches 190,000t of ore, at 0.32% U₃O₈, say 600 t U₃O₈, with good gold values.

The Swambo deposit could have the same origin as Kalongwe.

3.2.3. Shinkolobwe (deposit, with Ni and Co, nearly completely mined out).

This very important deposit, named previously Kasolo, is described elsewhere (9, 10, 11). It is located near the eastern end of the Monwezi extrusion.

Uranium was found in 3 R.2 fragments, among the 40 ones that are cropping out in the area (fig. 10):

- The first one, 140 m in length, is cut 100 m below the surface by the fault that bounds the extrusion to the north.
- The second one becomes larger in depth: 400 m on the surface, 580 m on the 255 level. It seems very deeply rooted. It is partly cut, 100 m below the surface, by an injected mass of brecciated siltstone (fig. 11).

- The third one does not crop out. It is known only by 30 holes, drilled east of the two other fragments, from the 255 mine level, between the 300 and 450 levels. It is divided in several small blocks separated by silty breccias.

The two first fragments are completely mined out. They provided about 35,000 t of U308. The third contains about 1,000 tons U308.

The fresh ore, found below the 80 level, consisted in thin veinlets of uraninite, irregularly oriented, crossing the lower Cu orebody, the RSC and the base of SD. Metric masses of uraninite were found, piled up below and against the siltstone injection, or locally along the fault that limits the fragment to the east. The uraninite veinlets are replaced below the 200 level by scarcer and scarcer small nodules, so that the ore disappears at almost 255 m depth.

The huge amount of contained uranium and the age determinations (620, 670, 706 My) do not indicate a per descensum origin, from the poor proto-ore held in the eroded part of the fragments. The metal could originate from the proto-ore held in important deep seated R.2 fragments, leached by hot brines. The permeable Kasolo fragment acted possibly as a wick deeply plunging down into the extrusion, forming a conduit for the ascending brines of a convection cell. As such a structure is probably exceptional, if not unique, the likelihood of finding an other Shinkolobwe is small.

Uranium is accompanied by the usual precious metals (Au, Pd, Pt) and rare earth bearing monazite. Abundant sulfides and selenio-sulfides of Ni, Co and Mo are also found in the fragment, spread into and outside the uranium orebody. Above 80 m depth, weathering produced quite a lot of beautiful minerals, among them the Cu bearing metatorbernite (12).

3.3. Uranium occurrence located outside the R.2 Group

3.3.1. Uranium in the coal-bearing Luena Basin

A structural depression occurs to the NW of the Kibaran Zilo Massif, 140 km NNE of Kolwezi (see fig. 1). It is filled with subhorizontal units of the Lukuga System, of lacustrine origin, that locally overlay the Kibara System. They are probably dissected in segments by Kibaran protuberances. One of these segments forms the coal-bearing Luena Basin, 60 km long and 15 km wide.

A radioactive anomaly was detected in 1970 when a helicopter flew over the Luena open-pit. Follow-up investigations gave the following results:

- A fragment of coal, particularly radioactive, gave 0.48% U308

- Channel sampling, perpendicular to the strata:

First coal seam and shale interlayer	0 - 1 m	106 ppm U308
Second id and idem	1 - 2.8 m	218 idem
Coal, base of second seam	2.8 - 3.5 m	2,761 idem
Lower shale	3.5 - 4.7 m	289 idem

This corresponds to 4.7 m at 0.06% U308, of which 0.7 m contains 0.27% U308.

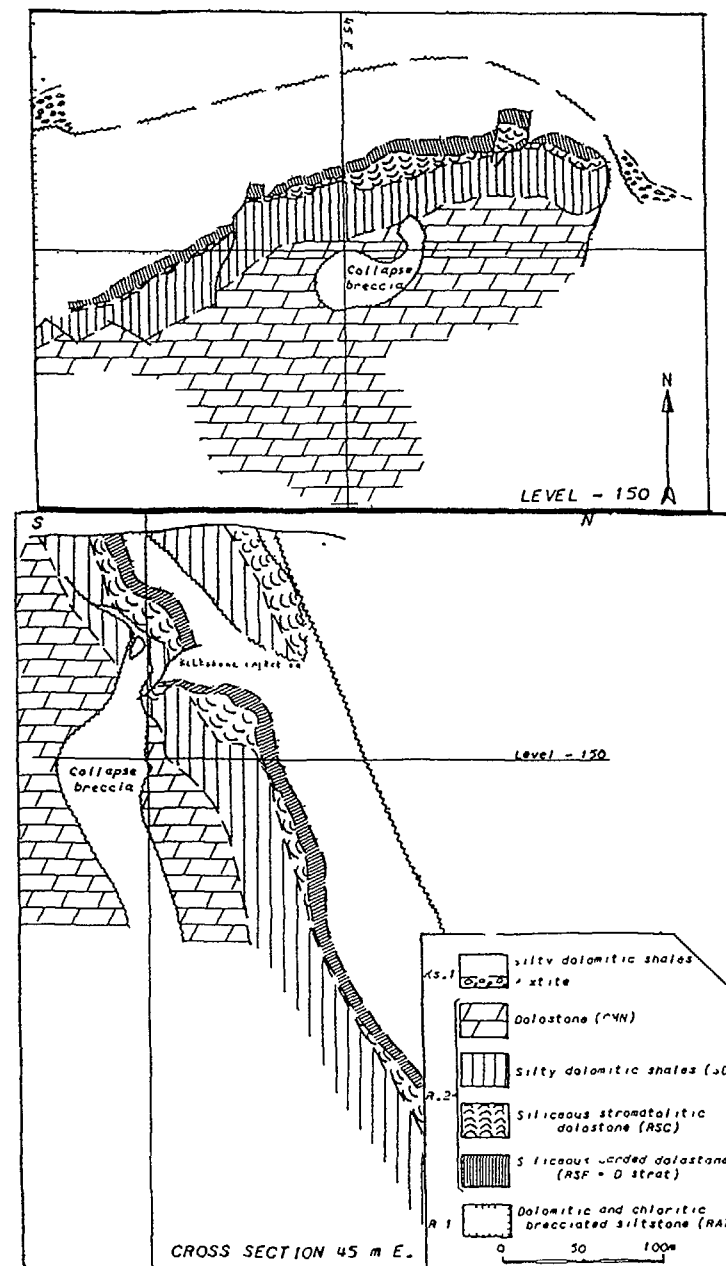


Figure 11 Shinkolobwe deposit

Later on, a drilling program to explore the coal over an area of 1 km² was conducted. 28 holes were drilled over a 200 m square mesh. One drillhole intersected 2.1 m at 0.24% U₃O₈, within 28.5 m at 0.08 % in situ (0.16% in the ashes). As the area covered by this drillhole is 4 ha, the related U₃O₈ reserve is about 2,000 t U₃O₈, in ore grading 0.08%. This is based on the assumption that the U mineralization is regularly distributed, what is very dubious.

Later on, a helicopterborn radiometric survey gave very vague indications, probably due to a thick overburden. To my knowledge, no other exploration was undertaken later.

3.3.2. Kipushi

Cu-Zn ore is mined in Lower Kundelungu dolostones, along the contact with a microsandy breccia. H. Schuiling mentions the presence of radioactive minerals (13). This is probably a mineralogical curiosity observed under the microscope.

3.3.3. Musoshi

Musoshi is a Cu deposit of Zambian type, located on the N flank of the Konkola Dome (fig. 1), which consists of granitic Basement. Some uranium occurrences were discovered in small pits and two drillholes. Later, underground crosscuts found 20,000 tons of ore at 0.12% U₃O₈, close to the surface. In addition, two uraninite veinlets, containing some molybdenite, were intersected by a drillhole at 363 and 370 m depths, just below the copper ore horizon (14), nearly as in the R.2 fragments of the Copper Arc. This indicates that the Zambian copper stratabound orebodies are likely approximately contemporary of the Shaban ones.

3.3.4. Kibaran Zilo Massif

During the helicopterborne radiometric survey carried out above the Luena district, the flight lines covered part of the Basement bordering the Lukugian to the SE. U and U+Th anomalies were detected, with amplitudes similar to those over the coal-bearing formations. The Kibaran was never prospected for uranium.

4. DATATION RESULTS AND ZONATION

The datations carried out in uranium deposits of the Copper Arc (4, 5, 7) suggest the following remarks

- a.- No datation gives a figure corresponding to a syngenetic or early diagenetic ore (approx. 1,000 m.y.?)
- b.- With the exception of Luishia, all the old ages (706 to 620 m.y.) relate to vein-shaped deposits, located along the southern limit of the Copper Arc, in a relatively recent tectonic structure, the Monwezi Extrusion.

c.- On the contrary, young ages (582 to 520 m.y.) are found in rather stratiform deposits located in the centre line of the Copper Arc, in early tectonic structures belonging to the Kolwezi phase.

The old ages are related to 3 tectonic phases, whereas the young ages are related to a late "thermal event" (4, 5 and 7). This seems not consistent with the fact that old ages are found in the youngest tectonic structure, while young ages are found in older tectonic folds.

Nickel is found instead of Cu in 3 deposits located along the southern limit of the Copper Arc only (Kasomp1, Swambo and Shinkolobwe). This fact suggested a N-S zonation Cu -- Cu+U-- Ni+U (15). However, one could point out that Kalongwe and Luishia, found in a similar location, hold Cu and no Ni.

5. POTENTIAL URANIUM PRODUCTION OUTSIDE SHINKOLOBWE

As said before, uranium production of Shaba stopped totally in 1961. Only limited tonnage was mined out in other places than Kasolo (Kalongwe, Kasomp1, Luishia and Luiswish1). The project to resume production arose in 1970, after considering the important amount of uranium found in the Musonoi open-pit, and especially in 1977, when the metal price quintupled.

For the reasons above-mentioned, the Kasolo eastern extension presents no interest. In other respects, the vein-shaped deposits Kalongwe and Swambo are too small and lie too far away from any railway and power-line to be economically mineable.

In the first place, small but steady production could come from the extraction of the uranium present in the iron precipitate produced in the electrowinning plants. The Luilu Plant (Kolwezi) produces yearly some 90,000 t of iron precipitate, grading 400 ppm U. By dissolving this material and using resin ion extraction column, 30 t U₃O₈ could be produced yearly as a by-product.

In addition, U production could considerably increase by processing the U ores found into the Cu-Co mines, especially the ones located in the Kolwezi Klippe, where more than 5,000 t of U₃O₈ have been mined out, in ores grading between 0.2% to 0.5% U₃O₈. For instance, an additionnal production of 170 t U₃O₈ would require direct leaching of 60,000 t ore per year. The U could be recovered by ion extractive column. In order to recover the valuable metals leached (some 3,000 t copper and 300 t cobalt), the solution should be introduced in the electrowinning plant circuit. The residue of filtration could produce interesting amounts of precious metals, as PM grades are high in the U ores: some 7 ppm Pd, 4 ppm Au and 4 ppm Ag.

Such an operation could go on more than 20 years with the known reserves. But it would have to overcome many difficulties, in the mining and metallurgy field.

In open pit, it is easy to exactly delimit the valuable ore by close-set trenches, and then to mine it out apart very carefully, using small mechanical shovels and dump trucks. The latter should be tested by scintillometers set out on a gantry. Such a direct selective mining can't be done in efficient underground operation. However, the U ore-bearing skips could be discriminated with scintillometers set around the winding shaft exit, or above the belt conveyor, after crushing. Of course, mining engineers don't like all these complications, and tend to throw a spanner in the works. The hindrances are much more important in metallurgy:

- considerable increase of the expensive sulfuric acid consumption.
- necessity to filter larger amount of iron hydroxide.
- necessity to inject poorly mineralized solution in the Cu-Co electrowinning circuit.

As the additional sale expected don't exceed 1.5% of the Cu-Co turnover (including precious metals), one understands that the process was never used.

Could such a project be undertaken in the future, notwithstanding that the U mined out in the past fed the concentrator? Probably yes, as iron precipitate is always produced, and as other uranium ore lenses will be probably found later in the untouched part of the Cu-Co deposits. From now on, these materials should be carefully stockpiled.

5. COMPLEMENTARY URANIUM EXPLORATION TO CARRY OUT IN SHABA

Nearly all the U occurrences known in Shaba are located in cropping out dolomitic fragments of a Roan megabreccia. How to find the unknown ones? Moreover, as it seems likely that many others are lying in depth, hidden by silty matrix, how to find the ones that are not too deeply buried?

Airborn radiometric survey, coupled with magnetometry, was carried out twice in the past. The first (1955) did only register the total count. It detected only one occurrence, Chabara, but missed many others. The second (1969) covered the main part of the Copper Arc (15,000 km²). The flight lines were spaced 500 m, flight height was 150 m. The equipment consisted of modern 4 channels spectrometer (total count, K, U and Th). This survey gave the following results:

- The radiometric anomalies of major importance which were found correspond to known surface U occurrences, namely: the Musonoi Extension open-pit, the Kasompi East and Swambo untouched deposits, the Shinkolobwe quasi-exhausted mine, and

all the stockpiles. Also the tailings from Shinkolobwe that cover the banks of the Kapare and Panda Rivers, the old stockpiles at Likasi, the Luishia and Luiswishu open-pits.

- On the contrary certain "hot spots", such as Chabara, or certain surface occurrences, such as Mashitu, were not detected.

- Half of the strongest anomalies found (34 of 76) are located in the Upper Mwashya and the Grand Conglomerat. They are partly carbonaceous formations, probably with adsorbed U. Another 20 or so anomalies are located in other formations of the Upper or Lower Kundelungu. It is very improbable that they are related to economic orebodies.

- The areas where the Roan is exposed give generally weak anomalies. This is surprising, since all known U occurrences of the Copper Arc are hosted in this Supergroup. Perhaps this is due to the overburden cover, which is always noticeable, except over the Cu-Co orebodies.

- Some 20 "hot spots" located over Roan or fault breccias were investigated by detailed ground radiometry, sometimes followed by trenching and pitting. Results were negative.

From this, airborne radiometry seems to be ineffective.

The best way to find missed or hidden U deposits could be to carry out dense exploration over the area covered by Roan formations only, and especially along the R.2 outcrops and over the extrusive structures. The method could be either helicopterborne radiometry at low altitude, with flight lines spaced 50 m, or handy and effective ground radiometry, like Alpha Cards method, carried out over a similar grid. Trenching, pitting and mainly drilling should systematically follow. Such an exploration could result in the discovery not only of uranium, but also of Cu-Co deposits.

In other respects, the potential of the Lukuga System and of the Basement should not be neglected.

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CASE HISTORIES AND NEW AREAS FOR URANIUM EXPLORATION IN BULGARIA

S.D. SIMOV
Committee of Geology of Bulgaria

I.B. BOJKOV
Uranium Company 'Rare Metals'

Sofia, Bulgaria

Abstract

Uranium exploration started in Bulgaria in 1945 and in 1946 a Soviet-Bulgarian Joint Venture was formed for the continuation of these activities. The first deposits were discovered during the late fifties. During 45 years of exploration the entire territory of Bulgaria (110 000 km²) was covered by Geiger Muller airborne gamma-ray surveys, and 80% of the territory by ground gamma-ray surveys. More than 18 million meters boreholes were drilled and 640 km galleries were constructed.

As a result 35 deposits were discovered during the past 45 years. Some of these deposits such as Buhovo, Eleshnitsa and Momino are districts consisting of several deposits. They are of medium size deposits with measured reserves between 5 000 - 20 000 t U. The remaining 32 deposits are small containing up to 5 000 t U. Seven deposits are currently in exploration stage, 14 in production and 14 are depleted. Two plants process the ore from vein deposits at Buhovo and from sandstone deposits at Eleshnitsa in southwestern Bulgaria. The capacity of the plants is sufficient to process the entire ore production as well as the loaded resins from in-situ leaching operations. The latter technique is widely used and 70 % of the uranium is recovered by this method. 75 % of the reserves occur in sandstone type deposits. High grade ore was mined from vein deposits such as Buhovo, Partisanska Poliana, Beli Iskar, and Smolian.

The reserves, the capacity of the mines and ore processing plants are sufficient to meet the uranium requirements to fuel the 6 operating nuclear reactors with a total installed electricity generating capacity of 3,54 GWe.

Uranium occurrences were known in Bulgaria at Buhovo at the Balkan Metallogenetic zone and at Streltsha in the Sredna Gora Zona since the early thirties. At the end of the second world war the Germans started to study the pitchblende mineralization at Buhovo and autunite concentrations in pegmatite veins in Streltsha. But systematic exploration started in 1945 by Russian geologists who discovered the first uranium deposits in the Buhovo region. Simultaneously with the exploration started the production of high grade ore which was shipped directly to the Soviet Union.

A Soviet-Bulgarian Uranium Joint Venture was formed in 1946 and remained was under Soviet management until 1956. Then the Bulgarian Uranium Enterprise "Rare Metals" was established. The headquarters of the enterprise are located at Buhovo town, 20 km east of Sofia. All the managers and specialists in "Rare Metals" were Bulgarians who were assisted by Soviet consultants until 1990. It was a difficult time for Bulgarian specialists as the uranium activities were considered top secret and the exchange of information was forbidden.

Systematic uranium exploration started in the early fifties and covered the whole Bulgarian territory (110 000 km²) with Geiger Muller airborne gamma-ray surveys. Despite its low sensitivity this survey turned out to be highly efficient and many uranium occurrences were detected. Some of them such as Eleshnitsa, Smolian, Planinetz, Partisanska Poliana and Beli Iskar were developed to deposits during the follow-up ground geological investigations.

Ground gamma-ray surveys were the main exploration method applied in regions with high gamma-ray background and suitable rock exposures. Almost 80% of the Bulgarian territory was covered by ground gamma-ray surveys. Radon surveys were only applied to evaluate the buried portions of known ore bodies.

Geochemical exploration and general geophysical techniques were used for a better understanding of the geochemical environment and the ore body structure. In the late sixties and early seventies a new exploration technique was developed for the geological evaluation of flat sedimentary strata. It consists of geophysical approaches to outline the sedimentary area, and morphology of the basin basement. Core drilling is applied to study the strata lithology and the geochemical environment, the oxidized zone and roll front development. The Momino, Belozem, Provslaven, Haskovo, Maritza, Navasen-Troian, Orlov dol, Isgrev, Okop-Teneva and Melnic deposits were discovered by this combined technique. All these deposits are buried under the soil and they cannot be detected by any ground surveys. The low gamma-ray energy of these ore bodies is mainly due to the low radium content generated during a short lifetime of the uranium minerals formed about 1 million years ago.

As a result of systematic exploration, 35 deposits were discovered during the past 45 years. Some of these deposits such as Buhovo, Eleshnitsa and Momino are in fact districts containing several deposits. Their size classes are indicated in the map (Fig. 1) and in Table I as medium with measured reserves between 5 000 and 20 000 t U. The remaining 32 deposits are small containing up to 5 000 t U measured reserves. 7 deposits are in exploration stage, 14 in production and 14 are depleted. Underground mining methods are applied at 12 deposits and uranium is recovered through in-situ leaching at other 8 deposits, while in 3 mines both technologies are applied.

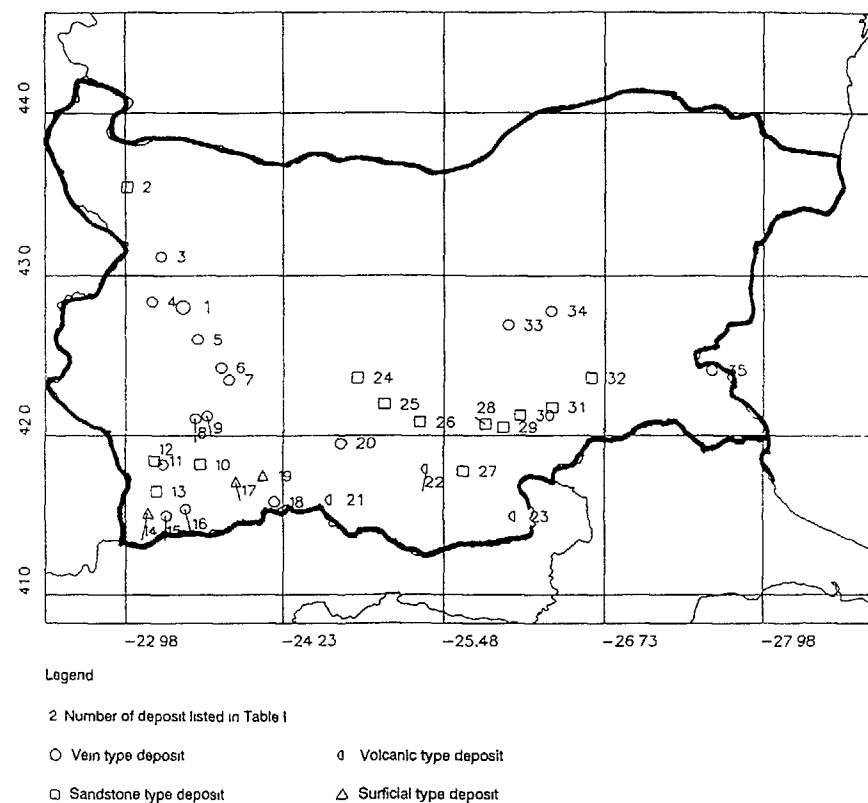


FIG. 1. Uranium deposits in Bulgaria.

The grade of the Buhovo deposit ranges from 0.1 - 1% U. The ore grade in the two deposits, Partisanska Poliana and Beli Iskar is high, between 1 and 2% U.

The deposits are located within or close to the regional uranium source such as Rhodope-Rila granites and acid volcanic rocks at Rhodope massif (Fig.1). Syenites enriched in uranium to 20 ppm were the source for uranium mobilization and subsequent ore bodies formation in the Balkan Metallogenic Zone (Buhovo deposit). Other granites containing 26-30 ppm uranium were the source for roll type deposits within sandstones and siltstones in the Thracien basin at Sredna Gora. The average uranium content in Rhodope-Rila granites is between 5,9 and 10,5 ppm with a low Th:U ratio between 1,1 and 2,0. This phenomenon characterizes repeated

TABLE I. URANIUM DEPOSITS IN BULGARIA

#	NAME	TYPE	HOST ROCK	SIZE CL	GRADE CL	U-MINERALS	ASSOC MIN	AGE	STATUS	MINING M
1	Buhovo	vein	bl schists	m	m	pitchbl	py gn cpy	U Carb	depl	UG
2	Smolianovtzi	sdst	sdst, pelite	s	m	pitchbl, brann	py-cpy	Permian	closed	UG
3	Probountza	vein	bl schists granite	s	m	pitchbl	py	Cretac	prod	UG
4	Kurnilo	vein	sdst, siltst	s	m	pitchbl	py gn cpy	U Carb	prod	UG
5	Gabra	vein	sdst, siltst	s	m	uranophan	py cpy	Permian	explor	
6	Biala voda	vein	granite	s	m	pitchbl, coffinite	py gn cpy	Pliocene	prod	UG/ISL
7	Kostenetz	vein	granite	s	l	pitchbl	py	Pliocene	explor	
8	Partusanska poliana	vein	granite	s	h	uranophan, apatite	stilbite	2 my	depl	UG
9	Beli Iskar	vein	granite	s	h	uranophan	stilbite	1 - 2 my	depl	UG
10	Eleshnitsa	sdst	sdst, congl, siltst tuffs	m	l	pitchbl, coffinite	py	Miocene	prod	UG
11	Sumiti	sdst	sdst, congl	s	l	coffinite	py	Miocene	prod	UG
12	Senekos	surficial	granite	s	l	pitchbl		Pliocene	prod	OP
13	Gradevo	sdst	sdst, congl, siltst	s	l	coffinite, apatite		Pliocene	explor	
14	Igralishte	surficial	granite	s	l	autunite	py	Pliocene	depl	UG/ISL
15	Pripetshen Deltachevo	sdst	sdst, congl, siltst,	s	l	pitchbl	py molybd	Pliocene	explor	
16	Melnik	sdst	sdst, congl, siltst,	s	l	pitchbl	py	Pliocene	depl	UG
17	Beslet	vein	granite	s	m	autunite	py	Pliocene	depl	UG
18	Dospat	vein, sdst	granite, sdst, congl,	s	m	pitchbl	py, molybd, cpy	Miocene	depl	UG
19	Selishie	sdst (surficial)	sdst, siltst, gneiss	s	l	autunite		Pliocene	depl	UG/ISL
20	Narretshen	vein	gneiss	s	l	pitchbl	py, molybd		depl	UG
21	Smolian	vein	pyroclastics	s	m	pitchbl	py, cpy	Miocene	prod	UG
22	Sarnitza	vein	pyroclastics	s	m	autunite	py	Miocene	depl	UG
23	Planinetz	sdst	pyroclastics	s	m	autunite	py	Miocene	depl	UG
24	Mormino	sdst	sdst, siltst	m	l	coffinite	py	Pliocene	prod	ISL
25	Belosem	sdst	sdst, siltst	s	l	coffinite	py	Pliocene	prod	ISL
26	Pravoslavlen	sdst	sdst, siltst	s	l	coffinite	py	Pliocene	prod	ISL
27	Haskovo	sdst	sdst, siltst	s	l	coffinite	py	Pliocene	prod	ISL
28	Marrutza	sdst	sdst, tuffs	s	l	coffinite	py	Oligocene	explor	
29	Navasen Trojan	sdst	tuffs	s	l	coffinite	py	Oligocene	prod	ISL
30	Orlov dol	sdst	sdst, siltst, congl	s	l	pitchbl, coffinite	py, molybd	Pliocene	depl	ISL
31	Isgrev	sdst	sdst, siltst	s	l	autunite	py	Pliocene	prod	ISL
32	Okop Tenebo	vein	sdst, siltst, congl	s	l	coffinite	py	Pliocene	prod	ISL
33	Sborishte	vein	granite	s	l	pitchbl	py, cpy, sphal		prod	UG
34	Sliven	vein	pyroclastics	s	l	pitchbl	py, cpy, sphal	Cretaceous	depl	UG
35	Rosen	vein	pyroclastics, monzonite	s	l	pitchbl	py, cpy, sphal	Cretaceous	depl	UG

SIZE CL size classes s = small = 500-5000 tonnes U
m = medium = 5000-20 000 tonnes U

GRADE CL grade classes l = low = 0.03-0.1% U,
m = medium = 0.1-1.0% U,
h = high = >1.0% U

STATUS expl = under exploration,
prod = in production,
depl = depleted

MINING M mining method UG = underground, OP = open pit,
ISL = in situ leach, UG/ISL = underground in situ leach

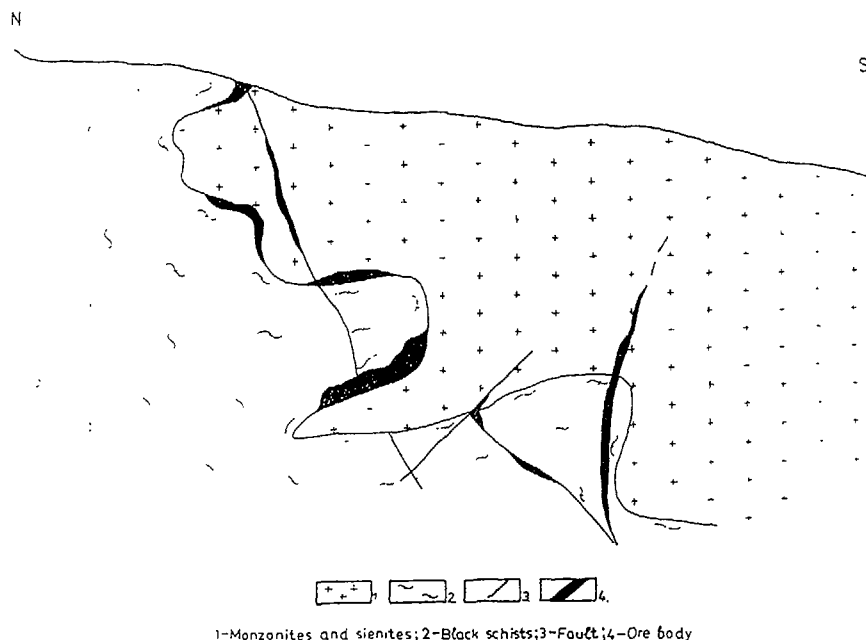


FIG. 2. Cross-section of the Buhovo deposit.

activation and differentiation of the primary rocks and formation of granites through granitization [1]. Another common feature of these granites is the high ratio of easily leachable uranium which accelerates uranium mobilization by ground waters. This is considered the reason why the ore bodies of Partisanska Poliana and Beli Iskar are of high grade.

The vein type deposits in the Buhovo region are located along the exocontact of a small quartzsyenite-monzonite massif. This intrusive and the host rocks consisting of Ordovician black schists are intruded by syenite porphyries dykes, all of them enriched in uranium. The ore bodies are located within the tectonized black schist at the monzonite contact. The morphology of the ore bodies is irregular in the form of stocks, pipes and lenses (Fig. 2).

The intragranitic vein type deposits were formed 1 - 2 million years ago in the Rhodope massif. The ore is high grade and the uranium is easily extracted from the ore consisting of uranophan and fluorine bearing apatite containing 3,72 % U (Fig. 2). The veins are banded, filled by layers of stilbite and uranophan.

Several vein deposits such as Smolian, Planinetz and Sarnitza are located within the volcanic rocks. The ore bodies - small lenses, veins and pipes - are found within tectonized rhyolites and tuffaceous sediments (Fig. 3). The ore bodies are made up of pitchblende, pyrite, markasite, quartz and calcite. There are also occurrences of sphalerite, opal and chalcedony in the halo surrounding the ore. The ore bodies are explored to a depth of 500 m (Figure 3).

Sandstone deposits are distributed within the Permian, Oligocene and Pliocene sediments. The Smolianovtzi deposit is hosted by grey alluvial sandstones and pelites in the base of the Upper Permian. The grey host rocks are intercalated with red sediments (Fig. 4). Pitchblende is the most common ore mineral associated with brannerite.

The largest sandstone deposits are located close to Eleshnitsa village within the Oligocene molasse strata. The lenticular and tabular type ore bodies are found at several levels within the medium grained grey sandstones and siltstones containing organic material (Fig. 5). The ore bodies are explored to 500 m depth. Pitchblende, coffinite, autunite and uranophan are found in the ore, associated with pyrite, markasite and zeolites.

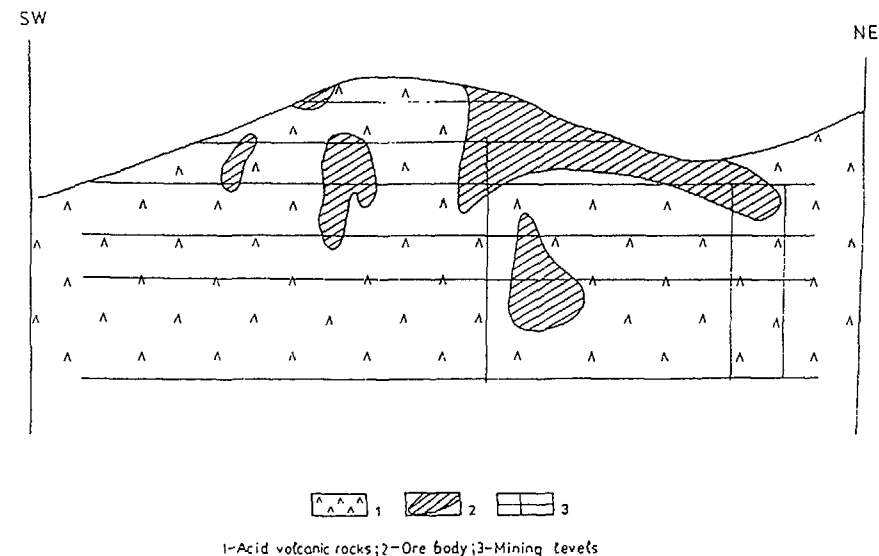


FIG. 3. Cross-section of the Smolian deposit.

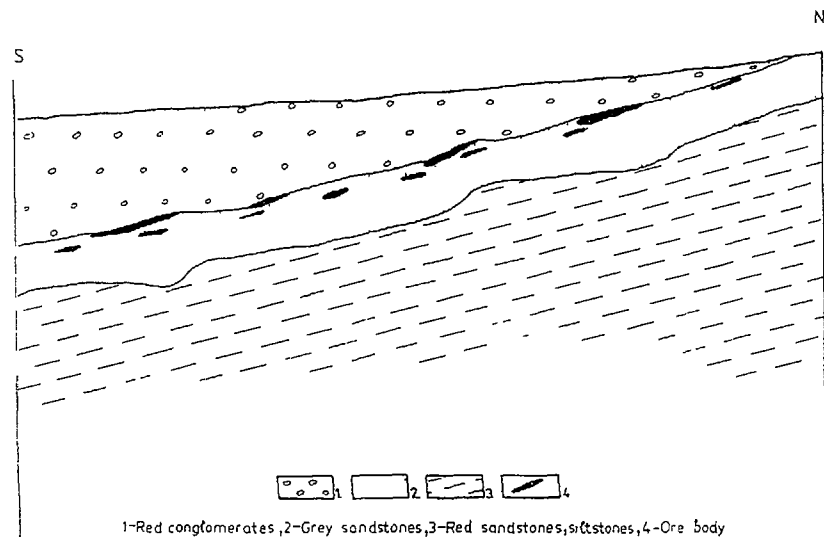


FIG 4 Cross-section of the Smolianovtzi deposit

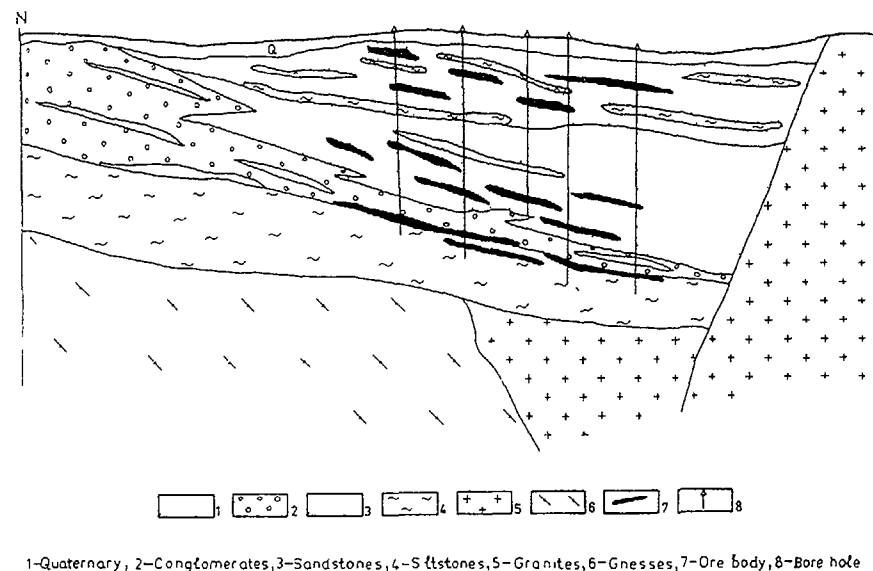


FIG 6 Cross-section of the Simitli deposit

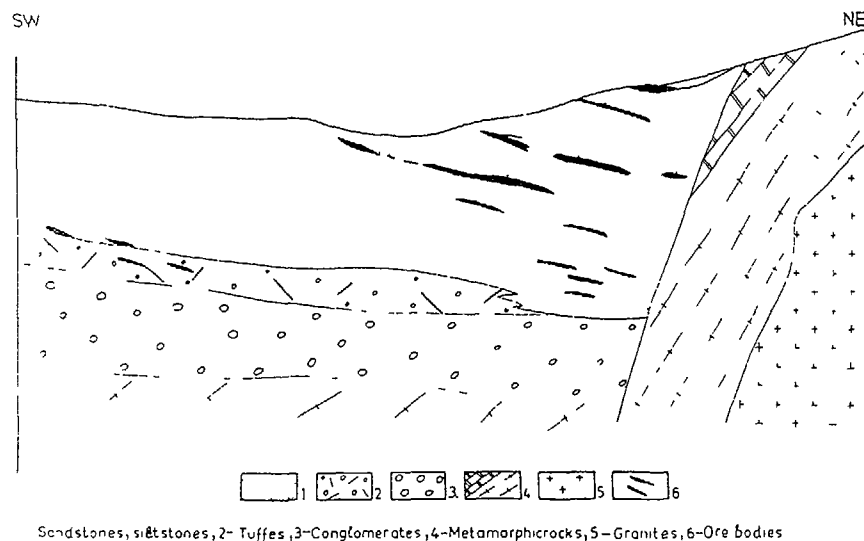


FIG 5 Cross-section of the Eleshnitsa deposit

The Simitli deposit is located within the Orranovo-Simitli sedimentary basin. Its basement consists of metamorphic and granitic rocks, covered by molasse sediments of Oligocene and Neogene age. The ore bodies are distributed in several levels within fine to medium-grained sandstones and conglomerates intercalated by siltstone strata. Coffinite is the main ore mineral associated with molybdenum and tungsten bearing minerals. (Fig. 6).

The development of the in-situ leaching technique led the geologist of the "Rare Metals" company to re-evaluate the uranium potential of the young sedimentary basins. Nine sandstone type deposits were consequently discovered in the Thracian basin. The Momino deposits are distributed along the east trending channel in Pliocene flat strata (Fig. 1). This channel is filled up with alluvial sediments which are covered by Quaternary deposits. Roll type ore bodies are located at the contact of oxidized and non-oxidized strata. The host rocks are poorly cemented sandstones and siltstones. The ore bodies are found at 100 to 260 m depth in several levels and their configuration is typical for the roll type ore bodies, having a few 100 m length (Fig. 7). Ningyoite is the main ore mineral associated with uranophan.

South of the Momino channel is another deposit with southeastern trend created by the Maritza river during Pliocene

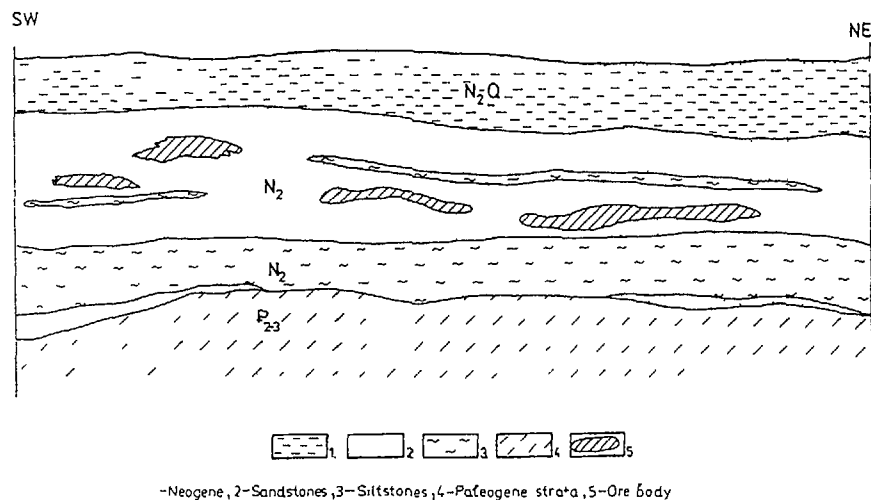


FIG 7 Cross-section of the Momino deposit

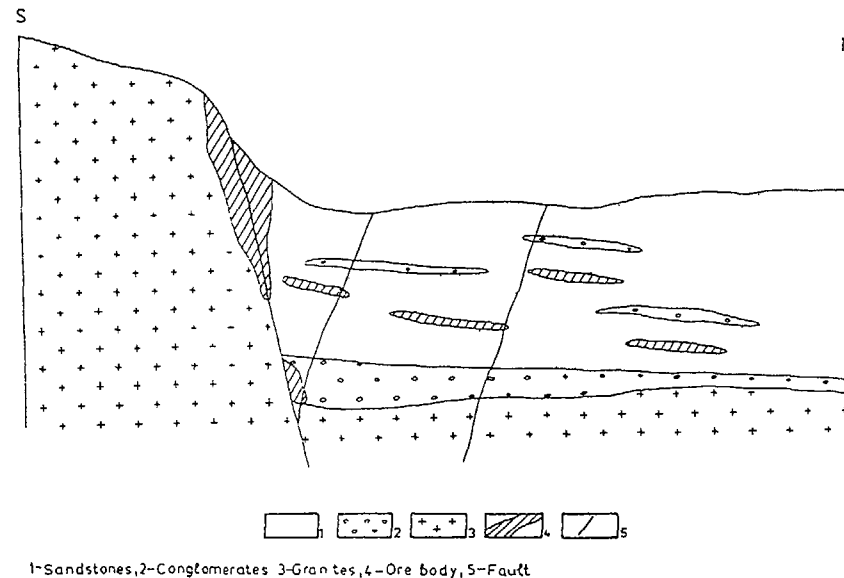


FIG 9 Cross-section of the Dospat deposit

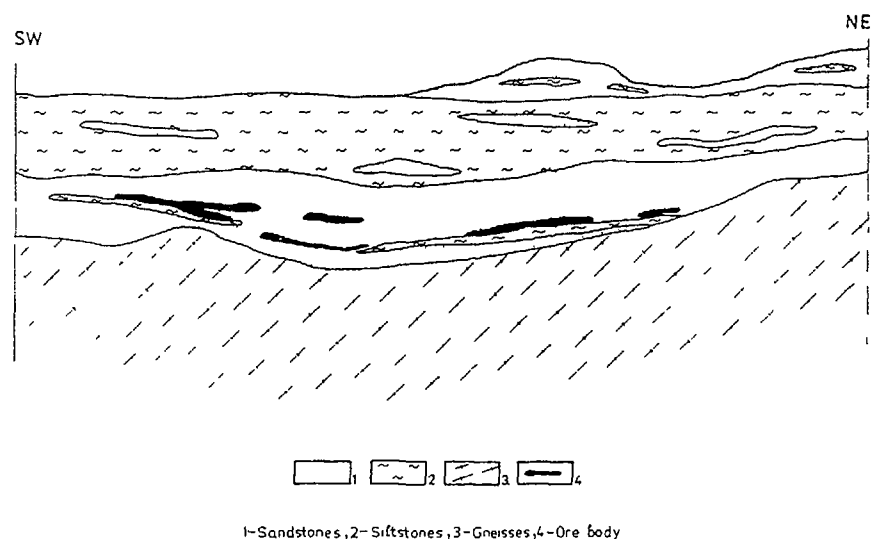


FIG 8 Cross-section of the Orlov dol deposit

and Quaternary. The Belozem, Pravoslaven and Haskovo roll type deposits are related to this channel. The ore is hosted by the same rocks as in the Momino deposits. Another channel with east trend formed by the Saslyika river at the southern border of the Pliocene basin hosts the Maritza, Navasen-Troian, Orlov dol and Izgrev deposits. The in-situ technique was first tested at the Orlov dol deposit and because of its high efficiency, was then applied along the contact between oxidized and non-oxidized sediments (Fig. 8). The linear type configuration of this geochemical trap is not typical for flat strata but is developed along young faults.

There are tabular sandstone ore bodies within the Miocene basin and veins at the basin basement of Dospat deposit (Fig. 9). This feature demonstrates the multiple mobilization of uranium distributed within the granitic basement and the formation of different type of ore bodies. Almost the same picture can be observed at the Senokos surficial deposit (Figure 10).

Uranium deposits are formed by different geological processes stimulated by tectonic and connected magmatic activation in southern Bulgaria. Tectonic-magmatic activation provided the best condition for crust differentiation and formation of magmatic rocks enriched in uranium which are referred to as regional

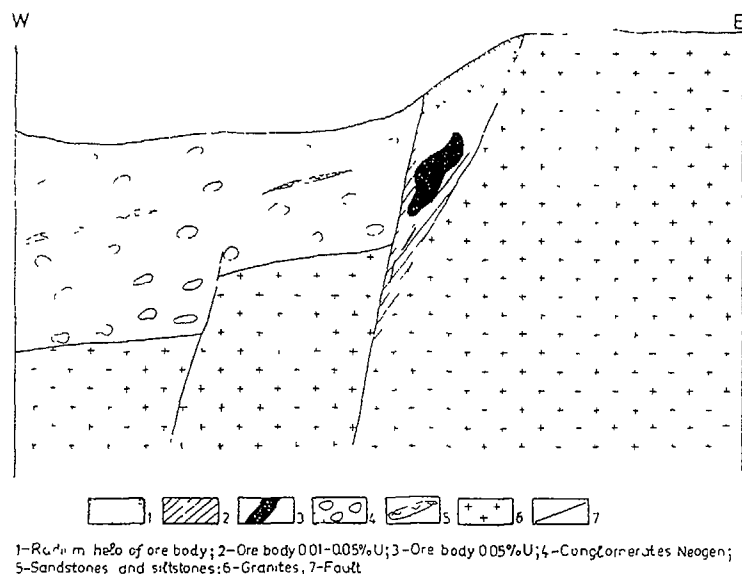


FIG. 10. Cross-section of the Senocost deposit.

uranium sources. The mobilization of uranium contained in this rock is enhanced by circulation of waters. Three tectonic-magmatic cycles are known which provided the best conditions for uranium deposits formation in Bulgaria. Depending on the ore formation depth and temperature, two main types of combined geological processes are proposed: In medium depth and temperature or close to surface under low temperature. Most of the Bulgarian uranium deposits including all sandstone types and most of the vein types were formed close to the surface under low temperature and pressure (Table II).

Two uranium ore processing plants located in Buhovo and Eleshnitsa process ore mined from the vein deposits in the Balkan Metallogenic Zone (Buhovo) and from the sandstone deposits in southern Bulgaria (Eleshnitsa). Loaded resins from in-situ leaching are also processed in the Buhovo plant.

The first nuclear electricity generating units with a total of 0,82 GWe were connected to the grid at the end of 1975. The estimated uranium fuel requirements for the nuclear reactors are shown in Table III.

The production capacity of the operating mines and of two processing plants is sufficient to meet the uranium requirement of the installed nuclear reactors. The Government is re-evaluating the existing nuclear electricity generating programme. It would be

TABLE II ORE FORMING CONDITIONS IN BULGARIA

Tectonic Magmatic Cycles	Geological Regions	Deposits formed under medium depth and temperature	Deposits formed under low temperature close to the surface
Late Alpine	Thracien Depression	-	<u>sandstone deposits:</u> Molino, Orlov dol, Boliarovo, Okop-Tenevo, Maritza, Navasen
	West Rhodope Block	-	<u>vein deposits:</u> Partisanka poliana, Beli Iskar, Piala voda, Kostenetz, Dospat, Gradevo <u>sandstone deposits:</u> Eleshnitsa, Simitli, Selishte, Melnik
Early Alpine	East Rhodope Block	-	<u>vein deposits:</u> Smolian, Sarnitza, Planinetz, Narretshen
Variscan	Balkan Metallogenic Zone	<u>vein deposits:</u> Buhovo, Probainitza Kurilo, Sliven, Sborrishte and Rosen	<u>sandstone deposits:</u> Smolianovtzi

TABLE III. URANIUM REQUIREMENTS

Year	Nuclear Reactor	Installed Capacity (GWe)	Reactor Requirements (tonnes U/a)
1987	Kozloduy 1 to 4	1.63	390
1988	Kozloduy 1 to 4	1.63	
	Kozloduy 5	0.93	
	Total	2.55	600
1989	Kozloduy 1 to 4	1.63	
	Kozloduy 5	0.92	
	Total	2.55	600
1990	Kozloduy 1 to 4	1.63	
	Kozloduy 5	0.92	
	Total	2.55	600
1991	Kozloduy 1 to 4	1.63	
	Kozloduy 5	0.92	
	Kozloduy 6	0.99	
	Total	3.54	850
	Kozloduy 1 to 4	1.63	
	Kozloduy 5	0.92	
	Kozloduy 6	0.99	
	Belene 1 cancelled		
	Total	3.54	850

Source: [4]

Note: Due to serious environmental problems the construction of the Belene 1 reactor was cancelled in 1990

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premature to foresee the installation of new reactors or prolong the lifetime of the first 4 reactors, keeping in mind the environmental problems and the potential hazards related to the operating of the old designed nuclear reactors such as the first 4 units of the Kozloduy nuclear electricity reactor.

URANIUM SUPPLY-DEMAND PROJECTIONS AND THEIR ANALYSES

M. GIROUX

Cogéma,
Vélizy-Villacoublay, France

E. MÜLLER-KAHLE, M. PECNIK
International Atomic Energy Agency,
Vienna

B. SOYER
Commissariat à l'énergie atomique,
Paris, France

D.H. UNDERHILL
Nuclear Assurance Corporation,
Norcross, Georgia,
United States of America

Abstract

The paper presents three uranium supply-demand projections, through 2005, 2020 and 2035 using different approaches and resource data. For the time through 2005, an adjusted Red Book approach is used, which compares the reactor related requirements with the expected future production including estimated imports from non-WOCA sources. For the second period, through the 2020, the methodology applied by Nuclear Assurance Corporation is used, which projects supply for two demand scenarios from firm projects and firm plus potential projects of its data base, by full cost category. The third approach, through 2035, uses the RAPP 3 computer supply simulation for the same demand scenarios both by producer country and resource category. The results from the different exercises for overlapping time periods coincide well as regards the drawdown of current stocks, the need for additional production centres as well as the sufficiency of known uranium resources. The consequences for uranium exploration and resource development are discussed.

1. Introduction

This study follows previous WOCA uranium supply - demand projections and their analyses, which were made by the International Atomic Energy Agency (IAEA) alone or in co-operation with the Nuclear Energy Agency (NEA) of OECD.

The latest of these exercises were published by IAEA [1,2] in 1986 and 1989 respectively, as well as by NEA(OECD)/IAEA [3] in 1990.

The time horizons for these projections were both short and long term, referring to 2000 or 2005, and 2030 or 2035, respectively.

In addition to the IAEA and NEA, the World Energy Conference 1989 within a general energy balance for the time 2000 - 2020 carried out by Frisch and co-workers, included also uranium supply - demand projections for the world, "the market economy zone" and "the centrally planned zone" [5].

In March 1991, the IAEA within a consulting meeting at which the authors of this report participated, laid the ground work for this study. The group concluded, that two approaches should be used, referred to as the Nuclear Assurance Corporation's method, a resource production cost oriented model for the time through 2020, and as RAPP 3, a previously used [1,2] computer programme, resulting in a producer and resource category oriented demand filling simulation through 2035.

To complete the supply-demand picture, a shorter term projection through 2005 connected with a brief review to 1984 was included in this presentation. Here, WOCA's historic and projected supply and demand data were used. For the projected supply the expected future uranium production based on the production capability from existing and committed production centers was used.

The sequence in which these projections will be presented in this report is according to their time horizon, from the shortest i.e. through 2005, to the longest, i.e. 2035.

2. Methodologies Used

The following chapter briefly describes the methodologies for the three supply - demand projections over the periods through 2005, 2020 and 2035.

2.1 Projection through 2005

For this period, also referred to as short term, the same approach has been used as in the Red Book 1990 [3], where a graphical comparison between the supply and demand projections was made in 5 years intervals.

The supply was based on the expected future production defined as the production capability by major producer country and the "rest of WOCA" minus 20%. The use of the expected future production concept was chosen, as it is considered to be a more

realistic measure than the production capability, which in a real life situation generally is not achieved.

The demand was equated with the reactor related uranium requirement, taken as aggregated total for WOCA. By definition [3] the reactor related uranium requirement is equivalent to the fresh uranium demand to the uranium mining industry. They take into account appropriate lead times for the different fuel cycle activities (refining, conversion, enrichment, fuel fabrication and site storage), but exclude any usage of recycled fissionable products such as plutonium and uranium.

2.2 Projection through 2020

The intermediate-term uranium supply-demand projections (through 2020) were made using the Uranium Supply Analysis (USA) System computer program. The USA System is a PC-based uranium industry data-base and analytical system developed and operated by Nuclear Assurance Corporation (NAC). A description of the USA System is given in the Appendix.

The USA System projects supply needed to meet the given demand. For this analysis, economic production under free-market conditions is used. Under economic production all requirements, including contracts, are met by the lowest cost producers. By assuming free market conditions all political restrictions, including the boycott of South African product and the Australian three-mine production limit, are lifted. Demand is met by production starting with the lowest cost producer. Increasing demand is met by production from the next higher cost producer.

Production is constrained only by production capacity and the identified project schedules. Projects do not come into production until their projected startup date. In market balancing, projects may be delayed, but never started early. The USA System does not develop additional projects based on the potential discovery of new ore deposits. Discovery of new resources is modeled in the long-term scenarios discussed in chapter 4.3 of this paper.

For this analysis, each project was allowed to produce at up to 100% of rated capacity. Aggregate analysis performed for the cases discussed indicates that capacity utilization is normally below 85%. Restricting individual projects to operate below 100% of capacity results in higher Full Cost projections than occur in the cases presented. This results from assuming that low cost, full capacity projects like Key Lake produce less product than they actually do, and by then replacing this product with production from a higher cost producer.

Results from the analysis of four cases are discussed: two cases each for both high and low demand scenarios. The two cases are referred to as "All Firm Projects" and "All Projects". Firm Projects are defined as all production centers for which plans as to production method, production level and first commercial

operation date have been announced. Analyses using All Firm Projects define the uranium supply capability and production cost structure of today's existing and/or planned industry through 2020. Analyses using All Projects (including All Firm plus All Potential Projects) define the additional capacity, together with its cost structure, required to meet demand through 2020. The scenario including production for All Projects is based on all identified resources in the USA System data-base.

The following assumptions were used in this analysis:

- the USA System data-base consists of resources as of January 1, 1991, as well as publicly announced project plans as of May 1991.

- the resource base for All Projects is identified resources by production center, consisting of RAR and some EAR-I. Recoverable resources are estimated by making appropriate reductions for losses in mine extraction and processing.

- Full Cost is the basis for all analyses. The authors assume that producers will increase production only when the market price provides for Full Cost recovery.

- an estimated annual non-WOCA supply of 3 000 t U increasing to 5 000 t U by 2000

- U-stocks in WOCA of 150 000 t, which can be drawn down to 1 year forward reactor requirement of about 40 000 t U

- the data for high and low demand scenarios

For Firm Projects the schedule is as stated by the owner. Standby and delayed projects awaiting market improvement are restarted when needed to satisfy demand. Startup dates are assigned for Potential Projects with no announced production schedule. These startup dates are based on estimates of the time necessary to plan, finance, permit, license, and construct the facility.

2.3 Projection through 2035

The long term uranium supply demand projections (through 2035) were made with the RAPP3 computer program. The program was originally developed by US DOE in years 1978-1982, modified for IAEA and installed on the Agency's mainframe computer where it is used for periodic uranium supply-demand analysis [1,2].

In general, the RAPP3 computer model uses given supply projection and estimates additional uranium production attainable from estimated resources by simulating the processes, events and time involved in the exploration, discovery, production and depletion of uranium resources, on a country by country basis. The objective of this supply modeling is the filling of the given demand projection.

The following given projections represent the input into computer model:

- the expected future production as explained above, based on existing and committed production capability projection [3]
- remaining RAR and EAR-I resources, recoverable at cost of up to \$80/kg U [3]
- undiscovered resources of the EAR-II and 12.5% fraction of SR categories assumed to be recoverable at cost up to \$80/kg of U
- an estimated non-WOCA supply
- U-stocks in WOCA of 150 000 t, which can be drawn down to 1 year forward reactor requirement, but is assumed to be adjusted
- the demand data for a high and low scenario

Furthermore a number of assumptions affecting the simulation of the supply process from exploration to production, was made. They include: the rate (or percentage) at which the resources are discovered and converted to reserves, the lead times for a number of events, such as time from start of exploration to the first discovery and time for the development of adequate resources to justify the construction of a production centre, construction time, and life time of the hypothetical production centre.

3. Data Bases Used

In this chapter the sources of the supply and demand data used for the various projections (-2005, -2020, -2035) are given.

3.1 Supply

The main sources for the supply data for the projections through 2005 and 2035 are the Red Book 1990 [3], as well as the Red Book Statistical Update 1990 [6].

The past supply (1984 - 1990) was taken from the actual production data of the WOCA producer countries augmented by an increasing import from non-WOCA countries (China, USSR) starting in 1986.

For the short term supply projection, through the year 2005, the concept of expected future production was used. In general, it is defined as 80% of the production capability from existing and committed centres, producing from low cost (up to \$80/kg U) known resources (RAR + EAR-I). This approach was introduced into the IAEA' work on uranium supply, as it appears to be more realistic than the production capability, which assumes a 100% utilization rate of the capacities of the mines and mills.

The basic data on the production capabilities are derived from the Red Book Statistical Update 1990 [6], but adjusted in view of recent mine closures. Also in addition to the WOCA supply projection, a growing import from non-WOCA was assumed.

For purposes of this paper, a reconciliation between the 1990 USA System data-base and the 1990 Red Book resource inventory was completed in order to better understand the similarities and differences between the two uranium resource data-bases. It was also done to increase confidence when comparing results of the intermediate- and long-term scenarios of this paper. The intermediate- and long-term scenarios were developed using the USA System and RAPP 3 analytical systems, respectively. The systems used the USA System and the Red Book resource inventories, respectively for all analyses presented in this report.

The reconciliation, for all resources with a production cost of less than \$130/kg U, indicates the 1990 USA System resource base is about 8% lower than the equivalent 1990 Red Book RAR and EAR-I inventory. A compilation of the 1990 USA System resource base is given in the Appendix. While there are significant differences in inventories between the two systems for countries with large resources (for example, Canada, Australia, and South Africa), these differences between countries are offsetting. The \$130/kg U limit on production cost was selected in order to limit the reconciliation to a practical range. The reconciliation was done based on the USA System "production cost" which is equivalent to the IAEA's "cost of production."

As noted above, resource inventories for individual countries differ between the USA System and Red Book. These differences between inventories occur primarily for two reasons: (1) The USA System is designed as a commercial system with a maximum forward project life of 30 years, and (2) uranium resource classification systems differ from country to country, while the operators of the USA System attempt to apply uniform standards for classification of resources of all production centers.

The Red Book resource compilation is meant to be inclusive; all estimated resources are included with no limit on maximum project life. Therefore, production centers with very large resources, such as South Africa's Nufcor and Australia's Olympic Dam, include all contributory RAR and EAR-I resources. These resources are capable of supporting production for more than 30 years. For practical reasons of financial analysis, USA System projects are limited to a maximum 30-year life based on existing and/or planned production capacity.

The Red Book inventory is a compilation of the resource inventories contributed to the IAEA and NEA by all of the reporting countries. The Ministry of Energy, Mines and Resources Canada (EMR) uses a more conservative resource classification

system than do most countries. The EMR reports as RAR and EAR-I only those uranium resources that are measured, indicated or inferred, and that are classified as minable. The USA System includes resources for some Canadian projects that are not classified as minable by the EMR. Projects that NAC believes have a high probability of future production, such as Cigar Lake and Midwest, are included in the USA System data-base. Resources of the Cigar Lake and Midwest projects were not classified as minable when the 1990 Red Book was compiled. They were therefore excluded from the 1990 RAR and EAR-I Red Book inventory.

There are differences between the 1990 USA System resource inventory and the January 1991 inventory used in the intermediate-term analysis presented below. The 1990 inventory was 2 036 7 + U million t U, while the January 1, 1991 total is 1 875 4 t U. This is a reduction of 161 300 t U, or 7.9%. Some of the changes are: 1990 production resulted in a reduction of the resource inventory. The inventory was also reduced to reflect adoption of a proposed highgrade mining plan for the Midwest project. Resources were lost or reclassified to Potential Projects with announced closures at Elliot Lake and in France. Addition of the McArthur River deposit in the Potential Projects category added about 58 000 t U to the inventory.

For the long term supply projection through 2035 using the RAPP computer model, the basic source was the Red Book 1990 [3] for the production capability projection, from which the expected future production was derived, as explained above. The computer simulated supply projection used first the low cost known resources not utilized for the expected production and subsequently after their exhaustion the undiscovered resources EAR-II and SR.

3.2 Demand Data

The demand data for the short term and longer term (-2020 and -2035) are derived from different sources compiled at different times. Therefore, a small discrepancy occurs in the data for the overlapping periods.

For the short term the data was taken from the Red Book Statistical Update 1990 [6] and adjusted for non-OECD-WOCA from [3].

The longer term projections, separated into a high and low scenario were based upon [3] and [7] and extrapolated through 2035. In detail, the two scenarios are defined as follows:

High: mid points between the high and low scenarios of the light water reactor (LWR) strategy [3],

Low : lower quarter point between the low 15% improved LWR and the low plutonium burning LWR strategies [7].

4. Supply - Demand Projections

This chapter summarizes the main input data and presents the results and analyses of the three supply - demand projections, through 2005, 2020 and 2035.

4.1 Through the Year 2005

This projection includes a brief historical review of the supply - demand situation between 1984 - 1990, to better illustrate the transition from a period governed by uranium overproduction to one projected to be dominated by underproduction. This short term projection, for the first time makes an attempt to incorporate a supply share from non-WOCA sources, including currently China and the USSR, but also from a number of prospective producer countries such as the CSFR and Hungary.

The WOCA installed nuclear electricity generating capacity grew during the period 1984 - 1990 from about 189 to 283 GW(e) and is projected to further increase from 287 in 1991, 335.8 in 2000 to 351 GW(e) in the year 2005. In parallel, the reactor related uranium demand grew or is projected to grow accordingly, as detailed in the following table.

WOCA HISTORICAL AND PROJECTED
URANIUM DEMAND
(Tonnes U)

1984:	37 000	1990:	43 800
1985:	37 000		
1986:	36 300	1991:	44 700
1987:	37 500	1995:	46 300
1988:	40 600	2000:	49 300
1989:	43 000	2005:	52 600

For the same period, the uranium supply defined as historical production or projected expected future production augmented by assumed imports from non-WOCA is declining from nearly 39 000 t U in 1984 to an expected low of less than 31 000 t in 1991. For the time through 2005, an oscillation in a range of 33 000 - 35 000 t U is assumed, as shown in the following table.

WOCA HISTORICAL AND PROJECTED
URANIUM SUPPLY
(Tonnes U)

1984:	38 844*/	-----	1990:	33 008 / 3 000
1985:	34 859 /	-----		
1986:	38 016 /	750**	1991:	30 710 / 3 000
1987:	37 742 /	1 000	1995:	33 500 / 4 500
1988:	37 991 /	1 500	2000:	36 000 / 6 000
1989:	36 371 /	2 000	2005:	33 000 / 6 000

* Denotes total supply, including estimated non-WOCA share

** Denotes estimated non-WOCA supply share.

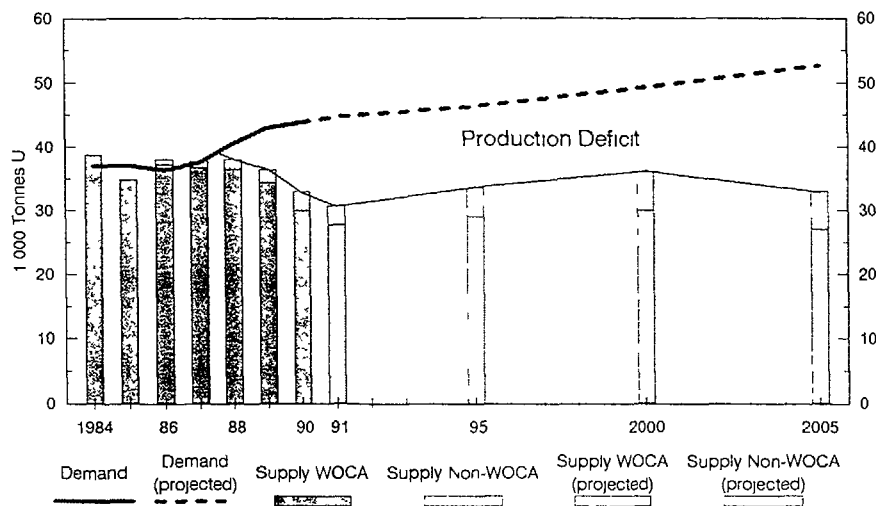


FIG 1 WOCA uranium supply-demand — history and short term projection

The supply - demand relationship using the above listed data (Figure 1) shows a production deficit in 1985 and from 1988 through 2005. Significant is the wedge-like widening of the production gap from about 2 500 t in 1988 to over 6 500 t in 1989 and nearly 11 000 t U in 1990. Using the above data, the production deficit appears to stabilize at about 13 000 t/year through 2 000 and to increase again reaching nearly 20 000 t in 2005.

The total production deficit according to the supply - demand picture detailed above, could reach 240 000 t U for the time 1984 - 2005. This is equivalent to nearly 25% of the cumulative uranium demand of this period.

The reasons that this production deficit has no impact on the current uranium price development, are the large uranium stocks in WOCA and non-WOCA alike. They were built up in times of overproduction mentioned above for WOCA. A similar development has taken place in China and the USSR which acquired the uranium produced in the Bulgaria, CSFR, the former GDR, Hungary and Romania.

At present, the volumes held in stocks are estimated to amount 160 000 t U in WOCA with increasing tendency. Of the total, due to lower concerns for assurance of uranium supplies, only 50 000 t are believed to be tied up as required inventories, while the remaining 110 000 t U should be available to contribute to the filling of the production deficit. The total non-WOCA stocks held in China and the USSR are very difficult to estimate,

due to the lack of historical production data. The Chinese inventories stemming from over 35 years of uranium production at variable levels and virtually zero demand could be around 50 000 t while the Soviet stocks are generally estimated at 200 000 - 250 000 t U.

In addition, the Eastern European production although decreasing due to the termination of the purchasing arrangements with the USSR, may still be about 10 000 t U/year the majority being produced in the USSR, while the present Chinese production is estimated at about 500 t U/year. Compared to this, the aggregate reactor related demand in Eastern Europe is about 10 000 - 12 000 t U in 1990.

In total, the Eastern European countries are in a very similar situation as there WOCA counterparts: the uranium mining production does not cover the demand and the difference is being drawn down from stocks and inventories, which also have to provide for the exports to WOCA.

For a world supply - demand projection through 2005 there are insufficient data especially for the supply side. The demand projection, however, can be of interest and may be justifiably included here, as an attempt to extend the analyses from WOCA to the world.

The world's nuclear electricity generating capacity for 1990 amounted to about 327 GW(e) and is projected by IAEA [8] and NUKEM [9] to reach 447 or 429 GW(e) respectively in the year 2005. The uranium demand from three estimates (IAEA, Nuclear Assurance Corporation (NAC) [10] and STEYN [11]) through the same period are summarized in the following table and graphically shown in Figure 2.

	WORLD URANIUM DEMAND (Tonnes U)			
	1990	1995	2000	2005
IAEA	52 480	59 400	64 800	71 720
NAC	59 120	66 390	79 360	79 360
STEYN	53 400	54 220	63 540	67 160
"Average" demand	55 000	60 000	69 230	72 750

In Figure 2 the separation between WOCA and non-WOCA indicated by the discrete lines, makes it evident that the major growth is expected to occur in the non-WOCA countries. On the other hand, it has to be accepted, that the uncertainties of the estimates lie in the non-WOCA portion of the world nuclear capacity projections.

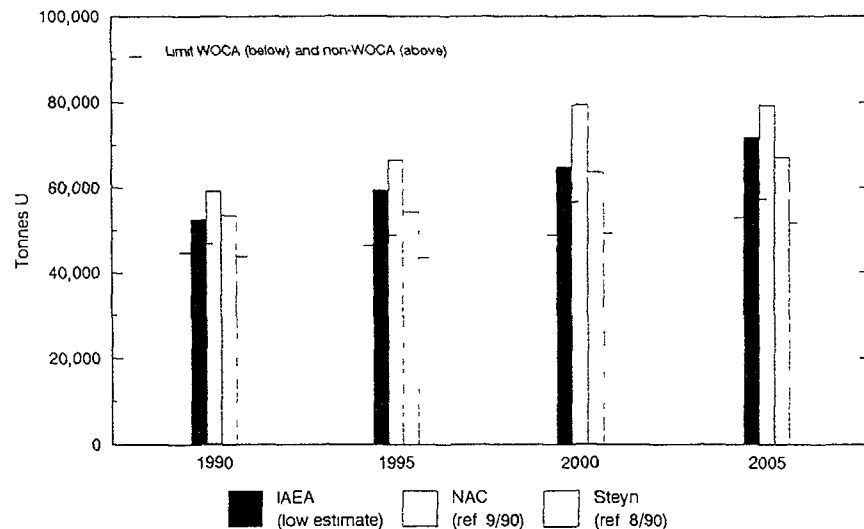


FIG. 2. World uranium requirement projections.

4.2. Through the Year 2020

For this intermediate-term analysis the underlying WOCA uranium demand projections are based on two different reactor strategies for meeting one nuclear electricity generation projection. This projection of electricity generation is the mid-case between the high and low demand scenarios through 2030 developed for the 1990 Red Book [3].

For the period of this projection, the nuclear capacity is expected to grow from 283 GW(e) in 1990 to 780 GW(e) in 2035. The data set in five-year intervals is listed in the following table.

Year	GW(e)	Year	GW(e)
1990	283.0	2010	462.5
1995	306.5	2015	527.5
2000	328.3	2020	597.5
2005	390.0		

The period of interest for the intermediate-term analysis is through 2020. The uranium high and low demand projections defined in chapter 3.2 show increases from 41 500 t U in 1990, for both cases, to 77 000 t U and 62 120 t U in 2020 for the high and low scenarios, respectively. More detailed listings of the annual demands are given in Table 1 and 2. Also shown are the cumulative

uranium requirements of the scenarios: for the high case, 1 896 000 tonnes U and for the low case, 1 684 000 tonnes U. Low case requirements are about 11% less than high case requirements.

The growth rates for both cases are very modest: less than 2% per annum for the high case and 1% per annum for the low case. This corresponds to the mid-case demand reported in the study published in 1989 [2].

The supply simulations of this study are based on two production assumptions: (a) by All Firm Projects, including Operating, Shut Down and Planned Projects, and (b) by All Projects, including Potential Projects. Potential Projects are those known resource deposits for which there are no announced production plans but that NAC believes, given sufficient demand, can and will come into production within the next 30 years. The sequence of this process is to first establish net demand by:

- estimating low and high demand cases
- reducing demand by estimated expected non-WOCA supply
- meeting about 110 000t U of demand over the next 6 to 8 years by drawdown of the WOCA stockpile;
- and then meet net demand from production.

This is done assuming free market economic production. Production is at Full Cost including capital and operating costs, with a small return to cover interest expense and a profit. All costs are in before-tax, 1991 dollars.

In the first analysis production is met from:

- All Firm Projects, including Operating, Shut Down and Planned projects
- production is from the production center with the lowest cost resources.
- additional production is then added from other producers in the order of increasing cost, until demand is met.

In the second analysis All Projects, including Potential Projects, produce to meet demand. No preference is given between All Firm and Potential Projects. However, a Potential Project is not allowed to produce before its estimated start up date. Production starts with the lowest cost resources and additional production is added from the next higher cost producer until demand is met. All of this production comes from the RAR and EAR-1 resource categories.

TABLE 1 LOW URANIUM DEMAND SCENARIO — ALL FIRM PROJECTS

Year	Demand Input to USA System (tonnes uranium metal)			Demand Fit	
	LOW DEMAND	NON-WOCA SUPPLY	INVENTORY REDUCTION	FIRM PROJECTS PRODUCTION	DEFICIT
1990	41 500	3 000	10 162	28 338	0
1995	47 800	3 000	15 618	29 182	0
2000	52 000	5 000	1 500	43 049	2 451
2005	55 270	5 000	0	32 268	18 002
2010	57 050	5 000	0	24 653	27 397
2015	62 450	5 000	0	17 733	39 717
2020	62 120	5 000	0	11 195	45 925
TOTAL	1 683 710	139 000	110 000	850 462	584 248

TABLE 2 HIGH URANIUM DEMAND SCENARIO — ALL PROJECTS

Year	Demand Input to USA System (tonnes uranium metal)			Demand Fit	
	HIGH DEMAND	NON-WOCA SUPPLY	INVENTORY REDUCTION	ALL PROJECTS PRODUCTION	DEFICIT
1990	41 500	3 000	10 162	28 388	0
1995	47 800	3 000	15 402	29 398	0
2000	55 500	5 000	0	51 530	0
2005	62 250	5 000	0	57 637	0
2010	69 750	5 000	0	64 264	0
2015	72 750	5 000	0	62 710	5 040
2020	77 000	5 000	0	52 999	19 001
TOTAL	1 895 750	139 000	110 000	1 578 432	68 166

The four supply-demand cases, including the two demand scenarios and associated supply projections, are given in Tables 1 and 2 and Figures 3, 4, 5, and 6.

4.2.1 Capability of the WOCA Uranium Industry through 2020

The high and low demand cases are nearly identical through the year 2000. The production shortfall throughout this period is met by the utilization of inventory. Annual inventory drawdown is nearly 10 000 t U in 1990, increasing to 15 000 t U in 1995. As the available inventory approaches zero (or is nearly consumed), production must be increased to meet demand. With the excess inventory of about 110 000 t U nearly consumed by the late 1990s, the supply and demand come into balance in about 2000. In the following discussions the term demand is equivalent to net demand or demand less non-WOCA imports.

Thousands of MTU

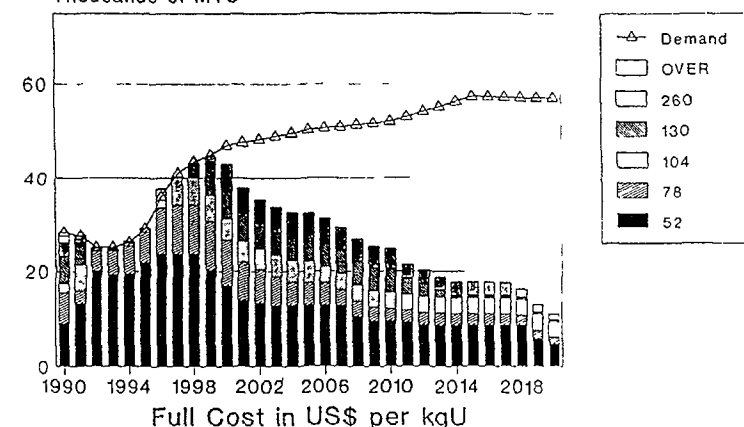


FIG. 3 Economic production in MTU — WOCA free market low demand (all firm projects)

Thousands of MTU

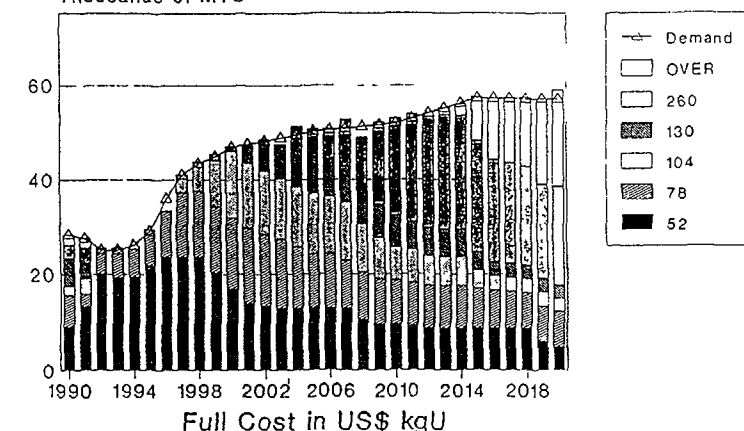


FIG. 4 Economic production in MTU — WOCA free market low demand (all projects)

By bringing All Firm Projects into production, it is possible, in both the high and low demand cases, to meet demand through about 1998. However, this requires a very rapid and dramatic increase in production levels through the second half of the decade. Annual WOCA production will have to increase from about 25 000 t U in the 1994-1995 period, to about 45 000 t U by 1998 to 2000. The required increase of 20 000 t U is equivalent

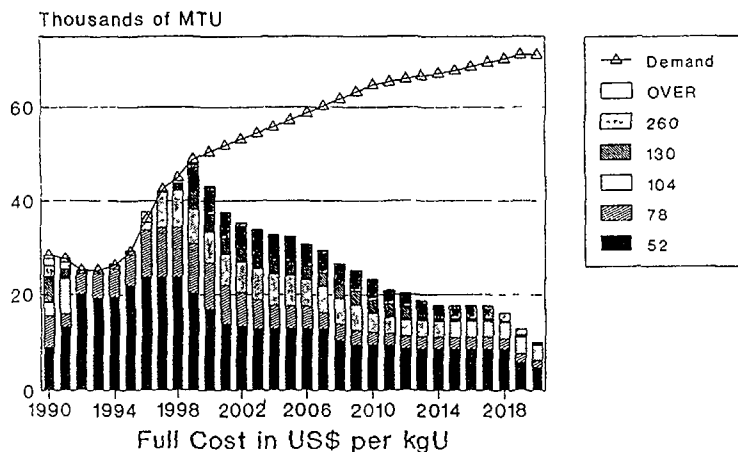


FIG 5 Economic production in MTU — WOCA free market high demand (all firm projects)

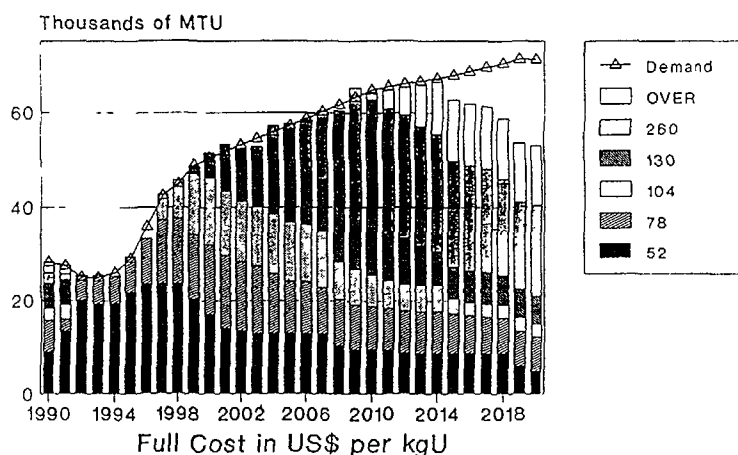


FIG 6 Economic production in MTU — WOCA free market high demand (all projects)

to production from four or five projects the size of Key Lake or Cigar Lake. This a production increase of 80% over a four to five year period.

While it may be initially possible to meet annual demand of 45 000 t U with production from only All Firm Projects, it will not be possible to sustain annual production at this level beyond

2000. With All Firm Projects operating at high levels, resources of some projects will be quickly exhausted resulting in project terminations and falling production. A shortfall would occur as early as the year 2000. By 2005 a production shortfall of 18 000 to 25 000 t U would occur in both the low and high demand cases, respectively. For the low-demand case, the aggregate production shortfall from All Firm Projects through 2020 is about 585 000 t U (see Table 1). For the high demand case the aggregate production shortfall from All Firm Projects is about 37% greater, or nearly 800 000 t U. The shortfall is 34% of low demand through 2020 and 52% of low demand over the same period.

Assuming that non-WOCA imports do not increase significantly beyond 5 000 t U (as is the assumption for the intermediate term analysis for both scenarios through 2020), it will be necessary to make up the deficit by producing from a large number of production centers for which there are no current plans. The new production centers will be required as early as the year 2000. Given the 10- to 15-year lead times necessary to discover new low cost deposits [5], it will not be possible to meet a significant portion of the demand in the 2000 to 2005 period with production from new discoveries. Because of the relatively short time available, it will be necessary to develop projects based on higher cost, already discovered resources. These are resources in the RAR and EAR-I categories included in Potential Projects of the USA System.

It is possible to fill all demand in the low case with production from All Projects. However, for this to occur, annual production would have to increase to between 55 000 and 60 000 t U by about 2015. For the high demand case, production from All Projects could meet demand until about 2015 by increasing annual production to nearly 70 000 t U. Beyond 2015 additional production from presently undiscovered resources will be required. Since this is over 20 years in the future there is sufficient time to develop new projects based on discovery of resources in the Estimated Additional Resources-II (EAR-II) and Speculative Resource (SR) categories. Long-term resource requirements are discussed in a later chapter of this report.

In conclusion, for both the low and high demand cases, production from known resources could meet demand for most of the period through 2020, but it will require the development of a large number of new production centers for which there are no current plans. The required Potential Projects include several Australian deposits with large resources. At present these deposits cannot be put into production because the Australian government policy allows only three uranium production centers.

4.2.2 Cost Structure of WOCA Uranium Industry through 2020

In addition to showing the production capability of the WOCA uranium industry, Figure 3, 4, 5 and 6 also show the distribution of production by cost category. All analyses in this chapter are

based on the Full Cost of production. Full Cost is defined as full recovery cost (Appendix) with return on investment based on sunk and forward costs determined by a discounted cash flow, rate of return (DCFROR) analysis using a specified rate of return. Full Cost was adopted for this study because it was assumed by the authors that new projects would be developed and brought into production only when the owners can expect to recover the full cost of production, including a minimal return on their investment.

The before tax rate of return (ROR) used in the Full Cost calculation is either 10 %, 12 % or 15 % depending on the degree of risk associated with the project. The ROR primarily reflects the cost of capital. Higher risk projects require a higher ROR to attract the capital investment necessary for continued project operation, expansion or construction. Projects currently operating to meet contracts have low risks and are assigned a 10 % ROR. Nonoperating projects are assumed to have higher risks and are assigned a 15 % ROR. A ROR of 12 % is assigned to operating projects with plans for capacity expansion and with insufficient sales contracts to absorb the added production.

At present, because of existing contractual obligations, uranium production comes from a variety of cost categories (see Figure 3). The cost distribution based on existing contracts is shown for 1990 and 1991. The distribution of cost for all future years is based on economic production without any consideration given for contracts.

With average annual WOCA production of about 25 000 t U expected through 1995, all demand could be met by production with a full cost of under \$78/kg U (\$30/pound U_3O_8). About 80 % of this production is in the under \$52/kg U category. Today there is little or no significant production with a full cost of under \$26/kg U. Therefore most of the production could come from the \$26/kg U to \$52/kg U (\$10 to \$20/pound U_3O_8) range.

The expected rapid growth of required production during the second half of the 1990s will be accompanied by a substantial increase in production cost. Reliance on only All Firm Projects means that by 1998 resources with production costs of between \$104 and \$130/kg U will be required. If All Projects are available, the maximum production cost in 1998 would be in the \$78 to \$104/kg U range (see Figure 4). There is no difference in demand between the high and low demand cases in this period. Therefore the only difference is between the Firm Projects and All Projects. In either case, new production with costs of up to between \$104 to \$130/kg U will be required. Nearly half of the production will have to come from resources with a Full Cost of over \$52/kg U.

For the low demand case, operation of All Firm Projects beyond 2000 means that production centers with Full Costs in the \$130 to \$260/kg U range would be in operation. By relying on All

Projects, the Full Cost of the highest cost segment would not escalate as fast. In the year 2000 the distribution of production from All Projects would be: 36% below \$52/kg U, 31% in the \$52 to \$78/kg U range, 30% in the \$78 to \$104/kg U range and only about 2.5% in the \$104 to \$130/kg U range (see Figure 4).

In 2005 of the low demand case, the distribution for All Projects would be: 26 % below \$52/kg U, 22 % in the \$52 to \$78/kg U range, 25 % in the \$78 to \$104/kg U range, 16 % in the \$104 to \$130/kg U range and 11 % in the \$130 to \$260/kg U cost range. This is equivalent to more than 50 % coming from above the \$78/kg U Full Cost category.

There is little difference in full production costs between the high and low demand cases when demand is met by All Firm Projects. Comparison of Figures 3 and 5 show that the high demand case relies on earlier production of high cost resources to satisfy demand.

Following is a description of the cost distribution of production when All Projects are used to meet the high demand case (see Figure 6). In the year 2000 the distribution is: 33% below \$52/kg U, 29% in the \$52 to \$78/kg U range, 28% in the \$78 to \$104/kg U range, and 10% in the \$104 to \$130/kg U range. About 62% is from the below \$78/kg U Full Cost category.

By 2005 the distribution is: 23% below \$52/kg U, 19% in the \$52 to \$78/kg U range, 22% in the \$78 to \$104/kg U range, 21% in the \$104 to \$130/kg U range and 15% in the \$130 to 260/kg U range. By 2005 only about 42% of the production is from the under \$78/kg U cost category. By 2010 less than 30% of the production is from the under \$78/kg U cost category. About 60% of the production will come from the \$104 to \$260/kg U cost category. By 2015 nearly 70% of the production comes from the \$130/kg U and above cost category.

In summary, within the next nine years the cost of uranium production will change from 80% coming from the under \$52/kg U cost category to only about one third coming from this category. By 2005 and beyond, about two thirds of the production will come from the more than \$78/kg U category. By 2000 the maximum production cost could exceed \$104/kg U and this could climb to above \$130/kg U by 2005.

4.3. Through the Year 2035

For this long term projection the underlying WOCA uranium demand projections are based on different reactor strategies for one nuclear electricity generation projection. This projection is the mid case between the high and low scenarios developed for the Red Book 1990 [3] through 2030 and extrapolated for the time to 2035.

For the entire time horizon, the nuclear capacity is expected to grow from 283 GW(e) in 1990 to 780 GW(e) in 2035. The data set five in years intervals is listed in the following table.

Year	GW(e)	Year	GW(e)
1990	283.0	2015	527.5
1995	306.5	2020	597.5
2000	328.3	2025	670.0
2005	390.0	2030	730.0
2010	462.5	2035	780.0

The uranium high and low demand projections defined in chapter 3.2. show increases from 41 500 t U in 1990 for both cases to 97 000 t U and 65 200 t U in 2035 for the high and low scenario respectively. More detailed listings of the annual demands are provided in Table 3 and 4. Shown are also the cumulative totals of the scenarios: for the high case 3.222 mill and for the low case 2.640 mill t U, or about 18% less than the high case.

The growth rates for both cases are very modest amounting less than 2% p.a. for the high case and to 1% p.a. for the low case, which corresponds to the mid demand case of the study published in 1989 [2].

The supply simulations made in two ways, a) by major supplier country and b) by uranium resource categories, are modelled in such a way, that the demand is filled. The sequence of this process is demand, which fits supply

- from expected future production referred to as committed production (Table 3 and 4) and associated

- non-WOCA supply,

- thereafter, the modelled demand fit from simulated production centres mining uses first the remaining RAR+EAR-I and then EAR-II and Speculative Resources (SR).

The four supply-demand cases including the two demand scenarios and associated supply projections by supplier country and by resource category, are compiled in Figures 7, 8, 9, and 10.

The high and low cases, nearly identical through the year 2000 exhibit the deficit known from previous projections. It amounts to nearly 10 000 t U in 1990, increases to 15 000 t U in 1995 and is reduced by the modelled demand fit reaching a supply demand-balance in about the year 2000. The cumulative total for this period amounts to about 130 000 t U.

To fill this demand in 2000, an increase of supply from nearly all present producers countries is shown (Figure 7 and 8).

TABLE 3. HIGH URANIUM DEMAND SCENARIO

Year	Input to RAPP3 model			demand fit	
	HIGH DEMAND	COMMITTED PRODUCTION	NON-WOCA SUPPLY	FROM RAR+EAR-I	FROM EAR-II+SR
1990	41 500	28 760	2 000	0	0
1995	47 800	27 550	2 500	0	0
2000	55 500	35 000	3 000	18 000	0
2005	62 250	29 730	4 000	27 500	0
2010	69 750	23 440	5 000	41 580	1 500
2015	72 750	16 610	6 000	36 080	14 000
2020	77 000	12 150	6 000	25 000	33 500
2025	85 250	8 700	7 000	21 000	46 450
2030	91 000	4 210	7 000	12 500	67 000
2035	97 000	2 720	7 000	12 500	74 000
TOTAL	3 222 000	808 780	224 500	951 300	1 073 700

TABLE 4. LOW URANIUM DEMAND SCENARIO

Year	Input to RAPP3 model			demand fit	
	LOW DEMAND	COMMITTED PRODUCTION	NON-WOCA SUPPLY	FROM RAR+EAR-I	FROM EAR-II+SPE
1990	41 500	28 760	2 000	0	0
1995	47 800	27 550	2 500	0	0
2000	52 654	35 000	3 000	13 500	0
2005	55 270	29 730	4 000	21 500	0
2010	57 050	23 440	5 000	28 350	0
2015	62 450	16 610	6 000	36 080	4 000
2020	62 120	12 150	6 000	30 230	14 599
2025	63 200	8 700	7 000	23 000	22 250
2030	64 100	4 210	7 000	14 000	39 000
2035	65 000	2 720	7 000	14 000	40 500
TOTAL	2 640 000	808 780	224 500	880 800	542 100

Between 1995 and 2000 the expected future production (committed production in Table 3) augmented by the simulated production (demand fit in Table 3) show a significant increase of several 100% for nearly all suppliers except non-WOCA, Namibia and the USA.

Between 2000 and 2035 the high and low cases show a more differentiated development, although following the same trend. In both cases the termination of supplies from Namibia in 2015 and Niger in 2030 (although a minor supply flow continues) seem important events. The Namibian supply is projected to be replaced by supply increases from Canada, while the Niger share seems to be taken over by increased US supply.

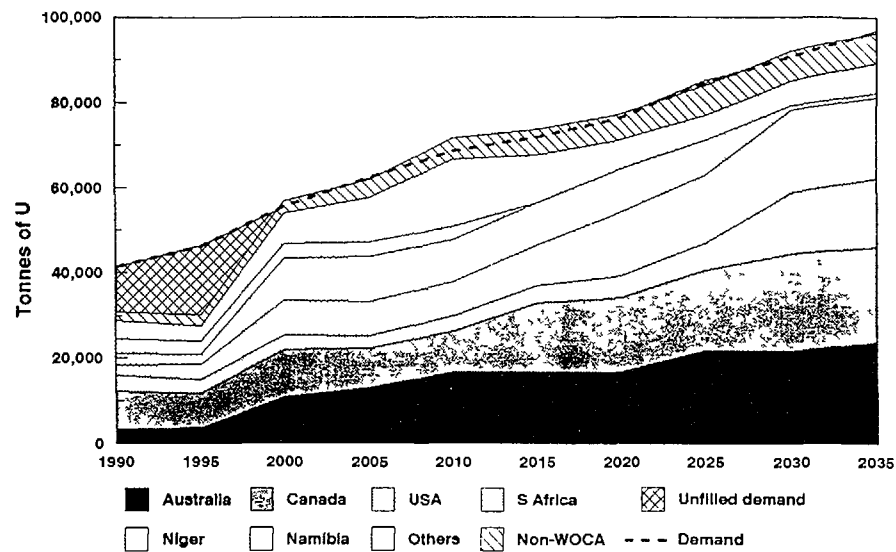


FIG 7 RAPP3 model, WOCA uranium supply-demand analysis — demand fit from various suppliers, high demand case

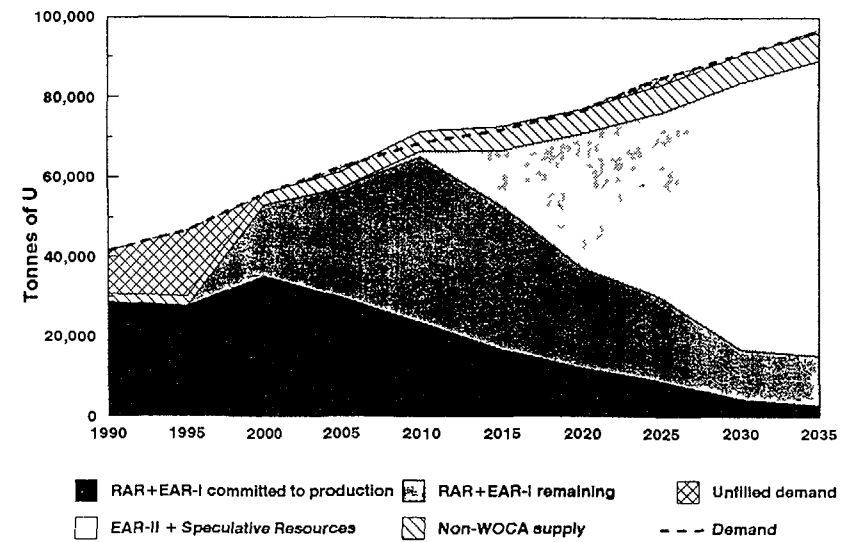


FIG 9 RAPP3 model, WOCA uranium supply-demand analysis — demand fit from various resource categories, high demand case

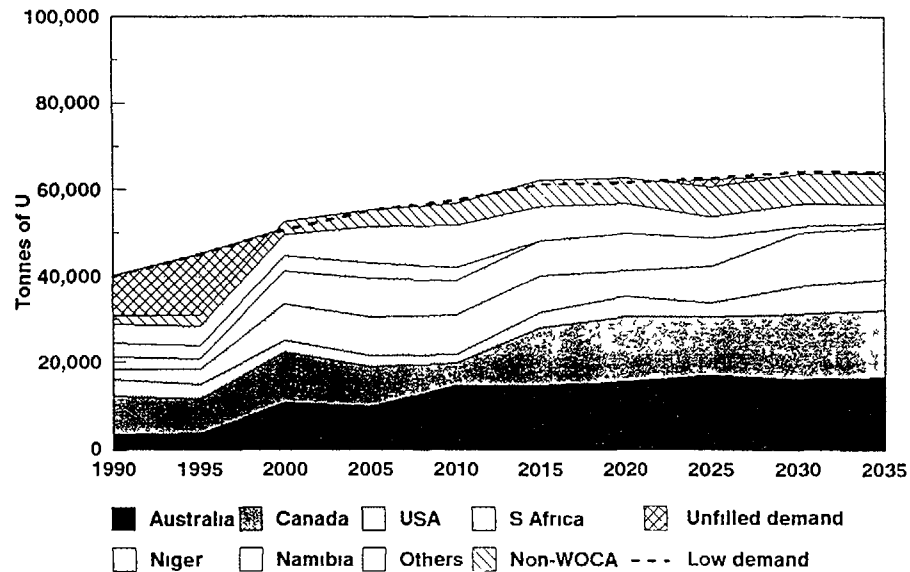


FIG 8 RAPP3 model, WOCA uranium supply-demand analysis — demand fit from various suppliers, low demand case

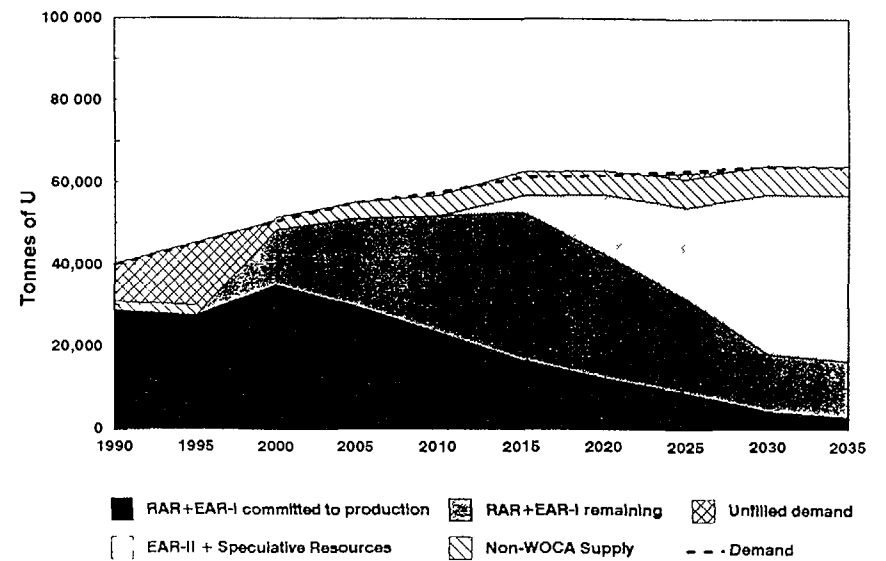


FIG 10 RAPP3 model, WOCA uranium supply-demand analysis — demand fit from various resource categories, low demand case

In both cases the continuation of supply concentration in few countries, which is already the current development, is modelled to increase in spite the constraints of a WOCA supply limit of 30% from any one country. In 2035, about 75% to 80% of the total supply is provided by the four countries Australia, Canada, USA and South Africa as compared to 65% in 1990.

This is in line with the present distribution of uranium exploration, which determines the supply sources for the future and which at present essentially concentrates in Australia, Canada, USA, France and India.

As regards the supply-demand projections by resource categories, reference is made to Figure 9 and 10. They show, that the supplies for the high and low demand cases through 2010 is filled by presently known uranium resources of the RAR and EAR-I categories recoverable at cost up to \$80/kg U. In more detail, the expected future production (referred to as committed production in Figure 9 and 10 and Table 3 and 4) is projected to be augmented by modelled supplies from remaining RAR and EAR-I at fast increasing amounts. For example, between 1995 and 2000 this supply source is shown to grow from zero to 18 000 t for high and to 13 500 t U for the low demand case.

In 2010, the last year for which the projected demand can be filled by supplies from known resources, the expected future production will amount to only 23 400 t U for both cases, (Table 3 and 4) while the remaining RAR and EAR-I contribute 41 580 t U and 28 350 t U respectively.

In the time after 2010 increasing supplies from presently undiscovered resources (EAR-II and SR) are needed as supplies, to take over from equally rapidly decreasing production from known resources. For the high case the undiscovered resources are modelled to supply 1500 t in 2010 to 74 000 t U in 2035 while for the low case the supply in 2030 from this resource is 40 500 t U.

As shown (Figure 9 and 10, Table 3 and 4) the cumulative supplies from undiscovered resources differ significantly for the two demand cases; for the high case they total over 1 million, while for the low case they amount to about 540 000 t U.

This projection implies that as for 2010 significant amounts from today undiscovered resources must be discovered and the preparation for this production be made. According to the RAPP3 simulation, these efforts would take place mostly in Australia, Canada, the USA and South Africa.

5. Comparison of Results

The findings of the supply - demand projections for the different time periods, through 2005, 2020 and 2035 using different methodologies as described above will be summarized and compared in order to arrive at meaningful conclusions.

Through 2005, the supply demand-demand projection shows a continuation of the production gap which started in 1985, under the assumption of supplies limited to presently existing and committed production centres.

The critical point in this time frame as it concerns the depletion of the available WOCA uranium stock is approximately the year 2000. Then supplies from presently not existing and uncommitted production centres are required, which for 2005 are estimated to be approximately 20 000 t U.

The second study concerning the time through 2020 shows in Figure 3, 4, and 5 that the net demand is filled with non-WOCA supplies and drawdown from WOCA stocks in both the low and high demand case, through the year 1998. This is in agreement with the results of the short term study through 2005 as mentioned above.

For the time after 2000 additional supply from potential projects are needed in amounts increasing from 17 000 t U for the low demand and 25 000 t U for the high case in 2005 to 45 000 t and 55 000 t U respectively in 2020 (Figure 4 and 6). The projected 2005 supply from currently firm and potential projects of 17 000 t and 25 000 t U respectively is again in close agreement with the gap projected in the short term projection, which had to be filled from the same presently not existing supply sources.

The study through 2020 also projects the full costs of the total supplies to fill the net demand as defined above (Figure 4 and 5). The full cost distribution for the supplies for selected years has been compiled in Figure 11 and 12 for the low and high demand cases respectively. Both supply curves are very similar for the low and high demand cases and show the following features:

In 1990, the cost distribution is very unfavorable considering the present price level: only about 30 % of the supplies are produced at or below \$ 52/kg U, and a cumulative 60 % is supplied at full costs of or below \$ 104/kg U. The consequences of this fact are being seen today in numerous mine closures in Canada, France, and the USA. The steps being taken at present, find their results in the situation

of 1998, which shows that over 50 % of the total supplies are produced at full costs of \$ 52/kg U and the entire supply at or below \$ 104/kg U;

in 2005, it becomes apparent that this full cost distribution cannot be maintained; the increase of supplies of about 15 % may contribute to the rapid depletion of low cost resources; the consequences are that the portion which can be produced at or below \$ 52/kg U decreased from over 50 % to less than 30 %, while the remainder is being produced at between \$ 52 and 260/kg U;

in 2014/15, the development towards higher full cost continues to a point where only 20 % are being produced at or below \$ 52/kg U and another 20 % are produced at above \$ 260/kg U.

The long term study through 2035 resulted in the projection of supplies by producer country and resource category. The supply projection by producer countries shows, that in order to overcome the supply gap in 2000, all present supplier countries have to increase their production. Significant features (Figure 7) are the projected depletion of the Namibian and Niger resources in 2015 and 2030 respectively. They are being replaced by increasing supplies from other countries such as Canada, South Africa and USA, all having significant presently undiscovered resources. This may explain the full cost increase projected for the years 2014/15 in the medium term study.

In the supply projection by resource category, the known resources including RAR and EAR-I will start to show signs of depletion in about 2010. From then on, undiscovered resources including EAR-II and SR have to be discovered and developed as supplies to fill, in increasing amounts the low and high demand projections. This point, the year 2010, coincides well with the results of the projection through 2020, where in the high demand case the first indications of supply gaps become apparent in the year 2012 and reach the level of about 5000 t in 2015 and close to 20 000 t U in 2020.

In summary, the following findings of the three supply-demand projections are significant:

a) the present production gap is being filled with material from available stocks through about 2000, then supplies from presently unplanned or potential projects must be available; these required supplies increase rapidly e.g. from virtually zero to 17 000 t and 25 000 tonnes U for the low and high demand cases in 2005 to 45 000 and 55 000 t U respectively in 2020;

b) the requirements from potential production centres after the year 2000 are projected to be accompanied by a significant increase of the full costs; the low cost share decreased from over 50 % at the end of the century to less than 30 % in the 2005 and a certain share of the production is expected to be produced at costs of up to \$ 260/kg U;

c) the known resources of the RAR and EAR-I categories recoverable at costs of up to \$ 80/kg U show a beginning depletion in 2010. At that point in time, either higher cost known resources or lower cost undiscovered resources of the EAR-II and SR categories must be developed, to fill the projected supply gaps.

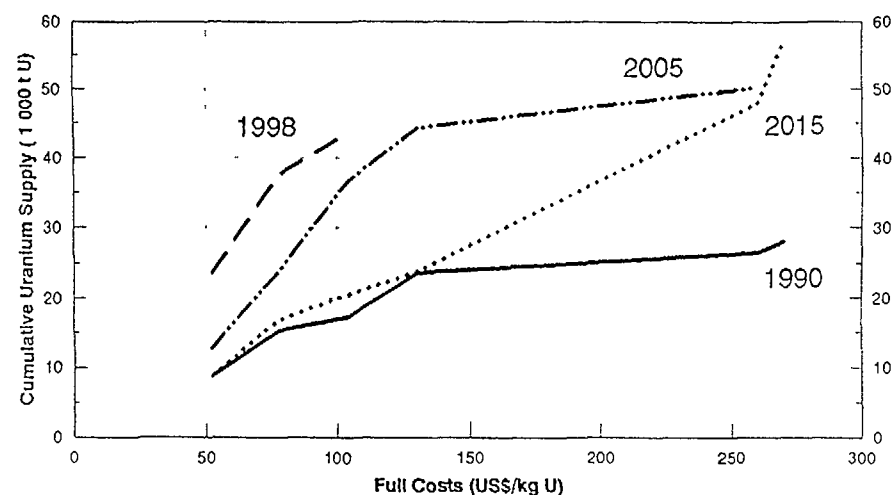


FIG. 11. Supply-cost curves — low demand (all projects).

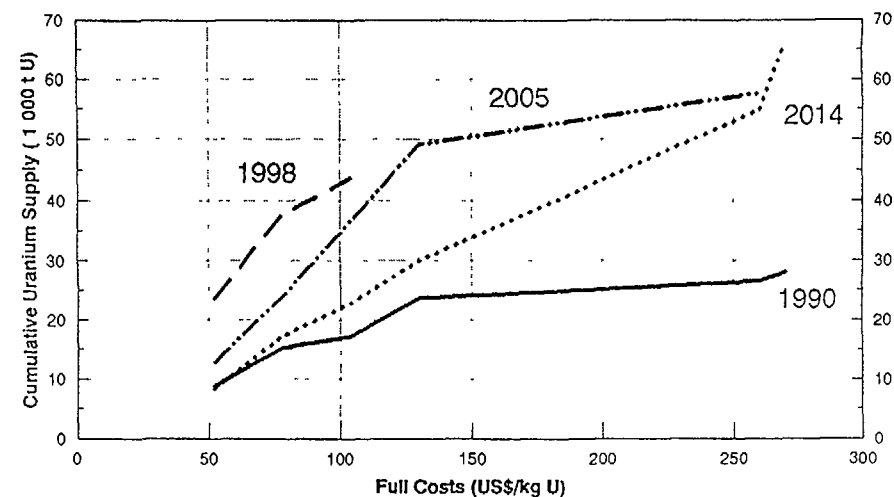


FIG. 12. Supply-cost curves — high demand (all projects).

6. Conclusions for Exploration and Uranium Resource Development

Based on the above findings there are a number of obvious conclusions as they concern uranium exploration and resource development. An important aspect in these aspects is the lead time needed to achieve a certain result. This has to be taken into account in the planning of future production and supply.

At present the development in the uranium industry is directed towards a decrease of the full cost as shown above. This will ensure a low cost supply through the end of the century but will also deny the industry the necessary returns for the development of new production centres over and above those which are firmly planned today. These will be needed with increasing cumulative capacities at the turn of the century.

The capital requirements for a financially exhausted industry will be significant and it is believed that this capital requirement for the future investments will increase the concentration among uranium mining companies.

The low cost known resources are expected to show signs of depletion in about 2010 even for the low demand case. Then, higher cost known resources must be prepared for production or lower cost undiscovered resources be developed. As the study includes known resources in countries where they will never be mined and never made available as supplies, the realistic resource base may be different and both the producer and consumer should make their own assessment. Exploration may have to be stepped up in the near future to discover and develop resources, which can sustain the production centres needed in the early parts of the next century.

Despite the inclusion of potential production centres in the supply-demand projections there will be a production gap in the high demand scenario in the year 2015, and this gap is expected to grow within 5 years to 20 000 t/year. This supports the call for resource development made above, but also shows that in about 25 years the production capabilities which include both the technical capacity and the resources required will not be sufficient to meet the estimated high demand which still is a modest projection. This indicates a stress on both resources and production despite a projected increase of the full production costs to over \$ 260/kg U for a considerable portion of the supply.

In summary, although this study cannot present any hard future, after all it is the future which is dealt with, a number of observations can be made on resource availability and production cost projection for two very modest demand scenarios. The study shows, that there is a need for a constant monitoring and evaluation of the demand situation both for industry and international organisations. The objective of such an exercise is to provide early signals, which will have to prompt the industry

to undertake the necessary actions. For this task, the industry must be prepared both in terms of human and financial resources. Unfortunately, both are in the process of being depleted.

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APPENDIX

THE URANIUM SUPPLY ANALYSIS (USA) SYSTEM

Nuclear Assurance Corporation

1. USA System Description

The Uranium Supply Analysis (USA) System includes a comprehensive data base of technical and financial information on uranium production collected by Nuclear Assurance Corporation (NAC) over the last 20 years. The USA System is a PC based interactive computer system for analysis of the uranium industry. The computer system has been developed and operated over the last 10 years.

The USA System has a data base containing available technical and financial information for up to 150 of the world's largest, existing, planned, and potential uranium production centers. The focus of the system is commercially based production centers. A production center normally consists of one processing center capable of producing uranium concentrate together with any and all contributory mines. The system includes all projects for production of uranium concentrate with attributable reserves. The USA System normally has a 15 year forward confidence range for analysis.

The uranium resource data base of the system includes known (identified) resources. For the purposes of the IAEA consultants meeting, the January 1, 1991 resource is 1 875 384 t U. This total is an aggregate made up of the resources contributory to individual production centers. For practical reasons, analysis of each production center is limited to a maximum reserve life of 30 years. Reserves normally include both RAR and EAR-1. No undiscovered resources (i.e. EAR-II or Speculative) are included in the USA System reserve base. Reserves are recoverable, based on appropriate adjustments of inground reserves for losses during mine extraction and processing.

The USA System includes primary uranium, as well as by-product and co-product production centers. Project capacity is based on the design or planned level. The USA System includes the cost of production for each production center, based on an average over the life of project (i.e. units of production basis). Cost information includes capital and operating costs which combine to equal production cost. This information is provided based on both forward and full cost. A rate of return is assigned to each project according to the level of risk associated with the project. The assigned rate of return is either 10%, 12% or 15%. For analysis reserves are classified project-by-project and distributed by production cost into categories of US\$20, \$30, \$40, \$50, and \$100 per pound U_3O_8 ; approximately equivalent to the IAEA categories of \$80, \$130 and \$260 /kilogram U.

2. Definitions of terms and parameters used in the USA System

Constructed Projects: Those production centers that have achieved some level of production in the past, though they may currently be inactive.

Delayed Projects: Those PLANNED PROJECTS for which commercial operation has been postponed, either by the operator or by the market projections.

Economic Production: The individual annual production levels resulting from a dynamic market balancing routine based upon competitive economic theory, technical limitations, and the total market demand requested by the user. The production projected to meet demand, regardless of contracts, with the lowest cost Uranium selected first and higher cost capabilities deferred. The most economic possible market conditions with all requirements, including contracts, being met by the lowest cost producers. This is premised upon high cost producers shutting down and procuring material from lower cost producers to meet their contractual obligations.

Firm Projects: All production centers for which plans as to production method, production level and first commercial operation date have been announced.

Operating Projects: Those production centers that have achieved some level of production in the past and are currently producing or are expected to produce (depending upon the results of the market balance subroutine).

Planned Production: The production schedule as announced by each given producer. No assumptions are made as to the duration of deferrals and shutdowns. These data are maintained by NAC and can be changed by the user only in YOUR OWN DATA BASE.

Potential Projects: Those resource deposits for which no production plans have been announced that NAC feels can and will come into production within the next thirty years.

Production Capacity: The expected maximum number of short tons U_3O_8 that can be produced in any given year during the life of the plant. Also referred to as design or nominal capacity. A data item input to the system and used only as a production limit.

Shutdown Projects: Those production centers that have achieved some level of production in the past and are currently not producing due either to the plans of the operator or the results of the market balance subroutine.

The following provides a listing and definition of financial parameters for worldwide production centers. All of the production cost figures are given in dollars per lb U_3O_8 in constant 1991 dollars and reflect before-tax costs. The

production cost estimates are based on the earliest attainable start-up year and maximum capacity. Descriptions of the column headings for the production cost parameters are as follows:

Capital Cost/lb U_3O_8 : Total capital cost per lb U_3O_8 on units-of-production basis, which does not include return on investment. Cost components included in the capital cost figures are acquisition/exploration, mine development, mill construction and environmental/infrastructure.

Operating Cost/lb U_3O_8 : Total operating cost per lb on units-of-production basis, which does not include return on investment. Cost components included in the operating cost figure are mining, hauling, milling production/property tax and environmental.

Production Cost/lb U_3O_8 : Sum of capital and operating cost. Figures in dollars per lb U_3O_8 on a units-of-production basis, which does not include return on investment.

ROR Percent: Rate of return in percent used to determine the forward marginal cost and full recovery cost. The rate of return used varies depending on the status of the production center.

Forward Cost: Forward production cost with return on investment based on ignoring sunk costs (expenditures prior to the current year) determined by a discounted-cash-flow, rate-of-return (DCFROR) analysis using the specified rate of return. Provides an estimate of the minimum revenue needed by a producer to continue operating or continue to move toward operation based on recovering all future expenditures with a return on investment.

Full Recovery Cost: Full recovery production cost with return on investment based on sunk and forward costs determined by a DCFROR analysis using the specified rate of return. Provides an estimate of the revenue needed by a producer to continue operation or continue to move toward operation based on recovering all past and future expenditures and a return on investment.

3. 1990 USA System Resource Inventory

	Low Cost RAR and EAR-I (in 1 000 t U) In 1/1/90 US\$ per kgU			
	52	78	104	130
Australia	346.2	54.5	61.5	36.9
Canada	326.5	24.3	84.2	0.0
U.S.A.	114.3	119.1	42.4	24.9
Other	<u>150.4</u>	<u>290.1</u>	<u>243.2</u>	<u>119.0</u>
Total	937.4	488.0	431.3	180.0

Grand Total: 2 036 700 t U

THORIUM DEPOSITS AND THEIR AVAILABILITY

F.H. BARTHEL

Bundesanstalt für Geowissenschaften und Rohstoffe,
Hannover

F.J. DAHLKAMP

Liessem/Bonn, Germany

Abstract

Thorium deposits are known from a number of countries and occur in a variety of geological environments. This paper gives a summary of the authors attempt to classify thorium deposits independent of their economic significance. The deposits which occur in igneous, metamorphic and sedimentary rocks are further subdivided according to their geological setting and type of host rock.

Significant thorium resources are found to be associated with carbonatite rocks (40 % of total known resources), mainly in those rich in niobium and rare earth elements. The second important type are vein deposits (31 % of known resources) where thorium occurs both in separate thorium minerals or in rare earth minerals. Next to vein deposits placers (unconsolidated beach placers and consolidated paleo-placers) account for 19 % of the known resources. Within this type thorium bearing minerals (mainly monazite) can be recovered as by-product of heavy mineral production (cassiterite, ilmenite etc.). Other deposits, such as pegmatites or alkaline intrusives are of minor importance. In a speculative order of economic importance the deposits can be classified as follows (1) unconsolidated placers, (2) consolidated paleo-placers, (3) veins, (4) carbonatites.

The total world annual output of thorium is estimated at around 400 t ThO_2 out only a portion is presently used. If commercial demand would increase due to wider application, e.g. in thorium fueled nuclear power plants annually around 2000 t ThO_2 could be made available by extraction from monazite concentrates recovered as by-product of heavy mineral mining.

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1. Introduction

Thorium is a lithophile element which is concentrated during magmatic differentiation. Because of the large ionic radius and the tetravalent charge thorium is enriched in residual melts and solutions. Thorium mineralizations are thus more abundant in late magmatic crystallizates, such as leucocratic granites, alkaline magmatic rocks, pegmatites, carbonatites, and hydrothermal veins, where it occurs in accessory minerals (e.g. zircon, thorite, thorianite, sphene and others). Thorium has a tendency to be concentrated in high-temperature fluids. Most of the thorium-bearing minerals are stable under weathering conditions, thus they can be transported over long distances as heavy minerals. They can be concentrated as placers at the banks of rivers or finally as coastal placers [1].

2. Classification of Thorium Deposits

The proposed scheme for the classification of thorium deposits is mainly based on published literature. As detail research on the metallogenesis and formation of thorium deposits is sparse compared to other metals the subdivision of the various thorium mineralizations into a valid and concise classification scheme can be only provisional. Nevertheless, the scheme proposed by the authors may serve as a guideline for the identification of thorium mineralizations and hopefully stimulate further research [2].

As shown in table 1, thorium occurs in a variety of deposits. For genetical reasons a division into igneous, metamorphic and sedimentary origin was made. For each of the three major types a subdivision can be made according to genetical differences or different host rocks [2].

The individual types of deposits are described briefly.

2.1 Igneous Intrusive Deposits

Thoriferous mineralizations in igneous rocks which may reach concentrations to call them deposits can be divided into three principal groups

Table 1: Tentative Classification of Thorium Mineralizations Deposits and their Geologic Setting

Type Origin	Subtype of deposit Host rock	Examples
igneous	carbonatites	Palabora, S. Africa; Araxa, Brazil; Mountain Pass, USA; Pass, USA; Telemark (Fen), Norway;
	syenites	Langesund, Norway;
	alkaline rocks	Poços de Caldas, Brazil;
	peralkaline rocks	Ilímaussaq, Greenland; Saima, China.
	granitic rocks	Jos Plateau, Nigeria,
	alaskitic and leucocratic granites	Rössing, Namibia; Kyzylomul/Tiënsan, USSR
igneous	volcanic and subvolcanic rocks	
	- acidic volcanics	Bear Lodge Mtns/Wyo., USA;
	- alkaline volcanics	Latium, Italy;
	- diatremes	Hicks Dome/III., USA
	pegmatites	Bancroft/Ont., Canada
	hydrothermal veins associated with	
igneous	- alkaline batholiths (Na or K-rich granites, syenites)	Central City and Wet Mountains, Colo., Boken Mtn., Alaska, USA;
	- carbonatite stocks	Steenkampskraal, S. Africa; Iron Hill-Gunnison, Colo., USA;
	- alkaline volcanics	Rexspar, Canada;
	- unknown origin (perhaps metamorphic segregations)	Grenville Province, Canada; Lemhi Pass, Montana, Idaho USA
	migmatites (pegmatitic-pegmatoid)	Travancore, India;
	thoriferous metasediments	
metamorphic	- charnockite	Sri Lanka; Sierra Leone; USA
	- pyroxenite	Fort Dauphin, Madagascar
	contact metamorphics and replacements (skarn, hornfels)	Mary Kathleen, Australia
	placers and residual concentrates (+ unconsolidated)	
	- coastal (littoral) placers	Egypt; Bahia, Brazil; Kerala, India
	- fluvial (alluvial) placers	Zimbabwe; Zaire; Nigeria; N + S Carolina, USA
sedimentary	- residual (eluvial) placers	Rio Grande do Norte, Brazil
	paleo-placers (consolidated)	
	quartz pebble paleo-conglomerates	Blind River, Canada; Sierra de Jacobina, Brazil
	- other ancient placers	Bighorn Mtn/Wyo., Marquette/Mich., USA; Musgrave Block, Australia; Roraima, Guyana-Venezuela
	other sediments	
	- dolomites	McLean Bay/NWT, Canada
	- black shales	Northern Pakistan

- occurrences in carbonatites
- occurrences in rocks of the syenite family (including alkaline and peralkaline types)
- occurrences in granites sensu lato

It is common to all types of deposits and occurrences in igneous intrusive rocks that thorium is erratically distributed, mainly in accessory minerals [3].

2.1.1 Carbonatites

Carbonatites are characterized by low silica content, high carbonate (> 50 %) content carrying minerals with rare earth elements, Nb, Ti, Cu, Zr, Th along with fluorite and phosphate [4,5]. Th enrichments are mainly found in carbonatites with higher contents of Nb and REE. Some typical examples where Th is recoverable as a by- or co-product are Araxa in Brazil (Nb), Mountain Pass, California (REE) and Palabora, Rep. of South Africa (Cu). Other examples are the carbonatites of Fen (Norway), Alnö (Sweden), Lueshe (Zaire) [2,13].

The thorium content of carbonatites in general is in the range of few hundred to few thousand ppm.

2.1.2 Syenites, Alkaline and Peralkaline Rocks

In general syenites and rocks of the alkaline and peralkaline family are enriched in Th, normally in the range from 30 to 100 ppm Th [6]. Only few examples have contents of more than 1000 ppm, like rocks of the alkaline complex of Poços de Caldas/Brazil (up to 2 % Th), or nepheline syenite of Ilimaussaq/Greenland (0,45 % Th). In some metasomatically altered rocks of alkaline and peralkaline composition the Th-content reaches few 100 ppm (Fen district, Norway) [7].

2.1.3 Granitic Rocks

Granites have Th-contents averaging 5 to 80 ppm, but Na- and K-rich types may have accumulations of several 100 ppm Th, e.g. the riebeckite granite of the Jos Plateau, Nigeria (200 to 300 ppm Th) along with enrichments of Nb. Similarly, in alaskitic granites, e.g. in the Kyzylompul Massif in Tianshan, USSR, the Th-content reaches 40 to 60 ppm [8,9].

2.2 Igneous Effusive Deposits

In effusive volcanic rocks no economic deposits of thorium have been found. In strongly acidic volcanic rocks enrichments of few 10 ppm have been observed. Alkaline volcanic rocks, e.g. in the Latium district, Italy enrichments of up to 240 ppm Th have been observed [10].

2.3 Igneous Epigenetic Deposits

2.3.1 Pegmatites

Certain types of pegmatites are enriched with thorium and other metals. In general only erratic enrichments in pockets or lenses can be observed. Examples were thorium occurred in considerable quantities are the uranium-bearing pegmatites at Bancroft/Canada and Mandrare in Madagascar [2,11].

2.3.2 Vein deposits

Vein deposits constitute an important type of thorium deposits because they often are of high grade, averaging several tenths of a percent and sometimes several percent thorium [1,2,3]. Reserves, however, are highly variable, ranging from a few tons to several thousand tons of thorium. In the past, vein deposits yielded an essential portion of the world's thorium production, particularly from mines in South Africa and USA [3,8].

Thoriferous vein deposits of are composed of a variety of minerals. Mineralogical composition and abundance of distinct gangue minerals permit a tentative classification into the following types [2,12]:

- quartz-alkali feldspar-iron oxide veins
- quartz-barite veins
- barite-fluorite veins
- calcite veins
- apatite-quartz veins
- carbonate veins associated with carbonatites.

Characteristics of thoriferous veins [13]: Quartz, and microcline, calcite and/or dolomite are common gangue minerals. Albite may be present, as well as biotite and muscovite. Barite, apatite, and fluorite are often observed. In many veins iron oxides like

limonite and hematite are abundant, in some magnetite and ilmenite. Rutile and other titanium oxides may be present in smaller amounts. Sulfides, mainly pyrite, are common in some veins. Rare earth elements are regular constituents and in some cases may amount to several percent. Thorium occurs as thorite and thorianite or in association with rare earth elements e.g. in monazite, brockite, bastnaesite, xenotime. Thorite is the most abundant thorium mineral, followed by monazite. It is remarkable that uranium is incorporated in uranothorite-uranothorianite or complex refractory minerals, and only exceptionally occurs in distinct uranium minerals like uraninite and pitchblende [2].

Vein mineralization occurs in shear and breccia zones, fractures, and small joints. Most of the veins form tabular bodies, some are lenticular or consist of vein-like impregnations or disseminations in brecciated country rocks. The dip is generally steep.

Veins with thorium mineralization are known to extend in length from a few meters to approximately 2 km, and in width from a few cm up to 20 m. Their depth may reach several hundred m.

Examples of hydrothermal vein deposits include: Lemhi Pass, Montana - Idaho (quartz-feldspar-iron oxide veins), Wet Mountains, Colorado (quartz-barite-hematite veins), both USA, Steenkamp's Kraal, South Africa (apatite-quartz veins), Kizilcaören, Turkey (barite-fluorite veins) [2,8,12,14].

2.4 Metamorphic Deposits

Limited information is available on thorium deposits of metamorphic origin. In principle, this type of deposit consists of thorium concentrations in metamorphosed or pyrometasomatized rocks, in which the thoriferous minerals are distributed in fractures or joints along schistosity planes or disseminated in distinct zones [15]. Thorium-hosting rocks include anatexites and migmatites, pyroxenites, gneisses and schists, and contact metamorphic rocks such as skarn, hornfels, and marble. In several cases, the mineralization is located in metamorphic rocks near contacts of granitic, syenitic, or pegmatitic intrusives [2].

All the reported metamorphic thorium occurrences are more of scientific than economic interest. The thorium-bearing minerals are erratically distributed, grades are rather low, in the order

of 10 to 100 ppm ThO₂, and reserves are negligible. Principal thorium minerals are monazite, thorite, and uranothorianite, but much of the thorium may be contained in accessory minerals.

Examples of thorium mineralization [2, 11, 15]:

Southern Travancore, India (thoriferous monazite in pegmatitic-migmatitic leptynite),

Sri Lanka (Th-rich monazite in graphitic shear zones in charnockitic rocks),

Sierra Leone (monazite-bearing migmatites),

Andrare River, Madagascar (uranothorianite in pyroxenite lenses within schists and gneisses),

Mary Kathleen, Queensland, Australia (contact metamorphic metasomatic U-rare earth element deposit),

Grenville Province, Canada (Th-bearing veinlike segregations in metamorphic rocks).

2.5. Sedimentary Deposits

Thoriferous occurrences in sedimentary rocks may be divided into three principal groups [1, 2, 15]:

1. Occurrences in unconsolidated placers
2. Occurrences in consolidated paleo-placers
3. Occurrences in other sediments

2.5.1 Occurrences in Unconsolidated Placers

According to their origin this type can be divided into

- residual or eluvial placers
- fluvial or alluvial placers
- coastal beach (littoral) placers

The most important enrichment of thorium in unconsolidated placers are found in coastal beach deposits [8]. They contain accumulations of heavy minerals, such as cassiterite, ilmenite, rutile, zircon, chromite, monazite, and others. The most common thoriferous mineral is monazite, less common are thorite, uranothorite and uranothorianite. Monazite contains generally between 8.8 and 10.5 %

Th and rarely up to 25 % Th. It has a specific gravity of 4.8 to 5.5 g/cm³ due to its varying composition. Monazite belongs to the light fraction of the heavy minerals and occurs in association with ilmenite, rutile, zircon, magnetite, garnet and sillimanite [16].

Beach deposits are considerably more extensive in area and volume and are higher in grade than fluvial placers. Beach placers extend for as much as several km in length and several hundred m in width. The ore grade may be up to 10 % ThO₂ and reserves can reach ten thousand tonnes of contained thorium. Beach deposits contain about 20 % of the world's resources of thorium.

Fluvial placer deposits are smaller in size, less uniform in composition, and more erratic in grade than beach deposits. Eluvial concentrations generally are of subeconomic magnitude in terms of tonnage and grade as well [2].

Examples of beach and fluvial placer deposits include Kerala, India, northeastern Brazil; North and South Carolina, USA [16,17].

The largest known beach placers are located along the west coast of India in the state of Kerala. They have been the world's principal source of monazite since early in this century. The deposits originated by deep lateritic weathering of a variety of monazite-bearing source rocks followed by natural sorting of the heavy minerals in coastal beach sands. Reserves amount to 4,000,000 t of monazite. The monazite contains 7 to 9.2 % Th [2].

Other extensive beach placers are located along the northeastern coast of Brazil. They differ from those of India in that the majority of the monazite sands are found in elevated beaches and bars behind the present coast line. Brazilian beach sands contain about 20 to 40 % heavy minerals of which 2 to 5 % is monazite containing between 4 to 5 % thorium. Less important beach placer deposits occur in Egypt, Sri Lanka, Korea, the United States, and in other countries [2].

Fluvial placer deposits are found in Carolina and Idaho in the United States, in Brazil, in many parts of Africa, and in the USSR.

2.5.2 Occurrences in Consolidated Paleo-Placers

Consolidated and fossilized placers which were formed by the same processes as modern placers may contain substantial amounts of thoriferous minerals. Two types can be distinguished [2]:

- Quartz pebble paleo-conglomerates of early Precambrian age, often containing detrital uranium oxide minerals,
- ancient placers of Proterozoic and younger age and virtually free of detrital uranium oxide minerals.

The quartz pebble paleo-conglomerate type is of economic significance since thorium can be extracted as a by-product of uranium and rare earth elements.

2.5.3 Occurrences in Quartz-Pebble Paleo-Conglomerates

The thorium and uranium host rocks is an oligomictic quartz-pebble conglomerate with a quartzitic matrix rich in pyrite. Dominant thorium-bearing minerals are detrital monazite, uraninite-uranothorianite, uranothorite, xenotime, and brannerite. The minerals commonly are matrix constituents of well-sorted conglomerate beds developed within basal depressions, possibly paleo-channels in the Archean-Lower Proterozoic basement surface, and deposited by braided and interfingering streams. Characteristically, the uraniferous conglomerate deposits are restricted to strata of uppermost Archean-lowermost Proterozoic age, i.e. to a time prior to the oxygenation of the atmosphere [2,11]. Examples include Elliot Lake, Canada and Sierra de Jacobina, Brazil.

Large resources of thorium (associated with uranium) occur in the Elliot Lake/Agnew Lake district, Ontario, Canada, where thorium and rare earth elements have been recovered as by-products of uranium in the past. The thickness of the conglomeratic layers reaches 1.5 to 10 m, and their lateral extension is 100 to 1000 m. The uranium ore contains about 0.044 % Th. The ratio of U to Th varies, in general in the producing deposits about 2:1.

2.5.4 Occurrences in Ancient Placers

This type is widespread and occurs in or adjacent to many crystalline massifs. In principal, it is a fossilized stream placer consisting mainly of monazite and other heavy mineral concentrations

in fluvial and littoral sandstones and conglomerates of similar dimensions as the other placers [2].

Examples include the Deadwood Formation, Bighorn Mountain, Wyoming, and the Goodrich Quartzite, Michigan, USA; Musgrave Block, Australia; Roraima, Guyana/Venezuela.

2.5.5 Other Thorium Occurrences in Sediments

Dolomitic limestone of Precambrian age is the host rock for thoriferous minerals at the McLean Bay, Great Slave Lake, NWT, Canada. It is uncertain whether this enrichment of uranium and thorium (pitchblende and monazite) is of syngenetic or epigenetic origin. The radioactive minerals are associated with concentric structures believed to be algae. The thorium content is estimated at about 0.025 % [2].

Black shales such as the Chattanooga Shales, USA, or the Swedish Kolm shales have low thorium contents (around 10 ppm). Graphitic black shales in northern Pakistan reportedly contain 100 to 200 ppm Th, with fault controlled enrichments of about 0.35 to 0.79 % Th.

3. Thorium Resources

3.1 Geographic Distribution of Thorium Resources

Thorium deposits and occurrences are described from the following countries.

Europe: Finland, Greenland, Norway, Spain, Sweden, Turkey

America: Argentina, Bolivia, Brazil, Canada, Chile, Colombia, Uruguay, USA

Africa: Egypt, Kenya, Liberia, Madagascar, Malawi, Mocambique, Morocco, Namibia, South Africa, Sudan, Tanzania, Zaire, Zimbabwe

Asia: Bangla Desh, India, Indonesia, Iran, Korea, Malaysia, Pakistan, Sri Lanka, Thailand

Australia and Oceania: Australia, New Zealand.

Thorium deposits in carbonatites are of major importance among the known resources, mainly in North and South America, parts of Africa, in Norway and Finland. Vein deposits account for significant resources in South Africa, the United States of America and Turkey. A large portion of the known resources are in beach sand deposits of monazite. Reserve figures have been reported from Australia, Egypt, India, and Liberia. Beach sand deposits not yet assessed are known from Brazil, the United States of America, Bangla Desh, Korea, Mada-gascar, Pakistan, South Africa, Sri Lanka and Uruguay. Thorium resources of magmatic or anatectic origin, present mainly in igneous alkaline intrusion, have been investigated in Brazil, Greenland, Iran, and South Africa. Unassessed deposits of this type exist in Canada, Finland, Morocco and other countries. A large amount of Canada's thorium resources is associated with quartz-pebble paleoconglomerates mined for uranium. Thorium is of minor importance in the South African quartz-pebble paleoconglomerates [2].

Since systematic exploration for thorium has been conducted on a limited scale and in restricted areas only, most thorium discoveries are a by-product of exploration for uranium and for placer deposits, and it has to be assumed that a large part of the world's total resources of thorium is not yet discovered, or at least not yet explored.

Quantitative figures of thorium resources are reported from only a limited number of countries. The resource figures given in Table 2 are primarily based on assessments of the world's uranium resources carried out since 1965 by the Nuclear Energy Agency, in conjunction with the International Atomic Energy Agency. The last report was issued in December 1986. Secondly, the compilation of the world energy resource data published in 1980 for the 11th World Energy Conference was used to obtain figures for thorium resources and production. The world total thorium resources given in table 2 refer only to countries addressed otherwise as "World Outside Centrally Planned Economies Area" (WOCA), since no reliable figures are available for countries in Eastern Europe, USSR, and People's Republic of China.

Resource figures used in table 2 are compiled from different sources, but are based mainly on the last issue of NEA/IAEA (1986) [18].

Table 2

WOCA Thorium Resources Compiled from Different Sources.
in 1000 t Th

area	resource category			% of WOCA total resources
	reasonably assured resources	additional resources	total resources	
Europe				
Finland	-	60	60	
Greenland	54	32	86	
Norway	132	132	264	6.4
Turkey	380	500	880	21.4
Europe, total	566	724	1290	31.4
America				
Argentina	1	-	1	
Brazil	606	700	1306	31.8
Canada	45	128	173	
Uruguay	1	2	3	
USA	137	295	432	10.5
America, total	790	1125	1915	46.6
Africa				
Egypt	15	280	295	7.2
Kenya	n.e.	n.e.	8	
Liberia	1	-	1	
Madagascar	2	20	22	
Malawi	n.e.	9	9	
Nigeria	n.e.	n.e.	29	
South Africa	18	n.e.	115	
Africa, total	36	309	479	11.7
Asia				
India	319	-	319	7.8
Iran	-	30	30	
Korea	6	n.e.	22	
Malaysia	18	-	18	
Sri Lanka	n.e.	n.e.	4	
Thailand	n.e.	n.e.	10	
Asia, total	343	30	403	9.8
Australia	19	-	19	
total WOCA (World excluding Eastern Europe, USSR, PR China)	1754	2188	4106	100

n.e. = no estimates are given for resource category.

Table 2 lists 23 countries with assessed thorium resources, of which six, Brazil, USA, India, Egypt, Turkey, Norway, accumulate more than 80 % of the known resources. Brazil has the largest resources accounting for nearly one third of the world's known total, followed by Turkey with approximately 20 % and the United

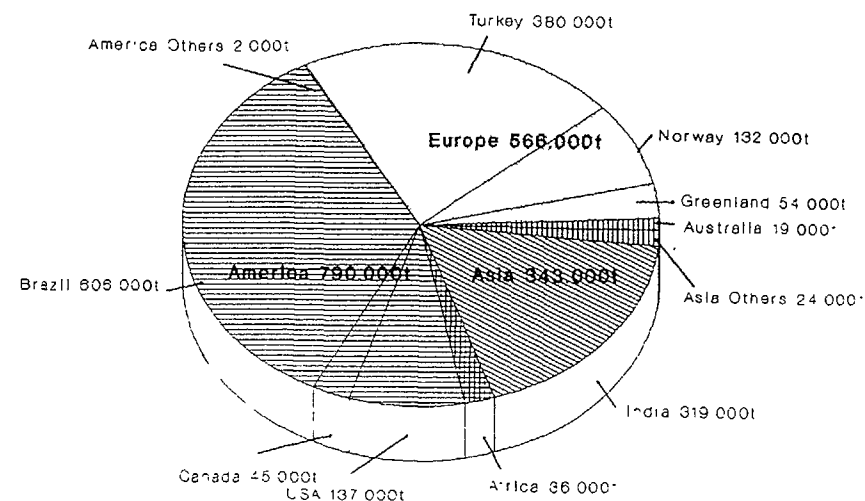


FIG. 1. Reasonably assured resources of thorium (total WOCA 1 754 000 t).

States with 10 %. Fig. 1 and 2 show the distribution of reasonably assured resources and total resources resp.

3.2 Geologic Metallogenetic Distribution of Resources

The geologic metallogenetic distribution of the thorium resources is based on the classification of deposits described in chapter 2. Main types of deposits include carbonatites, veins, placers, alkaline intrusive complexes, quartz-pebble paleo-conglomerates.

The first three types host the bulk of known thorium resources (Tab. 3). Carbonatites account for about 40 % due to large occurrences in Brazil and Norway. Vein deposits, mainly found in Turkey, South Africa, and in the United States, have a share of 31 %. Placer deposits in beach sands of India, Egypt, and of other countries account for 19 %. Occurrences in alkaline intrusive complexes and in quartz-pebble paleo-conglomerates each contribute 4 %. Other types of deposits such as pegmatites contain only 2 % of the known resources [2].

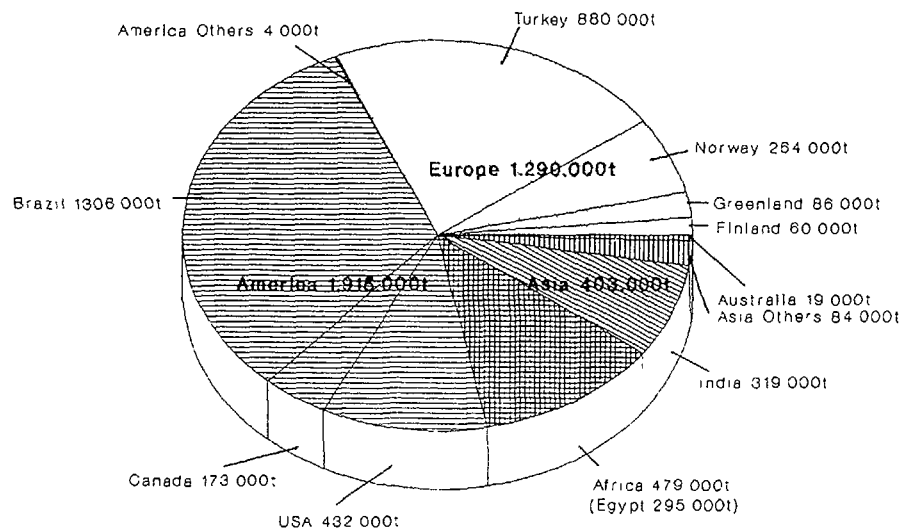


FIG. 2. Total resources of thorium (total WOCA 4 106 000 t).

Table. 3: Distribution of known thorium resources by type of deposit (see also fig. 3).

Carbonatites	40 %
Vein deposits	31 %
Placer deposits	19 %
Alkaline intrusive deposits	4 %
Quartz-pebble conglomerates	4 %
Others	2 %

4. Thorium Production and Availability

World-wide demand for thorium is limited, consequently there is only minor production. Current production is predominantly based on monazite recovered from placer deposits which are exploited primarily for titanium, tin, rare earth elements, and zirconium. Monazite is separated from other heavy minerals (ilmenite, rutile, garnet, zircon, magnetite) by selective flotation with

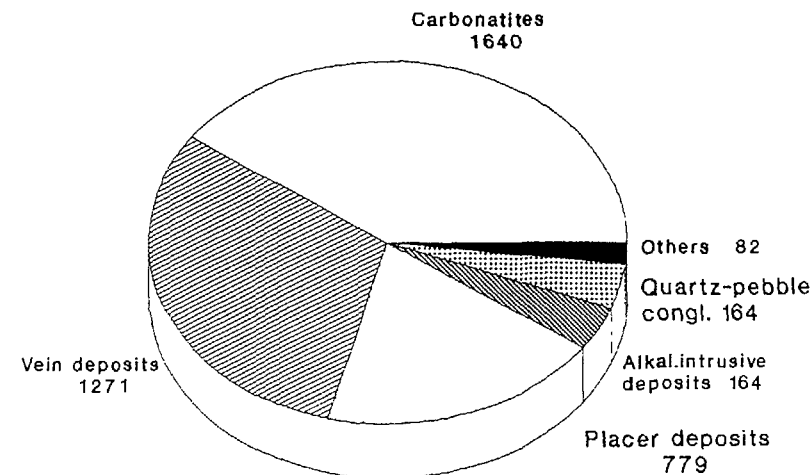


FIG. 3. Distribution of known thorium resources by type of deposit (1000 t thorium).

actinols (mixture of oleine and linoleine acid). By selective flotation, a concentrate of 75 % of monazite can be obtained, reaching a recovery of more than 90 %. The amount of thorium theoretically available from current monazite mining is in excess of present demand. Hence, in many countries, thorium is not recovered but is stored as "waste" or in tailings dumps.

4.1 Availability

In the United States, beach placers with monazite are mined in Florida, however, thorium is presently not produced for sale. In Brazil, thorium concentrates are produced during rare earth extraction and stored. In Australia, monazite is extracted from beach sands containing the equivalent amounts of 500 to 1000 t Th/year, but there is no recovery of thorium. In South Africa, monazite is recovered. Thorium production from heavy minerals concentrates has been reported from India. Monazite and xenotime as main thorium-bearing minerals are produced in Malaysia and Thailand as by-products of tin mining. Thailand has a stockpile of monazite concentrates of more than 5000 t at a grade of 1 to 8 % [16].

Table: 4 WOCA Production of Monazite (in metric tonnes of concentrate)

Country	1966- 1974	1975	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986	1987	1988	1989	1990 prelim.
Australia	34.300	4507	5310	9379	14992	16340	14079	13251	13100	15141	16260	18735	14822	12813	11872	13500	14500
Brazil	14.950	1452	1610	2440	2540	1900	2532	2200	2000	5256	3622	1895	3618	4332	2817	1900	2000
India	28.700	3000	3000	2734	3303	3254	3395	3704	4000	4000	4000	4000	4000	4000	4000	4300	4500
Korea Rep.	30	10	10	10	10	10	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
Malaysia	15.800	3285	1879	1977	1254	542	347	320	450	1051	4980	5808	5959	2908	2920	2948	3500
Nigeria	150	10	18	20	20	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.
South Afr.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	1000	1000	1000	1200	1200	1200	1200
Sri Lanka	200	10	n.a.	5	213	213	63	60	60	300	147	200	200	200	200	200	200
Thailand	1200	365	362	n.a.	845	32	152	150	100	277	298	663	1609	458	590	631	650
Zaire	400	300	240	96	77	90	51	50	50	15	2	-	7	97	168	175	170
Others	950	330	270	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	4	4	4	3	n.a.	n.a.	n.a.	n.a.
total	96.680	13.239	12.699	16.661	23.254	22.381	20.619	19.735	19.760	26.044	30.313	32.305	31.218	26.008	23.767	24.854	26.720
Estimated ThO ₂ content (estim.average grade 6.6% ThO ₂)	6.400	874	840	1.100	1.535	1.477	1.360	1.300	1.300	1.720	2.000	2.130	2.060	1.700	1.570	1.640	1.760

In the USA monazite is produced, but no data are available

Uranium ores in quartz-pebble paleo-conglomerates of the Elliot Lake/Agnew Lake area, Canada, contain thorium grades of several 100 to a few 1000 ppm. Thorium was recovered as by-product of uranium at Rio Algom's Elliot Lake operations from 1959 until 1968. Thorium sulfate was produced, grading from 31 to 35 % Th. Although all thorium goes to tailing dumps at present, in case of demand, some 1200 to 1600 t Th might be available annually from these sources.

Since there are no reliable figures on the actual availability of thorium, a best judgment on a theoretically producible quantity of thorium per year may be based on figures of monazite produced by heavy mineral beach sands exploitation.

Annual WOCA production of monazite since 1966 has climbed from about 13000 t in 1975 to a peak 32300 t in 1985 and dropped to about 24900 t in 1989. The corresponding thorium content of the ore, based on an estimated average grade of 6.6 % ThO₂, was 874 in 1975, 2130 t in 1985, and 1640 t ThO₂ in 1989. Leading monazite producers were Australia, Brazil, India and Malaysia. They accounted for more than 90 % of the monazite output (Fig.4). It becomes obvious from these data that thorium could be produced theoretically at an annual rate of about 2000 t ThO₂ from these sources only. In contrast, the actual annual recovery of Th is estimated at 150 t ThO₂ [19, 20].

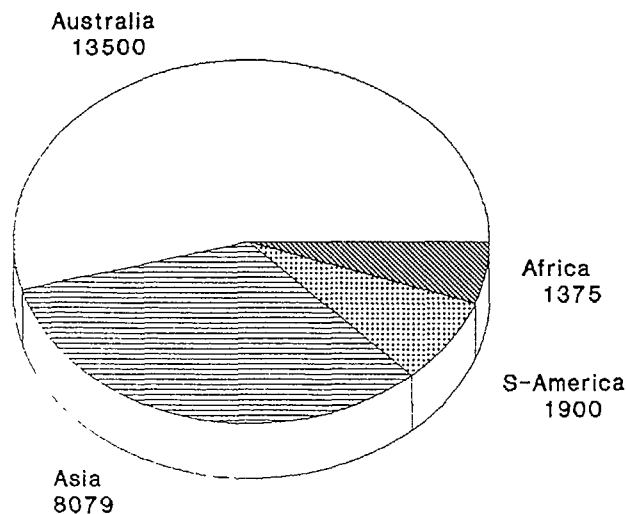


FIG. 4. Distribution of monazite production 1989 by continents (t monazite).

4.2 Principal Producers of Thorium

4.2.1 United States of America

In 1982, Associated Minerals Ltd., Inc., was reported to be a major producer of monazite from beach sands of Green Cove Springs, Florida. Other companies which reported monazite production from coastal heavy mineral sands of Florida and Georgia for the years 1971 to 1979, were Humphrey Mining Co. and Titanium Enterprise. The monazite is processed by Rhone-Poulenc at Freeport, Florida and Grace Davison Chemical at Chattanooga, Tennessee, for rare earth elements and for thorium. Thorium compounds are not sold. Estimated stock at the Grace plant site was about 5300 t of thorium oxide by the end of 1982. Thorium fuel for nuclear reactors was fabricated by several companies, e.g. Babcock and Wilcox, Gulf General Atomic, Kerr McGee, United Nuclear, and others.

4.2.2 Canada

At present, no thorium is produced in Canada. Thorium could be made available as a by-product of uranium recovery at a rate of 1200 to 1600 t ThO₂ per year from the Elliot Lake District,

Ontario. Thorium as thorium sulfate was produced at the Nordic Mill/Elliott Lake by Rio Algom between 1959 and 1968. The material was refined to Th metal, pellets, and powder.

4.2.3 Australia

Australia is one of the major producers of monazite (fig.4). Associated Minerals Consolidated Ltd. is the leading mining company operating beach sand dredges on the east coast. Five other companies, including Allied Eneabba and Renison Goldfields, are mining beach sands in Western Australia, the state with the highest total production at present.

4.2.4 India

The government-owned Indian Rare Earth Ltd. is the only producer of thorium and rare earth elements. Beach sand processing plants are operated at Manavalakurichi and Chavara and in the state of Kerala. A newly designed plant which could produce 4400 t of monazite annually is scheduled at Chatrapur, Orissa.

4.2.5 Brazil

Production of thorium from monazite dates back to 1895. At present, monazite is produced from heavy mineral beach sands by the government-owned company Nuclemon and controlled by the Comissao Nacional de Energia Nuclear. Heavy mineral operations are situated at Itabapoana (Rio de Janeiro), Cumuruxatiba (Bahia), and Guarapari (Espírito Santo). Potential by-product sources are large carbonate complexes, presently mined for niobium (Araxá), and alkaline complexes exploited for uranium (Poços de Caldas).

4.2.6 Malaysia

The Malaysia Mining Corp. is producing monazite from heavy minerals dredging operations in Perak and Selangor states. Their capacity amounts to 1500 t of monazite per year.

4.2.7 South Africa

Although no figures of actual thorium production from monazite are published, South Africa is reported to recover thorium sulfate as a by-product of copper and uranium at Palabora. A new plant

Table: 5 Prices for important Th-compounds in the USA (\$/kg ThO₂)

	Nitrate (mantle-grade) \$/kg ThO ₂	Oxide (99 % grade) \$/kg ThO ₂
1983	10.60	31.00
1984	10.10	35.85
1985	10.10	35.85
1986	13.60	40.00
1987	10.10	41.00
1988	13.80	45.00
1989	n.a.	n.a.
1990	16.55	55.00

Source: USBM, Minerals Yearbook, 1987-1990

for production of monazite is commissioned to produce 3000 t annually.

4.2.8 Egypt

The most important monazite concentrations are located in the Rosette area of the Nile Delta. Egypt is investigating the possibility of rare earth and thorium extraction. Pilot testing was undertaken but production has not begun.

4.2.9 Other Producers

Rhone-Poulenc is the principal thorium producer in France, processing imported ores for manufacturing of thorium compounds. Thorium compounds are also produced by Th. Goldschmidt, Essen, Federal Republic of Germany, by Nippon Yttrium Co., Japan, and by Rare Earth Product Ltd., United Kingdom.

Small productions of monazite are reported from tin mining companies in Thailand, from Negara Tambang Timah Sinkep, a government owned company in Indonesia, and from Sierra Rutile Ltd. in Sierra Leone.

4.3 Price

Prices for monazite, the main thorium ore, and for various compounds of thorium are published annually by the US Bureau of

Mines and by various mining journals (e.g. Engineering and Mining Journal).

According to these sources, the price for monazite has varied since 1969 between \$ 180 per ton to \$ 440 per ton in 1982 and \$ 695 per ton in 1990.

For comparison the prices for Th-nitrate and Th-oxide in the USA are shown in table.

In summary, it can be concluded, that sufficient thorium resources are available to meet an arising demand. If the extraction of thorium from monazite as a by-product of the recovery of REE is considered, theoretically about 1500 to 2000 t ThO₂ could be produced from this single source. If the annual demand would grow beyond that figure other sources of thorium, e.g. vein deposits, could be brought into production.

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MAIN TYPES OF URANIUM MINERALIZATION AND URANIUM EXPLORATION IN VIET NAM

NGUYEN VAN HOAI

Geological Department,

Ministry of Heavy Industries of Viet Nam

PHAN VAN QUYNH

Hanoi University

Hanoi, Viet Nam

Abstract

In Viet Nam, where uranium exploration started before the National Independence, a number of occurrences have been discovered. They belong to the sandstone, vein, volcanic, metamorphic and surficial deposit types. In addition, coal seams have been found to be uraniferous.

Uranium resources have been classified as follows: 200 tonnes U as EAR-I, recoverable at costs between US\$ 80 -130/kg U, 200 tonnes U as EAR-II, recoverable at costs between US\$ 80 -130/kg U, and 180 000 tonnes U as Speculative Resources, of which 100 000 tonnes are believed to be recoverable at costs of below US\$ 80/kg U and the remaining 80 000 tonnes at above US\$ 130/kg U.

MAIN CHARACTERISTICS OF URANIUM MINERALIZATION IN VIETNAM

The known uranium mineralization in Vietnam exhibits distinct characteristics as regards the distribution in space and time of formation. These features include the following:

1. The uranium occurrences known as of the end of 1990 are scattered over the whole territory of the country. However, some concentration occurs in the central and northern parts of Vietnam.

2. The age of the known uranium mineralization ranges from Proterozoic to Quaternary. However, those occurrences with a higher potential are of Mesozoic age.

3. The ore grade in the known occurrences ranges from 0.05 to 1.0 % U.

4. The occurrences and prospects belong to the following deposits types:

- sandstone
- vein
- volcanic

- metamorphic
- coal
- surficial in younger sediments.

DESCRIPTION OF TYPICAL OCCURRENCES AND DEPOSITS

The Nong Son sandstone type deposit (see location map): the Nong Son deposit is hosted in the Nong Son sedimentary basin located in the Quang Nam - Da Nang province in central Vietnam. The basin is considered a rift related basin formed in upper Paleozoic - lower Mesozoic. The basin is filled with three upper Triassic sequences, overlying unconformably Paleozoic sediments in the north and Proterozoic metamorphics in the south. The sequences include from top to bottom:

- conglomerates, red and grey sandstones with organic material and limestones
- sandstone-mudstone sequence, grey to red shales, and
- coal bearing sediments.

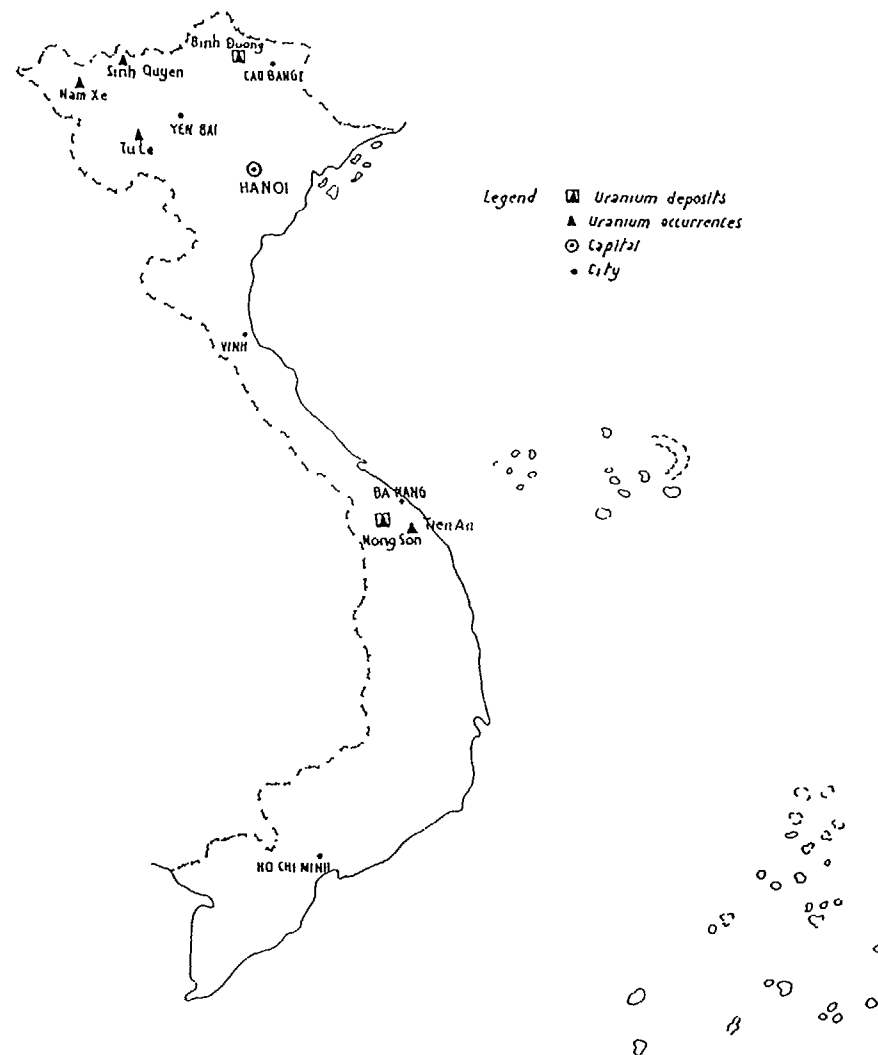
Paleozoic granites with 12 - 15 ppm U make up the rims of the basin.

The main uranium minerals are pitchblende, uranium bearing arsenates and vanadates, autunite as well as carnotite.

The Nam Xe uranium bearing REE occurrence and the Sinh Quyen Cu-U occurrence: These occurrences are located in northern Vietnam as shown in the location map.

The Nam Xe occurrence is located in the Lai Chau province. The mineralization occurs in Paleozoic marbles outcropping in the center of an anticline. The marbles are underlain by metamorphic and volcanic rocks. There are two types of primary mineralization: massive vein material as well as impregnations in the host rock. The ore grade of the vein mineralization is between 8 - 10 % rare earth oxides (REO) and that of the impregnated rocks 1 - 2 % REO. Because of the tropical weathering conditions the REE mineralization was enriched in the weathering zone consisting of dark brown soil. Here the average grade is between 4 - 5 % REO. The uranium mineral is pyrochlore and the main REE minerals are bastnaesite and parisite. The distribution of the individual REE is as follows: Ce = 38 %, La = 29.6 %, Nd = 16 %, Pr = 5 - 10 %, Y = 9.3 %, and Gd + Eu = 4 %. The pyrochlore contains up to several percent U as well Nb (2.7 - 12.4 %) and some Ta at a Nb:Ta ratio of 5:1.

The Sinh Quyen Cu-U occurrence is located in the Hoang Lien Son province. The occurrence is hosted by metasomatic rocks as parts of a Proterozoic metamorphic sequence. The mineralization is associated with silicification and the main ore minerals include chalcopyrite, pyrite and uraninite.



Principal uranium deposits and occurrences in Viet Nam.

The Tule uranium occurrence: This is located in the Sonla province in the northwestern part of the country (see location map). The uranium mineralization is contained in acid to alkalic volcanogenic sediments, derived from granitic porphyries, rhyolites, trachyte-liparite porphyry, tuffs and conglomerates.

The main minerals include uraninite, uranophan, uranium molybdate as well as molybdenite.

The Tien An occurrence is located in the central part of the country (see location map). Geologically, the uranium mineralization is associated with irregular graphite bodies in Proterozoic metamorphics.

Uranium occurrences in coal: This include the low grade U occurrence in the Nong Son sedimentary basin as mentioned above. Similar occurrences are known from a number of high grade coal deposits (anthracite, semi-anthracite) in Vietnam.

The Binh Duong uranium deposit: This deposit is located in the Cao Bang province in northern Vietnam (see location map). The deposit is hosted in Neogene - Early Quaternary sediments, which cover the contact between a Cretaceous two mica granite and Devonian limestones. The mineralization presents itself in irregular bodies and includes autunite and torbernite.

URANIUM RESOURCES

According to preliminary calculations the known uranium resources in Vietnam are distributed by deposit type as follows:

- sandstone type: 45 %
- vein type: 40 %
- volcanic type: 10 %
- others: 5 %

In order to improve the knowledge of the known occurrences, their further investigation is planned to be increased. In addition, special consideration is being paid to those deposit types, which are not yet discovered in Vietnam, but which may occur in the known geological environments. These types include the quartz pebble conglomerate and unconformity deposit types.

As of January 1991, the following uranium resources (in tonnes U) have been estimated.

	EAR - I (C ₁ + C ₂)	EAR - II (C ₂ + P ₁)	SR (P)
US \$ 80/kg U	---	---	100 000
US \$ 80-130/kg U	200	200	---
> US \$ 130/kg U	---	---	80 000
TOTAL	200	200	180 000

URANIUM EXPLORATION

The history of exploration for uranium and REE is parallel to the history of economic and social development in Vietnam. Therefore, the activities can be divided into three periods:

- Prior to 1955: exploration was carried out by French geologists of the Geological Department of Indochina. The main achievement of this period was the discovery of uranium phosphate veins in the Piaocac granite.

- Between 1955 - 1978: during this period uranium exploration concentrated in the territory north of the 17th Parallel and was carried out by two organisations, the Geological Survey of the Ministry of Industry and Trade Affairs and its successor, the General Department of Geology. Under the latter organisation, the occurrences Binh Duong, Nam Xe, Muong Hum (a U bearing REE occurrence not shown on the location map), and Sinh Quyen were found.

-after 1978: the main objectives were the conduct of genetic research as well as exploration in the entire territory of Vietnam. At the beginning of this period, activities were carried out under the General Department of Mining and Geology, which was transformed into the Geological Survey of the Ministry of Heavy Industries. These organisations were responsible for the discovery of the Nong Son sandstone type deposit and the occurrences Tien An, Nong Son (coal), and Nui Hong (a U bearing coal occurrence not shown on the map).

The exploration methods mainly used include an airborne radiometric survey at scales of 1 : 50 000 to 1 : 25 000, ground surveys at scales between 1 : 10 000 and 1 : 5 000 as well as underground exploration including drilling and some excavations.

The Vietnamese Government encourages the co-operation with foreign companies based on the principles of equality and mutual benefits as stipulated in the Foreign Investment Law.

THE DISCOVERY OF THE McARTHUR RIVER URANIUM DEPOSIT, SASKATCHEWAN, CANADA

J. MARLATT, B. MCGILL, R. MATTHEWS,
V. SOPUCK, G. POLLOCK
Cameco Corporation,
Saskatoon, Saskatchewan, Canada

Abstract

The McArthur River P2 North uranium deposit is in the southeastern portion of the Athabasca Basin, 70 km northeast of the Key Lake uranium mine. Mineralization is in Athabasca sandstone and fault-thrusted basement graphitic metasediments, at 500 to 600 metres depth and is structurally controlled by a regional, northeasterly-trending fault rooted in graphitic basement rocks. Several stages of hydrothermal alteration involving silica, illite, kaolinite, chlorite, and dravite preceded or followed the uranium mineralizing event at 1521 Ma. Early silicification, possibly related to basement quartzites, reduced sandstone permeability and limited the development of a clay alteration halo. The mineralization consists of uraninite without the cobalt-nickel-arsenide minerals found in some other unconformity-related uranium deposits. Uranium values of greater than 1 ppm form a pathfinder chimney to the bedrock surface. Mineralized sandstone boulders drew attention to this area in 1980, the regional conductor was defined geophysically in 1984-86, and drilling in 1986 located subeconomic mineralization 4.5 km southwest of the deposit. Continuing work in the area defined the characteristics of mineralization associated with the fault scarp and further drilling along the structure led to the discovery of the P2 North deposit late in 1988. A variety of geophysical techniques were used to define conductive trends and help outline alteration patterns and basement geology. Alteration patterns, structural observations and sandstone lithogeochemical signatures also guided the placement of the drill holes. The deposit has been tested by about 30 vertical drill holes spaced 50 to 100 m apart along its length of 1700 m. Preliminary estimates, at the end of 1990, indicate 200 million pounds at an average grade of 4% U_3O_8 . Partners in the McArthur River Joint Venture are Cameco Corporation, Uranerz Exploration and Mining Limited, Agip Resources Ltd., Interuranium Canada Limited, and Cogema Canada Ltd.

1. INTRODUCTION

In May, 1990, Cameco, as operator of the McArthur River Joint Venture, announced that further drilling at the McArthur River P2 North location had confirmed expectations of a new high-grade uranium deposit. The discovery is located in northern Saskatchewan, about 70 km northeast of the Key Lake uranium mine and 95 km southwest of the Rabbit Lake uranium mine (Figure 1). The McArthur River project is a joint venture between Cameco Corporation, Uranerz Exploration and Mining Limited, Agip Resources Ltd., Interuranium Canada Limited, and Cogema Canada Ltd. This paper reports on the status of the P2 North deposit to the end of 1990.

The project area is located close to the proposed Cigar Lake road and a newly constructed power line. Access to the area is by winter road or aircraft. Exploration is conducted from tent camps during the winter and summer months.

The P2 North deposit is located about 100 m from the property boundary on the northwestern side of the project area (Figure 2). At the end of 1990 the deposit had been tested by about 30 vertical drill holes spaced about 100 metres apart along its length of 1700 metres and in several cross-sections with holes spaced about 25 metres apart (Figure 3).

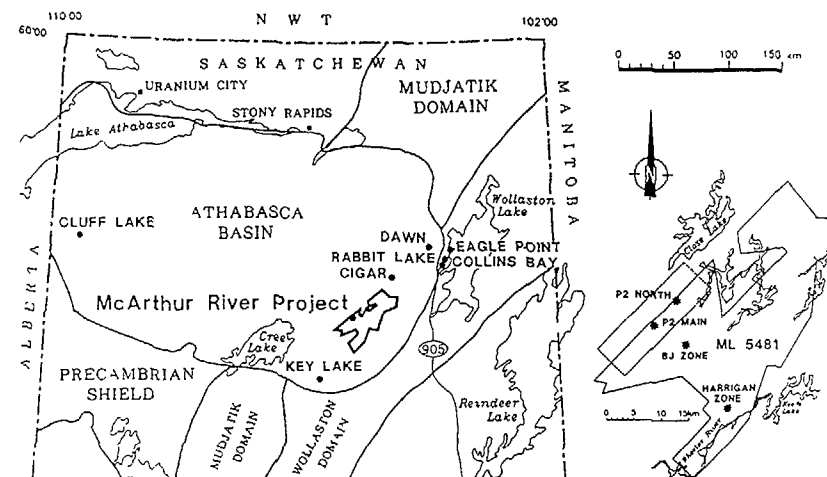


FIG 1. Uranium properties — location map.

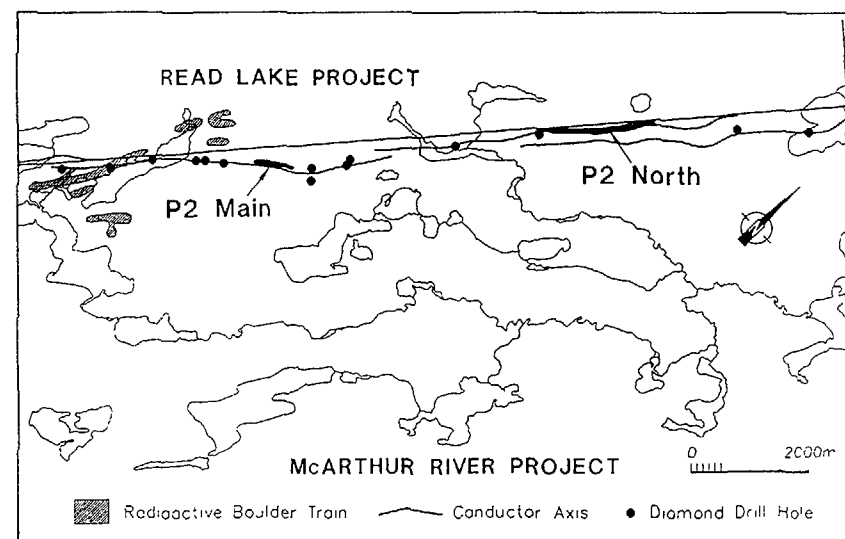


FIG 2. Deposit location map.

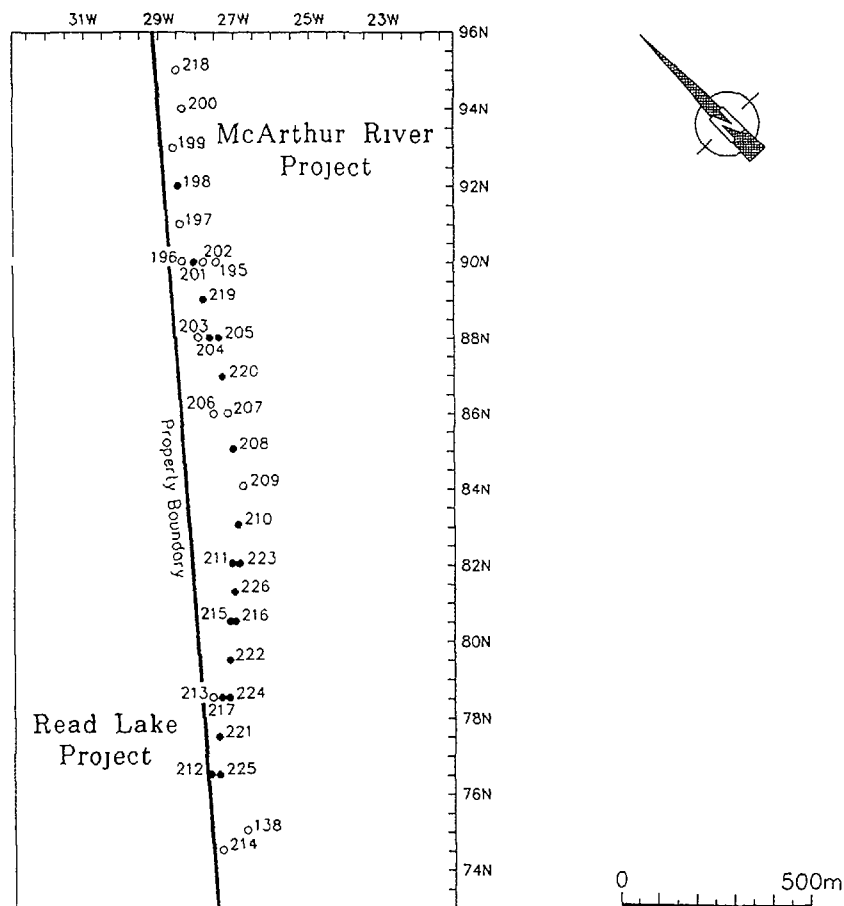


FIG 3 P2 North zone drill hole location map

Several holes encountered two or three mineralized intersections. The best intersection was 12% U_3O_8 over 45 m including individual assays greater than 77% U_3O_8 . Based on these results Cameco estimated a geological reserve containing 200 million pounds at an average grade of 4% U_3O_8 .

The uranium mineralization is associated with a reverse thrust fault at a depth of 500 to 550 metres and occurs mainly in silicified sandstone. The deposit does not have the extensive clay alteration and cobalt-nickel-arsenide minerals commonly present in a number of other Saskatchewan unconformity related deposits.

The deposit has the potential to be one of the larger discoveries of the prolific Athabasca sandstone basin, which hosts the Key Lake deposit, discovered in 1975, and the Cigar Lake deposit, discovered in 1981.

2. REGIONAL SETTING

The Athabasca Basin is situated within the southwest part of the Churchill Structural Province of the Canadian Shield. The province has been divided into subprovinces based on lithologic and structural parameters (Lewry and Sibbald, 1977, Macdonald, 1987). The McArthur River project is located in the Cree Lake Zone within the Wollaston Domain near the boundary of the Mudjatik Domain. The Wollaston Domain is underlain mainly by Archean granitoid gneisses which are unconformably overlain by metamorphosed shallow water metasedimentary rocks of the Aphebian Wollaston Group. These rocks are in turn unconformably overlain by unmetamorphosed flat-lying sandstones of the Helikian Athabasca Group. The P2 North mineralization occurs mainly in the sandstones near the unconformity with the underlying metamorphosed basement rocks of the Wollaston Domain, along a major post-Athabasca Group thrust fault which is coincident with an electromagnetic (EM) conductor axis.

The Wollaston-Mudjatik boundary separates the strong northeast patterns of the Wollaston Domain from the non-linear patterns characteristic of the Mudjatik domain. Graphitic gneisses along the boundary are the locus for both pre- and post-Athabasca Group faulting, controlling factors in ore formation.

3. EXPLORATION MODELS

The high-grade uranium deposits of the Athabasca Basin occur in a variety of positions in close spatial association with the sub-Athabasca unconformity (Figure 4). They may occur in altered basement rocks, at the unconformity in bleached and altered paleoweathered basement rocks or in altered Athabasca sandstone above the unconformity. Mineralization can extend down to 300 m or more into the basement.

Several styles of mineralization are recognized on the McArthur River property (Figure 5). At BJ Lake subeconomic mineralization is found in structurally disrupted basement quartzites and silicified sandstone and along faulted pelite-quartzite contacts. Focusing of the mineralizing fluids may have been related to structural disruption along paleotopographic ridges and the contact between quartzite and semipelitic gneisses. This style of mineralization is characterized by quartzite basement ridges, sandstone silicification and no EM response.

The P2 North deposit and the P2 Main mineralization are associated with the graphitic thrust fault that forms the P2 conductor but the thrust wedge is not present at P2 Main. The main differences between the P2 North model and deposits of the Key Lake-Cigar Lake type are the thick basement quartzite unit, the intense silicification across at least 100 m in sandstone, and the relative lack of nickel-cobalt arsenide and sulpharsenide minerals at P2 North.

Prior to the main mineralizing event at P2 North, alteration stages involved kaolinitization, silicification, dravutization and finally illitization. Whereas illitization is a

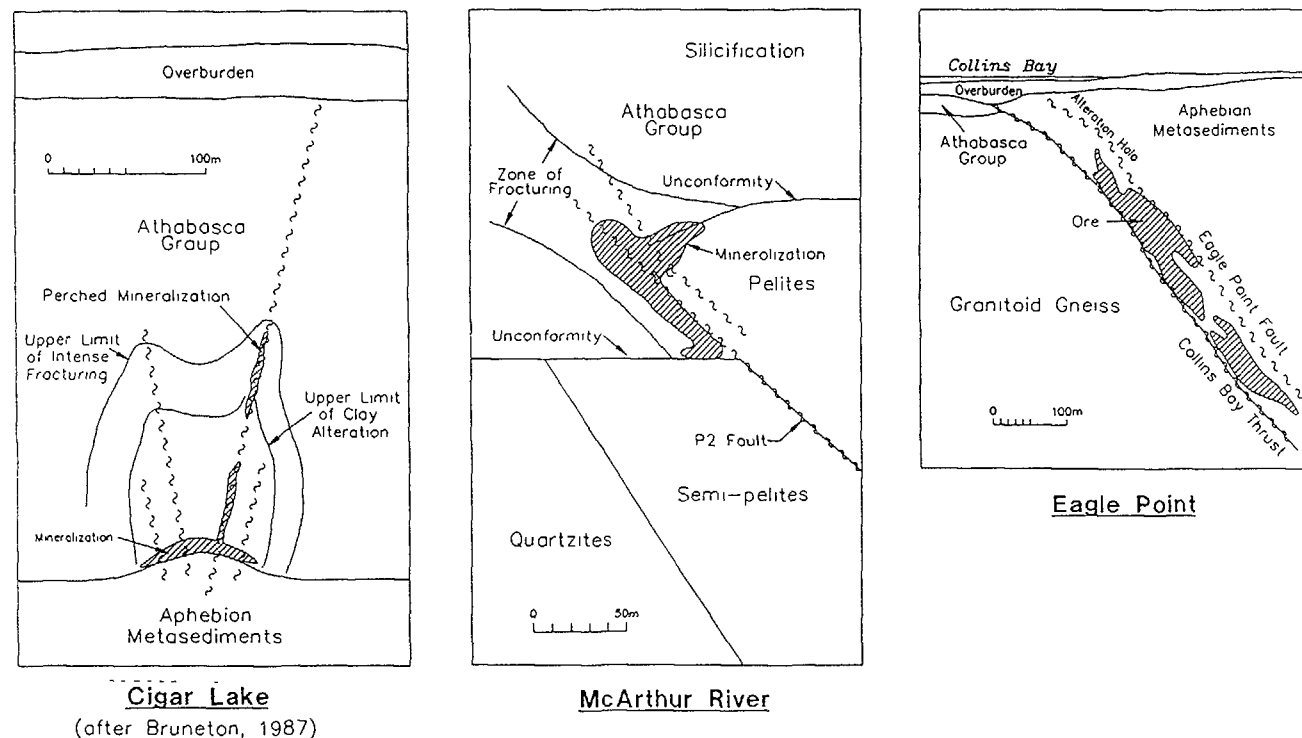


FIG. 4. Exploration models — Saskatchewan uranium deposits.

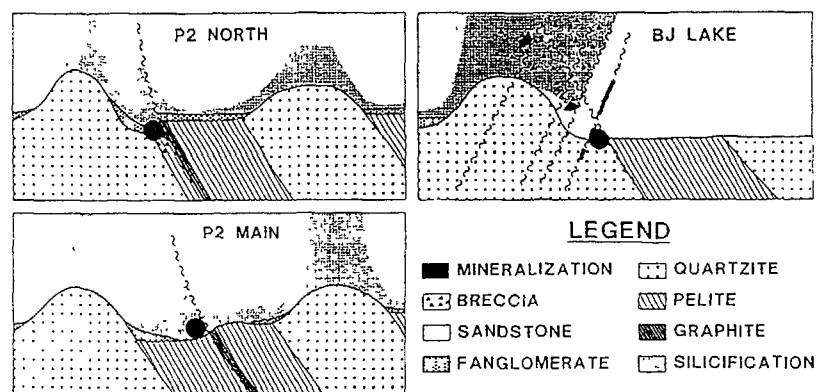


FIG. 5. Models for uranium mineralization — McArthur River project, quartzite relationships.

dominant alteration at Cigar Lake, the early silicification at P2 North appears to have limited the illitization event near mineralization and played an important part in the subsequent formation of the mineralizing system. Alterations postdating the main mineralizing event include fracture hosted dravite and euhedral quartz in vugs.

4. GEOLOGY OF THE P2 NORTH URANIUM DEPOSIT

4.1. Stratigraphy

The surface cover consists of a flat glacial lacustrine sand plain generally less than 10 m thick at the base of drumlins which reach heights of up to 100 m.

4.1.1. Athabasca Group

The Middle Proterozoic Athabasca Group, varying in thickness from 480 m over the hanging wall to 560 m over the footwall, consists of well sorted sandstone units and minor

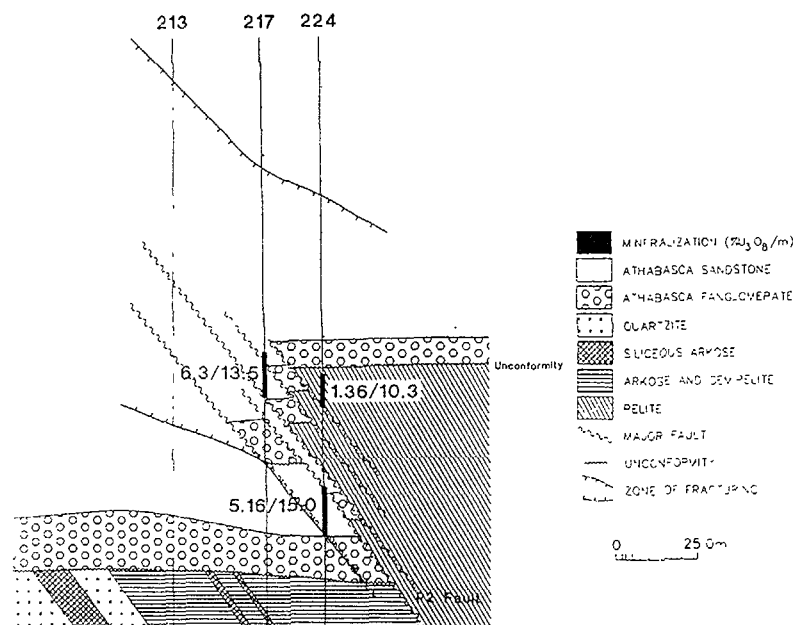


FIG. 6. Detailed cross-section (line 78+50N).

pebble beds of the Manitou Falls Formation. A basal fanglomerate containing pebbles and cobbles of quartzite unconformably overlies the crystalline basement (Figure 6).

4.1.2. Wollaston Group

The Wollaston Group metamorphic rocks below the Athabasca Group consist of pelitic gneisses on the hanging wall side of the P2 fault and of quartzites with subsidiary pelitic to arkosic gneisses on the footwall side.

The hanging wall sequence consists of a lower cordierite-graphite pelitic gneiss which hosts the P2 thrust fault, overlain by cordierite-garnet arkosic gneiss and quartz-rich arkosic gneiss with minor semipelitic gneiss. A middle member of garnet-cordierite±tourmaline semipelitic gneiss is occasionally overlain by a marker horizon of hornblende-dolomite-diopside calc-silicate. Garnet-cordierite semipelitic gneiss constitutes the upper member of the hanging wall sequence.

The footwall sequence is dominated by siliceous and arkosic rocks with a notable lack of graphitic units. Three individual quartzite units occur, separated by garnet- and cordierite-bearing semipelitic to pelitic gneisses and arkosic gneisses. These units which separate the quartzites are only correlatable over short distances, probably due to facies changes or tectonic thinning.

4.2. Structure

The two dominant structural trends at the deposit are 045° thrust faults dipping southeasterly at 40° to 45° and a series of steeply dipping transcurrent faults striking at 100° to 110°.

The P2 thrust fault formed at the base of the pelitic gneiss sequence and has lifted a wedge of Wollaston Group basement rocks into a position overlying the lower Athabasca Group sandstone (Figure 6). The maximum vertical displacement along the thrust fault exceeds 80 m at the north end of the deposit, decreasing to 60 m at the south end. The basal Athabasca fanglomerate is 15 m thicker over the footwall unconformity than over the hanging wall unconformity, suggesting that some displacement can be attributed to pre-Athabasca uplift of the hanging wall.

The P2 thrust faults have reactivated along several graphitic fault planes within the lower 25 m of the hanging wall basement rocks. These fault planes, parallel to foliation, rarely exceed 1 m in width within the basement rocks but expand into extensive zones of mylonitization, fracturing and brecciation in the overlying brittle sandstones. Net differential movement along the individual fault planes results in step-like offsets along the overthrust hanging wall unconformity. In one instance, a vertical offset of 16 m in the hanging wall unconformity occurs between 11 m spaced drill hole intersections. Rotated sandstone bedding is evident over a vertical distance of many tens of metres, often reaching the maximum rotation angle in the vicinity of the most intense faulting on the foot wall side of the P2 fault.

A 50 m to 75 m thick sandstone hosted fracture zone overlies the hanging wall unconformity. Rotated sandstone bedding and displacement features such as slickensides and sandy fault gouges are occasionally observed within the interval. The majority of the fractures are randomly oriented, the product of brittle fracturing from the uplift of the hanging wall thrust block.

Steeply dipping transcurrent faults occur frequently but irregularly over the entire length of the deposit, displacing the basement stratigraphy and the P2 fault.

4.3. Alteration

The most distinctive alteration characteristics of the P2 North deposit are the intensely silicified, weakly bleached sandstone and the weak development of hydrothermal clay alteration. These features are in dramatic contrast to the bleached, clay altered haloes associated with the majority of Athabasca uranium deposits.

Pervasive bleaching has affected the upper 225 m of sandstone. Below this, the sandstone retains a pink to purple colouration resulting from primary unbleached to weakly bleached, hematite. The upper sandstone is weakly and variably silicified, with the silicification gradually increasing in intensity with depth to 375 m, 125 to 150 m above the hanging wall unconformity. The intensity of sandstone silicification increases dramatically below 375 m, continuing to the basal fanglomerate unit.

Fine grained interstitial dravite, reflected in the boron geochemistry, occurs in the upper sandstone formations, increasing from northeast to southwest. Blue-green coloured, fracture hosted dravite and gray clay are present over the lower 25 m to 50 m of hanging wall sandstone. A 2-3 m thick zone of both fracture hosted and interstitial limonite usually occurs just above mineralization. A zone of pervasive grey alteration may be present above

the limonite, but it is always of limited extent and intensity. Fresh pyrite fracture coatings occur but are more common within mineralized intervals.

Alteration of the sandstone below the thrust fault is similar but due to an increase in structural disruption, much of which is post-mineralization, easy access by hydrothermal solutions has led to more intense alteration. Pervasive silicification is less intense although strongly silicified fragments of sandstone are often observed in zones of brecciation. Fracture hosted and interstitial dravite is present throughout. Euhedral quartz fracture coatings occur sporadically as does interstitial chlorite which is usually present as a matrix mineral within the basal fanglomerate. Intense chlorite alteration is restricted to the mineralized intervals.

The hanging wall basement rocks, particularly those over the lower sandstone, have been subjected to a greater degree of hydrothermal alteration than the footwall basement rocks. Footwall basement rocks show only weak alteration overprints of the paleoweathering profile. Alteration of feldspar and biotite to illite, sericite, and quartz is common in almost all rocks. The presence of dravite, apatite, and chlorite in the footwall rocks is not uncommon, but the intensity of these alteration features is low. Chlorite and illite alteration of the hanging wall basement rocks is strongest in the vicinity of fault zones, mineralized intervals, and occasionally at the unconformity. Occurrences of interstitial and fracture hosted dravite and pyrite are sporadic but may be locally intense.

4.4. Mineralization

The P2 thrust fault(s) controls the P2 North mineralization. The distribution of mineralization can be categorized into three groups as follows (Figure 6)

- (a) within the Athabasca Group and uppermost hanging wall basement rocks extending for a short distance beyond the tip of the wedge and upwards into the sandstone along the P2 structure
- (b) associated with narrow, paralleling fault planes within the hanging wall basement wedge (P2 faults and the steep transcurrent faults)
- (c) within the lowermost hanging wall basement wedge and the underlying sandstone, associated with the main plane of the P2 fault

Massive, high grade mineralization is restricted to occurrences of type 1 and 3 with the best intervals found in the sandstone adjacent to the hanging wall basement rocks.

Botryoidal uraninite masses and subhedral cubic uraninite aggregates constitute the earliest phase of mineralization. In polished thin section this mineralization possesses a high reflectance relative to younger, remobilized uraninite, hence the designations high-R and low R uraninite. The cores of the high R botryoidal uraninite masses often consist of blueish green clay chlorite containing disseminated cubic uraninite, galena, and angular sandstone fragments. Euhedral quartz crystals commonly grow in a radiating fashion outward along the concave external margins of the botryoidal uraninite. Textural evidence indicates that the blue green clay-chlorite, high-R uraninite, galena, and euhedral quartz were deposited during one continuous event. Pyrite, chalcopyrite, and minor nickel-cobalt sulpharsenides were also part of this process. Gold grains have been observed within the chlorite in a single polished thin section.

Low-R uraninite occurs with chalcopyrite and galena in fractures within brecciated high-R uraninite and euhedral quartz. Low R uraninite also occurs as irregular replacement aggregates of high-R uraninite. Chalcopyrite and galena were also deposited in the clay-chlorite matrix during this event.

A later fracture controlled modification resulted in oxidation of the early sulphides, especially chalcopyrite. Covellite replaced chalcopyrite along fractures which were in turn cut and replaced by chalcocite, carbonate, chlorite-clay veinlets. Fracture controlled goethite-carbonate veinlets, some with chalcocite, cut most of the above assemblages. The young fracture controlled low-R uraninite has also been partly replaced by goethite-carbonate during this event. The final modification involved further brecciation and introduction of goethite veinlets.

Microprobe analysis of the sulphides in core from one hole identified Ni-Co and Ni-Co-As bearing pyrite as well as a Co bearing chalcopyrite and an unknown Ni-Co arsenide mineral. This indicates that nickel, cobalt, and arsenic are present at P2 North but only on occasion does their content reach a threshold level allowing for formation of Ni-Co sulpharsenide minerals.

4.4.1. Age of Mineralization

Two samples containing abundant high-R uraninite from high grade hole MAC-212 were analyzed for age determinations using U/Pb methods. One sample yielded two separate ages from adjacent uraninite masses, differing in the quantity of pitted galena inclusions and possessing a subtle contrast in reflectance. The uraninite of uniform reflectivity with fewer galena inclusions yielded an age estimate of 1521 ± 8 Ma while the adjacent uraninite gave an estimated age of 1348 ± 16 Ma. A similar age of 1358 ± 19 Ma was established for a second sample. The younger dates are in agreement with the 1350 Ma age reported for the Key Lake uranium deposit as the primary mineralizing event (Trocki et al, 1984), however at P2 North it appears that the date represents a remobilization of the primary 1521 Ma mineralization.

5. GEOCHEMICAL SIGNATURE

A regional illite anomaly in sandstone extends for 100 km northeast from Key Lake (Earle and Sopuck, 1987). This zone, which coincides roughly with a belt of Aphebian pelitic gneiss on the western side of the Wollaston Domain, encompasses all of the known mineralization in the area from Key Lake to the McArthur River project. Sub-parallel zones of boron and chlorite enrichment were noted along the axis of basement quartzite ridges. The regional illite anomaly was ascribed to increased faulting and basement-sandstone fluid interaction along the trend. The McArthur River deposit was discovered toward the northeast end of the illite anomaly and along an anomalous boron zone.

Two distinct alteration patterns have been identified around uranium deposits of the Athabasca Basin (Figure 7). Mineralized zones in the north (Cigar Lake, Midwest Lake) are characterized by variable sized haloes of strongly illitized sandstones and small haloes of weak boron enrichment. The sandstone section has been desilicified. Illitic alteration chimneys are at least 100-300 m wide and extend vertically up through 400 m of sandstone.

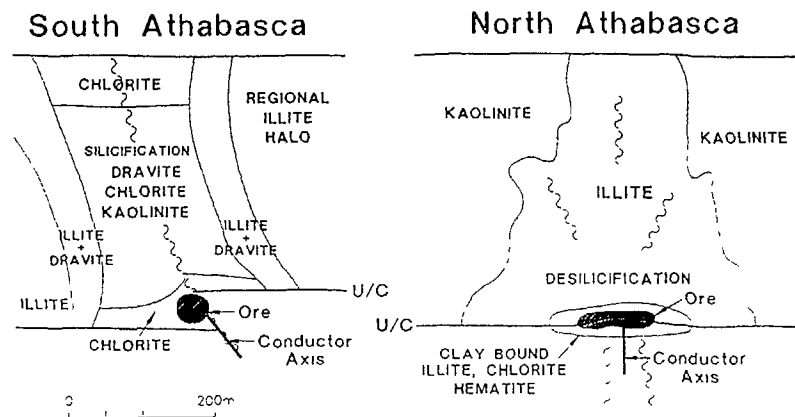


FIG. 7 Sandstone alteration models

at Cigar lake. These clay haloes may extend for appreciable distances along the strike of the conductor-related mineralization.

The southern area, at Key Lake and particularly on the McArthur River project is characterized by clay depletion and extensive areas of silicification. The dominant alteration related clays are dravite, kaolinite and chlorite. Illitization exists well outside of the mineralized zone. Silicification may be related to the interaction of hydrothermal fluids with rocks along basement quartzite ridges.

5.1. Drill Hole Lithogeochemistry

Anomalous uranium (Figure 8), greater than 1 ppm, is found from the unconformity to within 200 m of the bedrock surface at the northeast end of the deposit. The 1 ppm uranium anomaly encompasses the entire sandstone column over the higher grade southwestern portion of the deposit.

The proportions of illite and kaolinite clay components indicate that a consistent clay layering is present. Kaolinite dominates from surface to 250 m depth and from 330 m to the unconformity (Figure 8). A band of illitic sandstone separates the upper and lower kaolinitic layers. This clay layering probably reflects preferential migration of hydrothermal fluids along certain sandstone units.

The clay fraction of the upper kaolinitic unit generally has a high component of chlorite (>10%). Strong chlorite enrichment is also observed in the sandstone associated with mineralization.

Boron enrichment, in the mineral dravite, generally parallels the pattern of chlorite enrichment. Boron contents greater than 100 ppm are noted from the top of the sandstone to 220 m (Figure 8). A second boron-rich layer occurs in the vicinity of the thrust basement wedge, particularly below the P2 fault.

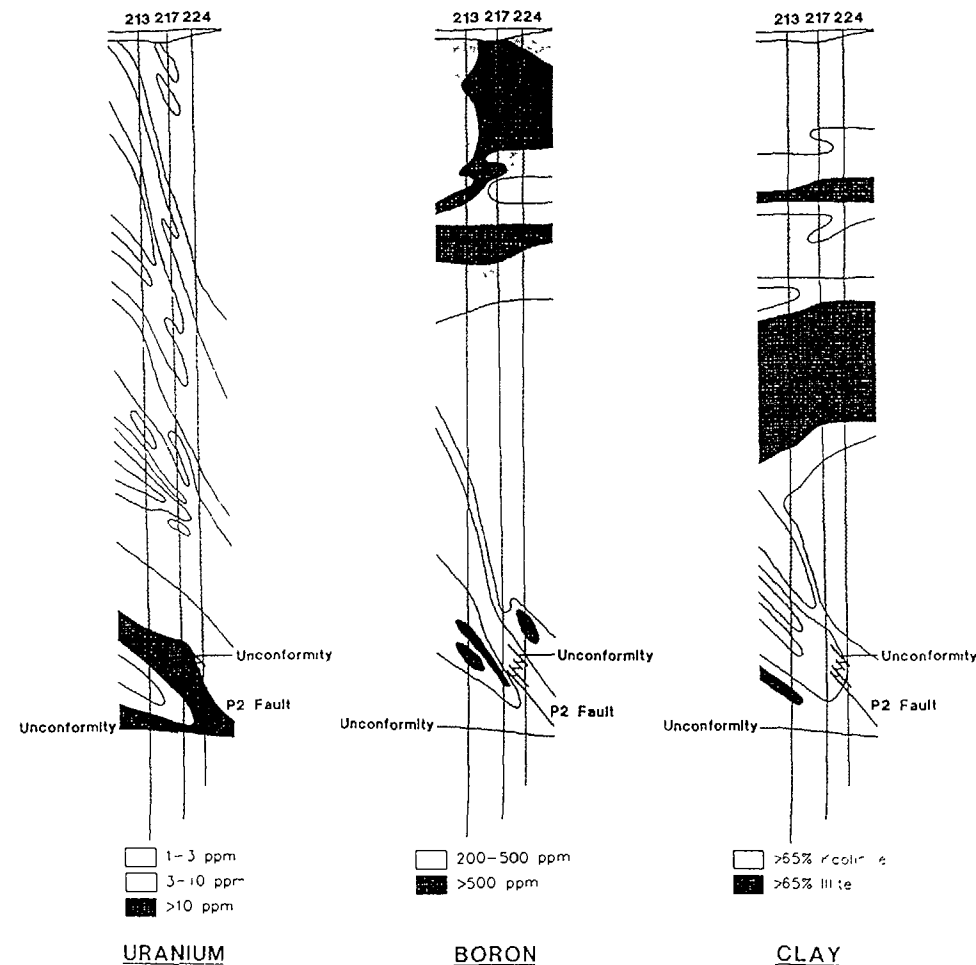


FIG 8 P2 North — lithogeochemistry (line 78 + 50N)

6. GEOPHYSICAL SIGNATURE

The P2 grid area lies within a broad magnetic low, interpreted to represent an Aphebian metasedimentary basin. The main conductor axis is associated with a low amplitude magnetic corridor and is cut by a series of east-west trending magnetic-structural breaks which appear to terminate the mineralized zone and also intersect the strongest mineralization. The deposit flanks a well defined resistive trend reflecting sandstone

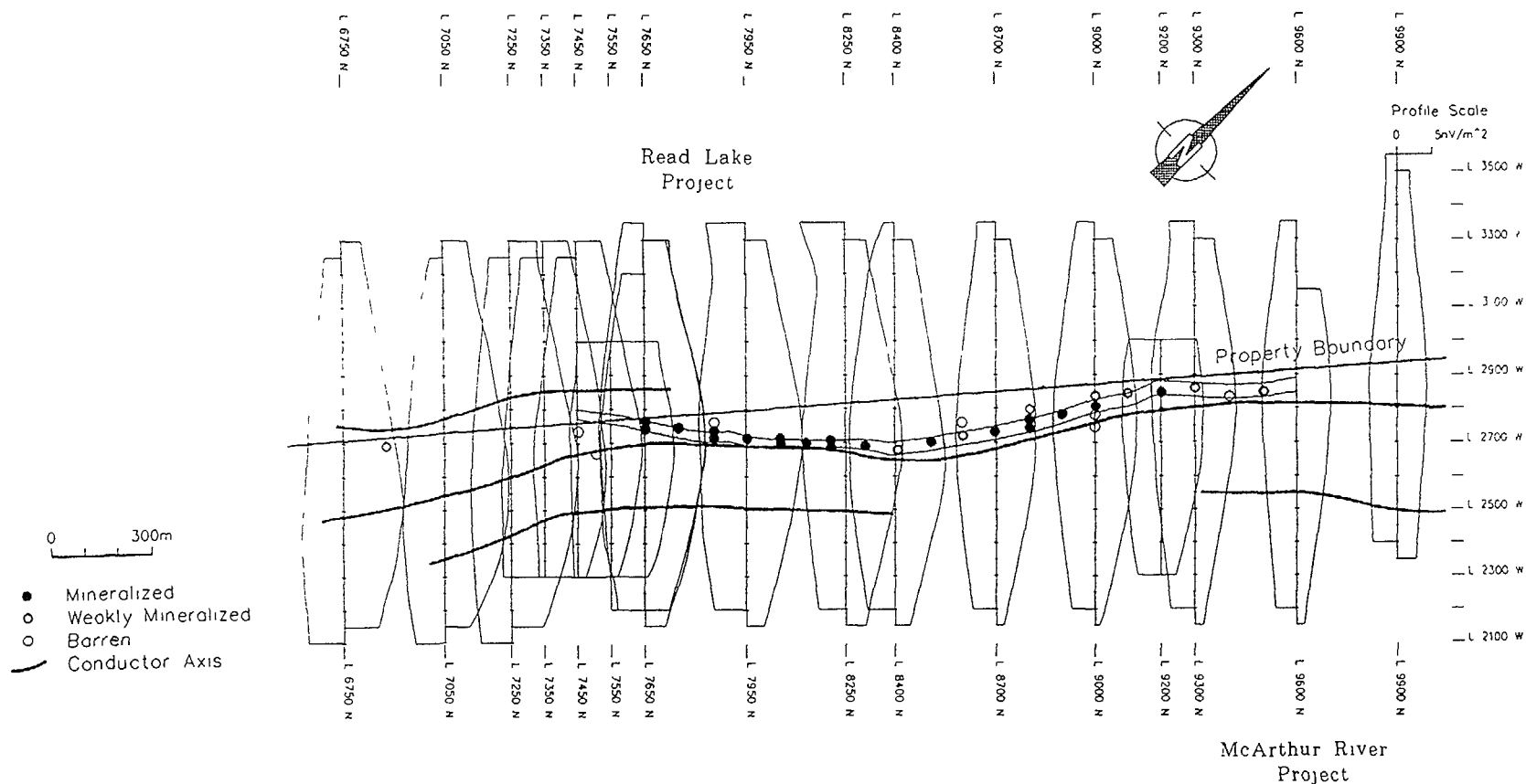


FIG. 9. P2 North — stacked EM-37 profiles, channel 15

silicification. Resistive sandstone is also indicated east of the conductive zone and is likely related to a basement quartzite unit.

Figure 9 shows fixed loop Time Domain Electromagnetic (TDEM) profiles, using the EM-37 system, channel 15 horizontal component, over the deposit, together with drill hole locations, interpreted conductor axes, and an outline of the mineralized zone. Figure 10 compares results obtained from fixed loop EM-37 and correlation processed (Polzer et al, 1989), moving loop UTEM stepwise array surveys. The P2 conductor is indicated at 27+50W as a strong response, which decays slowly and persists to late times. The late time decay constant is 3.4 ms which for an infinite thin sheet results in a conductance of 30 S. The asymmetry of the profiles and southeast migration of the axis at late times is due to the dip of the stratigraphy and indicates the presence of weaker subsidiary conductors and the

presence of conductive basement to the east of the P2 conductor. The southeast migration of the axis at late times is due to the dip of the stratigraphy.

Three resistivity techniques have been employed over the deposit; Controlled Source Audio Magnetotellurics (CSAMT) capable of mapping resistivity contrasts at depth, as well as EM-16R and Inductive Source Resistivity (ISR) surveys for mapping shallow resistivity features (Macnae and Irvine, 1988). Results from line 76+50N, on the southern part of the deposit, are presented in Figure 11.

The CSAMT pseudosection indicates conductive basement rocks between 23+50W and 27+50W, flanked by more resistive basement lithologies. The strong, narrow resistive feature immediately west of the conductive unit is interpreted to represent a shallow zone of sandstone silicification. Both the EM-16R and ISR plots define a similar zone of shallow

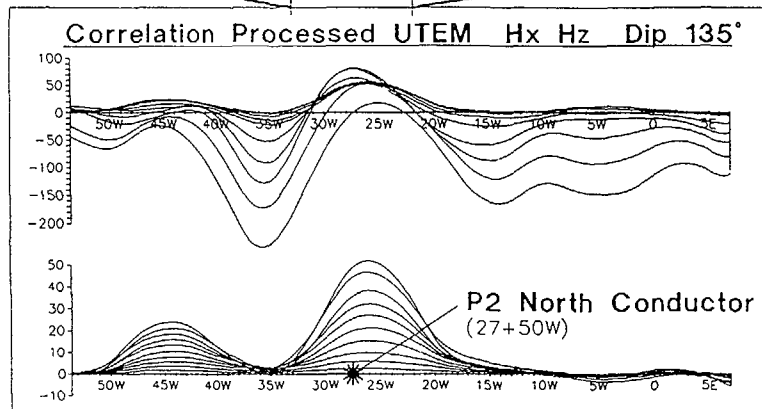
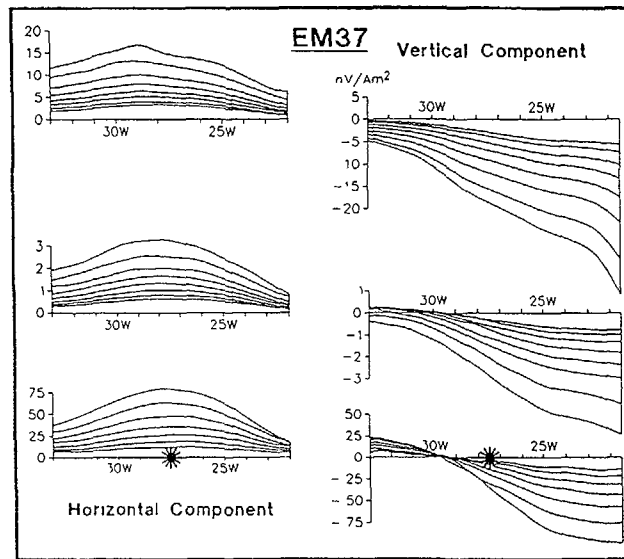


FIG. 10 EM profiles — filtered data (L90+00N)

silicification as well as elevated resistivities on the eastern end of the line. The two narrow ISR peaks at the west end of the line correspond to steep drumlins on surface.

Ground magnetic and gravity profiles along this line (not shown) indicate that the mineralization falls within a gravity and magnetic low indicative of a metasedimentary basin. Basement units to the east and west are interpreted to represent quartzite lithologies, with the eastern unit modelling as a paleotopographic basement ridge.

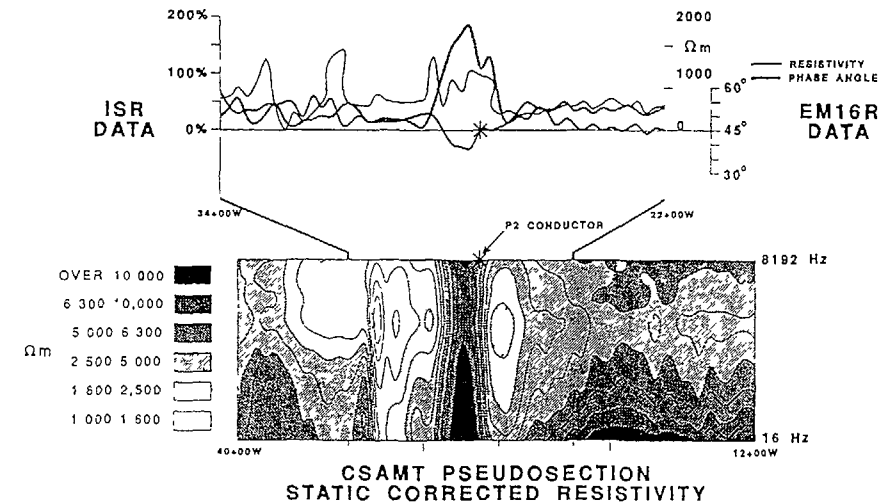


FIG. 11 P2 North (section L76+50N) — geophysical profiles

7. PROJECT HISTORY

Intensive exploration began in the McArthur River area on January 1, 1980 when Cameco's predecessor company, the Saskatchewan Mining Development Corporation, became the operator of the McArthur River Joint Venture. Following eight years of exploration the results of discovery hole MAC-198 allowed the project team to define the significance of weak mineralization encountered in the area by earlier drill holes. During the pre-discovery period numerous exploration targets were defined by geological, geochemical and geophysical methods and tested by 85,000 metres of drilling in 206 holes prior to the discovery of the P2 North deposit (Figure 12).

7.1. Geophysical Surveys

Airborne EM surveys were flown in 1977 and 1978 using the Mark VI INPUT system. These outlined conductive zones, primarily in the eastern and southern portions of the project area, which were followed up on the ground in succeeding years. In view of high noise levels encountered, the deeper, western portion of the project was reflighted in 1981. Although a number of 2 to 3 channel anomalies were located and interpreted to have a potential basement source, scattered 1 and 2 channel responses in the vicinity of the P2 North deposit were not interpreted as basement conductors.

Regional total field magnetic surveys are the most effective method of mapping the sub-Athabasca basement geology. A high resolution total field and gradiometer airborne survey was carried out in 1982 over the eastern part of the Athabasca basin. The airborne

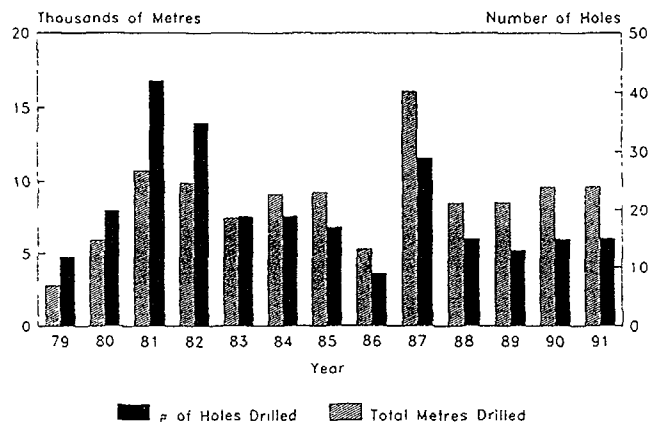


FIG. 12. McArthur River project — drilling statistics.

magnetic coverage was used as an aid in the recognition of metasedimentary basins, to assist in developing a structural interpretation for the area, and in planning ground geophysical surveys.

Ground radiometric prospecting and the magnetic setting of the airborne EM trends were also used to define areas for ground geophysical survey follow-up. Initially ground follow-up of airborne EM anomalies included fixed loop TDEM surveys, primarily DEEPEM, together with magnetic and VLF coverage to assist in mapping lithologies and structure. Gravity and a variety of resistivity mapping methods were also used on a more limited basis.

As the exploration of the property progressed to basement depths in excess of 300 m, large fixed loop TDEM coverage was used in a reconnaissance mode (McMullan et al, 1987) to locate basement conductors. In these deeper areas gravity often proved effective in locating suitable basement geology, and resistivity methods were employed to aid in the definition of alteration zones. The primary follow-up tool has been large fixed loop TDEM coverage. Since 1980 approximately 1500 km of TDEM coverage has been completed from more than 300 loops ranging from 200x200 to 1000x1200 m in size.

At McArthur River, as in other Athabasca Basin projects, borehole Pulse EM surveys have been carried out to detect the presence of conductors missed by drilling, or to resolve a sequence of conductors at depth.

7.2. Technical Landmarks

7.2.1. Harrigan Zone (1980 - 1981)

During the first year of exploration, basement hosted uranium mineralization was discovered at the Harrigan Zone near Vollhoffer Lake. Follow-up during 1981 showed that

the zone was sub-economic. The best grades were 0.52 and 0.55 %U₃O₈ over 5.3 and 4.0 m respectively.

7.2.2. Boulder Trains (1980 - 1984)

Both the Key Lake and Midwest Lake uranium deposits had been discovered by locating the bedrock source of glacially transported boulders.

During the summer of 1980 prospectors discovered 30 - 40 radioactive sandstone boulders on a small drumlin. During 1981 an additional 450 glacially transported radioactive sandstone boulders were identified over 7 km along an esker system. Some assays were greater than 1% U₃O₈. This location is near the northwestern property boundary about ten kilometres southwest of the P2 North deposit. An on-property bedrock source of these boulders was never demonstrated. However, considerable exploration efforts were aimed at this portion of the property.

In 1981 prospectors discovered 128 glacially transported radioactive sandstone boulders over a strike length of 2.4 km at BJ Lake. The head of the boulder train disappeared off of a drumlin and into a sand plain covering the bedrock mineralization source. A second boulder train (BJ East) discovered in 1982 also headed under cover, in this case swamp.

In 1984, in the same area, the "dravite" boulder train was recognized. Since then, hydrothermally altered, but non-radioactive, boulder trains have become important targets for exploration follow-up.

7.2.3. Early Drilling (1982-1984)

During the fall of 1982 a number of holes were drilled at the head of the BJ Main boulder train. These intersected weak sandstone-hosted mineralization with grades up to 0.6% U₃O₈ over 0.5 m. Although basement depths of 400 m had been anticipated, these drill holes intersected basement rocks at 170 m indicating a geological setting of unexpected complexity.

Drilling in 1984 at the head of the BJ East boulder train defined unconformity related uranium mineralization at a faulted pelitic gneiss-quartzite contact below an intense zone of sandstone brecciation. Assay values of 1.44 %U₃O₈/6 m and 0.84 %U₃O₈/6.5 m were returned. There was no basement conductor in this area. Following this discovery, models involving quartzite-related mineralization controls were developed.

7.2.4. Identification of Deep Conductors (1984)

Prior to 1984 geophysical techniques were not being routinely used to map conductive lithologies where the sandstone cover exceeded 400 metres. It was thought that conductors would not be identified at these depths. As a result of improved instrumentation and methodology basement conductors can now be routinely detected under more than 700 m of sandstone cover.

Exploration in the P2 area accelerated in 1984 with the discovery of the P2 conductor using reconnaissance DEEPEM coverage employing loop dimensions of 400x800 m. In this area unconformity depths exceed 500 m. Definition of the entire P2 conductor target system

was completed in 1986. The open-ended conductor stretched for 12 kilometres on the property and became a high priority exploration target.

7.2.5. P2 Footwall Concept (1985)

The first indications of significant structural and hydrothermal alteration coincident with the P2 trend were identified during 1985. Drilling confirmed significant vertical offsets of the Athabasca unconformity along the P2 fault and found impressive structures and hydrothermal alteration on the footwall side of the fault. Future drill holes were collared to test terrain located some 50 - 100 metres from the geophysical conductor axis on the northwest (footwall) side.

7.2.6. P2 Main Zone (1985 - 1987)

Drilling of the footwall side of the P2 fault during this period culminated in the discovery of significant unconformity related mineralization at the P2 Main Zone. Assays of up to 1.38 %U₃O₈/7.3 m were returned from this sub-economic deposit which has a strike length of 500 m. The alteration and structural characteristics at P2 Main are similar to those identified at BJ Lake and a common genesis was postulated. The P2 Main Zone marked the P2 fault trend as having the potential to host significant concentrations of uranium mineralization.

7.2.7. A Near Miss (1985)

Several reconnaissance holes tested the conductor in the vicinity of what is now the P2 North deposit. Hole MAC-138 was drilled 100 metres away from the deposit on the hanging wall side of the P2 fault. The hole encountered elevated lithogeochemical uranium and boron values. The drilling results were not viewed as being significant enough to warrant immediate drilling followup, attesting to the elusive nature of a deep Athabasca Basin discovery.

7.2.8. Further Drilling (1987-1988)

In 1987 both the BJ and P2 Main mineralized areas were drilled in more detail. The results were disappointing and both zones were classified as being sub-economic. The 1988 program was intended to complete first pass drill testing of the P2 trend. The general consensus was that if results were negative, then further drilling along the P2 trend would fall to a lower priority. Additional drilling along this structure could well have been delayed for several years.

7.2.9. The Discovery (1988)

In July, 1988 drilling on the northern portions of the P2 conductive trend intersected intense sandstone hosted structural disruption and hydrothermal alteration in hole MAC-195. A major structure related alteration system had been intersected on the hanging wall side of the P2 fault. Hole MAC 196 collared 100 m to the northwest of hole MAC-195 intersected weak uranium mineralization and strong sandstone-hosted alteration and

structure on the footwall of the P2 fault. The next hole, MAC-197, intersected material of somewhat higher grade, but it was not until hole MAC-198 intersected grades of 4.27 %U₃O₈ over 10 m that a major high grade deposit was suspected. This was confirmed by subsequent drilling of 13 holes in 1989 and 15 holes in 1990.

8. CONCLUSION

Further drilling in 1991 has completed holes 50 metres apart along the southern third of the deposit which contains the highest grade ore, together with one cross section of 5 drill holes spaced about 10 metres apart. Although chemical assays are not available for this recent drilling program, radiometric probe data indicate that uranium reserves will exceed the original estimate of 200 million pounds at 4% U₃O₈.

The timing of mine development will depend on market conditions and other factors. The joint venturers intend to prepare an environmental statement and mining proposal based on the assumption that McArthur River ore will be processed at the existing Key Lake facility where reserves will be depleted before the end of the century.

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SERMINE: A SOFTWARE FOR ORE DEPOSITS EXPLORATION AND ESTIMATION

J P BENAC, D DELORME,
C DEMANGE, H SANGUINETTI
Cogema,
Vélizy-Villacoublay,
France

Abstract

SERMINE is a Cogema software built in response to internal needs and has currently been used for seven years in exploration, mining survey and estimations

SERMINE integrates a complete and coherent chain of programs whose application fields range from acquisition of basic data describing orebodies to the most advanced geostatistical estimations and simulations, with the continuous assistance of graphical displays

The system is user friendly, integrated, open and efficient

1 - GENERAL PRESENTATION OF THE SERMINE SOFTWARE

1.1 WHAT THE SERMINE SOFTWARE IS . ORIGINS AND DEVELOPMENT

SERMINE software is a complete tool capable of assisting the mining geologist in carrying out all his tasks.

In fact SERMINE integrates under a single menu some thirty modules which contain over one hundred programs whose application field ranges from the input and acquisition of the basic data to the most advanced geostatistic methods, such as the anamorphosis of distribution laws or conditional simulations

The considerable scope of these applications is fortunately supplemented by a *very user-friendly interface giving the programs a simplicity of implementation rarely achieved in this type of software product*. This enables most of the basic tasks to be entrusted without any problem to users with little or no training in data processing

This convenience and simplicity stems from the history of the SERMINE software, which was designed and developed by a team of geologists and geostatisticians of a mining company, COGEMA, within its "Studies and Reserves Division"

This project began in 1984 in response to the requirements of the mining divisions and the international subsidiaries of COGEMA

Since that time, the software has benefited from the suggestions and criticisms of the users which have greatly contributed to its convenient use

Since that time also, the "Studies and Reserves Division" of COGEMA has supplemented this software as a result of new working methods developed chiefly by the Geostatistical Centre of the National Mining College of Paris in Fontainebleau

1.2 MINIMUM HARDWARE REQUIRED

The system was first developed on HP 9000 serie (HP 200, 300) with HP Pascal System. It was then extended on the same serie to HP 400 with Unix (X-Window). In the same time, the software was supplied on Harris (series 800 and 1000) with VOS or Unix system. More recently, one version was realized for IBM PC and compatibles with MS-DOS system. Currently, it is also operating on Intergraph workstation with Unix

In each case the minimum configuration required is

- * RAM memory, 640 Kb minimum
- * Mathematic coprocessor of the type Intel 80387 or equivalent
- * VGA graphic card, 256 Kb or more
- * Highly recommended colour monitor
- * Hard disk with a minimum capacity of 20 Mb (the executable files and the DOC files, accounting for approximately 10 Mb)
- * Diskette unit, format 3 1/2" 1.4 Mb or 5 1/4" 1.2 Mb
- * Microsoft or compatible mouse recommended

Useful devices

- * 80 or 240 column parallel printer for the printer outputs and check printouts
- * HP-GL compatible plotter for creation of maps and other graphics. However, since the plotter outputs are provided in the HP-GL file format, it is possible to transfer these files onto a diskette and to make a plot of them from another MS-DOS system equipped with a plotter supporting the HP-GL language

1.3 DESCRIPTION OF THE USER INTERFACE

1.3.1 Choice of Language

It is possible to select the language for the menus and messages displayed by the SERMINE software (excluding of course the system messages). Two selections are available : French and English

1.3.2 Menus of the SERMINE programs

Despite the very large variety of functions performed by the programs in the SERMINE software, the menus were designed to be displayed in a similar manner to avoid the frequent reference to the instructions, or tedious learning

The standard menu consists of a list of options

1.3.3 Access to the programs

Two methods of access are possible :

- the use of a general menu, listing the programs in the software, with a summary of the functions provided ;
- directly calling the programs by their name from the system command line. After execution, return to the system command line.

1.3.4 Work units

- * Unless otherwise indicated, the SERMINE programs use the data in the files as such without distinction of unit other than for the angle data (necessarily degrees in the file, even if it has been possible to use other units during inputs).

However, in order to display the symbols and names of units on the graphs, listings, messages, it is necessary to indicate the work units if they differ from the default units.

The default units are :

for lengths : the meter (m)

for grades : the "per thousand" (‰)

It is possible either to change the symbol only, or to use a conversion factor in order to have a "display unit" which is different from the "user unit".

1.3.5 Usual messages

- Saving of the parameterisations

Several programs save the last parameters which have been supplied to them, in a file format, and propose them, as defaults, for subsequent execution (even in the case of another work session provided that it is in the same directory).

- Activation of the check printout

These are listing outputs ("draft" quality) enabling the current operations to be checked. They are generally used to detect errors (wrong input file, etc.).

- Validations

Most of the programs and options include a validation at the end of the menu or parameterisation ; working through the menu may sometimes seem tedious. In this case it is possible to skip the parameterisation by means of the direction arrows and stop at the VALIDATION question, with an acoustic signal.

2 - THE DATA FILES IN THE SERMINE BASE

DESCRIPTION AND RECOMMENDATIONS FOR USE

2.1 THE TWO FILE FORMATS ; TEXT AND BASE

All the data files handled by the SERMINE software have the same overall organisation.

Two types of information coding coexist :

- The TEXT format, corresponding to ASCII files, in which the data are in "plain text", visible and handled by "text editors".

This format is particularly useful for interfacing with other systems and software, and is recommended for recording.

- The BASE binary format in which the data are in the form of a code, depending on the system used.
In order to read them it is necessary to use suitable programs.

Most SERMINE programs operate with these more quickly accessible and more compact formats, in terms of memory.

These BASE files are organised according to the Sequential indexed type with forward linking.

These technical aspects of the BASE format are almost transparent to the user thanks to the provision of a certain number of utility programs enabling him to read and modify the files as if they were in "plain text".

2.2 ORGANISATION OF THE DATA FILES

2.2.1 Structure of the files

Generally speaking a data file comprises :

- an area specifically designed for the system and for the software (see reference manual), transparent to the user, and not described here.
- a "data" area consisting of a set of data registers, this area being in some respects the part of the file which is "visible" to the user.

Each register contains the information relating to the same type of data and/or data of the same origin, etc.

For a drill hole, for example, there will be different registers for :

- the scroll (location information, dates, etc.)
- the technical heading area (casing, etc.)
- the deviation measures

- the measures or analyses
- the text descriptions or others
- etc

2.2.2 Structure of the registers

A data register is divided into two areas :

- the HEADER area, containing the numbers, locations etc , on three file lines.
- the VALUE area containing the numeric, alphanumeric or text information and the numeric elements enabling this information to be identified geometrically relatively to the point of origin recorded in the header (drill hole collar, etc)

2.2.3 Classification, selection and masking of the registers

Classification of file registers :

The acquisition of data and "creation" of a file from other files may be executed in a sequence totally different from the drill hole numbers, locations, data types, etc. It is therefore often useful to be able to classify the registers to obtain ordered lists, and the classification is even mandatory for executing certain operations such as merging several files into one

Moreover, certain graphic programs will be much faster to execute when used on the basis of classified files.

Selection and masking of the registers of a file :

The same file may contain registers relating to information of different origin and type.

It is often useful to make selections from locations, numbers, analysis types, etc.

Numerous programs in the SERMINE software provide a SELECTION option for all or part of the KEY, and also for other parameters (coordinates, values, etc.)

It is sometimes more useful to use the MASKING, which is the logical complement of the selection

For example, with SELECTION the registers which meet the fixed criteria are extracted ; with MASK the registers other than those meeting the same criteria are extracted.

2.2.4 Organisation of the measures in the registers

A "measure" means a "line" in the "measure" area of the register

The maximum length of the "line" is 99 words of 4 bytes

Where appropriate the measure "line" will be subdivided into true file lines, in order to conform to the maximum length of 80 characters permitted in a certain number of manipulations.

2.3 GENERAL PRESENTATION OF NETWORKS AND NETWORKS FILES

2.3.1 Definitions

In the SERMINE program a network is a discretised representation of two or three dimensional spaces using regular networking. This networking divides this space into rectangular cells which can be given a thickness for calculating tonnages (2D) or in parallelepipedic blocks (3D)

To these blocks or cells will be assigned variable values defined in the space thus produced (grade, grade thickness, ore quantity, etc)

3 programs proceed to network 2-D or 3-D data

LOGNEC splits and organizes drill hole data within a 3 D block network,

- PACNEC transfers impacts in a 2-D block network,
- ISOS composites punctual data into a 3-D network

The networks are to be coded in BASE or TEXT files in the same way as all other SERMINE data

However, the structure of the network file registers will be particular, and specific tools will be necessary to carry out computations and transformations on them.

2.3.2 Network location parameters

The general case for 3D networks is considered here, the case for 2D networks being a special one of the former.

The elements of the networks (the blocks) are arranged in columns indexed along the x axis, in rows indexed along the y axis and in benches indexed along the z axis (x, y and z of the network) and mapped relatively to the point of origin of the network.

The method of mapping used enables the network to be "adjusted" on objects to be displayed by breaking free of the constraints of the geographical coordinate method, while retaining the option of reestablishing the link at any time.

This is important, not only from the geographical viewpoint but also from the viewpoint of optimising the size of files, which is very appreciable, particularly with the DOS system, where the size of the 3D files can quickly become critical because of the available memory size. Moreover, the computation time which depends on the number of blocks may also be reduced through this method.

3 - INPUT PROGRAMS

Four programs in this category permit the creation of data base file for drill hole or mining data. These programs are very easy to use, with friendly looking menus, default values proposed, checking of the data down the screen.

The DONLOG program allows to input casing descriptions (depth, diameter and thickness of the casing, fluid), measures with incremental or variable steps (samples analysis, deviations) and text descriptions (lithologs, colour).

The GRILLE program is designed for a quick acquisition of drill hole data in the systematic grid of a bench in an open pit.

PRODUITS was initially intended to automatize the computation of trucks production of uranium ore, sampled radiometrically with a scanner. It can be used for any other production data.

IMPACTS allows the input of impacts from drill holes, or stopes.

4 - BASE FILE MANAGEMENT

This module is essentially made by classical utilities operating on file: change name, create new, merge, delete, give information on file or system, list of file, extract, correct by way of a powerful editor.

The most interesting options are probably the selection and masking and especially the possibility of generating a Base File using an uncompressed ASCII file data for an other database.

More specific are the programs of transformation of the drill hole data. UTILOG enables: erosion above minimum, compositing on fixed length, merging with deviation measures geological or sampling information, transformation of coordinates.

DEFPAC defines interactively and graphically the stopes corresponding to drill hole impacts.

5 - OUTPUT PROGRAMS

They are essentially graphical tools.

PLAN is for horizontal maps of drill hole data with possibilities of window, grid, colouring function of the grade, slice definition (Fig 1).

GEOS makes all cross-sections and maps, with any strike or dip and with creation and modification of patterns used (Fig 2).

PLOTPAC allows projections of drill holes impacts with the same options as PLAN.

CROSSNEC visualizes and edits horizontal and vertical sections of a network (Fig 3).

ISOS enables isocontouring of networked punctual data (Fig 4).

STARLOG shows the drill hole logs (Fig 5).

But there are also programs to compute classical statistics (histograms, (Fig 6), correlations) or grade thicknesses.

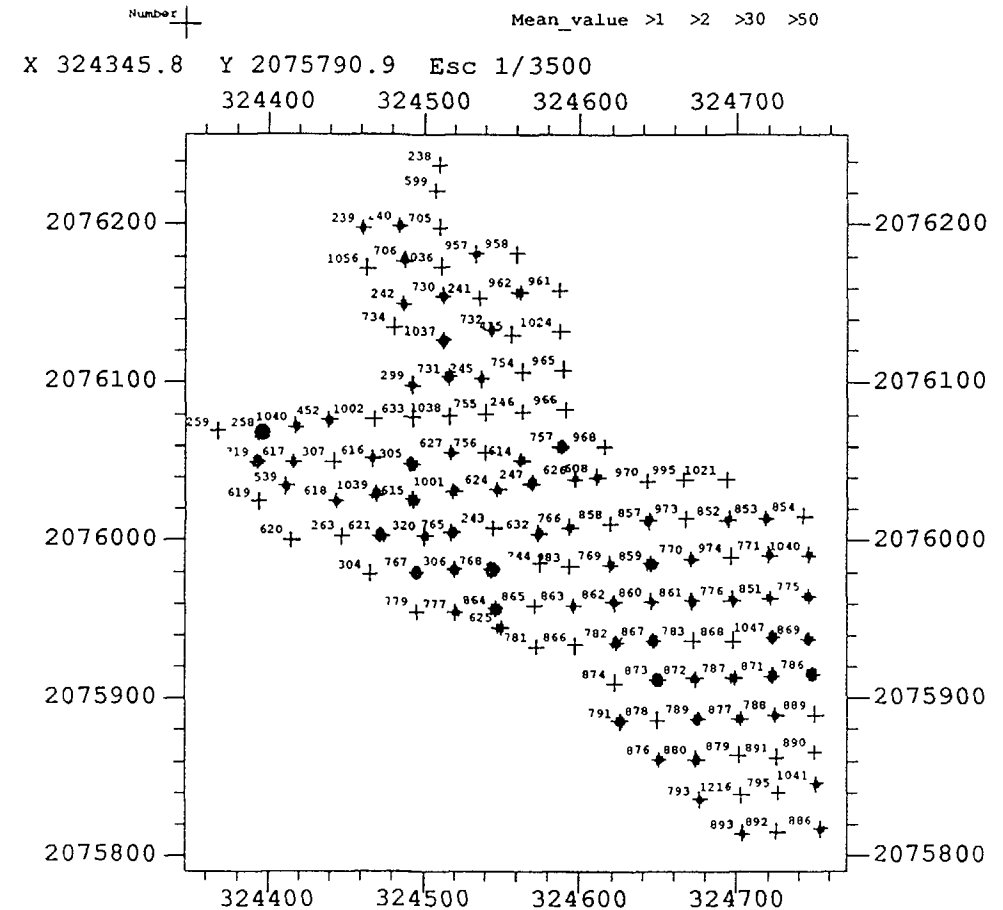


FIG 1 Holes location map

Strike=0 Dip=90 X=800 Y 750 Z 850

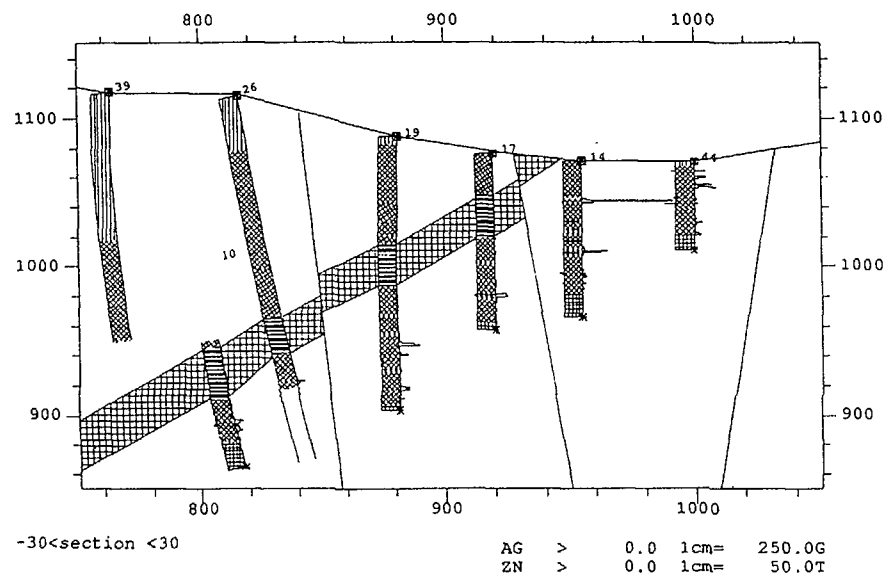


FIG. 2. Geological cross-section.

6 - NETWORK HANDLING

This module is necessary to carry out any estimation and includes 2 main programs :

- UTINEC - edits network values,
- enables the geometric characteristics of the network to be completely redefined,
 - transfers, extracts and merges networks,
 - directly computes network values.

SELNEC is particularly efficient and defines selections from the values of a network variable by :

- mathematical operators,
- morphological operators,
- interactive graphic plots,
- transposition of another selection.

Network

KRIGED GRADE

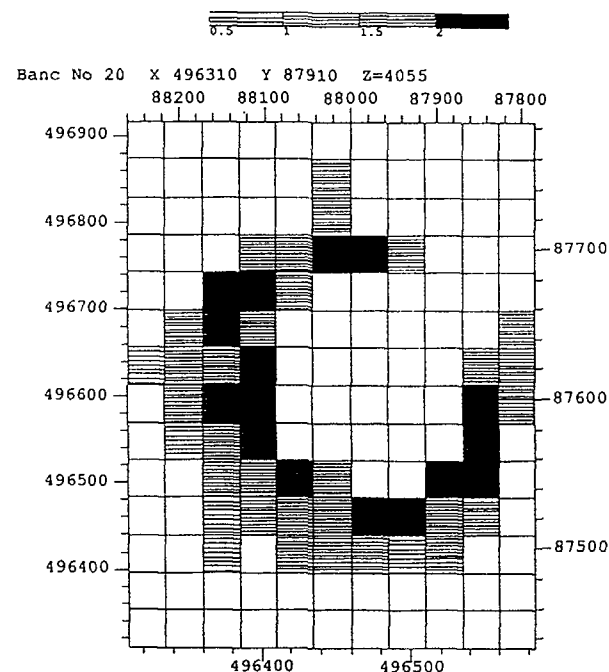


FIG. 3.

7 - LINEAR GEOSTATISTICS PROGRAMS

VARIOS enables the variogram to be computed and fitted on the basis of all the numeric data types and all the regularization modes offered by SERMINE. (Fig 7)

The computing is carried out according to the main and diagonal directions of the network used. It is possible to introduce an angle and distance tolerance (0 to 100%), to select the maximal length to be computed (maximal lag), the computing lag of the variogram, and to select only the values beyond a certain threshold for the calculation.

The experimental variogram is fitted by means of a spherical function, with the possibility of using four structures, including nugget effect, and to introduce anisotropy coefficients for each structure.

The variogram is fitted directly on the screen by positioning a cursor on the experimental curve.

- ALTITUDE OF OVERBURDEN
X 585005 Y 6470200 Z=314.5

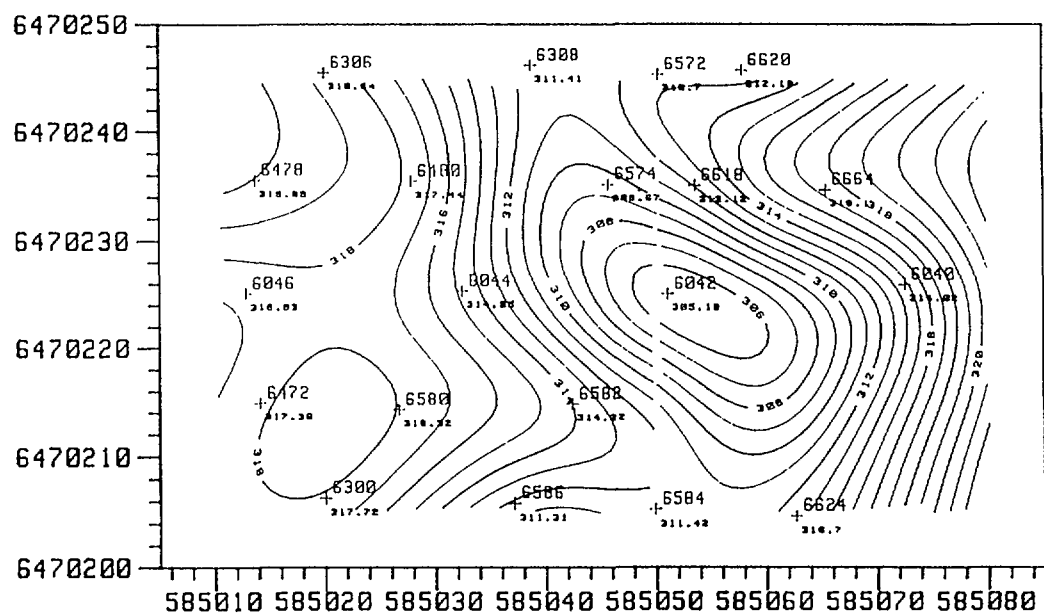


FIG. 4. Contour maps for geological interpretations, grade visualization and topographic surveys.

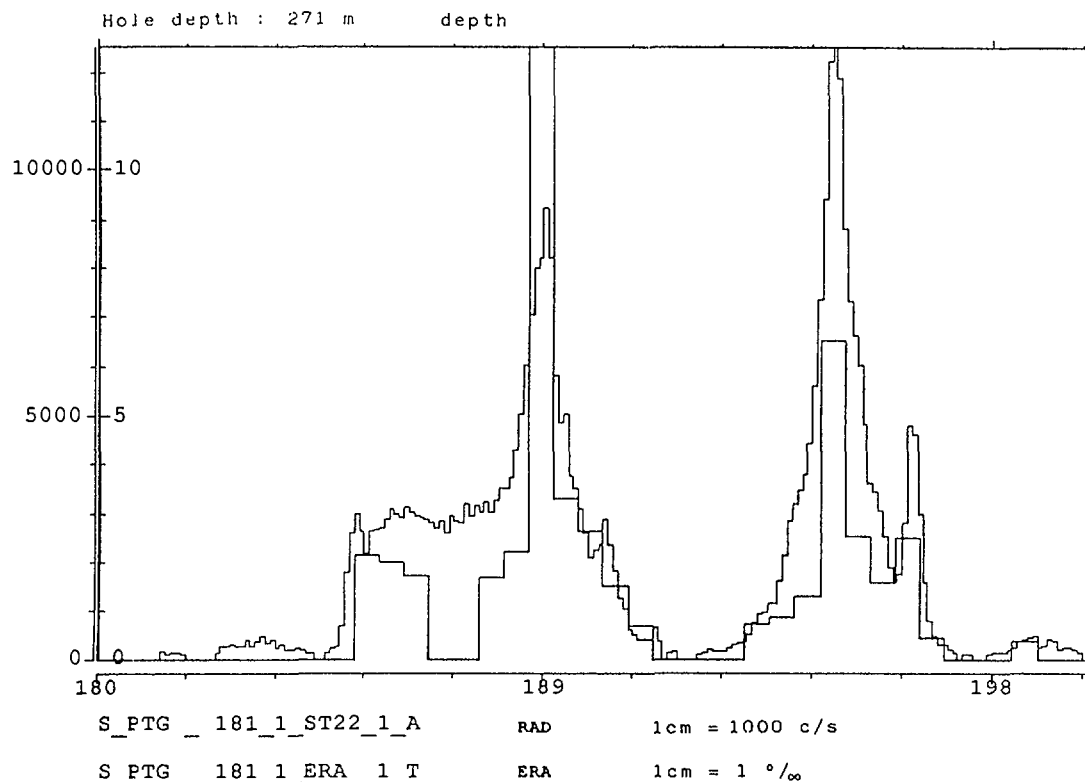


FIG. 5. Radioactivity and radium equivalent logs.

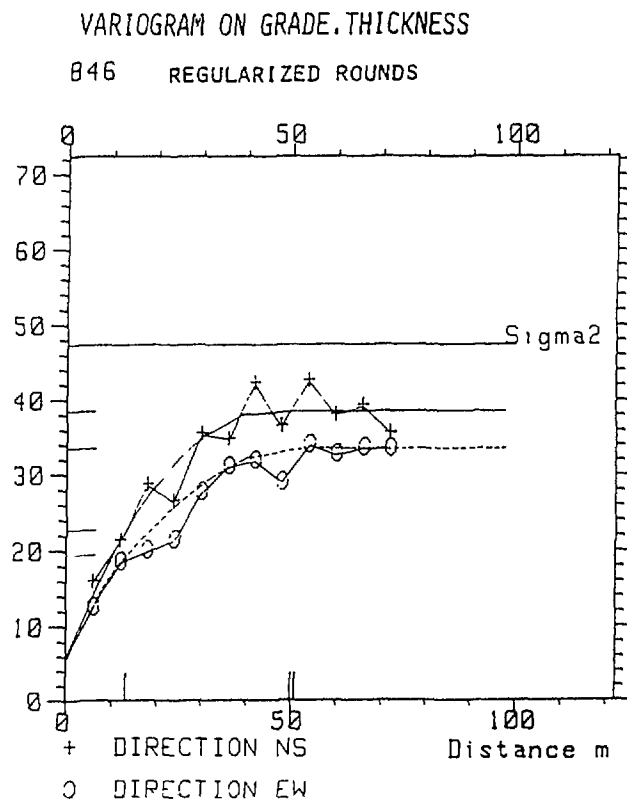
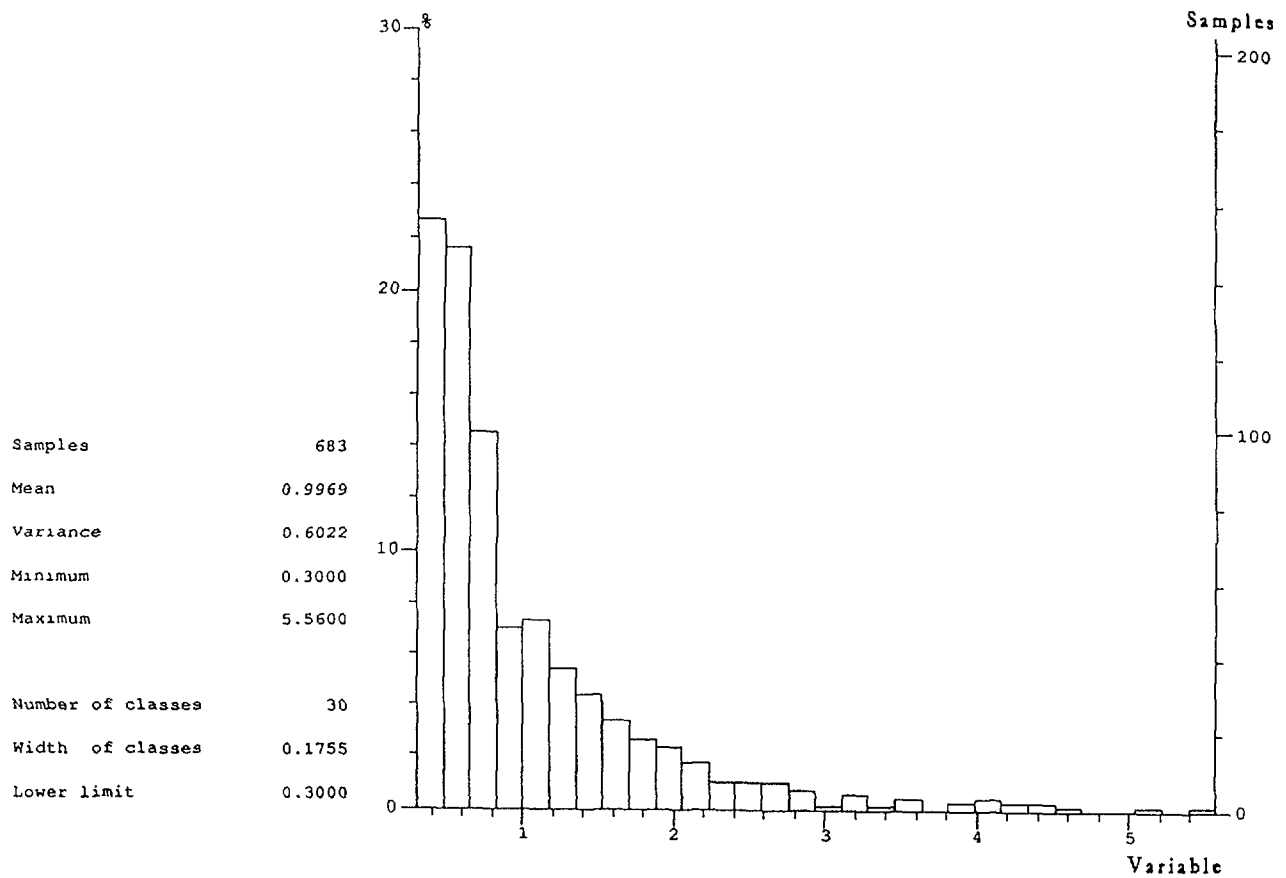


FIG. 7. Interactive variogram fitting: preliminary stage of geostatistical calculation.

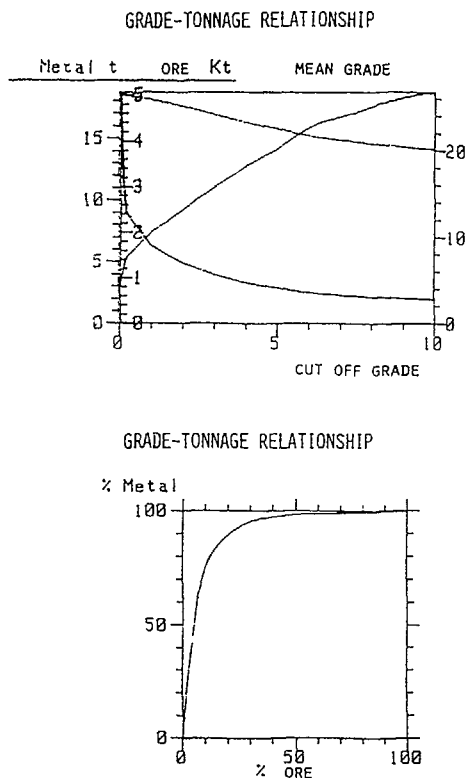


FIG. 8. Grade-tonnage curves: a useful graphic output from ore deposit estimation.

Estimation is made by KRINEC.

The kriging is carried out on all unmasked blocks according to the criteria imposed in selecting the kriging configuration. There are 10 pre-established configurations. The kriging may be simple (assignment of one weight to the mean value) or ordinary.

The maximum number of informing data for one block is set to 27. It is possible to store the variance and print the weights simultaneously.

The TONNAGE program is used to print the results. One can select up to 10 cut-off grades. Several graphs can be obtained : metal/ore, metal-ore/cut-off. (Fig 8)

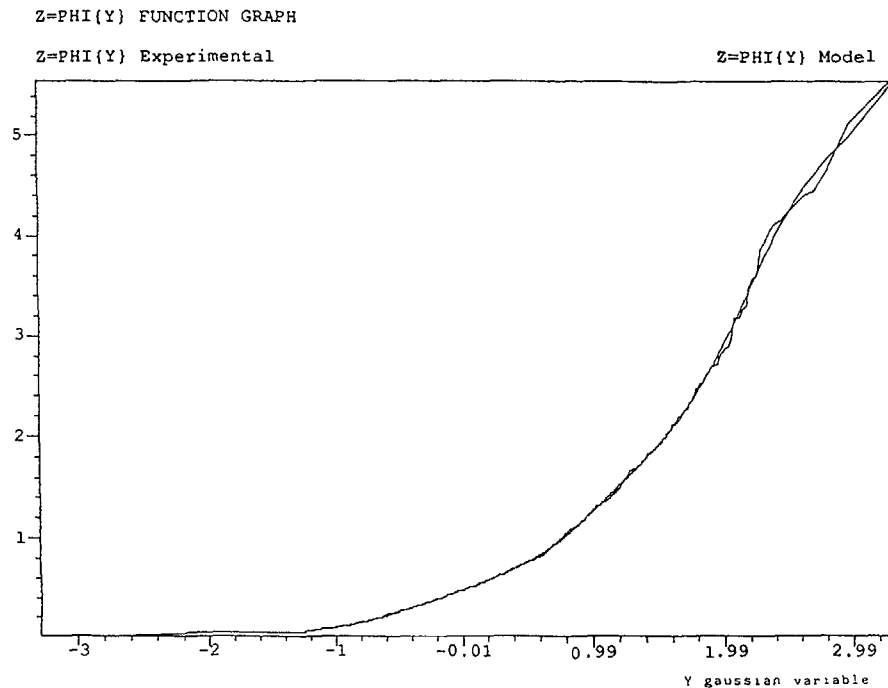


FIG. 9. Anamorphosis.

8 - NON LINEAR GEOSTATISTICS

The program UTIGAM allows computation of average variogram by using auxiliary functions.

ANAGAUS program groups the operations using Gaussian anamorphosis, and its fitting in Hermite polynomials, particularly for executing a global estimation of the recoverable reserves. (Fig 9) The change of support coefficient enables to proceed from sample histogram to a block histogram. It is then possible to compute the grade-tonnage relationships by applying different cut-off grades for different block sizes. Covariances from raw values or Gaussian values are also available.

Three methods are used for local estimation : VUTIL defines two service variables in a Gaussian model ; after study of variograms, these variables are kriged. This method was first developed by Cogema. CUNIF is an application of the well known uniform conditioning method and KD makes disjunctive kriging.

9 - CONCLUSION : The future of Sermine

SERMINE is a software undergoing constant development. Different projects currently studied can be cited:

- Connection of Sermine data with Intergraph workstation data. The Argos Software that displays mining operations and geological data in 3-D is presently tested at the Lodeve Mine in France.
- Other tools for estimation are under study: 2-D and 3-D conditional simulation, gamma models, orthogonal Indicator Residual Models.

RECOVERABLE RESERVES ESTIMATION USING SERMINE SOFTWARE: CASE HISTORIES

C. DEMANGE, H. SANS

Cogéma,
Velizy-Villacoublay,
France

Abstract

Two case studies illustrate the use of the Service Variables method for the estimation of local recoverable reserves.

In the first one the estimation of a deposit can be followed step by step.

In the second one the estimation and production figures are compared and the discrepancies analysed.

1 - THE PUY DE L'AGE DEPOSIT

1.1 Geology and development

The small uranium ore bodies of Puy de l'Age are currently mined by Cogéma, in the northern part of the French "Massif Central" (Fig. 1).

Several larger deposits occur in the vicinity: Bellezane, Peny, Margnac and Fanay among others.

All these deposits are located in the granitic complex of Saint Sylvestre, intruding metamorphic series: anatectic gneisses of Thaurion and micaschist. On its northern margin, the Saint Sylvestre granite is in contact with the older Guéret granite.

The batholith of Saint Sylvestre comprises three units from west to east:

- the granitic gneiss of La Brame
- the leucogranite of Saint Sylvestre, geochemically enriched in uranium (17-24 ppm) and host of the uranium deposits
- the albitized Saint Goussaud leucogranite

The area of Puy de l'Age extends over three hundred meters from north to south and extends 200 m down-dip.

The leucogranite is cut by mafic subvolcanic rocks (lamprophyres or "minettes") which define three main mineralized areas.

The mineralisation occurs in veins or episyenites with subvertical enriched columns. The diameter of the columns is generally small (5 to 10 m) but their grade can be locally very high.

The southern part of the deposit was discovered first. Underground mining methods were used to mine the deposit below the 80 m depth while the upper part will be exploited by open pit.

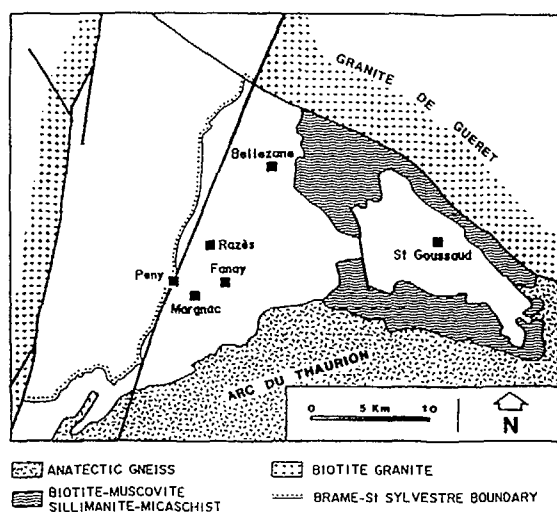
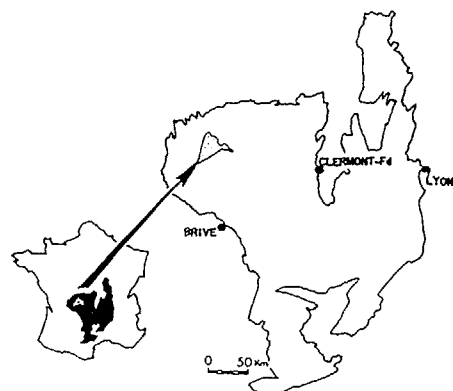


FIG. 1. Saint Sylvestre granitic complex.

1.2 Data

The deposit has been studied with 150 inclined drill holes on a regular 10 x 8 m spacing (fig 2). Radiometric data were collected by down hole probing every 10 cm. In a first step the radiometric data are transformed into grades through the correlation curve :

$$T = 0.7 Ra^{1.05} \quad \text{with } T = \text{grade in } \text{‰} \\ Ra = \text{radioactivity in AVP} \times 1000$$

The grades are then composited every 2.50 m and classified in a network comprising 15 x 19 x 14 panels of 8 x 5 x 5 m.

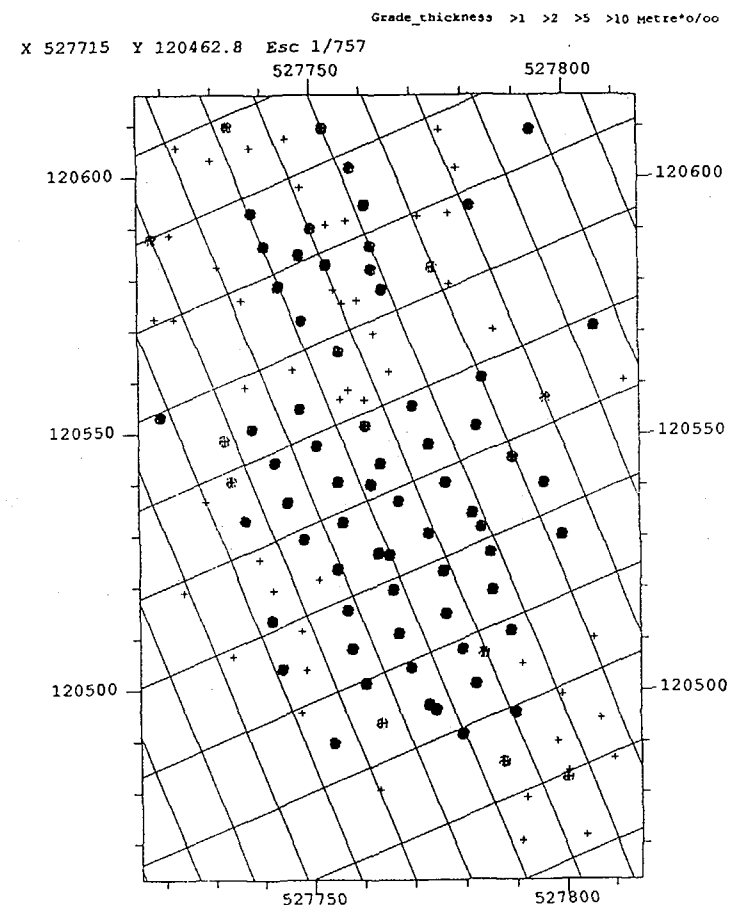


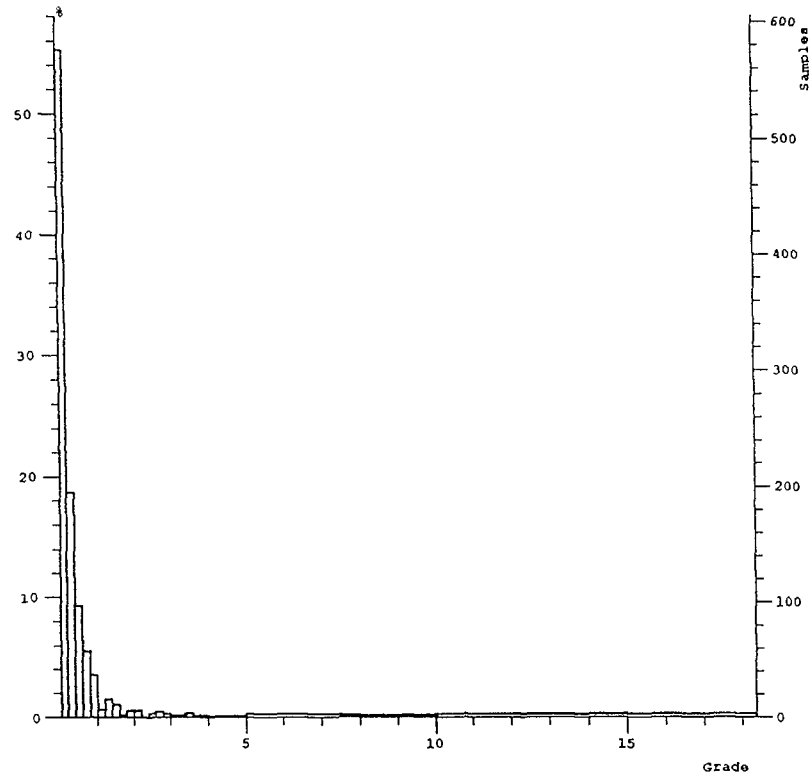
FIG. 2. Puy de l'Age location map.

The limits of mineralization are then defined on benches and sections, taking into account the geological interpretation of the deposit.

1.3 Statistics

The global statistics are displayed in Fig 3 : number of samples 1044, mean 0,42 ‰, variance 1,11 (‰)². The histogram is very skewed and the grade dispersion is important.

$$\text{ratio } \frac{\sigma}{m} = 2.54$$



Samples	1044
Mean	.4152
Variance	1.1126
Minimum	.0200
Maximum	18.3700
Number of classes	24
Width of classes	variable
Lower limit	.0200

FIG. 3 Puy de l'Age experimental histogram.

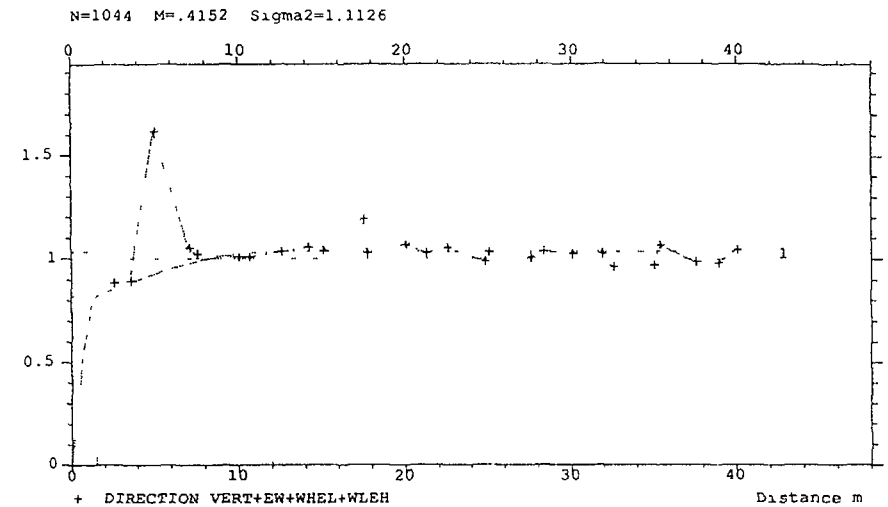


FIG. 4. Puy de l'Age variogram

1.4 Variogram of 2.50 m regularized grades

The variograms are calculated along the main directions of the network : NS, EW and vertical. No peculiar anisotropy can be defined and an isotropic model is adjusted according to two spherical nested structures with respective ranges of 1.5 and 12 m (Fig 4) and sills of 0.83 and 0.28.

The variogram reproduces the erratic nature of the mineralization. The short range (1.5) deduced from short distance information along the drill holes, and also from experience in the underground mine, represents 75 % of the global variability

1.5 Global estimation of recoverable reserves

1.5.1 Gaussian anamorphosis

The grade $Z(x)$ on 2.5 m support does not correspond to a simple known distribution law that can be parameterized (for example a log normal law is not sufficiently dispersed). It is therefore necessary to apply to the experimental distribution a transformation called anamorphosis which transposes each $Z(x)$ into a gaussian equivalent and allows to work with an easily parametered law.

In practice, one works with cumulated histograms called function of repartition. to each z is associated a y of identical cumulated frequency. Then the transformation Φ relating y and z (anamorphosis function) is modeled with a Hermitian polynomial development.

$$Z(x) = \Phi^{-1} Y(x) = \sum_{i=0}^n \frac{\psi_i}{i!} H_i Y(x) \quad \text{with } n = 20$$

Z=PHI(Y) FUNCTION GRAPH

Z=PHI(Y) Experimental

Z=PHI(Y) Model

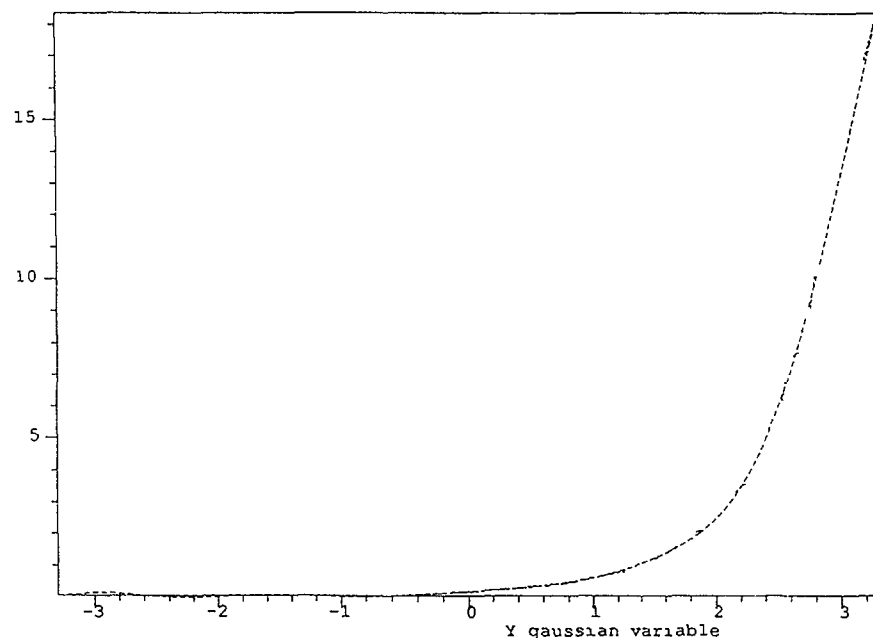


FIG 5 Puy de l'Age anamorphosis

1.5.2 Change of support

The objective of this step is to choose the appropriate block size v that corresponds to the selectivity of the mining method. Such a block has not strictly a physical meaning since it integrates several dilutions and sortings occurring between the blasting and the stockpiling.

A size of $2.5 \times 2.5 \times 2.5$ is adopted here.

The variogram modelization allows to calculate the distribution variances of the blocks v using the Krige's relationship :

$$D^2(Zv) = D^2(Z(x)) - \bar{\gamma}(v,v)$$

with $D^2(Zv)$ = dispersion variance of blocks v

$D^2(Z(x))$ = dispersion variance of the regularized samples (2.5 m)

$\bar{\gamma}(v,v)$ = mean variogram within block v

$$\text{Application } D^2(Zv) = 1.11 - 0.741 = 0.369 \text{ (}\text{°/°}\text{)}^2$$

The variance has diminished by 67 %

The change of support coefficient r can therefore be calculated

$$\text{given } D^2(Zv) = \sum_{i=1}^n \frac{\Psi_i^2}{i!} r^{2i}, \quad r = 743$$

Knowing the coefficient r allows to calculate the block distribution according to the relations :

$$Z(v) = \Phi v(Yv) = \sum_{i=1}^n \frac{\Psi_i}{i!} r^i H_i(Yv)$$

1.5.3 Results

The global recoverable reserves can then be calculated for different cut offs. As a matter of fact, applying the cut-off z_c to the bloc distribution is equivalent to applying the cut-off $y_c = \Phi^{-1}(z_c)$ to the gaussian distribution.

then for the ore : $T(z_c) = 1 - G(y_c)$
for the metal $Q(z_c) = \int_{y_c}^{\infty} \Phi v(y)g(y)dy$ easily computable with the Hermitian polynomials

g = density of the gaussian law

G = cumulative gaussian distribution

The results are displayed in Fig 6 and compared with those corresponding to a selection on 2.5 m samples. This last selection represents an ideal but technically unfeasible result that is taken into account in a polygonal estimation.

SUPPORT	ORE kt	METAL (t)	GRADE °/°°
2.5 m composites	148.1	124.8	0.84
blocks $2.5 \times 2.5 \times 2.5$	182.2	121.1	0.66

1.6 Local estimation of recoverable reserves using the Service Variables Method

Two variables are estimated in each $8 \times 5 \times 5$ m panel

- the probable proportion of mineralized $2.5 \times 2.5 \times 2.5$ m blocks (TMIN variable)
- the corresponding quantity of metal (TREC variable)

This method is described by J.F. Bouchind'homme in his thesis (1) and summarized in appendix

The variables TMIN and TREC are calculated for each 2.5 m sample, given its grade and a 2 °/°° cut-off

Samples and 2.5*2.5*2.5 blocks

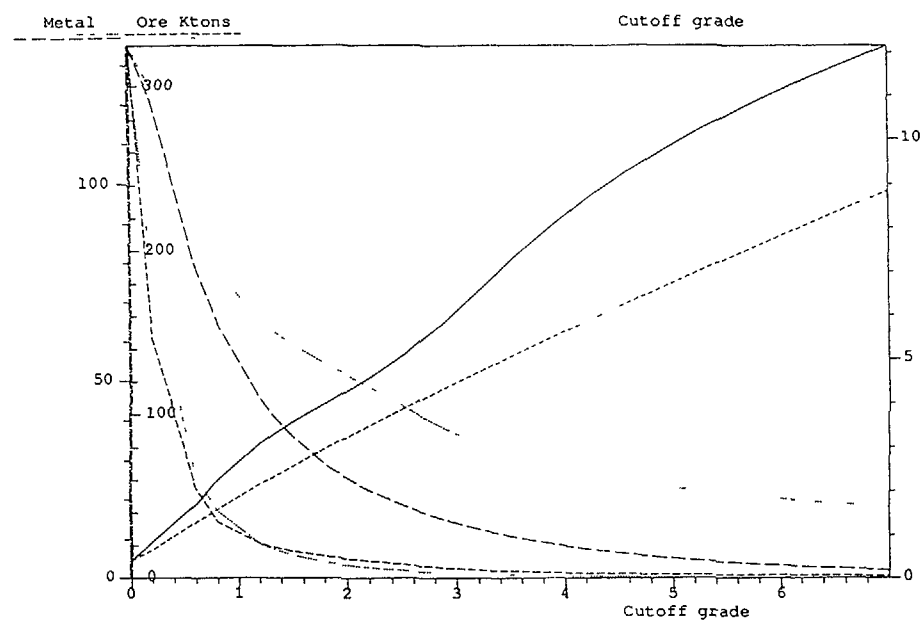


FIG 6. Puy de l'Age — grade-tonnage relationship

1.6.1 Statistics

	TMIN	TREC
mean	55.29	37.02
variance	1158	2182

It can easily be demonstrated that the average recoverable grade tm can be expressed as

$$tm = \frac{TREC}{TMIN} = \frac{37.02}{55.29} = .67 \text{ which is very close to result for global reserves}$$

1.6.2 Variograms TMIN - TREC (Fig 8)

The variograms of service variables TMIN and TREC are calculated and modeled with two isotropic spherical structures

		Spherical 1	Spherical 2
TMIN	sill	650	509
	range	3	18
TREC	sill	1500	682
	range	3	16

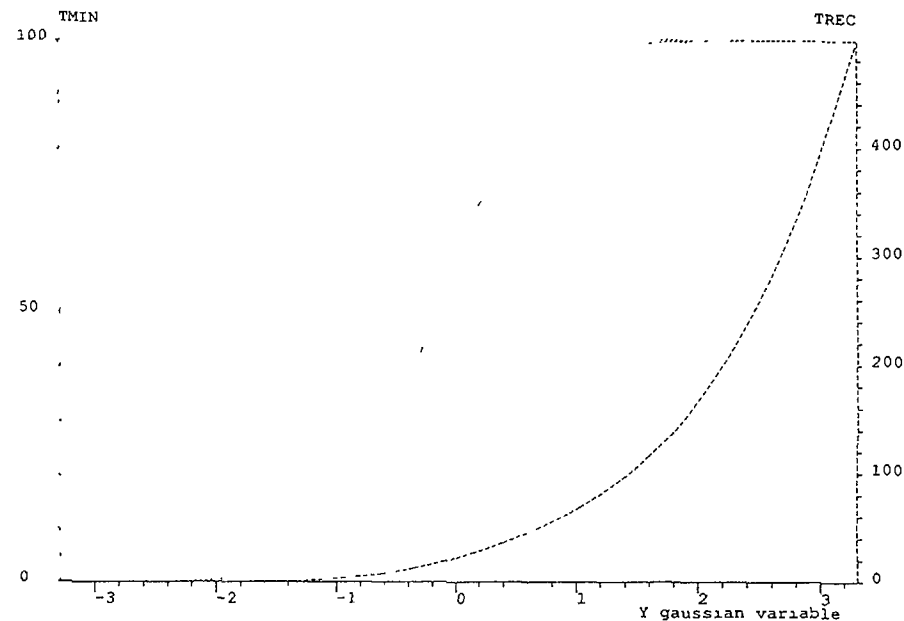


FIG 7 Puy de l'Age — TMIN and TREC graph as a function of Y

The TREC variable is less structured than the variable TMIN. As a matter of fact the ore variable (TMIN) corresponding to the envelope of the mineralization at a .2°/°° cut-off is more continuous than the metal variable that incorporates the grade variations within the envelope mineralization.

1.6.3 Kriging of TMIN and TREC

The average values of each service variable are estimated by kriging in each 8 x 5 x 5m panel. The following neighbourhood is chosen :
5 x 5 m panels coplanar with the kriged panel
3 x 3 m panels on the first upper and lower benches

The ore and metal recoverable by panel as well as the average grade are easily computed from the kriged values of TMIN and TREC.
The sums of pannels values give a result consistent with the global estimation :
(180 kt ore, 0.67 °/°° - 121,7 T metal)

1.7 Conclusions

This method has been applied at Cogema for 10 years in several types of deposits including sedimentary and vein type environments. It estimates satisfactorily as far as the geological interpretation is not neglected and can by the way provide a good definition of the mineralized limits.

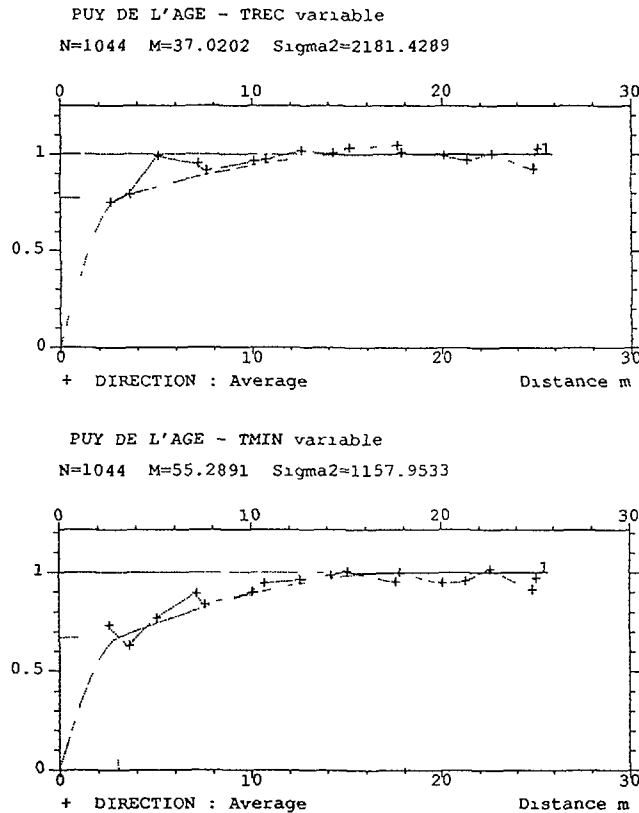


FIG 8. Puy de l'Age — TREC and TMIN variables

2 - DOMINIQUE-JANINE DEPOSIT

2.1 Geology and development

The orebody is currently mined by open pit (fig 10) by Cluff Mining, a partnership between AMOK Ltd. (80 %) and CAMECO Corp. (20 %), near Cluff Lake (Northern Saskatchewan). It was discovered in 1984, on the west side of the Dominique-Earl River Complex dome. The mineralization lies immediately below the overburden to the east and as deep as 65 m to the west. The ore is associated with subvertical fractures and faults as well as westerly dipping thrust faults within a mylonite zone, near the contact between the Peter River Gneiss and the Earl River Complex.

The mineralization is primarily confined to a wide tectonic zone ("zone à boules") characterized by large rotated and rounded blocks of barren gneiss enveloped in mineralized gouge and/or clay alterations.

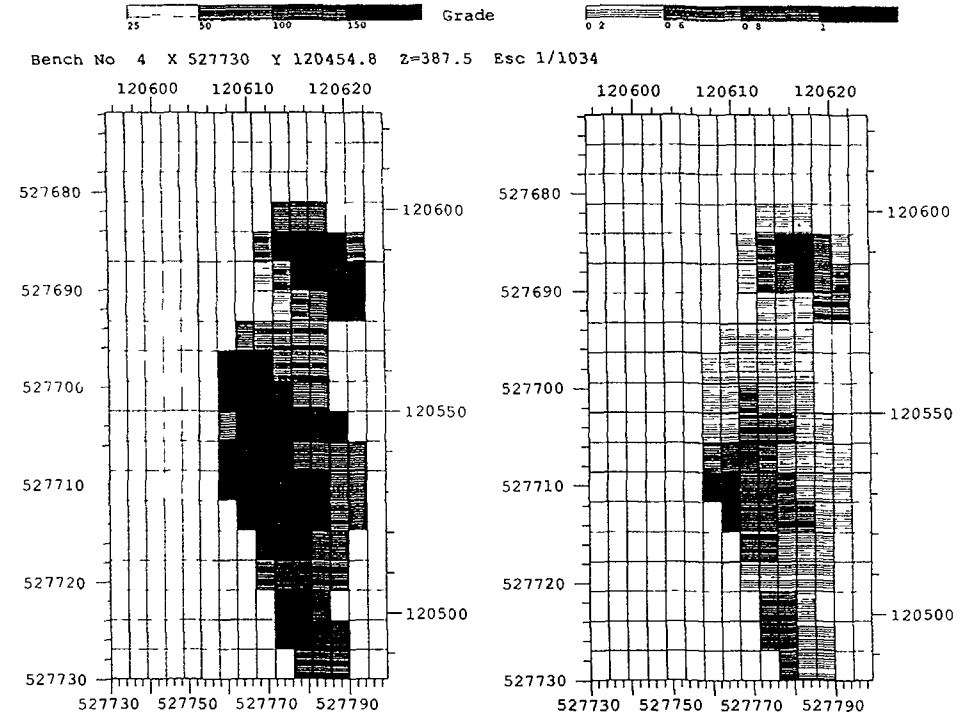


FIG. 9. Puy de l'Age — metal.

Ore grades tend to change rapidly over short distances, which makes the evaluation of such a mineralized area difficult. From 1984 to 1986, the deposit was defined by 28 diamond drill holes (1,686 m) and 314 percussion drill holes (18,958 m). The drilling grid was 10 x 10 m and locally 5 x 5 m on a cross oriented N10 and N90. Generally the holes were striking E-W and dipping east.

Stripping of the pit started in 1989 and the first production occurred in October of the same year. Mining is now almost completed.

Rapidly, slight discrepancies appeared between estimation and production data. This induced the company to undertake a more precise study using data from blastholes. The result of this study will be used to make more accurate estimations of other pits in the vicinity.

2.2 Presentation of the study

2.2.1 First estimation

It was completed in 1986 with the data from the development drill holes (Fig 11). A radioactivity- grade relationship was calculated. Structural analysis was made on a 10 x 10 m grid ; same profiles had a 5 m spacing.

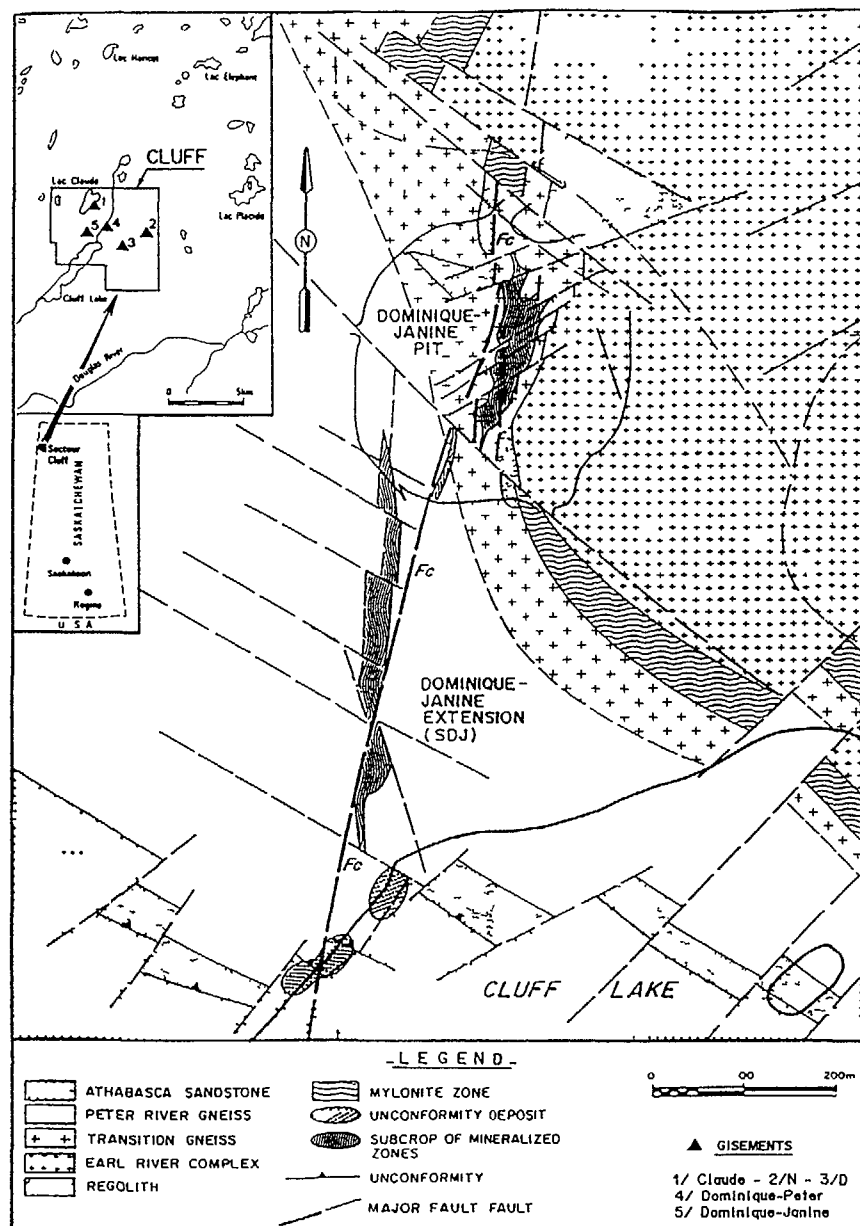


FIG 10 Dominique Janine and Dominique Janine extension (SDJ) deposits

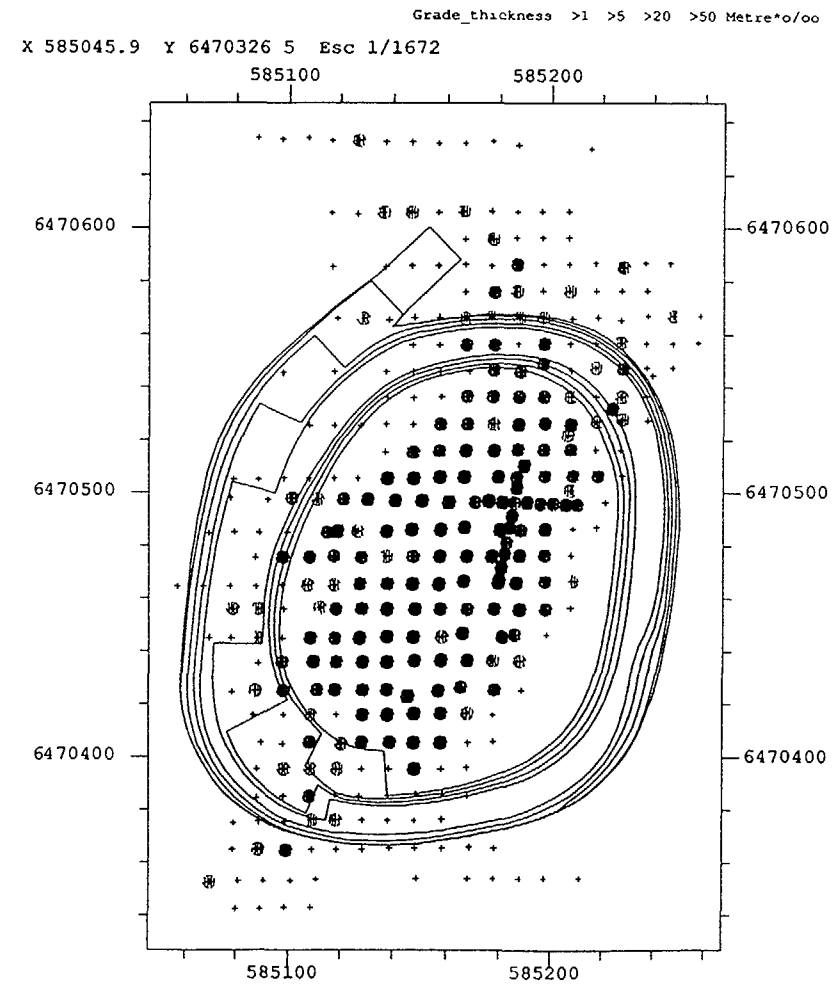


FIG 11 Dominique Janine — development drill holes

The main estimation parameters were

- compositing over 1 m
- kriging of panels 10 x 10 x 1 m (or 10 x 10 x 3 by grouping)
- global recoverable reserve on selection blocks of 3 x 3 x 1 (or 3 x 3 x 3 by grouping)
- local recoverable reserves, using Service Variables method, for different cut offs

Another estimation was then computed with compositing over 3 m and direct computing with panels 10 x 10 x 3 m and blocks 3 x 3 x 3 m. An open pit optimisation was also performed.

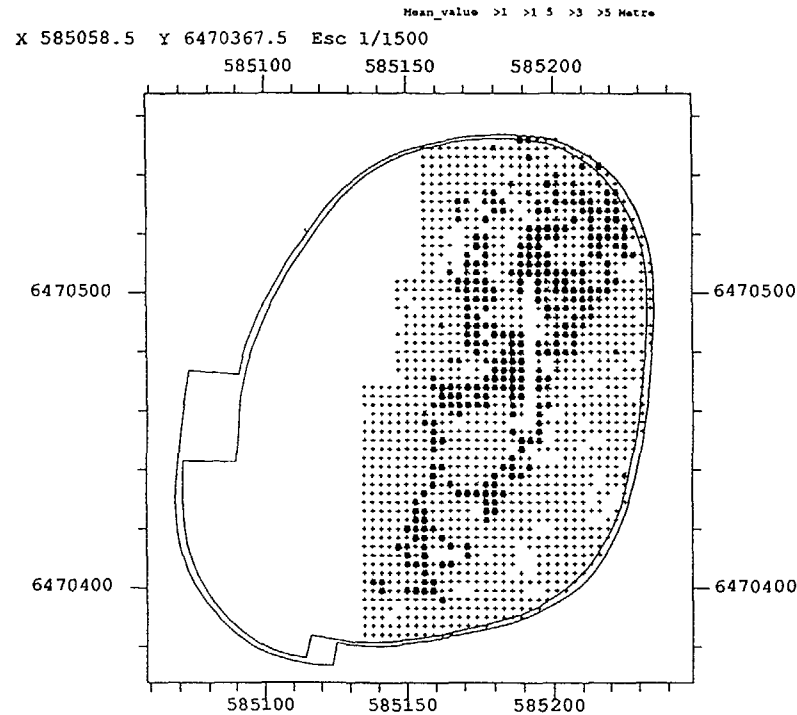


FIG. 12. Dominique Janine — blast holes location (bench 312).

Principal parameters of the estimation : mean 0,54 % ; variance 4,07 (‰)² ; variogram with 3 isotropic spherical structures :

- ranges 3,3/7,4/25,3 and sills 1,37/0,48/2,21
- variance of selection blocks : 2,87
- change of support coefficient : 0,925

2.2.2 Second estimation

A second estimation is performed at the beginning of 1991. At this time the radiometric data of some 12371 blastholes, probed at a 0.1 m step, are available. They represent 9 benches of 3 m height with a drill hole grid spacing of 3 x 3 m (Fig 12). A new radioactivity-grade relationship is computed. Regularization is made over 3 m. A mask is used, in such a way that the kriging does not spread to the waste areas on the margins of the deposit. Missing information between blasts, are reconstituted by kriging.

Some 4906 blocks (3 x 3 x 3 m) are informed with a mean grade of 1.59 ‰ and a variance of 12.15. A 1-3 x 3-1 kriging configuration is used.

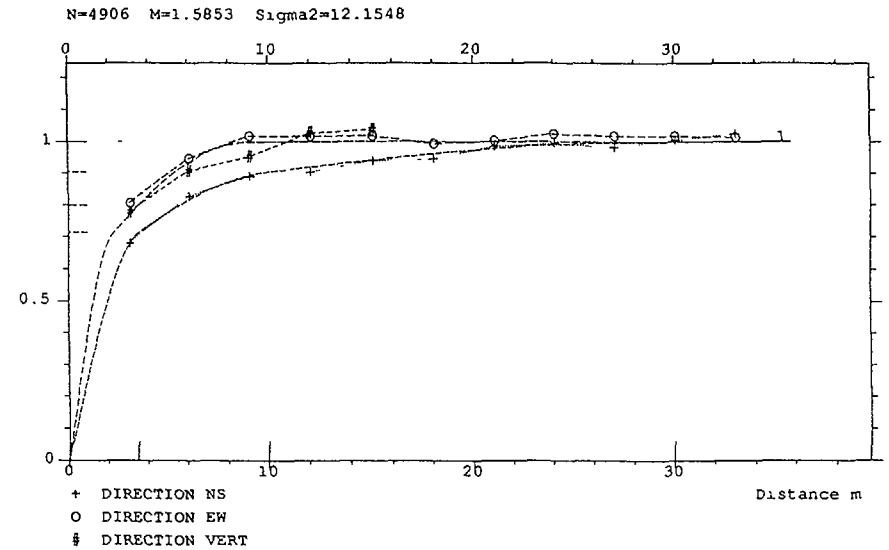


FIG. 13. Dominique Janine variogram.

The variogram is fitted with 3 spherical structures with anisotropy (Fig 13).

	1 structure	2 structure	3 structure
Sill	6,61	3,30	2,24
Range NS	3,5	10	30
EW	2,2	8,3	10
Vertical	3,5	10	30

The results of estimation by kriging are very close to those of first estimation.

2.2.3 Production data

Every ore truck is scanned outside the pit by an automatic scintillometric scanner which records the bench and blast numbers, and calculates the grade with a radioactivity-grade relationship.

The data from the scanner has been adjusted to balance with those of the mill.

2.2.4 Results

Fig 14 shows bench by bench comparisons for ore, metal and grade between production and first estimation. In order to account for the fact that the ramp is ignored by the first estimation. The actual open pit is digitized and only the panels inside this outline are computed (sometimes weighted by the percentage of volume in the contour).

The global result on these 9 first benches shows an overestimation of 18 % on ore, 23% on metal and 3 % on grade.

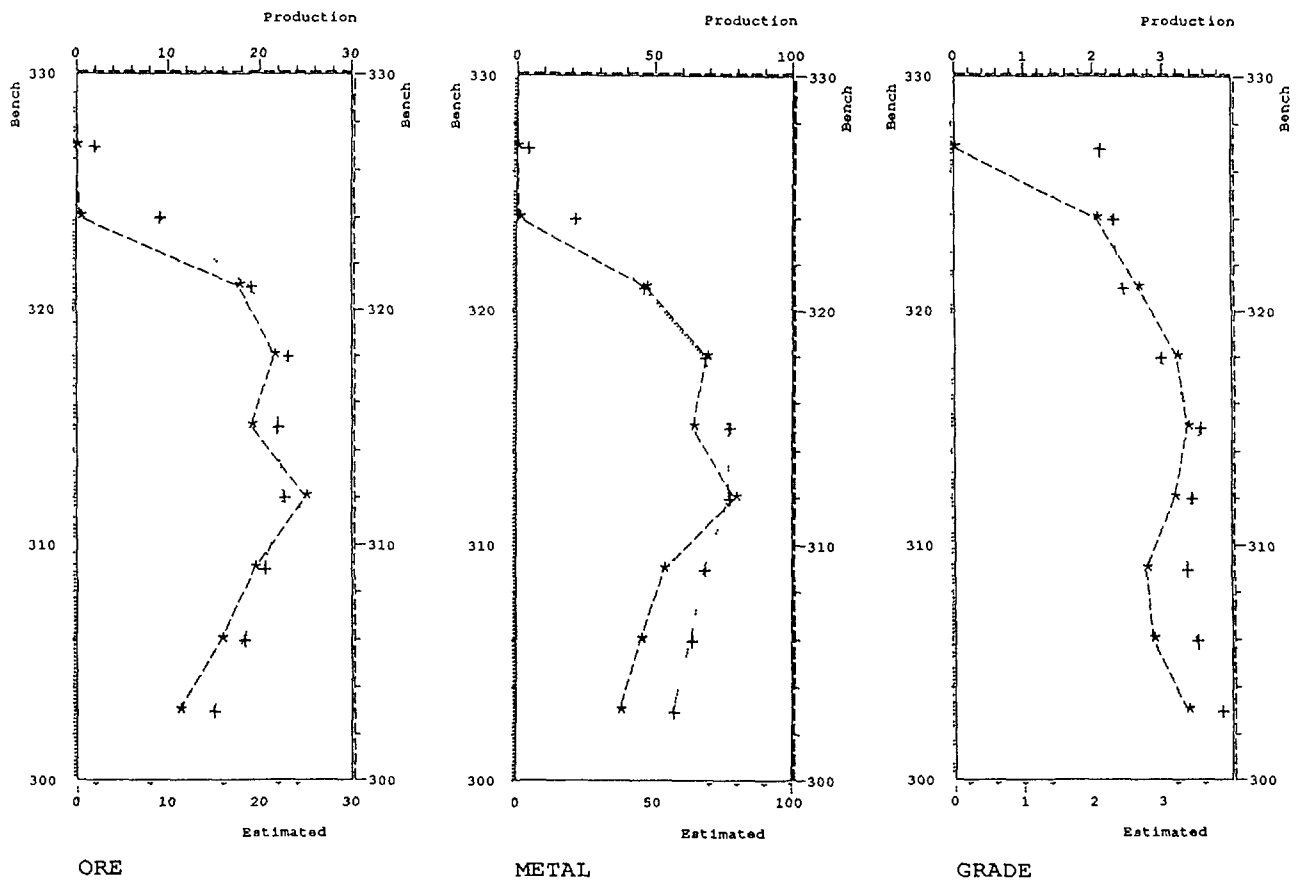


FIG 14 Dominique Janine — comparing estimated grade with production

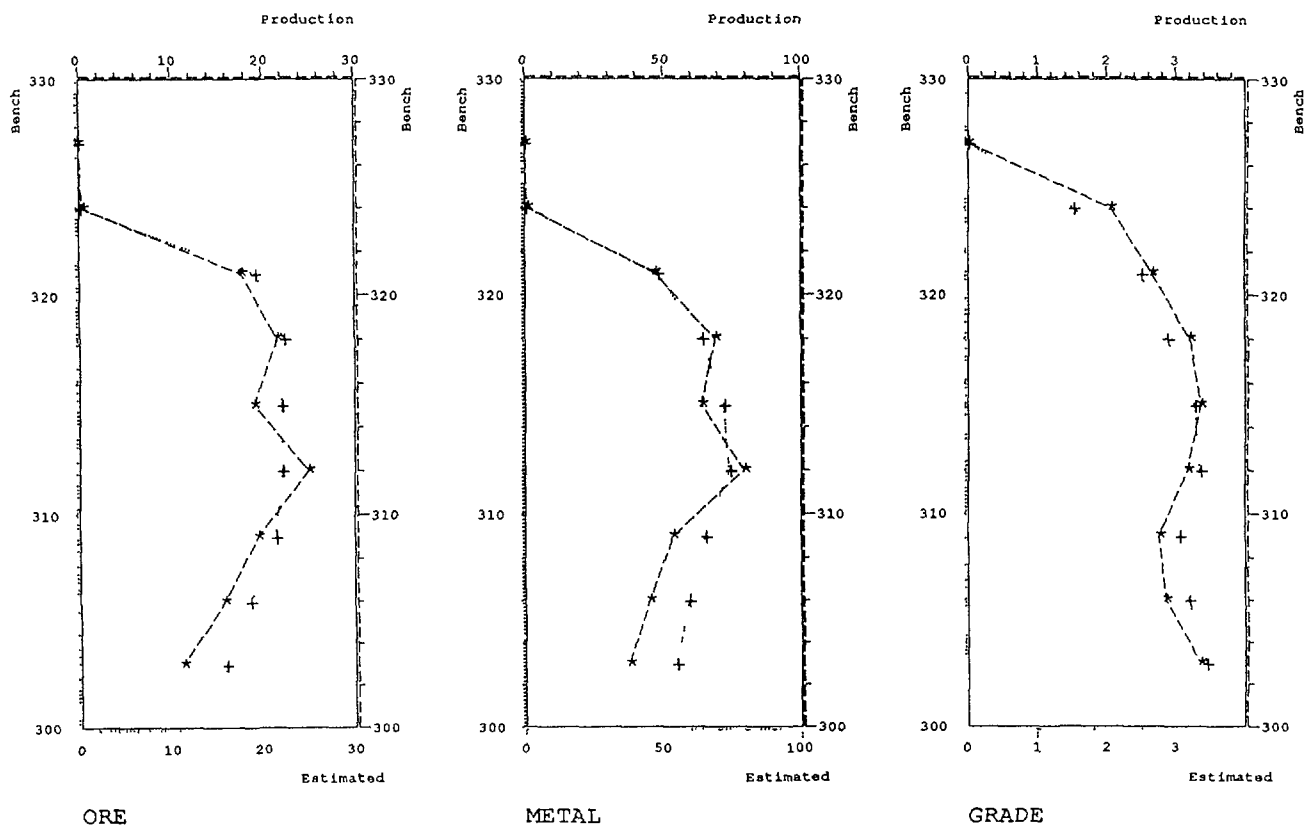


FIG 15 Dominique Janine north — comparing new estimation with production

Curves of estimated grade overprint those of production, with a classical smoothing peaks are overestimated and gulfs underestimated. But the ore and metal curves display a systematic gap that has to be explained. The grade of the first bench is poorly estimated because it corresponds to a small volume of ore.

The ore and metal in the second bench are overestimated. This is related to high values spread by kriging, and occurs especially on the borders of the mask. The phenomenon is attenuated when panels are masked after kriging rather than before, usually it disappears with a closer spacing in drilling.

In the last three benches the differences for metal are particularly important combining discrepancies on grade and on ore.

2.3 Discussion

2.3.1 Representativity of the 10 x 10 m grid

The estimation of the mean grade of the deposit (a priori unknown) is a major step in the evaluation. Uncertainties of the exact value of this parameter have two origins:

- the model of evaluation itself
- the representativity of the sampling by drill holes

This latter effect diminishes naturally with a closer spacing in drilling. It is therefore interesting to compare, the statistical parameters of the grids. For this purpose some blasts done in the waste were not taken into account. In the 10 x 10 estimation the bias resulting from an overinformation in richest areas (5 x 5 m² spacing) was avoided. The conformity of information in each panel was insured through weighting by the number of samples.

For similar geometries, the respective parameters are close to each other (identical mean). On a bench by bench comparison basis, the grade of regularized samples in drill holes does not differ significantly from the blast hole values taken as reference.

	N	mean	variance	/m
Blast holes	19511	0,44	3,68	4,36
Drill holes	1756	0,44	3,51	4,25

2.3.2 The variogram models and the selectivity

The computation of the local recoverable reserves, by the service variables method implies the choice of a size of selectivity blocks, which is realistic for mining (here 3 x 3 x 3 m³). It is therefore important to estimate the diminution of variance when the support changes from sample to blocks.

This estimation is calculated with a model of variogram, the fitting of which requires care, in particular near the origin (grid too large compared with first structure, not enough points for short distances, high variability of grade).

So far, at a given selection block size, significant variations in the estimation of variance diminution can be observed according to on the fitting.

At the time of the first estimation (grid 10 x 10 m), variograms used to be fitted on the gaussian anamorphosis value of grades. Such an adjustment tends to smooth the variograms, enhances the ranges but underestimates the diminution of variance for the first structure.

Since 1986, SERMINE has been able to infer the variogram adjustment on experimental values, from the model fitted on anamorphosed values. This correction implies a 48 % diminution of variance (instead of 29 %) previously which reduces from 3 to 0.7 % the difference between estimation and production grades, but the differences for ore and metal are now 25 % and 24 % respectively. SERMINE also allows now better regularisations, taking into account the length (and so the representativity) of the samples into the blocks. This improvement brings the difference on ore from 25 % down to 9 % (Fig 15).

However, it is clear that the remaining variation on metal results only from differences on ore.

2.4 Conclusion

The comparison of production and estimation histograms showing few discrepancies, it is in the global estimation of ore that the difference lies.

For the remaining difference, several hypothesis can be proposed:

- problem of density
- grade-radioactivity relationship: a higher slope would induce a smaller selective block size (with a 2 x 2 x 3 m support the difference on ore is reduced to 4 %)
- problems in computing production: underestimation of the truck weights, bad adjustment of the cut off
- maybe the limits of a gaussian model that should be replaced by others more adapted.

It is probable that the gamma model gives better results for highly disseminated grades. The indicator residual model is also probably more efficient to modeling the geometry of the deposit.

References

- 1 J.F. Bouchind'homme: estimation de l'uranium récupérable sur les gisements sédimentaires stratiformes exploitables à ciel ouvert. Thèse de docteur ingénieur, Institut National Polytechnique de Lorraine, 18 avril 1980.
- 2 Amok internal reports.

APPENDIX

SERVICE VARIABLES METHOD

In a Gaussian model, two service variables are defined :

- TMIN is the expectation of the proportion of small blocks higher than the cut-off grade, knowing the grade of the central section.
- TREC is the metal recoverable from these blocks.

1/ Determination of the cut-off Y_C

Knowing the raw cut-off Z_C to be applied to the block its Gaussian equivalent Y_C to be applied to the Gaussian histogram of the block v is computed, knowing that $Z_C = \Phi^{-1}(Y_C)$.

2/ Determination of $\text{cov}(Z_O, Z_V)$

Covariance between raw grade of the central sample and raw grade of its block v .

$$\text{cov}(Z_O, Z_V) = D^2(Z) - \bar{\gamma}(Z_O, Z_V)$$

$D^2(Z)$ = sill of the variogram = dispersion variance of samples in the ore body.

$\bar{\gamma}(Z_O, Z_V)$ = mean variogram between central sample and its block.

3/ Determination of $\text{cov}(Y_O, Y_V) = r_O$

It is assumed that the pair (Y_O, Y_V) is Gaussian.

r_O is the correlation coefficient between Gaussian grade of block V and Gaussian grade of central sample. r is the change of support coefficient.

r_O is computed by applying to the n coefficients of the anamorphosis, the coefficient r_O x r , which correctly reconstitutes $\text{cov}(Z_O, Z_V)$.

$$\text{cov}(Z_O, Z_V) = \sum_{i=1}^n \frac{\psi_i^2}{i!} (r \times r_O)^2$$

knowing r_O we can obtain conditional law of Y_V knowing Y_O .

4/ Computation of TMIN and TREC at sample level.

The following is considered :

- For the ore : the random variable $TM = 0$ if $Z_V < Z_C$
 $TM = 1$ if $Z_V > Z_C$
- For the metal : the random variable $TR = 0$ if $Z_V < Z_C$
 $TR = Z_V$ if $Z_V > Z_C$

Using a Gaussian distribution allows to obtain a conditional law of TM , knowing $Z_O = \Phi^{-1}(Y_O)$ grade of the central sample.

Consequently : $TMIN = E(TM/Z_O) = E(TM/Y_O)$
 and $TREC = E(TR/Z_O) = E(TR/Y_O)$

We compute for each sample value Y_O according to its development in terms of Hermite polynomials.

5/ Estimation of TMIN and TREC at panel level.

After modelling the TMIN and TREC variogram the mean values TMIN* and TREC* are estimated by kriging for each panel.

6/ Computation of the tonnages and grade of recoverable material.

For each panel :

- Recoverable ore = TMIN* x Volume of panel x density
- Recoverable metal = TREC* x Volume of panel x density
- Recoverable grade = $\frac{TREC^*}{TMIN^*}$

7/ Change of cut-off.

Back to 1/

THE CERRO SOLO PROJECT

P.R. NAVARRA

Comisión Nacional de Energía Atómica,
Godoy Cruz, Argentina

Abstract

The objective of the Cerro Solo Project, which was initiated in 1990, is to perform the assessment of the Co. Solo uranium ore deposit in a sequential manner. The deposit, which is located in the Patagonia Region, Argentine Republic, belongs to the sandstone type, and to the subtype coarse-grained, high energy stream. Uranium occurrences are peneconcordant, lenticular shaped and irregularly distributed in the fluvial sediments of the cretaceous Chubut Group.

Resources estimated using up to present data, in tonnes of uranium recoverable at costs up to \$80/kg U, are:

Reasonable Assured Resources (RAR):	300 tU
Estimated Additional Resources -	
- Category I (EAR-I):	900 tU
Estimated Additional Resources -	
- Category II (EAR-II):	2400 tU

Resources of this ore deposit are significant to the uranium requirements in Argentina.

INTRODUCTION

The main objectives of the Comisión Nacional de Energía Atómica (CNEA - National Atomic Energy Commission) in the field of uranium exploration are to increase knowledge concerning uranium favourability of regional geologic environments in order to estimate potential resources and select areas to explore, and to perform assessment programs on target sites that possibly could increase economic resources. At the present investment in assessing target sites must be justified by fulfilled conditions which will allow for the reduction of uranium concentrate production costs. The Cerro Solo ore deposit was selected by the CNEA because of their promising grade and amount of known and potential resources significant to Argentina's uranium requirements. The first chapter of the report deals with the framework in which this project is carried out, regarding national uranium requirements and resources in Argentina.

URANIUM REQUIREMENTS AND RESOURCES IN ARGENTINA

In the short term, uranium requirements will be directly related with the fuel needs of the nuclear power plants listed in the following table:

Plant	Capacity (MW)	Starting year	Status
Atucha I	340	-	Operating
Embalse	600	-	Operating
Atucha II	700	1995	Under construction
Central IV	700	2000	Proposed

A total installed capacity of 2340 MW is expected by the end of the decade.

The annual uranium requirements related with the plants previously mentioned, starting with the present 150 tU/year will grow to 240 tU/year before 1995 and to 340 tU/year by 1999. All of the plants will require approximately 10000 tU during their useful life.

Regarding the long term period, technical studies carried out by specific planning committees assigned by both, the Secretaría de Energía (Energy Office) and CNEA highlighted that an important growth of nuclear capacity is expected to meet the Argentina's energy requirements in the first two decades of the next century. Furthermore, they forecasted an additional nuclear capacity of 12000 MW up to the year 2020, to reach the foreseeable share of nuclear in the energy output that will be needed (1). As a conclusion, about 60000 tU would be required to fuel all the plants considered in that study during their useful life.

Concerning the possibilities of supporting the uranium needs in the short term using indigenous resources, the reserves of the Sierra Pintada ore deposit can cover most of the needs. Furthermore, contributions of other smaller deposits which are in different stages of exploration or development will be available to reinforce supplies, if they are needed. However, taking into account the present situation of the increasing gap between national and international uranium prices, attention must be focussed on the consideration of alternatives for the reduction of national prices.

In the long term, the level of known resources which could be categorized as competitive costs are far from covering the foreseeable requirements. To be in a position to confront the challenge presented by

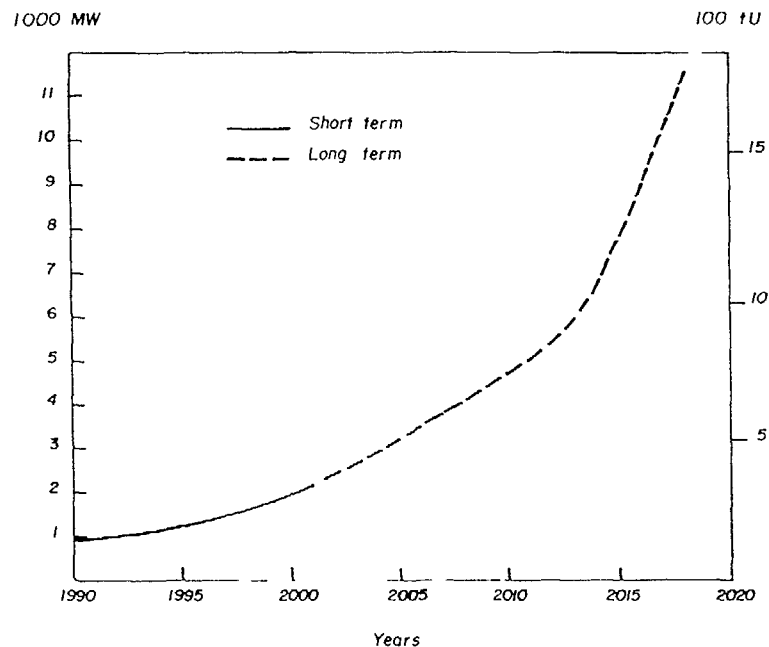


FIG 1 Projected installed nuclear capacity and estimated annual uranium requirements in Argentina

increasing future uranium output it is very important as a first step, to improve the geological and uranium favourability knowledge. In that way, the CNEA's Gerencia de Exploración (Exploration Branch) has completed the conceptual stage of a project which goal is to make a contribution to geological knowledge as a way to lead exploration investment, so as to produce new discoveries in known and unknown uranium districts.

Figure 1 shows the expected increase in nuclear installed capacity and related annual uranium requirements, in the periods up to the year 2000, and also between the years 2000 through 2020

GEOGRAPHIC LOCATION AND GEOLOGICAL SETTING OF THE ORE DEPOSIT

The Co. Solo ore deposit is located in the Patagonia Region, 1900 km from Buenos Aires to the SSW, and 430 km W from the CNEA's headquarters of

Trelew City, Chubut Province. Schematic geographic and geologic location maps are shown in figures 2 and 3.

The Chubut Group stratigraphic unit, which hosts the uranium mineralization, was deposited in the intracratonic San Jorge Gulf Basin, subsiding since the early Cretaceous. At this time an important volcanic activity took place a few hundred kilometers to the West, and served as a source for the deposition of the continental Chubut Group

The San Jorge Gulf Basin is classified higher level in the country regarding uranium favourability, as indicated by numerous anomalies and ore deposits known throughout a great number of places mainly in connection with the Chubut Group (2).

In the region of the middle Chubut River, including both slopes of the Pichiñán ridge (see Fig. 2), the Chubut Group is predominantly comprised of fluvial conglomerates and sandstones belonging to the Gorro Frigio Formation which fill, in places reaching 300 m in thickness, the depressed areas of the underlying paleozoic and mesozoic formations. Conformably, there are tuffs and tuffaceous mudstones varying between 10-50 m thick overlapping the fluvial member as shown in figure 4.

The Co. Solo ore deposit belongs to the sandstone type, and to the subtype coarse-grained, high-energy stream. Sedimentologic patterns fill into the fluvial braided stream characteristics and considering the Miall classification corresponds to the multichannel system, low sinuosity and significant slope, dominating traction transported sediments which make up the bed load (3).

Uranium occurrences are peneconcordant, lenticular shaped and distributed irregularly or erratically in some places. Frequently the host sediments are anomalous within most of the thickness and are of low grade within tens of meters. Intermediate to high grade layers are disseminated throughout the fluvial member, both in the fringes and in the thickest places. This shows that the mineralization process affected almost the entire fluvial member, reaching high grade in specific places. When conditions were adequate however has been preserved

Occurrences of the borders of the paleochannel are oxidized. Uranium minerals include: autunite, meta-autunite (Los Adobes ore deposit) and tyuyamunite, carnotite, uranophane (Co. Condor ore deposit) (3). The

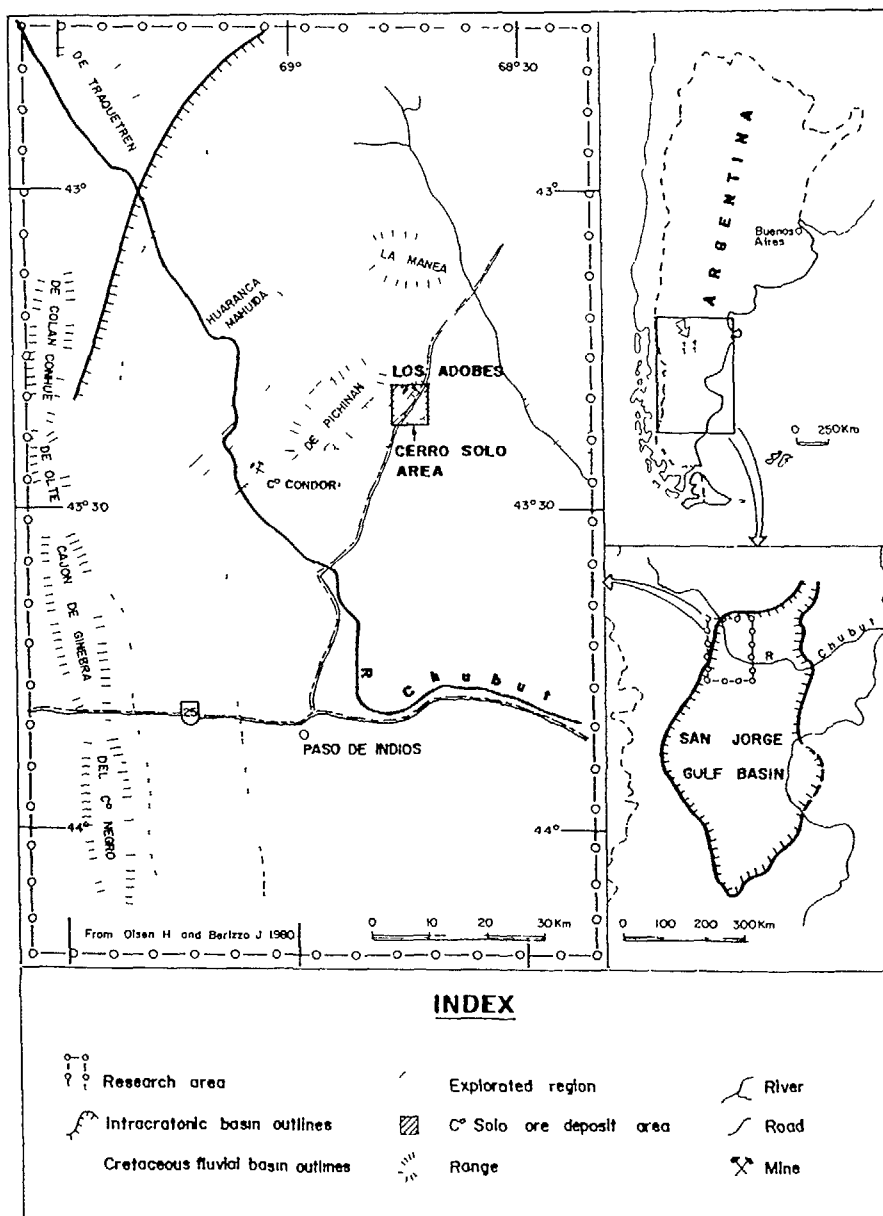


FIG 2 Schematic geographic and geological location maps

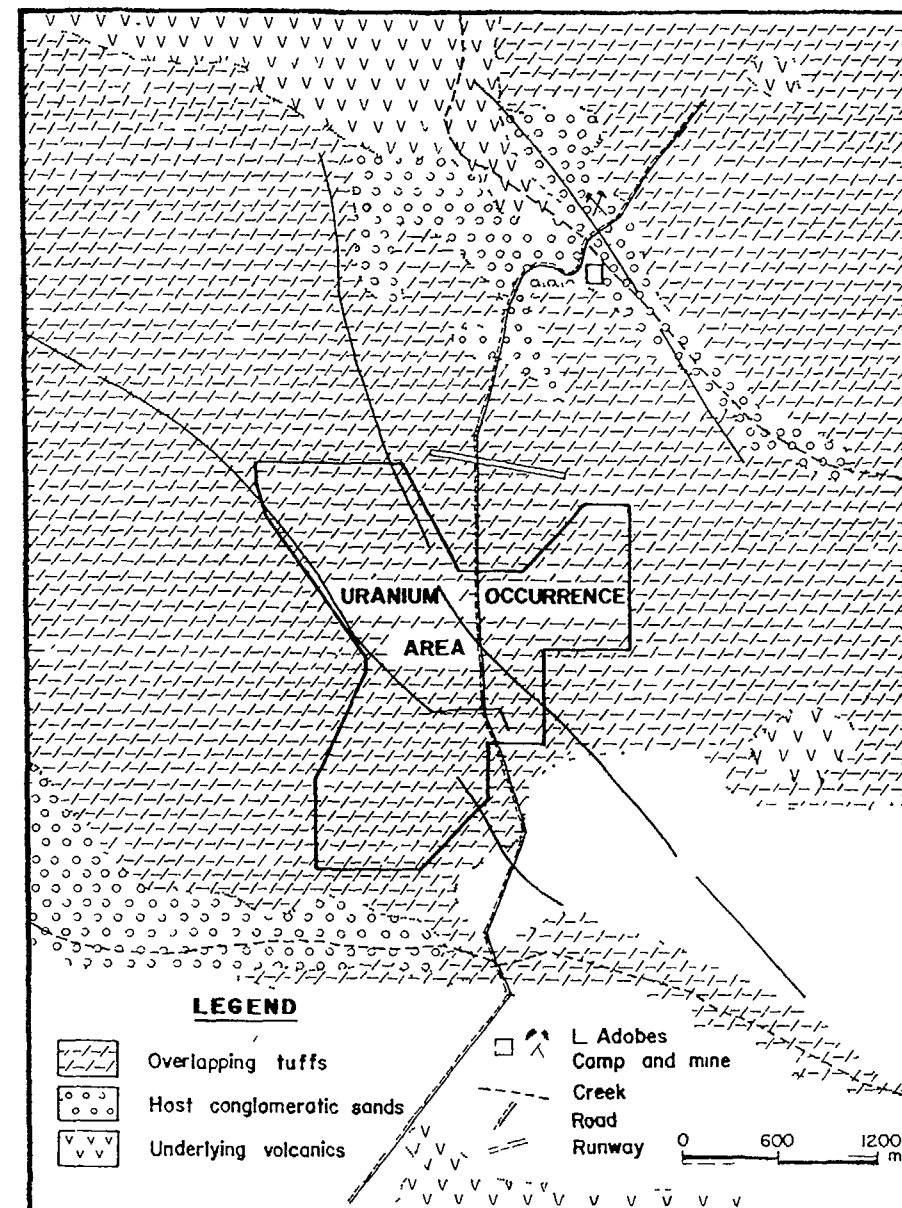


FIG 3 Location map of the Cerro Solo ore deposits

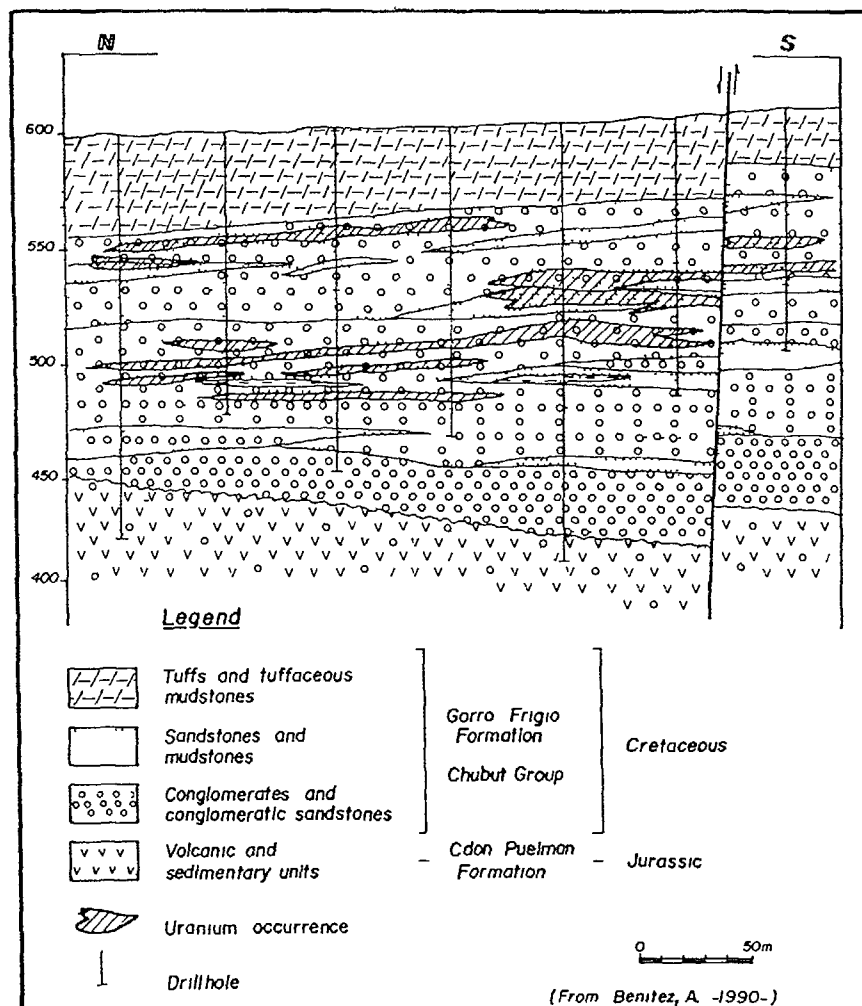


FIG 4 The Cerro Solo ore deposits schematic cross section showing the geological setting of the uranium occurrences

deposits of Los Adobes and Co. Condor, both outcropped, were mined in the 1970's. The blind deposits laying in the deep section of the Cretaceous paleochannel, remain dominantly reduced. Uranium minerals observed include uraninite, pitchblende and coffinite being associated organic material, abundant pyrite and occasionally calcite (3)

The rock hosting mineralization varies from coarse conglomerates to thin mudstone layers, but conglomeratic sandstones are clearly more abundant.

Two important faults, belonging to the main system affecting this region in the Tertiary could have played an important role in preserving mineralization.

The Pichiñan ridge uplift gently downdipped the beds to the present east side. After the uplift occurred the reoxidation of the sediments initiated.

The most probable source of uranium is the leaching of overlying tuffs. Uranium minerals were deposited undoubtedly during reduction processes, with the organic material, which occurs as disseminated plant debris or finely divided, playing a main role as shown by the close association with mineralization.

EXPLORATION HISTORY OF THE PICHINAN RIDGE REGION

In the Pichiñan ridge region the host fluvial sediments were explored in an area 35 km in length from East to West and 8 km wide from North to South. These explorations were conducted since the 1960's by various research programmes. The geology and topography of the entire surface of the Pichiñan ridge region was surveyed using scales varying from 1:20000 to 1:5000. Airborne and pedestrian radiometric and geophysics surveys as well as sedimentologic studies were performed. A number of different drilling exploration programmes were carried out covering selected sites over all the region (about 109000 m as a total)

Presently the eastern slope of the Pichiñan ridge which declivities to the creek named A° Perdido is considered a promising area for development. Drilling results have made it possible to eliminate those areas devoid of or containing only minor deposits, and to establish the location of some target sites. The Co. Solo area is considered the most significant of them.

DESCRIPTION OF THE CO. SOLO PROJECT OBJECTIVES AND UP TO PRESENT RESULTS

The conceptual stage of the project was formulated by analyzing data gathered on the site previous to 1987. The assessment process is divided into 3 separate operative stages. An evaluation step is used after

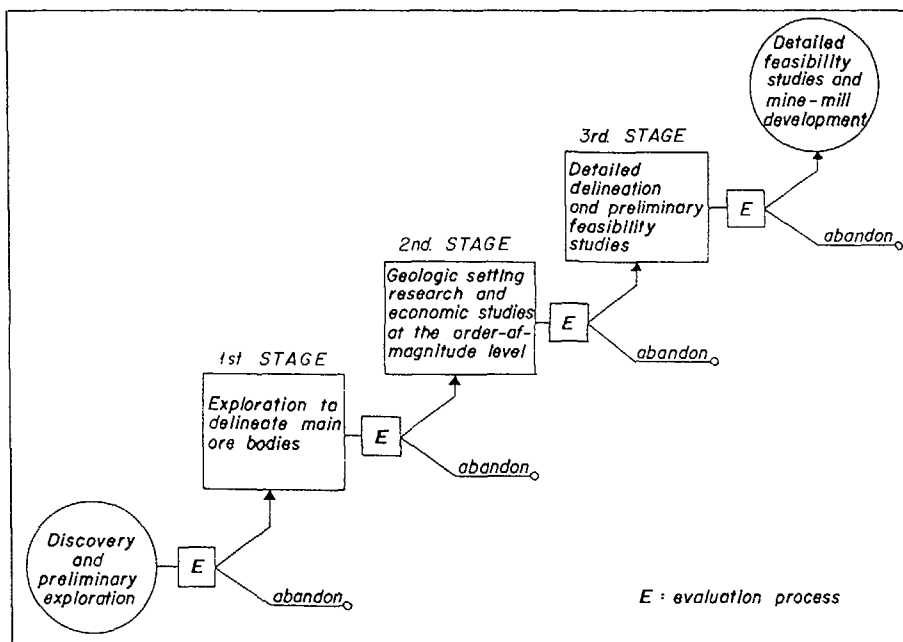


FIG. 5. Cerro Solo project: sequential diagram.

each stage has been completed to assess results and to consider whether to continue or abandon the project (see Figure 5).

The main goal of the first stage which was initiated in November 1990 and completed in June 1991, was to determine the approximate boundaries of the main ore bodies. The appropriate drill holes grid was determined by considering the characteristics and distribution of mineralized lenses on two restricted areas of the deposit explored using grids of 50x50 and 25x25 m wide. Thus, 9500 m of drillholes distributed in a grid of 100x100 m were utilized in the first stage, in order to obtain cutting samples, gross gamma and electric logging. The results were succesful both in determining the location of main ore bodies and in outlining most of the ore deposit areas, thus confirming the model of the ore lenses distribution previously elaborated. The first stage also was completed within budgeted time and costs limits.

1) RESOURCES

REASONABLE ASSURED RESOURCES R A R		ESTIMATED ADDITIONAL RESOURCES CATEGORY I - E A R - I	
Recoverable Up to \$ 80/Kg U	at costs Between \$ 80-130/Kg U	Recoverable Up to \$ 80/Kg U	at costs Between \$ 80-130/Kg U
300	75	900	160

2) ADDITIONAL RESOURCES

ESTIMATED ADDITIONAL RESOURCES - CATEGORY II E A R - II	
Recoverable Up to \$ 80/Kg U	at costs Between \$ 80-130/Kg U
2400	400

FIG. 6. The Cerro solo ore deposits: uranium resources (t U as recoverable quantities).

According to present knowledge it is possible to define areas in which resources can be estimated, taking into account the frequency, distribution, and the characteristics of mineralization. Within these areas the individual lenses size varies widely but more commonly between 10 to 50 m in length, and lie irregularly distributed at a depth of some 70 to 125 m (98 m average). Assuming 1 m minimum, they average in thickness 1.9 m, with a grade of 2.4 %U.

Resources were estimated using standard, conventional methods. The results were classified into the IAEA system and are shown in Figure 6. As the amount in each of the categories indicates the RAR and EAR-I categories are subordinated to the EAR-II category, because of the relationship between the sample distribution and the ore lense irregular distribution. An important goal of the second stage is delineating the ore deposit using a

closer drillhole grid, to improve the reliability of the estimation. About 18000 m of drillholes will be bored.

Additional goals of this stage are to improve knowledge regarding detailed geological characteristics of the ore deposit in connection with more reliable resource estimates, and to conduct an economic study at the order-of-magnitude level.

The third stage will include about 20000 m of drillholes and pilot mining and milling studies. The objectives of stage 3 are detailed delineation of the significant individual lenses, and mining-milling-economic studies at the preliminary feasibility level.

A multidisciplinary effort which began in the first stage includes studies on: geological setting of mineralization, resource estimates, costs of mining and milling methods and economic analysis. In each successive stage this studies are planned with increasing detail in order to attain higher reliability of results.

The sequential method has proved to be very effective at reducing the risk of investment, and at the same time improving efficiency in operational aspects and optimizing results.

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IMPROVEMENTS TO THE QUALITY OF THE ESTIMATES OF US URANIUM RESERVES

Z.D. NIKODEM

Nuclear and Alternate Fuels Division,
Energy Information Administration,
United States Department of Energy,
Washington, D.C.,
United States of America

Abstract

Extensive work has been done in the United States on the estimation of uranium reserves. The government's role in uranium raw materials shifted from support of military programs to assessing the supply available for commercial power generation. A comprehensive system evolved in which government staff estimated reserves for each property over a range of cost levels using standardized estimation methodology and criteria. The program was assigned to the Energy Information Administration (EIA) of the Department of Energy in 1983 which has the responsibility for reporting on energy resources. As uranium supply had increased and demand had decreased, there was less concern about the adequacy of resources. In this situation, and with reduced staffing levels, the EIA adopted a two part interim approach to preparing reserve estimates. One used questionnaires to obtain uranium company estimates of their economic and subeconomic reserves, with company determined economic criteria. A second approach modified the earlier detailed government property estimates to account for production. The EIA developed a new system with the assistance of consultants and the uranium industry. The goal of the new system is to produce one set of estimates at various cost categories for each property based on a rigorous adherence to EIA criteria. Initial information is gathered from the industry through a revised annual questionnaire. Company estimates that conform to EIA standards are incorporated into the EIA reserve data base. Additional information is gathered for those estimates requiring clarification, primarily through detailed technical conferences with company staff. The EIA has the capability to prepare independent reserve estimates from basic drill hole data when required. Uranium reserves estimated for 1990 by the EIA include the initial results from the new methodology. The cooperation and support of the uranium industry have been excellent. Detailed evaluation of properties is continuing. Further work is being directed toward improving estimation techniques and analyzing production levels obtainable from reserve levels at various cost categories.

Introduction

The United States has a long history of estimating uranium reserves. This information was of fundamental value in the planning of procurement programs in the early days of defense nuclear programs. Subsequently it was vital for the development of civilian nuclear power programs. Reserve data are basic for understanding both the near and long term outlook for uranium supply and the potential economics of that supply. Such information can lead to development of adequate supplies and to employment of appropriate energy generation technology and production facilities. While there is much less concern currently about the adequacy of uranium supply, fundamental questions remain about the magnitude of uranium resources, and their costs of production and availability. Monitoring and study of uranium reserves and resources must continue if future shortages are to be avoided and sound planning by both the producers and the users is to be assured. To meet the changing needs of the users of reserve data, the Energy Information Administration (EIA) has developed a modified uranium reserve evaluation program.

History

Initial work on estimation of uranium reserves in the United States was done in the 1940's as part of the Manhattan Project, the wartime effort to develop nuclear weapons. This activity largely concerned the uranium vanadium deposits of the Colorado Plateau which contained most of the known deposits of uranium in the US at that time.

As the work of the Atomic Energy Commission (AEC), which succeeded the Manhattan Project, proceeded there were increasing needs to understand the extent and nature of uranium ore reserves. The demand for uranium was far in excess of known resources. Systematic reserve evaluations provided a gauge to assess the success of exploration and a basis for planning uranium procurement.

In 1952, a small ore reserves group was established at the Grand Junction (Colorado) Office. This group undertook the process of evaluating the reserves of all known deposits. The group established procedures and criteria for consistent evaluation of the resources based on generally accepted engineering methods. At that time reserves were only a few thousand tons of uranium in many small deposits. Exploration activity was almost entirely done by the AEC. Drilling data and information gathered during the examination of privately owned deposits were the basis for the reserve estimates.

The AEC provided incentives for uranium exploration and production through guaranteed markets and prices, and through financial and technical assistance. These incentives, coupled with the discovery of larger deposits in other geologic environments, soon attracted many mining companies and individuals to the uranium business. As activity expanded the problem became one of estimating reserves from company developed basic data. The AEC was the sole buyer of uranium. A requirement to provide ore deposit basic data to the AEC was included in the procurement contracts. A close working relationship on monitoring and evaluation of reserves developed between the AEC and the industry. This relationship was maintained over the years as the industry expanded and new companies entered the business.

Procedures for estimating reserves evolved that recognized the nature of uranium deposits and the geologic controls, the technology for mining and processing, and the costs of production. Reserves were estimated in various categories of reliability and at various costs of production. The categories of Indicated Reserves (which included Measured Reserves) and Inferred were used over the early years of the activity. The basis for economic evaluation initially was the AEC price schedule for uranium ores. Later reserves were estimated at \$8.00 per pound of U₃O₈ and then at a range of cost levels. In the 1970's estimates were made at \$10, \$15, \$30 and \$50 per pound of U₃O₈ (\$27, \$40, \$80 and \$130 per kg U). These cost categories were modified as prices changed and to reflect inflation.

Techniques were developed that allowed processing of the increasing amounts of data being generated and employed a variety of estimation and evaluation methods. Computers, which were of the main frame type, and statistical techniques, were increasingly used. AEC engineers and geologists developed the software needed, as there were no commercial programs available.

Some idea of the growth of the size of the effort involved can be gained by reviewing a few statistics. See Table 1

Table 1. Drilling, Reserves and Production, 1950 and 1978

	1950	1978
Meters Drilled	238,000	14.6 Million
Reserves, Tonnes U	2,300	530,000
Number of Reserve Properties	less than 100	1,500
Mine Shipments Tonnes U	600	15,500

In 1950, about 238,000 meters of surface drilling for uranium exploration and development were completed. Reserves were about 2,300 tonnes of uranium contained in less than 100 properties. Mine shipments of ore contained about 600 tonnes of uranium. In 1978, during the height of uranium activity, some 14.6 million meters in 104,400 holes were drilled. Most of these data were collected by the AEC and converted to digital form and processed in the reserve program. Reserves producible at the \$80 per kg U cost level had increased to 530,000 tonnes uranium. These reserves were in 1,500 properties in 15 different states. Some 15,500 tonnes of uranium in ore were shipped from 391 different sources.

The emphasis of the earlier programs was placed on the estimation of ore reserves. As it became apparent that projected needs were much greater than available reserves, there was an increasing interest in understanding the possible extent of resources beyond those meeting the restrictive criteria of reserves. Consequently the resource program developed into a fully integrated evaluation of uranium resources of all categories of reliability and economics. The concern was about the potential supply and economics of uranium for the long term, some 30 or more years ahead and producible at costs well above prevailing prices. This information was needed to support decision making relative to deployment of the light water reactors and for programs to develop improved reactor types such as the breeder reactor.

A skilled work force in reserve and resource appraisal was developed in the AEC and extensive files of data on all US uranium deposits was assembled. Efforts were expanded to study the nature and extent of uranium resources world wide. This work included cooperative efforts with the Nuclear Energy Agency and the International Atomic Energy Agency.

Estimation Procedures to 1983

A consistent approach to national uranium ore reserve estimation was employed by the AEC and its successor agencies, the Energy Research and Development Administration (ERDA) and the Department of Energy (DOE) through 1982. In this procedure government staff engineers and geologists, prepared reserve estimates for each deposit using basic sample and cost data from the mining companies. Data were gathered in field offices established at different locations around the country. During the 1960's, the field office staff made the initial reserve estimates using procedures and criteria set out in an ore reserves manual. Reserve estimates were reviewed in Grand Junction

and, upon acceptance, included in the national reserve data base. In addition to the study of resources, evaluations and projections of the production capability that the resources could support, and as planned by the industry, were also made. With the increasing reliance on computers in the late 1960's, deposit evaluation became more centered in the Grand Junction office.

A key aspect of the program was the close contact with the mining and exploration companies, through the field offices and through frequent visits and conferences with the Grand Junction staff. The data, estimation criteria, procedures, and results were reviewed annually in meetings with the mining companies' key staff. This assured the completeness and accuracy of the results. Annual presentations of the findings of the various studies, were provided to the public through annual conferences at Grand Junction and by the issuance of a variety of reports.

In the early 1980's, there was a diminishing concern about adequacy of uranium supply. Growth in nuclear energy and uranium demand and future projections of need were greatly reduced. Uranium supplies seemed adequate for an extended period. The U.S. uranium industry was undergoing a severe retrenchment. Exploration and development had decreased sharply, from the 104 400 holes and 14.6 million meters drilled in 1978 to only 9,970 holes and 1.6 million meters in 1982 (Table 2).

Table 2. Exploration and Development Drilling, 1978 and 1982

	1978	1982
Number of Holes Drilled	104 400	9,970
Meters Drilled	14 630 000	1,646 000

In addition to the changing uranium outlook, there were pressures to reduce government staffing levels. As a result, a major change in the uranium program occurred in 1983. The uranium resource program was transferred from the Nuclear Energy program of DOE to the Energy Information Administration (EIA). This component of the DOE has the basic assignment of gathering and reporting national energy related data including coal, and oil and gas. With the transfer of the activity, reserve program staff levels were greatly reduced and the activity moved from Grand Junction, Colorado to Washington D.C. The extensive files and data base on the uranium industry were also moved to Washington.

EIA Estimation Procedures 1984 to 1989

The procedure adopted by the EIA for reserve estimation was considered to be an interim approach pending development of a permanent system. The modified approach was also influenced by the limited staff available and a changing view of information needs. It was judged that less detail on reserves was needed, and more information on other topics that were indicative of the viability of the domestic uranium industry was required. The new approach had two components. The first component was to employ questionnaires to gather industry estimates of

their economic and subeconomic reserves. The second was to revise the previous DOE property by property estimates at various cost categories, primarily to account for production.

A questionnaire was sent annually that requested company estimates of their reserves. The companies were to use their own criteria as to what was economic or subeconomic, considering their sales contracts and their assessment of the uranium market. In addition, data on criteria and costs used in estimating reserves, and a variety of exploration, property, production and sales data were also requested. The reserve estimates provided by the mining companies were totaled and reported by the EIA as national economic and subeconomic reserves.

For the second component of this procedure, the EIA continued to report reserves in various cost categories derived from the property by property reserve estimates previously made in Grand Junction. These revised estimates were made primarily by subtracting production from the estimates and by proportionately reducing reserves at other cost levels. No modifications were made to reflect changes in mining methods or additional exploration results, either from surface drilling or mine development. As exploration, production and mine development were at low levels such adjustments provided reasonable estimates of reserves. As time progressed, however, the cumulative effects of such procedures became less reliable.

Problems in Methodology

Questions arose concerning the reliability of reserves reported by the companies. The economic criteria used by the companies varied widely, as the price levels received in existing contracts ranged from less than \$20 per kg U to well over \$100 per kg U. In addition there were differing interpretations of the prices that might be attainable in the market in the near and longer term future. In addition, there are many variations in estimation methods and criteria and in company policy, concerning reserve estimations. Aggregate estimates of the reserves reported by companies provided little insight into the availability of uranium at various cost or price levels or to the reliability of the estimates. During this period, the EIA staff had limited direct contact with the industry and little opportunity to exchange views on reserve data. It became clear that the estimates were becoming of questionable reliability and that the estimates were not being used by the industry as they had prior to 1984.

The revisions to deposit estimates by subtraction of production from a previous reserve estimate could only be expected to be reliable for a few years for active deposits. Modifications to the estimates to reflect new knowledge on the character and ore distribution in the deposit must be made from time to time to reflect the current situation. Mining invariably will show that an ore deposit is different from the interpretation at the time of the previous estimate. Some ore will be disproved, other ore may be added. The mining experience may show differing costs and recoverability. Low prices may cause changes in mining methods from that previously practiced or planned. Inflation would change costs. Mining of low cost resources during periods of low price might lead to loss of parts of the deposit or make remaining reserves available only at much higher costs. Surface or underground exploration and development drilling may add considerably to partially delineated deposits. Technological advances, as in the case of in-situ leaching, can change costs and the approach to production for some types of deposits. The accumulation of these factors tended to make the adjusted reserve estimates less certain and of less use in reserve and supply analysis.

In view of these problems there was increasing dissatisfaction with the reserve estimation procedures. A new approach was necessary.

A New Approach

During 1987 and 1988, the EIA reviewed the problem with assistance of consultants and the industry. This review led to the development of a new approach to estimation of national uranium reserves. The basic new strategy seeks to develop a more consistent and reliable appraisal of US uranium reserves. This is done by maximizing the use of company information and employing techniques that can lead to a variety of reserve estimates with a minimum of industry and EIA staff effort. Basic to the program is a closer working relationship with the industry. This leads to a better understanding of the procedures of reserve estimation followed by the uranium companies. In addition, the goals and criteria of the EIA for uranium reserve estimation are more fully defined, with the objective of getting industry support to reach the goals.

Emphasis in the new approach is on the major uranium properties, and on properties with changes in mining methods. The value of this approach can be seen in Figure 1. This shows the distribution of US \$80 per kg uranium reserves for the 100 largest reserve properties. The 100 largest deposits contained 98 percent of the reserves. The 30 largest properties contained 76 percent of the reserves. By concentrating work on these properties the overall reserve picture can be more quickly reassessed. Similarly, concentrating on the 50 largest reserve controlling companies would cover about 86 percent of US reserves.

The basic components of the new program can be summarized as follows,

- 1 Use of a revised annual questionnaire for gathering company reserve and collateral data. A simplified questionnaire focuses on the fundamental information desired. EIA reserve needs and estimation criteria are

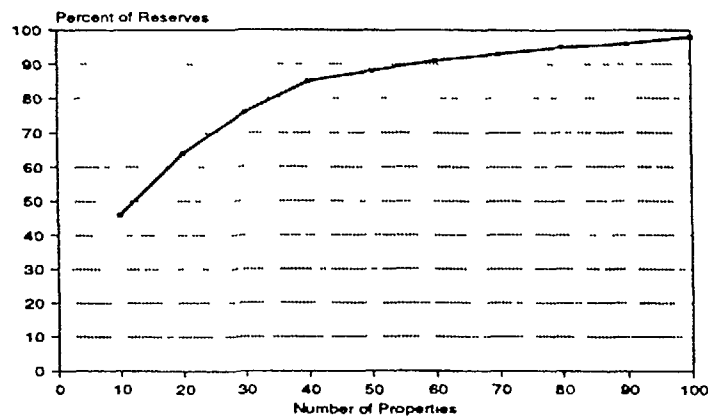


Figure 1. Reserve Distribution Properties (\$80/KgU)

clearly set out. Reserve data are requested for the EIA cost categories, if available, or for the cost levels adopted by the company. Flexible company responses are encouraged, in recognition of the varying problems and approaches in the industry.

- 2 Analysis of the information provided in the questionnaires and of the companies' procedures and criteria for reserve estimation through on site technical reviews with company staff.
- 3 Where company estimates are found to meet EIA criteria, acceptance of the company estimates, and inclusion into the EIA national reserve data base.
- 4 Where company criteria do not conform to EIA criteria and where possible, modification of company estimates to meet the EIA criteria.
- 5 Where adjustment of company reserve estimates is not possible to conform to EIA criteria and needs development of independent EIA estimates of deposit reserves using basic data provided by the companies.
- 6 Development of improved basic information needed for reserve estimation, such as cost data and improvement in reserve estimation techniques.
- 7 Compilation of the various accepted estimates into a national reserve appraisal and reporting of the results annually.

The interrelationships of the major program components are shown in Figure 2, from questionnaire to compilation of national reserve estimates. Each major component of the program is discussed in more detail below.

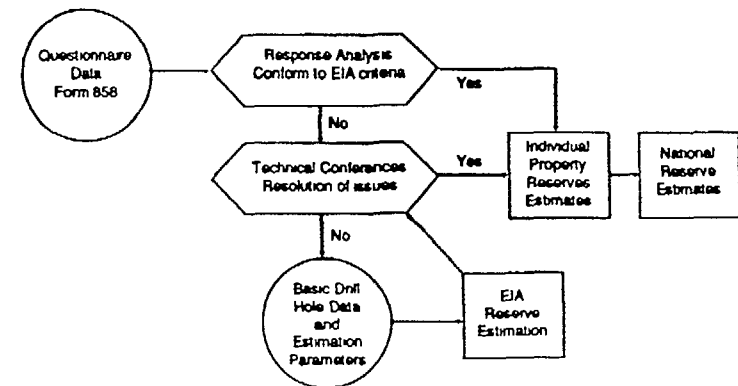


Figure 2. Major Program Component Relationships

Questionnaire

The primary approach of the EIA in gathering energy information is by annual questionnaires sent to the industry. In the uranium area, the questionnaire is Form EIA-858, "Uranium Industry Annual Survey." The current version of the form dealing with uranium reserves has been considerably shortened and simplified. The revision should

particularly ease reporting for non-conventional production, such as in-situ leach and by product operations. The goal was to focus on the essential information needed under the new strategy, to ease the burden on respondents and to allow more flexibility in the information provided. This will lead to more thorough and useful information from the companies and provide the basis for a closer working relationship with the industry. The final questionnaire reflects industry comments and suggestions from review and discussion of draft versions.

As in previous practice all information is closely held within the EIA and treated as "Company Confidential." The importance of confidentiality is well understood and special care is taken to assure that the data is protected.

Form 858 contains a general instruction section and two main parts seeking data. Schedule A covers uranium raw materials activities and Schedule B covers uranium marketing activities such as sales, imports, prices, inventories, and supply commitments.

The improved survey gives the respondent the option to use a microcomputer version of the form. This should ease filling out the form for the companies and in using the data by the EIA. The forms provided to the companies are preprinted with previous data for each property under control of the company. This allows the respondent to mark in corrections, thereby reducing the need to fill in repetitive information. Data can be transmitted to EIA by paper copies transmitted by mail, through facsimile machines or by computer diskette.

Schedule A is subdivided into four parts. Part I covers exploration and development activity, Part II covers reserves and mine production, Part III covers milling and processing and Part IV covers employment. There are 17 sections or "items" of information requested in Schedule A. There is a section that provides an opportunity for comments on any aspect of the information presented in Section A. A glossary provides the respondents with the EIA definitions of key terms.

Schedule A, Part II, (Items 7 through 12 of the questionnaire) is the portion of interest in this paper. This section has been extensively rearranged and simplified as part of the new EIA approach to reserve estimation. Data is collected for each property on the topics listed in Table 3. The series of questions requests company estimates for individual properties and information on the criteria and methods used in making the estimates, as follows.

Item 7, Property Identification and Ownership, requests the names and location of the property and the current ownership and control. Status requests the current stage of development and activity at the property, that is, in exploration, development or production stages, and the types of reserve and feasibility studies completed.

Item 8, Reserves, requests company reserve estimates for the property, reported separately for open pit, underground, in situ leach, and other types of operations. Reserves are requested by forward cost categories of \$15, \$30, \$50 and \$100 per pound of U_3O_8 (\$40, \$80, \$130 and \$260 per kg U) to the extent available, or for cost categories used by the company.

Item 9 requests the operating costs per ton of ore used in the reserve estimates presented. Cost per pound is obtained for in-situ leach operations. Cost elements include mining, haulage, royalty, milling, and indirect operating costs.

Table 3 Sections of Revised Form 858, Schedule A, Part II; Reserves and Mine Production

Item 7	Property Information Identification Ownership Control Status of Development
Item 8	Property Reserve Estimates
Item 9	Operating Costs
Item 10	Capital Costs
Item 11	Reserve Estimation Parameters
Item 12	Mine Production

Item 10 requests capital costs for mine development, and mill and plant construction.

Item 11 requests information on the parameters used in the reserve estimation, such as cutoff grades and thickness, tonnage factors, area of influence and recovery factors. Descriptive data, such as number of holes involved, average grade, thickness, and depth, are also obtained.

Item 12 requests data on mine production from the property for the year in ore and in contained uranium. Data on vanadium production are also requested for those properties with vanadium values.

The initial mailing of the new version of the form was made in December 1990 seeking data on 1990 activities and year end reserve and property status. Schedule A of form 858 was mailed to 104 companies. Responses have been received and reviewed.

Technical Review

The industry responses to Form 858 are reviewed in detail to decide if estimates provided can be accepted without modification or what additional steps should be taken. A fully completed form may provide an adequate basis for such a determination. The review is supported by other information available to the EIA on the deposit in question, and knowledge of the company reserve estimation practice. Sometimes, it is necessary to obtain additional details by telephone or by written request. For the early years of the program, particularly for major uranium companies and deposits, an in-depth technical review with company staff will be needed. The incorporation into the EIA data base of some of the property reserve estimates provided by the companies will thus be deferred until technical meetings have been held.

The technical reviews with the companies seek to establish a thorough understanding of the company practice and procedures in reserve estimation. The meetings will also establish a better understanding of the company policies and problems, and very importantly, build a better working relationship with the company. The goal is to enlist the company's assistance in developing sound national uranium reserve estimates.

The mutual meetings seek to involve high level company officials to explain the EIA program and goals and to assure corporate support. Subsequent meetings are with technical staff involved in reserve estimation. Such meetings are usually at active field sites and may include mine and plant visits.

Proposed agendas for the meeting are provided to the company in advance to assure understanding of the scope of the meeting. This also assures that the proper staff are present and that they are prepared for the meeting. Typical meeting agendas cover the topics shown in Table 4.

Table 4 Company Technical Conference Agenda

EIA Role in National Energy Resource Information
EIA Uranium Program Plans and Strategy
Company Reserve Estimation Procedures, Methodology and Criteria
Company Reserve Estimates by Property
In Depth Review of Selected Properties
Company Capital and Operating Costs
Way for EIA to Handle Company Reserve Estimates
Company Production and Exploration Activities and Future Plans
Future Actions by EIA and the Company

A more detailed check list has been developed for EIA staff on each topic to assure that all pertinent matters are covered in the meeting. The meetings are informal and encourage company comments and suggestions.

During 1990 meetings were held with 12 key companies involving over 100 properties. The properties were estimated to contain about 60 percent of the \$80 per kg reserves for the U.S., as of January 1, 1984. During the conferences, reviews were made of properties already in the EIA records and new properties were added. About 50 percent of the properties examined during the company conferences have been reevaluated since January 1, 1991.

The response and cooperation of the companies have been excellent. They have strongly supported the new EIA approach to national reserve estimation. Their positive response provides assurance that the program will be

successful. The input to the program from the companies has been very helpful and has led to increased industry participation in the national uranium reserve program.

Modification of Company Estimates

Review of company responses to Form 858 and information from the technical reviews may indicate the need for EIA to modify company estimates so they conform to EIA criteria. The most common problem expected is that company estimates do not conform to the cost criteria selected by the EIA. Knowledge of company procedures and of the nature of the reserves in the ore deposits in question can provide a suitable technical basis for modification of estimates or development of additional estimates with alternative criteria. Parametric relationships for deposits amenable to extraction by different types of methods, such as where in-situ leaching can be used in place of conventional mining, are being studied to help in this activity. The goal is to find means to get to an acceptable estimate without doing a complete estimate of the reserves. This would require much more time and manpower.

Independent EIA Reserve Estimates

If no suitable means to modify company estimates can be developed it will be necessary for the EIA to prepare independent estimates. These independent estimates also can provide the information needed to develop the means for modifying future company estimates. Thus it may not be necessary to continue to do the independent EIA estimates once deposit and procedure relationships are developed and well understood.

The EIA will make independent reserve estimates using company supplied data and data in EIA files on the deposit. As many companies are using computer methods for handling sample data and for reserve estimation, acquisition of data should be simplified from earlier times when the basic analog hole log records on paper were usually obtained. Such logs require considerable effort in digitizing and entering into the computer records. Some logging records now include digital magnetic data as a routine part of the logging procedure.

The basic computer programs now in use were derived from the programs developed in Grand Junction for the uranium ore reserve program. These programs have been modified to run on IBM compatible "PC" computers, which now have the capability of doing computing jobs only possible previously on main frame computers. The modification of the programs was supported by the IAEA and published in 1988 as TECDOC 484, entitled "User's Guide for the Uranium Ore Reserve Calculation System URAD." The EIA has made some additional modifications to ease use of the system, to allow use of a variety of basic data formats and to allow plotting of data used in the system. These modifications will ease the estimation of reserves from company data.

The validity of the basic computer techniques and programs was well established through their extensive use in the Grand Junction program. Many changes, however, have been made to allow their use on PC's and to accommodate additional data formats. Test cases have been run to verify the reliability of the modified software. As part of this review, cooperative studies with industry have been carried out, comparing results from the use of different

estimation procedures and computer programs and different estimation criteria. These studies have provided confidence in the programs and techniques now in use.

As the EIA program will be dealing with different types of ore deposits to be mined in varying ways and with data formatted and processed by the mining companies, it is important that the system used by EIA have considerable flexibility in data handling. To aid in developing that flexibility, additional software, including commercial programs in use by the mining companies, is being investigated. Improvement and modification of software will be a continuing activity.

Improved Reserve Estimation Data and Techniques

To assure well founded evaluation of company provided information and to support independent EIA estimates, efforts continue to improve a variety of basic and ancillary data needed for reserve estimation. These data relate to an understanding of the nature of the deposits, which influences areas of influence, mining and processing technology, which effects costs, recoveries, and mining thicknesses, and to industry production practice and costing.

Our goal is to have reserve estimation methods that produce estimates meeting EIA criteria for a variety of parameters and that require a minimum of labor and time. They also should have the flexibility to handle different types of data from geologically variable deposits, which are to be mined in various ways. These data and technique developments will be pursued in parallel with the estimation program and will investigate different approaches.

Estimate Compilation and Reporting

As estimates are accepted as conforming to the EIA standards, the data are entered into the national uranium reserve data base. This allows for flexibility needed for the EIA's analytical and data reporting requirements. The data are used to assess the current status of the U.S. uranium industry, as well as to form the basis of estimating future supply capability.

The *Uranium Industry Annual* report, published by the EIA, contains a variety of information on the U.S. uranium industry, including updated resource estimates. In addition, information will be presented in the publications of the OECD Nuclear Energy Agency and the International Atomic Energy Agency and in papers presented at industry meetings, such as the annual Uranium Seminars of the U.S. Council for Energy Awareness. All data will continue to be reported in an aggregated form so as not to divulge information on any one company.

Results of the New Approach

The results of applying the new approach to estimating reserves for 1990 are summarized in Table 5. As the new approach has not covered all the properties and companies involved, these estimates only partially reflect the new findings. At the end of 1990, the reserve estimate for the \$80 per kg U forward-cost category was 101,900 tonnes

Table 5. Changes in \$80, \$130, \$260 per kg U Forward Cost Reserve Estimates During 1990 (Tonnes Uranium)

Item	\$80/kg U	\$130/kg U	\$260/kg U
Reserves, End of 1989	106,500	369,000	591,000
New Reserves	7,300	8,800	10,400
Reevaluations ^a	(9,600)	(19,200)	(15,800)
Depletion (Production and Erosion)	(2,300)	(3,500)	(4,600)
Reserves, End of 1990 ^b	101,900	356,000	581,000

^aNet additions and subtractions

^bDoes not include reserves from byproduct facilities

uranium held in 227 properties. The \$130 per kg U reserve estimate was 356,000 tonnes uranium held in 568 properties.

The net decrease in reserve estimates for 1990 compared to 1989 was established by the EIA staff largely through the reevaluation of known properties (Table 5). The reevaluation process included (1) the modification of company data to meet EIA criteria and (2) the results of the EIA conducting independent reserve estimates from basic drill hole data received from companies at technical meetings. "New" reserves, or those reserve estimates for properties that were added to the EIA data base as a result of findings made at company meetings, contributed to 7 percent of the total estimated reserves for the end of 1990.

Based on the evaluation of company data, the EIA assessed the distribution of reserves most likely to be extracted by various types of mining methods. Conventional underground mining continues to be the most dominant class, comprising over 50 percent in each cost category. The share of reserves estimated to be amenable to recovery by in situ leaching at the end of 1990, however, has increased compared to 1989 by 32, 18, and 16 percentages, respectively, for the \$80, \$130, and \$260 per kg U forward-cost categories.

Plans

Work is continuing on all aspects of the new approach. The goal is to develop sound and accurate national estimates of uranium reserves at a variety of cost levels that are well understood, and arrived at with a uniform set of standards. Working closely with industry, improvements in data gathering and analysis will be sought to produce acceptable estimates with a minimum of effort by industry, and by the EIA. Meetings will continue to be held with industry staff to complete our technical reviews of the principal companies and deposits involved.

Improvements in analytical and reserve estimation procedures will be pursued, including improvement in current software and acquisition of additional programs. The EIA seeks cooperative activities with the industry and

internationally to improve reserve estimation technology. Development of better understanding of the parametric relationships of ore reserves, including costs, will continue. Analysis of the production levels attainable from reserves at various cost levels will be undertaken.

Conclusion

The new approach to developing national uranium reserve estimates adopted by the EIA is expected to provide a reliable set of data that will have the confidence of the industry. The method will draw on company data to the greatest extent possible. Company estimates will be used where possible, or they will be modified to EIA criteria. As necessary, independent estimates will be made by the EIA using company data. Priority is given to the larger deposits and the companies with the largest reserve holdings. This will provide the fastest improvement to the reserve estimates.

A close working arrangement with the industry is fundamental to the program. The revised program of data acquisition and technical review is now well underway. Industry acceptance has been very good. A closer working relationship has been established that will benefit the uranium industry and those who rely on EIA data on uranium.

GRADE AND TONNAGE MODELS FOR URANIUM RESOURCE ASSESSMENT AND EXPLORATION

W.I. FINCH

United States Geological Survey,
Denver, Colorado

R.B. McCAMMON

United States Geological Survey,
Reston, Virginia

United States of America

Abstract

Grade and tonnage models for uranium developed thus far are of three varieties: (1) the standard grade and tonnage models of the U.S. Geological Survey (USGS) used in the MARK3 program to estimate undiscovered uranium ore, (2) the authors' deposit-size-frequency and average grade distribution model used to estimate undiscovered uranium endowment, and (3) the U.S. Department of Energy (DOE) grade-tonnage model used to estimate undiscovered potential uranium resources. All three of these are based on descriptive geologic models for the type of deposit being assessed. Results and much of the input data are expressed in probability distributions. The standard grade and tonnage models, which consist of two graphs, express average grade and tonnage of ore in proportion of number of deposits plotted against a log-scale of grade and tonnage, respectively, along the abscissa. Statistics for these visual graphs and the subjective estimates of numbers of deposits expected in the favorable area are used in the MARK3 computer simulation program to calculate probability estimates of the tonnage of ore and contained U_3O_8 in undiscovered uranium deposits. The deposit-size-frequency (DSF) method of estimating undiscovered uranium endowment uses a matrix of the probable numbers of deposits estimated in established size classes (expressed in tons of mineralized rock at endowment grade) combined with the distribution of the average grade of U_3O_8 for deposits in a selected control area to produce probabilistic estimates of the contained U_3O_8 in undiscovered deposits. A comparison of estimates of undiscovered uranium endowment obtained using the DSF method and the estimates of undiscovered uranium ore obtained by the MARK3 method for solution-collapse breccia deposits of the Grand Canyon region in Arizona USA shows that the probability estimates for the two

methods are similar. The larger (mean=15,000 tonnes U_3O_8) DSF estimates for undiscovered endowment are for deposits with a lower grade cutoff and include numerous small deposits whereas the smaller (mean=12,000 tonnes U_3O_8) MARK3 estimates for undiscovered ore are for deposits having a higher grade cutoff and includes deposits only of larger sizes. Considering these differences in the two methods, the results are remarkably compatible. The DOE grade-tonnage model consists of plots of the percent of uranium inventory (pre-production tonnage) versus grade-cutoff and the resulting average grade at different grade-cutoffs. Three mathematical parameters of this model are used in the DOE URAD (Uranium Resource Assessment Data) cost model to calculate the estimated potential uranium resources in various forward-cost categories. For a selected uranium deposit type, the USGS standard grade and tonnage models can be used to predict the grade and target sizes of undiscovered deposits for exploration, especially in frontier areas. Construction of standard grade and tonnage models for some types of deposits may reveal subtypes by segregation of either grade or tonnage values into distinct distributions that can be related to specific geologic environments, thus possibly revealing relations not suspected before the modelling.

INTRODUCTION

Grade and tonnage models derived from known uranium deposits provide the basis for quantitative assessments of undiscovered uranium resources as well as to serve as useful guides in exploration. The grade and tonnage models developed thus far are of three varieties: 1) the standard grade and tonnage graphical models of the U.S. Geological Survey (USGS), 2) deposit-size-frequency (DSF) data-matrix models of the USGS used in a method modified after one that was used for the U.S. Department of Energy (DOE) National Uranium Resource Evaluation (NURE) program, and 3) the grade-tonnage curves of DOE. All are based on the concept of a descriptive deposit model generally in the format of the systematic arrangement of information that describes the essential attributes of a given type of uranium deposit. Each of these models was developed for specific uses described below. Although all three may guide exploration, the standard grade and tonnage models are probably most useful.

We acknowledge the aid of W.D. Grundy (U.S. Geological Survey) in plotting the graph shown in Figure 5.

GRADE AND TONNAGE MODELS

Standard USGS grade and tonnage models

Sixty standard grade and tonnage models are published in the "Mineral Deposits Model" book of the U.S. Geological Survey (Cox and Singer, 1986); only two of these are for uranium deposits. A third uranium deposit type, solution-collapse breccia pipe uranium deposit, model is in press (Finch, Pierson, and Sutphin). The standard USGS grade and tonnage models show the average grade and tonnage of ore in proportion of the number of deposits as the ordinate plotted against a log scale of grade and tonnage, respectively, along the abscissa. Average grades and the associated tonnage based on production, reserves, and resources at the lowest possible cutoff grade are used to prepare standard grade and tonnage models (Cox and Singer, 1986). Each plot represents an individual deposit cumulated in ascending proportion. Smoothed curves based on the mean and standard derivation of the data are drawn through the plots (Singer, in press). By convention, the log scale is identical for a given metal and the size of the graph is the same. This allows visual comparisons to be made between different models. A computer program to plot the models is given in Singer and Bliss (1990). On the graphs for each model, the value of grade and tonnage is shown at the 90th, 50th, and 10th percentiles.

The grade and tonnage models for unconformity-related deposits are shown in figure 1. These models by Mosier (1986a) show that 50 percent of the deposits have an average grade of at least 0.49 percent U_3O_8 and a size of at least 260,000 tonnes of ore. Only 10 percent have an average grade greater than 2.0 percent U_3O_8 or a size greater than 9,600,000 tonnes of ore.

The grade and tonnage models for volcanogenic uranium deposits are shown in figure 2. These models by Mosier (1986b) show that 50 percent of the deposits have an average grade of at least 0.12 percent U_3O_8 and a size of at least 340,000 tonnes of ore. Only 10 percent have an average grade greater than 0.25 percent U_3O_8 or a size greater than 5,500,000 tonnes of ore.

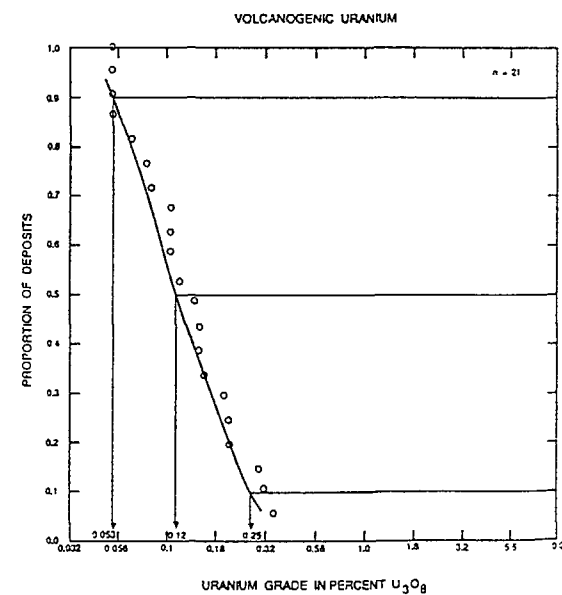
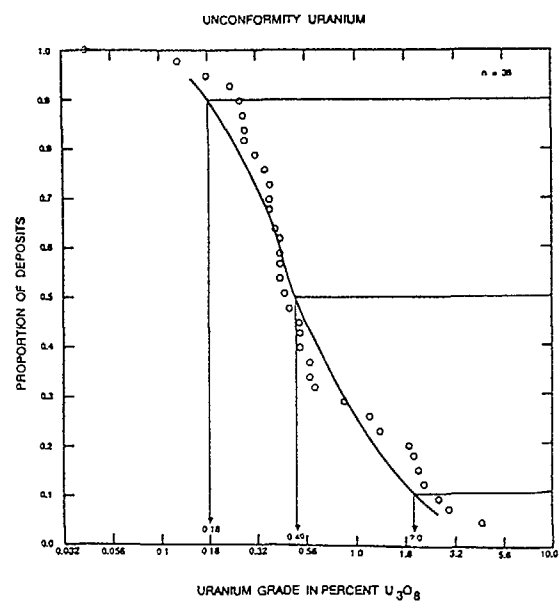
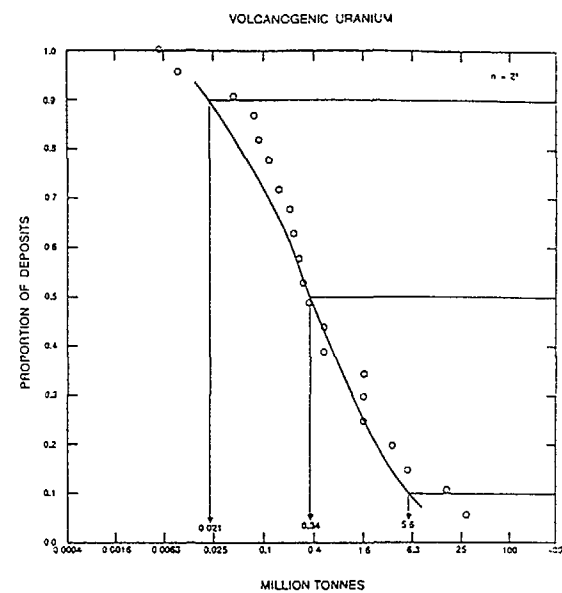
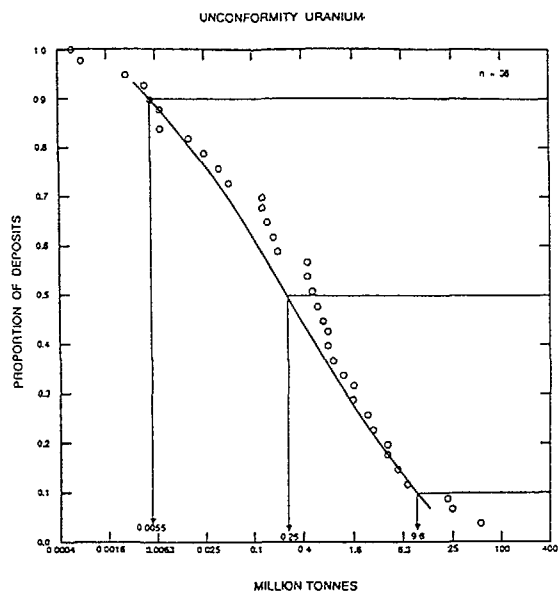


Figure 1. Grade and tonnage models for unconformity-related uranium deposits (from Mosier, 1986a)

Figure 2. Grade and tonnage models for volcanogenic uranium deposits (from Mosier, 1986a)

The grade and tonnage models for solution-collapse breccia pipe uranium deposits are shown in figure 3. These models by Finch, Sutphin, and others (in press) show that 50 percent have an average grade of at least 0.56 percent U_3O_8 and a size greater than 230,000 tonnes of ore. Only 10 percent have an average grade of at least 0.66 percent U_3O_8 or a size greater than 500,000 tonnes of ore.

Deposit-size-frequency (DSF) models

The distribution of grade and tonnage for the DSF model is shown by grade in percent U_3O_8 expressed at the 95th percentile, most likely value, and 5th percentile and by a matrix of numbers of deposits estimated in the same categories for size classes ranked from smallest to largest. Geometric or log-geometric size classes are most commonly used. The size classes are established for a given type of deposit and for a specific area (commonly well explored), which is called a control area. The DSF model takes into account the spatial density of deposits according to their size. Examples of DSF models are shown in table 1 for solution-collapse breccia pipe uranium deposits in the Grand Canyon region of Arizona, and in Otton (1987, table 3) for young organic-rich uranium deposits in the Lake Gillette area, Washington USA. In the Hack-Pinenut control area, the number of deposits in the size classes is based on production, reserves, and endowment of known deposits and on subjective estimates of the grade and tonnages of unexplored identified pipes. The model in table 1 shows that 50 percent of the deposits have an average grade of at least 0.17 percent U_3O_8 and a size range of 1 to 20,000,000 tonnes. Only 5 percent have an average grade greater than 0.44 percent U_3O_8 .

DOE grade-tonnage curves

During the National Uranium Resource Evaluation (NURE) program of the U.S. Department of Energy, grade-tonnage curves were constructed for deposits of various types, but mostly sandstone-type deposits in the U.S.A. Curves for roll-front sandstone-type and classical vein-type deposits are published in an International Atomic Energy Agency (IAEA) Technical Report (Finch, Harris, Ruzicka, and Müller-Kahle, in press). A set of curves for tabular sandstone-type deposits in the Ambrosia Lake control area (Hetland and Grundy, 1977), New Mexico, is shown in figure 4.

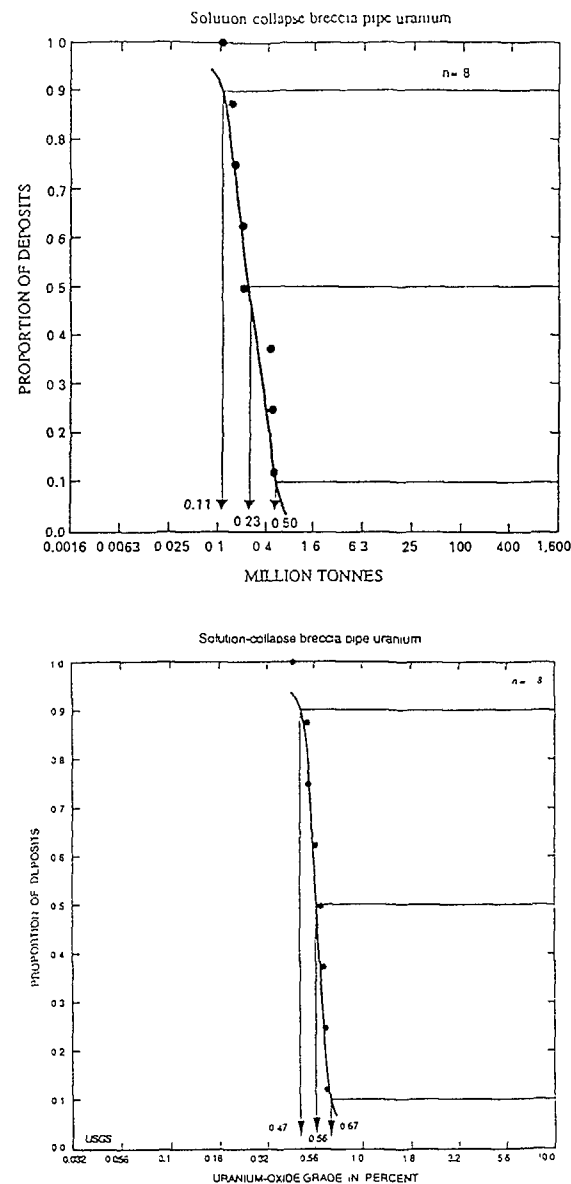


Figure 3. Grade and tonnage models for solution-collapse breccia pipe uranium deposits (from Finch and others, in press)

Table 1. Estimated grade distribution, G , and size-frequency distribution for solution-collapse breccia deposits in the Hack-Pinenut control area in the Grand Canyon region of Arizona USA (Finch and others, 1990)

Grade Distribution (G)			Size-frequency distribution						
Percent U_3O_8 at 0.01% cutoff			Size class (k)	Size-class interval (tons of mineralized rock above cutoff of 0.01% U_3O_8)			Number of deposits ¹		
Lower (0.05)	Most likely value	Upper (0.95)		Lower (0.05)	Midpoint ²	Upper (0.95)	Lower (0.05)	Most likely value	Upper (0.95)
0.06	0.17	0.44	1	1	1.4×10^2	2×10^4	5	14	16
			2	2×10^4	6.3×10^4	2×10^5	3	4	6
			3	2×10^5	6.3×10^5	2×10^6	2	3	4
			4	2×10^6	6.3×10^6	2×10^7	0	1	1
TOTAL							10	22	27

¹Odds are 9 to 1 that the stated interval contains the true mean value

²Midpoints of size-class intervals for size classes 1-4 represented by the geometric mean of the upper and lower limits

The curves are based on drill-hole data of the inventory, which is pre-production tons U_3O_8 at and above a minimum grade of 0.01 percent U_3O_8 contained in discovered mineralized material, for either a single deposit or collective ore for a set of deposits in a well-explored control area. The percent of inventory above cutoff grade forms the left ordinate. One curve is the average grade of inventory at various cutoff grades plotted from the right ordinate, cumulative average grade versus increasing cutoff grade, and the second curve is percent of inventory at various cutoff grades, cumulative tons mineralized material versus cutoff grade. From these curves, values of average grade at various cutoff grades can be predicted. For example, at a cutoff of 0.09 percent U_3O_8 the average grade is 0.18 percent U_3O_8 and the percent of total tons of U_3O_8 inventory is 48 percent (figure 4)

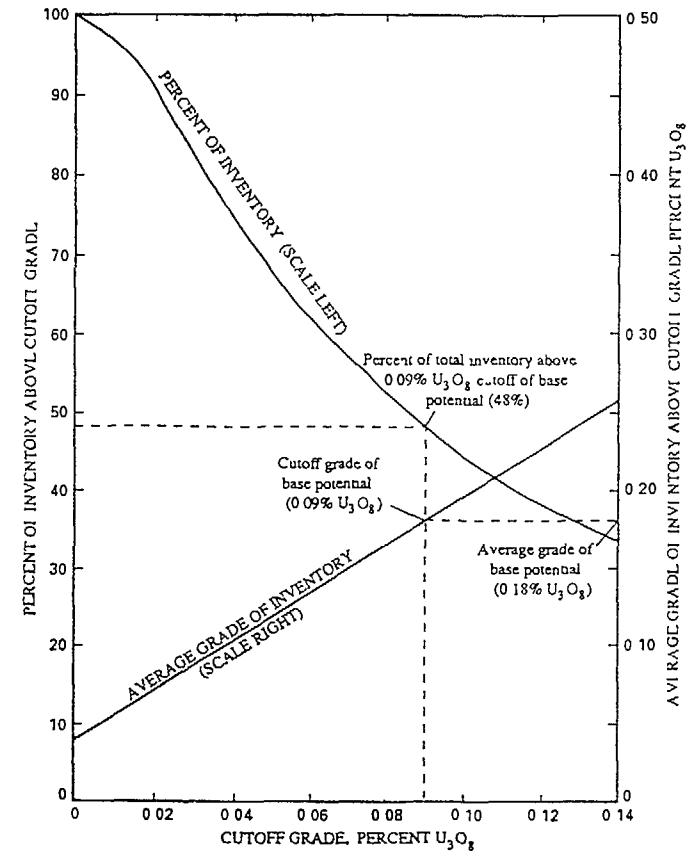


Figure 4 Grade tonnage (expressed in percent of inventory) curves for the Ambrosia Lake control area, New Mexico, USA (from Hetland and Grundy, 1977)

DESCRIPTIVE MODELS

Descriptive models are of many formats but in general have in common the name of deposit type or some other identifier, commonly related to classifications of mineral deposits, and have various elements of the geological environment, such as rock types and textures, age of mineralization, and tectonic setting, and the deposit description, including its geochemistry,

geophysical characteristics, geometry, ore controls, zoning patterns, and other features (Singer and Cox, 1988). Descriptive models of four types of uranium deposits are in the "Deposit model book" (Cox and Singer, 1986), and others less formalized are scattered in other publications (Mathews and others, 1979; Otton, 1987). The principal uses of descriptive models are to identify possible deposit types for a given environment, to evaluate the favorable portions of an area being assessed, to evaluate the degree of favorability of permissive portions, and to help estimate parameters for the various assessment methods, such as expected number of deposits in an area for the size-classes in the DSF method and for the MARK3 simulation process.

COMPARISON OF RESOURCE ESTIMATES OBTAINED USING THE DSF AND MARK3 METHODS

The DSF method is a modification of the standard NURE estimation equation, $U = A \cdot F \cdot T \cdot G^1$, (A=area, F=fraction of area mineralized, T=tons per unit area, G=grade) by replacing the factors F·T by a single factor that represents the tonnage of the total number of deposits in all size classes. Use of the DSF method requires knowledge of the size frequency of deposits and distribution of average grade in a well- but not completely- explored control area (table 1). The favorable area, A, is measured in appropriate units. In the DSF equation, a likelihood factor, L (generally 1 or less than 1), is used to express the similarity of the favorable area to the control area. The probability distribution estimates of undiscovered uranium endowment are calculated by entering the required data into the DSF equation (see Finch and McCammon, 1987) and by using the TENDOWG computer program (McCammon and others, 1988), a modification of the program by Ford and McLaren, (1980).

The MARK3 method uses a Monte Carlo simulation process to calculate the undiscovered uranium ore estimates in the form of percentiles (for example, the 90th, 50th, 10th) based on

¹ The NURE equation was developed because typical sandstone-type uranium deposits cannot be defined well enough to be individually counted. A cluster concept has been proposed to allow the DSF methods to be used for these sandstone deposits (Finch, 1991; Finch, Grundy, and Pierson, in press).

subjective estimates of the probable number of deposits in the region being assessed and on the statistics used to construct the appropriate grade and tonnage models (Drew and others, 1986; Reed and others, 1989; Root and Scott, 1988). For the area being assessed, the estimates of the number of deposits is judged partly on the size of the area but more importantly on the geology of the area compared to that of known districts and on exploration history. Attention is paid to the abundance of structures and other features known to localize the deposits.

A comparison of the results obtained using the DSF method with those obtained using the MARK3 method for solution-collapse breccia pipe uranium deposits in the Grand Canyon region in Arizona is shown in figure 5. The mean value for the undiscovered uranium endowment for the DSF method is 14,905 (about 15,000) tonnes U_3O_8 (Finch and others, 1990), and mean value of undiscovered uranium ore for the MARK3 method is 11,666 (about 12,000) tonnes U_3O_8 . The DSF result includes smaller deposits than the MARK3 method and considering these basic differences in the kind of material estimated by the two methods, the results are remarkably similar.

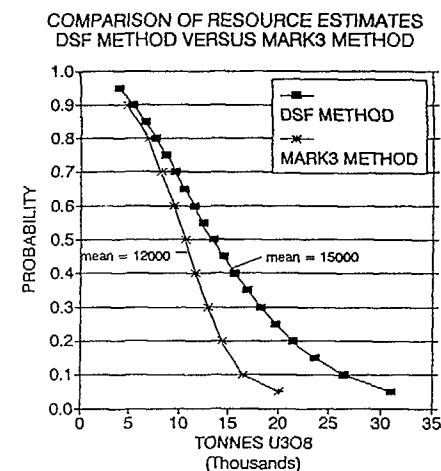


Figure 5. Graph showing the probability estimates of unconditional uranium resources based on the deposit-size-frequency and MARK3 methods

USE OF DOE GRADE-TONNAGE CURVES TO ESTIMATE POTENTIAL URANIUM RESOURCES

In the past, DOE has applied economic factors to the uranium endowment estimate to obtain the potential (mineable) resources in three forward-cost categories (Ford and McClaren, 1980; Blanchfield, 1980; Das and Lee, 1991). The grade-tonnage curves are used to define a probability distribution for grade that is used to calculate the probability distribution of tonnage of undiscovered resources. The three parameters from the grade-tonnage curve are (1) the average grade of the endowment (grade cutoff=0.01 percent U_3O_8), (2) slope of the "average grade of inventory" line (generally the linear segment above 0.04 percent cutoff grade), and (3) average grade of ore at a cutoff grade of 0.04 percent U_3O_8 (see Appendix B, p. 75 of Blanchfield, 1980 for details). These are combined with various physical and economic market factors in the DOE URAD [Das and Lee, in press; not to be confused with IAEA URAD (Uranium Reserves and Data) in IAEA-TECDOC-484] to calculate the estimated potential uranium resources for each of \$30, \$50, and \$100 per pound U_3O_8 cost categories in each favorable area. Summaries of the potential resources for the U.S. are published each year by Energy Information Administration of DOE (Energy Information Administration, 1990).

USE OF STANDARD USGS GRADE AND TONNAGE MODELS TO PLAN AND EVALUATE EXPLORATION

The standard USGS grade and tonnage models coupled with descriptive models are useful to plan and execute exploration. Pre-exploration study can be guided by these models. In a frontier area, the models for a given type deposit are an indication of the target size, and one can plan a drilling program that takes into account the expected sizes of undiscovered uranium deposits. Once a discovery has been made and size of the deposit determined, one can plot its grade and tonnage on an appropriate model graph to give some idea about probable grade and tonnage of the remaining undiscovered deposits. For example, if the geology indicates that unconformity-related vein deposits can be expected, the model to use is shown in figure 1. In mature areas, plotting of the distributions of grades and tonnages of known deposits and

comparing the plots with available models provides an insight into possible grades and tonnages of undiscovered deposits.

Grade and tonnage models offer a way to compare distributions between deposit types, and thus have a bearing on management decisions to explore one type over another type. Relations of grade and tonnage among the three types of deposits modelled in figures 1-3 are shown in figures 6 and 7. These figures show significant differences: grades for unconformity-related deposits have a much wider range than those for volcanogenic deposits, and breccia pipe deposits have a narrower range of grade at about the midpoint of the grade for unconformity-related deposits. The sizes of the three types are grouped more tightly.

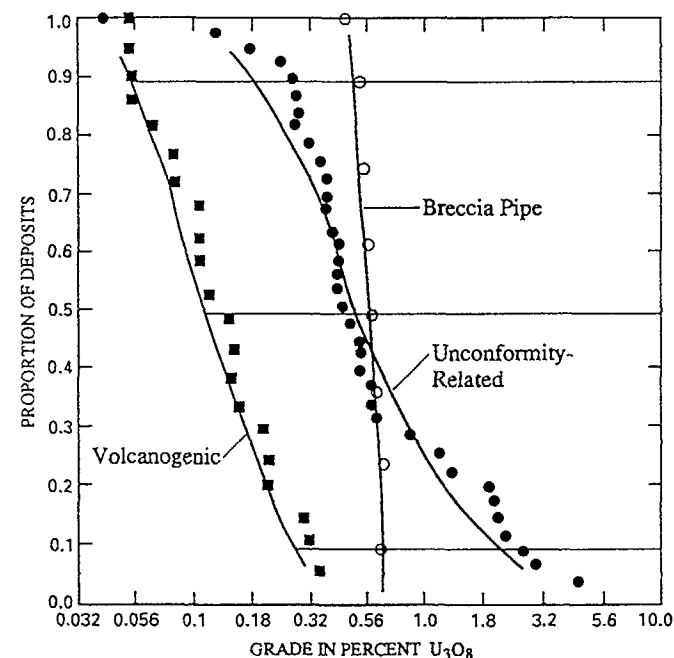


Figure 6. Comparison of grade models for unconformity-related, volcanogenic, and breccia-pipe uranium deposits

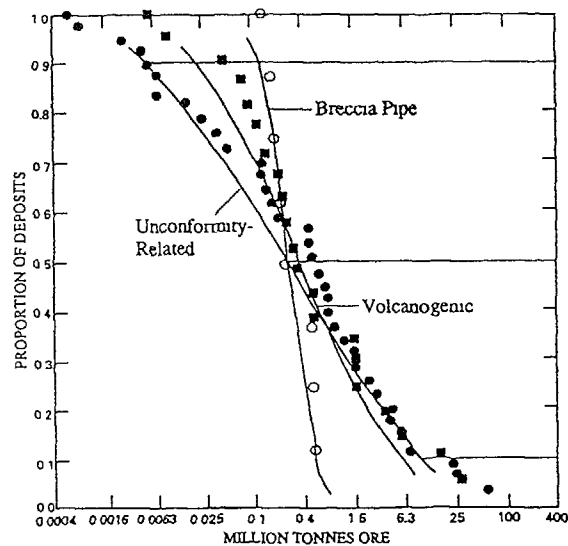


Figure 7 Comparison of tonnage models for unconformity-related, volcanogenic, and breccia-pipe uranium deposits

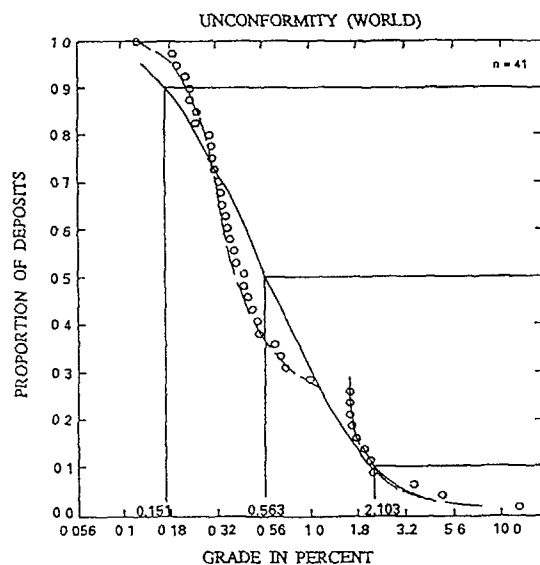


Figure 8 Grade models for unconformity-related uranium deposits for Canada (lower dashed line) and Australia (upper dashed line) (from Ruzicka, 1990)

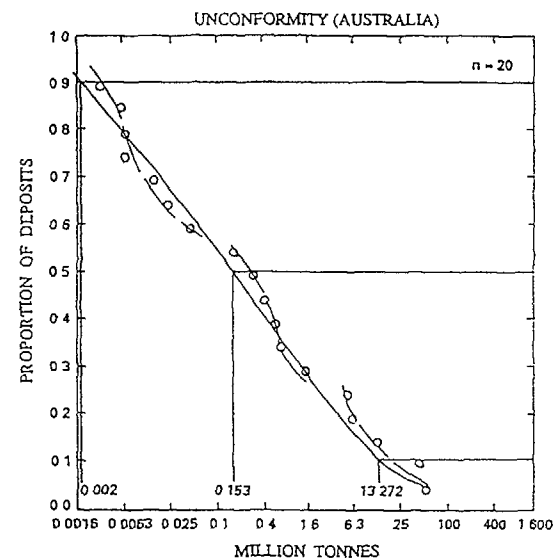


Figure 9 Tonnage model for Australian unconformity-related deposits in solid line, lower dashed line for deposits hosted by metasedimentary rocks adjacent to major Archean/Proterozoic granitic complexes, middle dashed line for deposits associated with metasedimentary rocks near to smaller to distant from large granitic complexes, upper dashed line for small deposits associated with volcano-sedimentary rocks (from Ruzicka, 1990)

ADDITIONAL CONSIDERATIONS

Of the three types of models discussed in this paper, the standard grade and tonnage models offer additional insights into the geologic processes affected the intensity of mineralization. Construction of grade and tonnage models may reveal subtypes within a type thought to be of fairly uniform occurrence (Singer, in press). This is true of the unconformity-related deposits modelled by Ruzicka (1990; also in Finch, Harris and others, in press) and shown in figures 8-10. The grade of Canadian deposits is distinctly higher than those of Australia (fig. 8). For deposits in 8) For deposits in Australia, the tonnages segregate into three distributions relative to their geologic environments: granitic, metasedimentary, and volcano-sedimentary rock terranes (fig 9)

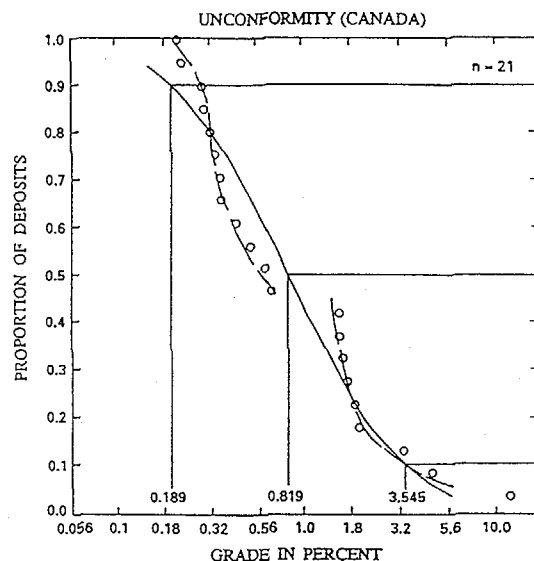


Figure 10. Grade model for Canadian unconformity-related deposits in solid line, lower dashed line for deposits hosted by basement rocks, upper dashed line for deposits hosted by sandstone (from Ruzicka, 1990).

For deposits in Canada, the grade distribution is in two groups related to the host of the ore, basement rocks or sandstone (fig. 10). In these cases, it may be necessary to plot separate grade models for each group. Although not shown here, Australian deposits exhibit a much larger range in tonnage of ore than Canadian deposits (Ruzicka, 1990).

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GEOLOGICAL FEATURES AND NEW DEVELOPMENT OF THE XIASHUANG URANIUM ORE FIELD IN SOUTH CHINA

Feng SHEN
Bureau of Geology,
China National Nuclear Corporation,
Beijing

Yongzheng PAN, Zhigen GONG
South China Bureau of Geological Exploration
of Nuclear Industry,
Shaoguan

Jiashu RONG
Beijing Research Institute of Geology,
Beijing
China

Abstract

Granite-related vein type uranium deposits are of the main uranium type in China. Xiazhuang uranium ore field, a typical one among them, lies in uranium-enriched two-mica granites in the eastern part of Guidong massif, South China. Within the two-mica granites, diabase dykes and later formed silicified fracture zones of different trend occur in swarms, which have roughly the same distance to each other respectively. Pitchblende-bearing microquartz veins are usually concentrated in the location where diabase dykes are intersected by microquartz veins and in the vicinity of contact zone of two-mica granites with muscovite granites. Several deposits are also found in the exocontact zone and in the overlying red basins as well. The development process of the ore field with the advancement of one's knowledge is emphasized.

1. INTRODUCTION

Exploration for uranium in China has been underway since 1955. The Bureau of Geology (BOG), China National Nuclear Corporation (the former Ministry of Nuclear Industry) has been undertaking the responsibility for survey, location, evaluation, exploration and final handing-over the technically feasible and economically viable uranium deposits for commercial exploitation. During the past three and a half decades,

BOG has successfully identified hundreds of uranium deposits and thousands of uranium occurrences of various types of mineralization, so that the uranium resources which have been established can well meet the demand for uranium fuel in China's nuclear industry and nuclear power programme.

Up till now, with the exception of unconformity-related type and quartz-pebble conglomerate type of Lower to Middle Proterozoic period which are of typical high-grade, large tonnage, almost all the other major types of uranium deposits in the world have been discovered in China. Among them four significant types are granite type, volcanic type, sandstone type and carbonate-siliceous-pebbly type. The geological environments of the uranium deposits found in China are quite different from the ones occurring in other countries. While most of uranium deposits in the world exist in the Lower structural layer (in Archean and Proterozoic rocks), or in the Middle structural layer (in Caledonian, particularly in Hercynian rocks), most of the uranium deposits in China appear in the Upper structural layer (in Mesozoic and Cenozoic rocks), with very young mineralization ages (usually < 90 Ma).

Most of the successful exploration programmes are often based on proper application of exploration strategy and judicious combination of different techniques; any single technique can rarely succeed. Generally, there always is a long or short tortuous exploration phase during the development of the uranium ore fields or even each single uranium deposit discovered in China. During the exploration process of uranium deposits, it is quite difficult to work out a suitable strategy for exploration, and to select the most effective exploration methods to suit a given geological environment, but this is something must be done for the success of the exploration programme. In this paper the geological features and metallogenesis of one of the most significant discovery in uranium exploration in China—Xiazhuang uranium ore field in Guangdong Province, South China—are involved, the case history, the evolution of exploration strategy and exploration methods adopted, and the knowledge derived from there, are reviewed and discussed.

2. GEOLOGICAL FEATURES AND METALLOGENESIS OF XIAZHUANG URANIUM ORE FIELD

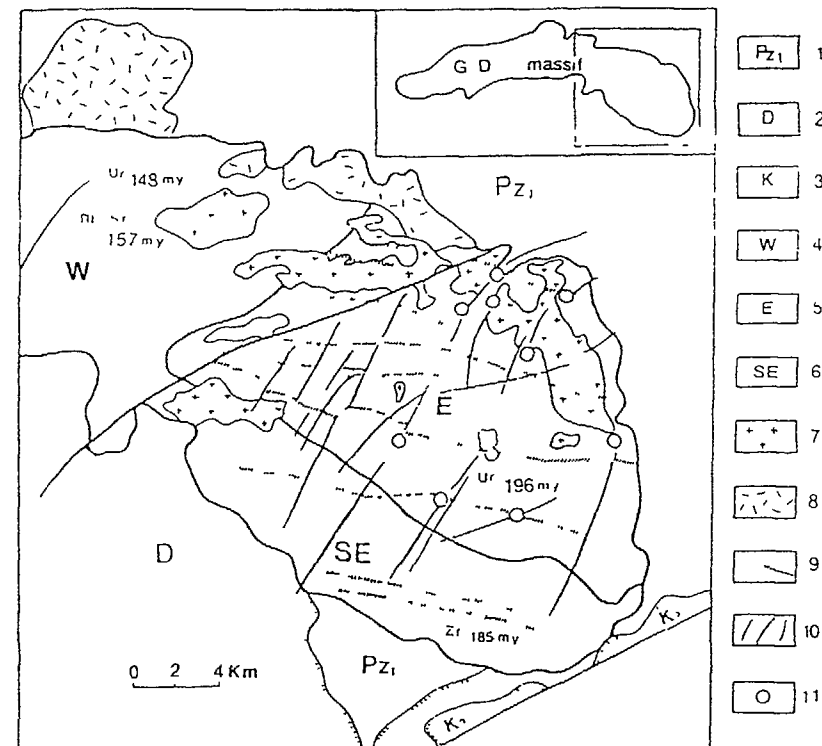
South China Uranium Province is located to the south side of North China Platform. Tectonically, it belongs to Yangzi Platform and South China Caledonian orogenic belt. The granite type uranium deposits which are of great economic significance, are widespread in four provinces and autonomous region, namely, Guangdong, Jiangxi, Hunan and

Guangxi, while less amount of them is developed in other regions. Among them, Xiazhuang uranium ore field, being an important base of uranium resources, an important part of South China uranium province, and one of the first target areas for uranium exploration programmes, is located in Wengyuan county, north of Guangdong Province, South China. It mainly occurs within or around the eastern part of Guidong granite massif of Indosinian–Yanshanian period, which is 68 km long, 12–18 km wide, covering an area of 1009 sq kms, situated at the southeastern fringe of Cathaysia post–Caledonian uplift, the central part of South China Caledonian orogenic belt, and distributed in the interior of the matured continental plate where the continental crust has undergone repeated crustal movement and intense multiple magmatism. The country-rocks of the granite massif mainly comprise of Cambrian and Devonian epimetamorphic rocks—argillite and sandy rocks. The uranium content in Cambrian and Ordovician series distributed in the eastern and northern parts of the massif ranges from 8 to 9 ppm, in Devonian and Carboniferous series from 5 to 13 ppm.

The study carried on in recent years shows that the major intrusive body of Guidong massif instead of a single one consists of three intrusive bodies of different stages, namely, Xiazhuang medium-grained two-mica granite body (196 Ma) in the eastern part, Luxi coarse-grained porphyritic biotite granite body (185 Ma) in the southern part and Aizi medium-coarse grained hornblende–biotite granite body (157 Ma) in the western part, followed subsequently by multistage supplementary intrusions (mainly of fine-grained moscovite granite) and multiple dykes intruded into the massif.

Being a uranium and uraninite-enriched massif, Xiazhuang two-mica granite body has an average uranium content of 20 2ppm, $Th/U \approx 1$, and uraninite content of 10g/t. The granite massif has experienced various types of intense autometamorphism, e.g., albitization, potash-feldspathization, moscovitization, quartzification, and had been subjected to the intense compression of early stage prior to the intrusion of western granite body, therefore the rocks are of high brittleness, and can easily be cataclased. This is probably why diabase dykes, lamprophyre dykes, as well as multiple quartz veins and hydrothermal alteration widely spreaded in later stage are well developed only within Xiazhuang (and Luxi) granite body under the action of regional stress.

During the fault block movement taking place at late Cretaceous age, Xiazhuang granite body was once uplifted and eroded, subsequently subsided and partly covered by down-faulted basins. The ore field, with an area of 320 sq kms, is located within Xiazhuang granite body and the overlying basins, as well as in the exocontact zone in metasedimentary rocks.



- 1 Early Palaeozoic, 2 Devonian system, 3. Late Cretaceous red bed, 4 Western body—Hb-Bi granite, 5 Eastern body—two-mica granite, 6. Southeastern body—giant porphyritic biotite granite, 7 medium-fine grained two-mica granite, 8 dacite-porphyry, 9 diabase dyke, 10 quartz vein, 11 uranium deposit.

FIG 1 Geological map of the Guidong granitic body

The uraninite in Xiazhuang granite body has partly suffered dissolution, the uranium leaching rate is quite high, coming to nearly 60% of the total uranium in rocks. Lead-isotope dating shows that a great loss of uranium (20–80%) took place in Xiazhuang granite body compared with the calculated primary uranium content, so Xiazhuang granite body acted probably as a uranium source.

After the acidic magmatism had been over, in the scope of Xiazhuang and Luxi granite body, a lot of basic dykes, quartz veins, silicified fracture zones were well developed.

among them 5 groups of NWW trending diabase dykes and schistosed silicified fracture zones, 2 regional NEE trending giant quartz veins and 5 groups of NNE trending silicified fracture zones were successively formed, comprising the network structural framework of the ore field

The diabase dykes occur in swarms with an approximately equal distance of 3-5 km between each swarm. Each single dyke, with an average width of 2-10 m, a maximum width of 100 m, and with the dip over 50°, is quite stable. The silicified fracture zones are filled with microquartz. Furthermore, also exist a series of NEE-NWW trending schistosed silicified fracture zones. Silicified fracture zones are predominant ore control factor, while diabase, episyenite, silicified granite and carbonaceous sandstone are favourable host rocks for uranium mineralization.

Two types of uranium deposits can be distinguished according to the types of ore-control structures. (1) uranium deposits located at the superimposition of NWW trending silicified fracture zones and the same trending compressed zones or block sandwiched between diabase dykes and schistosed zones. The attitude of ore bodies is similar to the NWW trending compressed zones. The width of each single ore body ranges from 1 m to more than 10 m, and the length can reach to 240 m. The ore-grades of this type vary depending on the difference of host rocks. In episyenite the grade ranges from 0.07% to 0.1%, in silicified granite from 0.1% to 0.17% and in silicified diabase from 0.155% to 0.205%. (2) uranium deposits occurred at the intersection of NNE trending silicified fracture zones and NWW trending diabase dykes. Ore bodies are strictly controlled by structure features, they occur in tabular or pillar shapes, and often in swarms. Each single ore body is 1-3 m wide and 40-50 m long.

Ore bodies also occurred at the place where silicified fracture zones cut across the contact interface of two stages of granite rocks, probably due to the difference of physico-mechanical properties between them.

Three stages of hydrothermal process can be recognized. First stage, pre-ore stage, forming white, fine grained quartz, second stage, ore stage, resulting in red, black microcrystalline quartz and later dark purple fluorite (=calcite) bearing mainly pitchblende and sulfide, coffinite in small quantities, and third stage, post-ore stage, producing banded microquartz, white quartz-comb, calcite and parti-colored fluorite.

The uranium ore-forming period in Xiaozhuang ore field falls into 86-60 Ma, while the main intrusion, the uranium-rich two-mica granite was formed at 196 Ma, the latest

product of acidic magmatism, the muscovite granite, at 135 Ma and the diabase dyke at 110-100 Ma. Considering the great 'time interval' between granitic intrusion and ore formation it is more reasonable to suggest that the uranium in hydrothermal solution might probably be derived from consolidated uraninite-rich granite rather than from magmatic differentiation. The diabase dyke might seemingly act as thermal terrain for heating deeply circulating ground water to extract uranium from uranium-rich granite and precipitate it at the upper part due to the destruction of uranyl complexes during sudden decreasing pressure at the ore stage.

3 THE DEVELOPMENT OF XIAOZHUANG URANIUM ORE FIELD

At the beginning of uranium prospecting in China, based on the experience of foreign countries it was considered that granite massifs especially the interior parts of huge granite batholiths is unfavourable for the formation of uranium deposits and the prospecting and exploration programmes for uranium in South China were mainly placed in the area of metasedimentary rocks. Only when we conducted a passing survey into the interior area of the granite massif during the prospecting in metasediments outside of Guidong granite massif and found two promising uranium occurrences in December 1956, did we begin to pay more attention to the granite massifs. One of these two uranium occurrences situated in diabase dykes was later developed into a medium-size deposit. With this important discovery, prospecting and exploration for uranium in the granitic area was commenced. In the early 1958, in silicified fracture zones within granite batholith, a more promising deposit was discovered. Considering that it gave us more confidence in prospecting uranium deposits within or around granite massifs, we named it "Hope deposit", and operating construction was started later in 1960, becoming one of the first uranium producing mines in China. Afterwards, from June 1958 to 1960, a series of deposits which occur in silicified fracture zones in Guidong granite massif and adjacent massifs were found. Over the past 35 years, in Xiaozhuang ore field, uranium reserves of 12000 tons has been ascertained.

Reviewing the exploration process in Xiaozhuang uranium ore field, one can find its development is not plain sailing, there are periods of breakthrough, expanding and also stagnation as well. The development can be roughly divided into five stages. (1) From the late 1950's to the middle 1960's, it was realized that uranium mineralization is controlled by silicified fracture zones, great success was achieved in the investigation. With the location of several significant deposits, the ore field began to take an embryonic form. (2) From the late 1960's to the early 1970's, as the deposits with surface outcrops of ore veins almost disappeared in the ore field more attention was paid to new target area outside the field.

but little achievement was made. (3). From the early 1970's to the middle 1970's, after detailed study of the ore-control features, it was recognized that deposits are controlled by paralleling NWW trending diabase dykes and schistosed silicified fracture zones, exploration in the ore field especially in the northern part was strengthened, resulting in discovery of a new deposit. (4). From the middle 1970's to the late 1970's, because no new recognition on metallogenesis was achieved, came another period of stagnation. (5). Since the late 1970's, the knowledge of intersection control on mineralization has been mastered, many favourable occurrences at the intersection of silicified fracture zones and diabase dykes were found, some of them have been developed into deposits. By using effective geophysical methods, it is easier to determine whether ore body at the intersection exists or not, and its pitch direction. During this stage, in addition to the location of many intersection type deposits, a lot of occurrences and some deposits occurring in exo- and endocontact zones in eastern part of the ore field have been found. At present, the exploration at depth is underway, the expanding of the ore field also shows promising in the future.

URANIUM RESOURCES OF THE UNION OF SOVIET SOCIALIST REPUBLICS

N.P. LAVEROV, V.I. VELICHKIN
USSR Academy of Sciences

V.I. VETROV, V.V. KROTKOV
Ministry of Atomic Power and Industry

A.L. LAPIN, S.S. NAUMOV
Concern Geologorazvedka,
Ministry of Geology of the USSR

M.D. PELMINEV
Ministry of Geology of the USSR

M.V. SHUMILIN
Moscow University

Moscow, Union of Soviet Socialist Republics

Abstract

The paper presents a first overview over the uranium resources of the USSR. The areas where resources occur are referred to as "uranium - ore areas" and "uranium - bearing areas", the difference being the confidence level of the resource estimates and consequently, the resource development. A total of nine uranium - ore areas and six uranium - bearing areas are described as regards their geological environments, deposit types and uranium resources according to their resource category and cost category. In addition the distribution of the total uranium resources recoverable at costs of up to \$ 120/kg U, by deposit type is described.

1. INTRODUCTION

The history of the Soviet uranium geology is closely associated with the name of one of its prominent earth scientists of the 20th century, V. I. Vernadsky, who, already in 1919 suggested the necessity to study the properties and resources of radioactive minerals in Russia. During the period 1914 - 1940 a number of scientific expeditions was organized on Vernadsky's initiative. As a result of these expeditions the first uranium deposits were discovered and investigated in Central Asia.

After the Second World War, an extensive systematic search for uranium deposits in the USSR as well as over the world was initiated. These activities can be separated into three historical periods. The first one, from the beginning of uranium exploration to approximately 1950, included radiometric surveys

of known mining areas. During the second period, through 1960, mainly airborne radiometric methods were employed, while in the sixties and seventies, referred to as the third period, the assessment of previously collected data and applying this data, the estimation of undiscovered resources was made which lead to the search for blind ore bodies.

It has to be emphasized that the Soviet school of uranium geologists developed isolated from the experiences in uranium geology gained in other parts of the world. In addition, the first uranium deposits discovered in the USSR happened to be very different from those found in other countries. All this explains the peculiarity in the development of the geological concepts and methodologies, which, however, proved to be sufficiently effective in practice.

2. NOTES ON THE URANIUM PRODUCTION CAPABILITIES IN THE USSR

In parallel with the development of the uranium resource base in the USSR, production capacities were created. At the present stage, the production capability of the Soviet uranium mining industry includes the following large scale facilities:

- the Vostochny (Eastern) Mining and Metallurgical Complex in Zheltye Vody,
- the Prikaspiy Mining and Metallurgical Complex in Shevchenko,
- the Navoiy Mining and Milling Complex in Navoi,
- the Vostochny (Eastern) Rare Metal Complex in Khodzhent,
- the Tselinny Mining and Chemical Complex in Stepnogorsk,
- the "Yuzhpolimetall" Production Association in Bishkek,
- and the Priargunsky Mining and Chemical Complex in Krasnokamensk.

All production facilities are part of the "Minatomenergoprom" Ministry of the USSR, also referred to as Ministry for Atomic Power and Industry (MAPI).

3. URANIUM RESOURCES OF THE USSR

At present, there are nine "uranium-ore areas" and six "uranium-bearing areas" known on the territory of the USSR. The "uranium-ore areas" are defined as areas which contain deposits currently being mined or mined in the past. However, in the majority of these areas there are some deposits which have not yet been developed for mining. In detail, these "uranium-ore areas" include Kirovograd, Krivorozh, Stavropol, Zacaspiysk, Kyzylkum, Karamazar, Pribalkhash, Kokchetavsk, and Streltsovsk. Their location is shown in Fig. 1.

The "uranium-bearing areas" refer to those regions which contain deposits which are not yet mined and are considered as

reserved for future production. The geological knowledge of these areas is at a lower level than those of the "uranium-ore areas" and the resource estimates have a lower degree of confidence. These areas include Zauralsk, Yeniseisk, Vitimsk, Central-Transbaikal, Onezhsk and the Far East. The location of these "uranium-bearing areas" is also shown in Fig. 1.

Large parts of the northern and far eastern portions of the USSR are still insufficiently explored for uranium. The only uranium deposit in the Far Eastern USSR, which is of economic significance, is Lastochka (Fig. 1). There are, however, numerous uranium occurrences in areas between the northern part of the Ural and Chukotka and the Ussuriy region. The geological data collected so far suggest that new uranium districts may be found in this extensive territory and that the total uranium resources of the USSR are not limited to the known resources.

In the following, a description of the "uranium-ore areas" as well as of the "uranium-bearing areas" is given.

3.1 "URANIUM - ORE AREAS"

The areas Krivorozh and Kirovograd are located between the Dnieper and Bug rivers. The city of Zheltye Vody, located in the center of these areas, houses the Vostochny (Eastern) Mining and Milling Complex, which is supported by the resources of these areas. The geographical proximity and their similar geological settings justify their grouping into one district.

The geological environment of Krivorozh (Fig. 2) is determined by a N - S trending geosynclinal folded trough between an Early and Late Archean block. The trough is filled by lower Proterozoic rocks including conglomerates, iron quartzites and schists. The uranium mineralisation concentrated in the two main deposits Pervomaiskoye and Zheltovodskoye, is associated with faulted schists and iron quartzites, which are affected by an intensive alkali metasomatism (albitisation). The mineralisation, 1.8 billion years old, consists mainly of uraninite and brannerite and contains admixtures of P, Zr and Sc.

The Kirovograd area is located in a granite gneiss dome (2.6 billion years) with a granitic core (Fig. 2), consisting of two granitoid complexes of different ages and composition. The northern part of the core includes rapakivi granites and labradorites, 1.8 - 1.7 billion years old, while the southern part is made up of anatectic and intrusive potassium granites about 2 billion years old. The uranium mineralisation is related to metasomatised albitites controlled by fault zones and confined to the gneisses of the peripheral parts of the dome and the southern portion of the younger granitic massif, and forms stockwork type deposits. The main deposits are Michurinskoye, Severinskoye and Vatutinskoye closely related to fault zones E and W of the central part of the dome. Mineralogically the ore

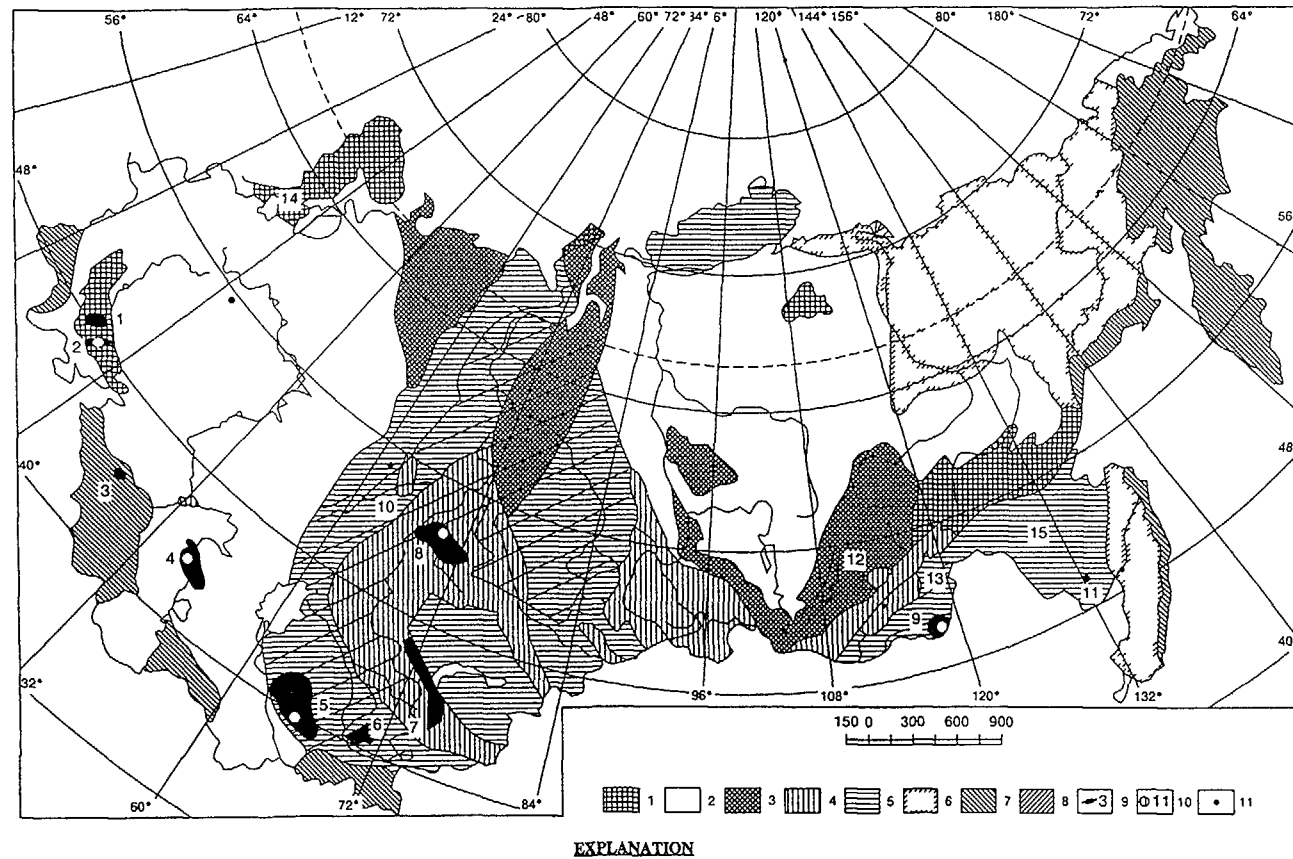


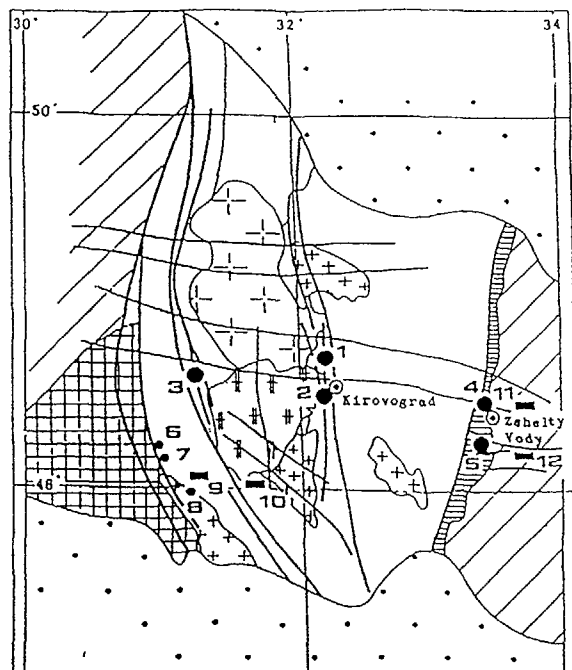
Fig 1. Location of USSR uranium resources

consists of uraninite and brannerite, but contains carbonates which poses certain metallurgical problems, the reason why the resources are placed into the higher cost category.

In addition to the albititic deposit types of the Krivorozh and Kirovograd areas there are some smaller still undeveloped deposits (Kalinovskoye, Lozovatskoye and Yuzhnoye) which are

associated with K metasomatic processes, as well as deposits in the overlying Quaternary sandstones (Devladovo, Bratskoye, Surskoye, Safonovskoye etc), some of which were mined by ISL methods.

The largest deposit in these areas is Severinskoye with about 50 000 tonnes U. The Pervomaiskoye and Zheltovodskoye



EXPLANATION

- 1: Platform complex (K - P)
 2 - 5: metamorphic complexes (AR - PR): 2: protogeosynclinal (Pt), 3: granite-gneiss of early Archean age, 4: gneiss of late Archean age, 5: granulite (AR)
 6 - 8: proto-orogenic granitoids (PR): 6: rapakivi granites, 7: granites, 8: subalkaline granites
 9: fault
 10 - 12: uranium deposits: endogenic types: 10: in soda metasomatites-albitites:
 1: Severinskoye, 2: Michurinskoye, 3: Vatutinskoye,
 4: Zheltovodskoye, 5: Pervomaiskoye
 11: in potassium metasomatites: 6: Kalinovskoye,
 7: Lozovatskoye, 8: Yuzhnoye;
 12: exogenic types: 9: Sadovoye, 10: Bratskoye,
 11: Novogurievskoye, 12: Devladovo

Fig. 2: Geological map of the Krivorozh and Kirovograd areas

deposits are mined out to a considerable degree, while Michurinskoye and Vatutinskoye are being mined.

The total resources of the two areas are listed as follows (Table 1 and 2):

TABLE 1

KNOWN URANIUM RESOURCES¹⁾ OF THE KRIVOROZH AND KIROVOGRAD AREAS (Tonnes U)

	< \$ 80/kg U	\$ 80 - 120/kg U
Krivorozh	28 200	2 600
Kirovograd	44 000	71 500

TABLE 2

ADDITIONAL RESOURCES²⁾ OF THE KRIVOROZH AND KIROVOGRAD AREAS (Tonnes U)

	< \$ 80/kg U	\$ 80 - 120/kg U
Krivorozh	2 200	2 900
Kirovograd	71 500	111 600

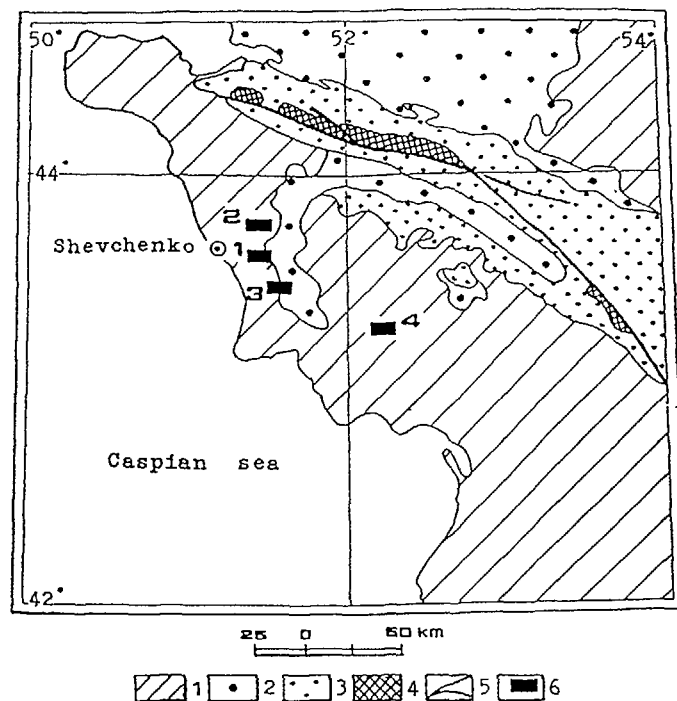
¹⁾ Equivalent to the Soviet resource categories B + C1 + C2, and equivalent to the NEA(OECD)/IAEA resource category RAR + EAR-I

²⁾ Equivalent to the Soviet resource category P1 and equivalent to the NEA(OECD)/IAEA resource category EAR-II.

The **Stavropol** area is located at the northern foothills of the Caucasus in the upper part of the Kuma river.

The area is underlain by a remnant of crystalline basement uplift, which is covered by sediments. In late Miocene this cover was intruded by a number of subvolcanic laccoliths and intrusive bodies of rocks ranging in composition from granite porphyry to quartz syenites. The mineralisation is associated with granite porphyries of this intrusive suite and large bituminous xenoliths of the intruded cover sediments and consists of sulfides and nasturan oxidized to considerable depth.

There are two uranium deposits in this region, Beshtau and Bykogorskoye. Their total resources totalled 5 300 tonnes U of the below \$ 80/kg U cost category, but are depleted.



EXPLANATION

- 1 - 3: Platform complexes (K - Q): 1: Neogene - Quaternary sediments
 2: Paleogene sediments
 3: Cretaceous sediments
- 4: folded complex (T - J)
 5: fault
- 6: uranium deposits of the exogenic type: 1: Melovoye, 2: Tomakskoye,
 3: Tasmurun, 4: Taybagar

Fig. 3: Geological map of the Zacaspiysk area

The **Zacaspiysk** area is located on the eastern shore of the Caspian Sea on the Mangyshlak Peninsula, close to the city of Shevchenko, the headquarter of the Prikaspiy Mining and Metallurgical Complex.

The area is underlain by a folded sedimentary sequence of Permian-Triassic and Jurassic ages (Fig. 3), which is covered by sediments ranging in age from Cretaceous to Neogene.

The uranium is located in the younger sedimentary cover in accumulations of phosphatized bone detritus of fossil fishes in pyrite-bearing clays. The ore is generally low grade, 300 to 500 ppm U, however, a preconcentrate of 2 - 3 times the original grade can be made by separating the fish bones. At the same time, the P content can be increased considerably. In addition to U and P, REE and Sc are recovered from the ore.

The largest deposit in this area is Melovoye with resources of 43 800 tonnes U. Other deposits are Tomakskoye, Tasmurun and Taybagar. The total known resources of this area are estimated at 64 400 tonnes U recoverable at costs of \$ 80 - 120/kg U.

The area of **Kyzylkum** is located between the Amu Darya and Syr Darya rivers in the Kyzyl Kum desert, which is traversed by the extreme Northwestern foothills of the Tien Shan - Pamir mountain ranges. In the southern part of the area is the town Navoi with the Navoiy Mining and Milling Complex where the uranium ores of the area are being treated.

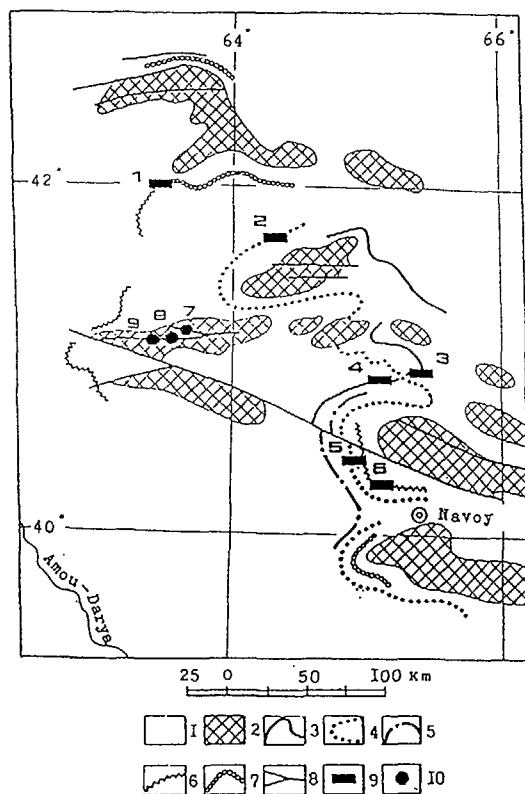
The area is tectonically a basin and range system (Fig. 4), the basins being filled with sediments (conglomerates, sandstones, mudstones) ranging from Cretaceous to Neogene, while the ranges or uplifts consist of pre-Mesozoic basement.

Two types of uranium deposits have been found in this area; an older one in Silurian-Ordovician black shales, referred to as polygenic and a younger one in the clastic sediments of the basins, referred to as exogenic (Fig. 4).

In the black shale deposits, the ore occurs either in stratiform bodies (Rudhoe) or as complex stockworks (Koscheka, Dzhanthuar) associated with V. The mineralisation was formed in the Paleogene and thus was affected by the Alpine orogeny. The U grade is between 0.1 - 0.2 per cent, the V content can reach 1 - 3 per cent.

The sandstone type deposits are of the rollfront type associated with the oxidation-reduction interfaces. The mineralisation contains also Se as well as rare elements. The uranium grade varies from the first hundredths to tenth of a per cent U. The rollfronts occur in a number of stratigraphically different horizons ranging in age from middle Cretaceous to Tertiary and can be traced for hundreds of kilometers. The deposits of this type are Uchkuduk, Sugrally, Ljavljakan, Beshkak, Bukinay and Kanimeh. Mining is done using conventional methods (Uchkuduk, Sugrally) and underground ISL (Bukinay, Beshkak, etc.).

The uranium resources in the black schists deposits are relatively small and therefore assigned to a higher cost category, while the resources in the sandstone type deposits are larger (e.g. Uchkuduk with 50 000 tonnes U and Sugrally with about 40 000 tonnes U) and belong to the lower cost category. A summary of the resources in the Kyzylkum area is given in Table 3.



EXPLANATION

- 1: Platform complex (K - Q)
- 2: folded complex (PZ)
- 3 - 7: oxidation - reduction fronts in various stratigraphic horizons of the platform complex: 3: in Paleogene sediments, 4: in Coniacian-Santonian sediments, 5: in Campanian - Maastrichtian sediments, 6: in upper Turonian sediments, 7: in lower Turonian sediments
- 8: fault
- 9: uranium deposits of the exogenic type: 1: Uchkuduk, 2: Sugraly, 3: Ljavljakan, 4: Beshkak, 5: Bukinay, 6: Kanimch;
- 10: uranium deposits of the endogenic type: 7: Rudnoye, 8: Koscheka, 9: Dzhanuar

Fig. 4: Geological map of the Kyzylkum area

TABLE 3
TOTAL URANIUM RESOURCES OF THE KYZYLKUM AREA
(Tonnes U)

	< \$ 80/kg U	\$ 80 - 120/kg U
Known Resources	155 600	9 700
Additional Resources	106 000	8 300
Prognosticated Resources ¹⁾	100 000	60 000

¹⁾ Equivalent to the Soviet resource category P2 and equivalent to the NEA(OECD)/IAEA resource category Speculative Resources.

The Karamazar area is located in the northwestern part of the Tien Shan mountains and covers both the ridges of the Chatkal and Kuramin ranges and the adjoining part of the Fergana valley. The town of Khodzhenst with the Vostochny (Eastern) Rare Metal Complex, which processes the uranium ore extracted in the area.

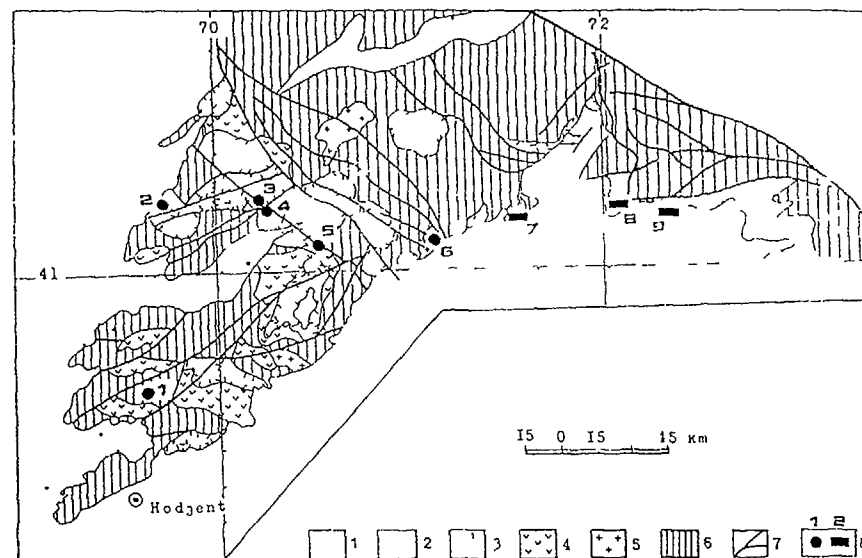
The geology of the Karamazar area (Fig. 5) consists of the Chatkal - Kuramin uplift made up of pre-Paleozoic crystalline rocks and of early to middle Paleozoic continental carbonates, which are overlain by Carboniferous - Permian acid volcanics or cut by their subvolcanic equivalents, and the Fergana depression filled with Mesozoic - Paleogene marine sediments and coarse grained clastics of Neogene - Quaternary age.

The uranium deposits of the Karamazar area belong to two types; to the hydrothermal vein-stockwork type related to the volcanic structures, and to the stratiform type in the Paleogene carbonates which contain hydrocarbons as reductants.

The hydrothermal deposits are of Permian age and controlled by circular and layered features of the acid volcanics, as well as by the unconformity with the underlying folded basement rocks. This deposit type includes the deposits Alatanga, Kattasay, Chauli, Maylikatan, Charkasar and Taboshar, one of the very early Soviet uranium discoveries. Mineralogically these deposits are not uniform, as their ores have different mineral associations, such as U - Mo (Alatanga, Chauli), U-Cu-Pb-Zn (Maylikatan, Taboshar) and U-Bi (Adrasman). The ore grade in this deposit type is in the range of tenths to hundredth of a per cent.

The stratiform deposits located in the Fergana depression include Maylisu, Shakaptar and Maylisay. They are hosted in water-and-hydrocarbons bearing carbonates and are of Miocene age. These deposits are relatively low grade.

The total resources of this area were 20 000 tonnes U recoverable at costs of less than \$ 80/kg U. These are, however,



EXPLANATION

- 1 - 2 Platform complex (K - Q)
 1 continental sediments
 2 continental carbonaceous sediments (K - P₂)
- 3 - 4 orogenic complex (C₃ - P₁)
 3 late orogenic intermediate to acid volcanics
 4 early orogenic basic to intermediate volcanics
- 5 - 6 Pre-orogenic complex (PZ₁ - PZ₂)
 5 granites
 6 folded sequences
- 7 fault
- 8 uranium deposits: endogenic types: 1 Taboshar,
 2 Chauli, 3 Alatanga, 4 Kattasay,
 5 Maylikatan, 6 Charkasar
 exogenic types: 7 Maylisu, 8 Shakaptar
 9 Malisay

Fig 5. Geological map of the Karamazar area

mined out. The largest deposits were Alatanga and Chauli with total resources of 4 500 tonnes U each.

The **Pribalkhash** uranium-ore area is located west of the Lake Balkhash and north and northwest of the town Bishkek, in whose vicinity the Production Association "Yuzhpolimetal" processed the uranium ores mined in this area.

Geologically, the area is related to the Chuili - Kendyktas uplift made up of gneisses and schists of the pre-Paleozoic basement and younger continental basic to acid volcanics of late Silurian to Devonian age (Fig. 6). This volcanic sequence is affected by volcano-tectonic structures controlled by deep seated faults and separated by basement domes which in turn, are cut by subvolcanic intrusions of acid composition.

These acid (liparitic) volcanic complexes are the host for the uranium vein-stockwork type mineralisation. The deposits (Botaburum, Kyzylsay, Kurday, and Dzhydell) are very similar as regards their geological settings. The Kurday deposit, however, is hosted in a granite, which is structurally associated with a volcanic neck. The remaining deposits are directly located in volcanic rocks.

The location of the ore bodies is controlled by a combination of tectonic features and the contacts of volcanic lithofacies. The mineralogical association includes uranium and molybdenum minerals. The ore grade varies between 0.1 and 0.3 per cent U. In the Dzhydell deposit some high grade blocks with plus 10 per cent U were found.

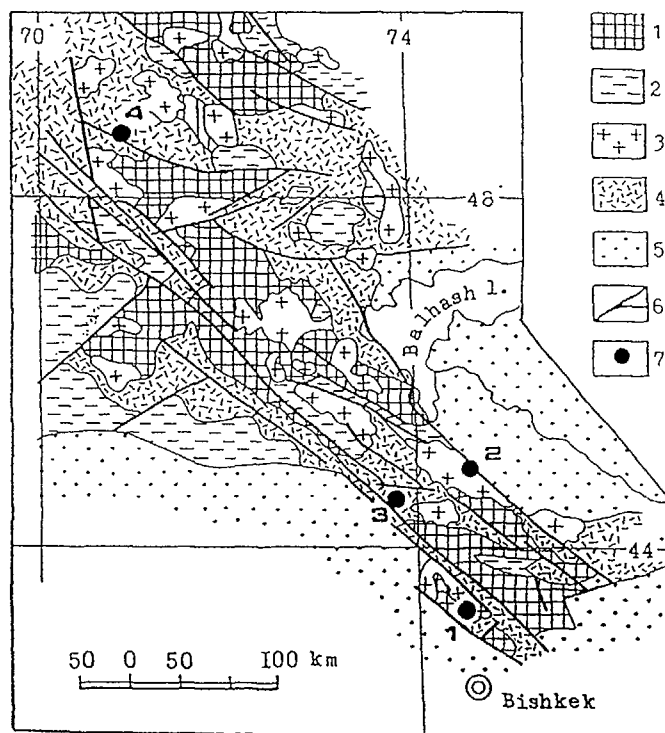
The largest deposit in this area is Botaburum with about 10 000 tonnes U. The total resources of the area are summarized in Table 4.

TABLE 4

TOTAL URANIUM RESOURCES OF THE PRIBALKHASH AREA
(Tonnes U)

	< \$ 80/kg U	\$ 80 -120/kg U
Known Resources	17 900	4 000
Additional Resources	50 000	---
Prognosticated Resources	50 000	---

The **Kokchetavsk** area is located at the southern edge of the West-Siberian plain between the rivers Irtysh and Ishim. The town of Kokchetavsk lies in the center of the area and Stepnogorsk in its eastern part. This town is the location of the Tselinny Mining and Chemical Complex, which processes the uranium ores produced in the area.



EXPLANATION

- 1: Platform complex (MZ - Q)
- 2 - 3: orogenic complex (PZ₁-PZ₂)
 - 2: acid volcanics
 - 3: granites
- 4 - 5: pre-orogenic complexes
 - 4: folded continental volcanogenic rocks (PZ₁ - PZ₂)
 - 5: metamorphics (PR ?)
- 6: fault
- 7: uranium deposits of the endogenic type: 1: Kurday
2: Botaburum, 3: Kyzylsay, 4: Dzhideli.

Fig. 6: Geological map of the Pribalkhash area

Geologically, the area is associated with the Kokchetavsk massif of Caledonian age (Fig. 7). Its basement consists of folded Precambrian crystalline gneisses and slightly metamorphosed continental volcanics of Cambrian - Ordovician age, which are cut by multiple Silurian - Devonian intrusions of granitic composition. Some depressions in the folded basement are filled with Silurian - Devonian volcanogenic sediments and post-Devonian (Carboniferous - Jurassic) platform cover sediments. Of importance for the genesis of the uranium deposits are deep seated faults, which cross the folded basement of the Kokchetavsk massif.

The uranium deposits in this area belong to two types; hydrothermal vein-stockworks of Silurian - Devonian age, and stratiform sandstone type deposits in the paleo-valleys of the pre-upper Jurassic peneplain buried under younger sedimentary cover.

The hydrothermal type deposits also referred to as endogenic (Fig. 7) are mainly restricted to the peripheral parts of the granite gneiss core of the Kokchetav massif. They include the deposits Ishimskoye, Vostok, Balkashinskoye, Grachevskoye, Zaozernoye, Tastykolskoye and Manybay.

The stratiform deposit, referred to as exogenic in Fig. 7 is Semizbay, is located in the eastern part of the area.

The largest deposits are Vostok, Manybay, Grachevskoye, Zaozernoye, and Semizbay each with about 20 000 tonnes U, while Balkashinskoye and Tastykolskoye have resources of 1 000 to 3 000 tonnes U each. The total resources of the Kokchetavsk area are shown in Table 5.

TABLE 5

TOTAL URANIUM RESOURCES OF THE KOKCHETAVSK AREA
(Tonnes U)

	< \$ 80/kg U	\$ 80 - 120/kg U
Known Resources	72 400	26 800
Additional Resources	9 200	30 400
Prognosticated Resources	---	70 000

The uranium-ore area **Streltsovsk** lies in the Eastern Transbaikalian region on the left bank of the middle part of the Argun river. The town Krasnokamensk is approximately 15 km northwest of the mining district. This town houses the Priargunsky Mining and Chemical Complex which processes the ores mined in the area.

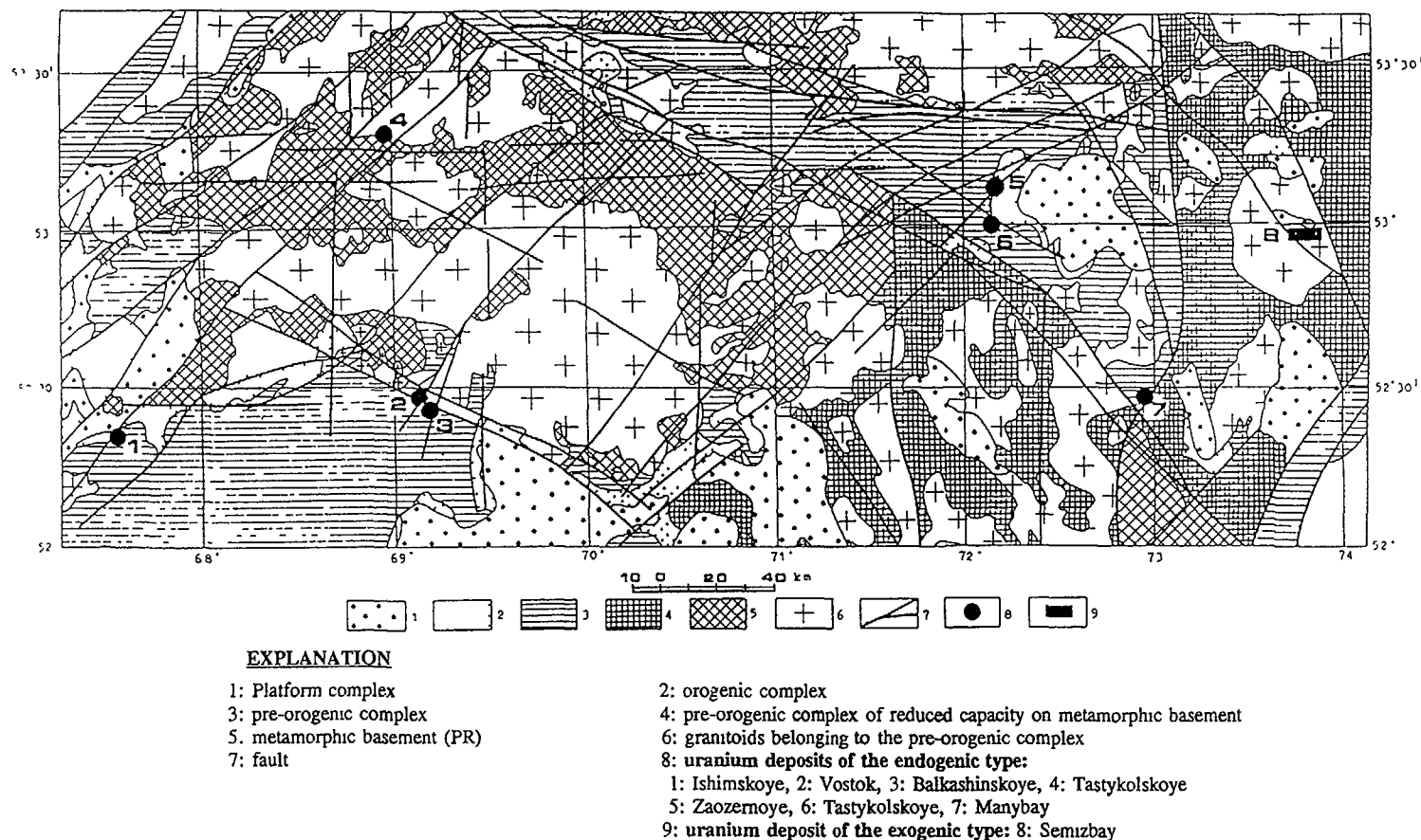


Fig. 7: Geological map of the Kokchetavsk area

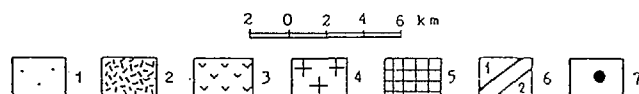
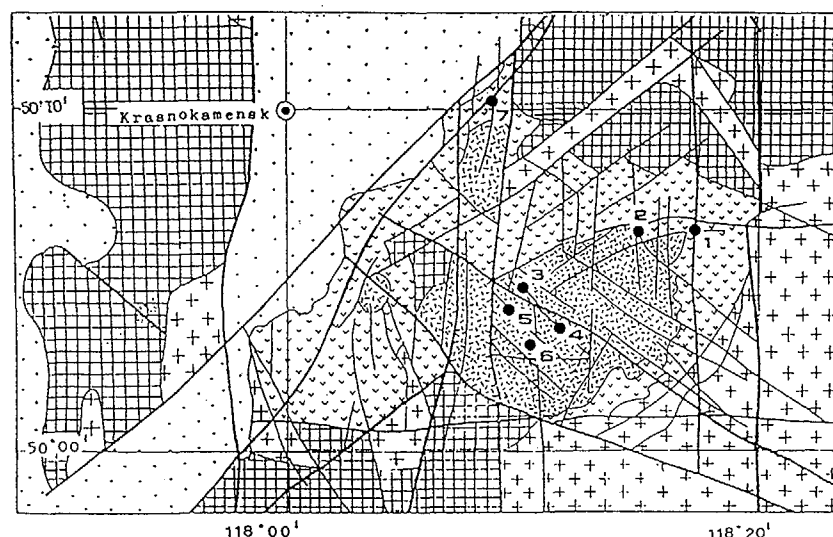
Covering a Paleozoic basement of the Uralo-Mongolian fold belt are upper Jurassic volcanics and late Cretaceous graben structures filled with coal bearing sediments (Fig. 8). The volcanic areas include a large 20 km diameter caldera filled with volcanics and derived sediments, including andesites, dacites, basalts as well as sediments and tuffs. This complex is transversed by numerous fault systems providing a network of channels.

The mineralisation is controlled both by the tectonic features and the porosity and permeability of the affected rocks, including a wide variety ranging from the basement granite to

sandstones and tuffs. The consequences of these conditions are a variety of ore bodies including veins, stockworks and stratiforms.

Two mineralogical ore types are known; a uranium ore and a U-Mo ore type. The U grade varies and depends on the host rock. In general, the ore runs 0.2 per cent U, it can reach 0.6 per cent in large stockworks and several per cent in veins.

The largest deposits in the area are Streltsovskoye, which is predominantly a vein deposit with total resources of plus 60 000 tonnes U, and Tulukuevskoye, a large stockwork type with



EXPLANATION

- 1: Platform complex (K₁)
- 2 - 3: orogenic complex
 - 2: younger subcomplex (felsites, ignimbrites, liparites, tuffs, sandstones)
 - 3: older subcomplex (sandstones, ignimbrites and dacitic tuffs, andesites, basalts, conglomerates)
- 4 - 5: pre-orogenic complexes
 - 4: granites (PZ)
 - 5: metamorphics (PR)
- 6: fault
- 7: uranium deposits: 1: Shirondukuevskoye, 2: Streltsovskoye, 3: Tulukuevskoye, 4: Yubileynoye, 5: Vesennee, 6: Novogodnee, 7: Dalnee

Fig. 8: Geological map of the Streltsovsk area

total resources of plus 35 000 tonnes U. Smaller deposits are those with stratiform mineralisation, including Yubileynoye, Novogodnee, and Dalnee, each with resources in the 4 000 to 10 000 tonnes U range.

The total resources of the Streltsovsk uranium-ore area are compiled in Table 6.

TABLE 6
TOTAL URANIUM RESOURCES OF THE STRELTISOVSK AREA
(Tonnes U)

	< \$ 80/kg U	\$ 80 - 120/kg U
Known Resources	119 200	4 800
Additional Resources	84 800	2 200
Prognosticated Resources	50 000	20 000

3.2 "URANIUM - BEARING AREAS"

The Zauralsk area is located some 80 km east of the town Cheljabinsk.

Geologically, it lies on the western edge of the Siberian platform consisting of a basement affected by Hercynian folding. The basement is covered by sedimentary sequences ranging in age from Jurassic to Paleogene-Neogene (Fig. 9).

In this sedimentary environment there are two types of sandstone uranium deposits; in upper Jurassic and in Recent sediments, the older ones being the more important deposits (Dolmatovskoye, Dobrovolskoye).

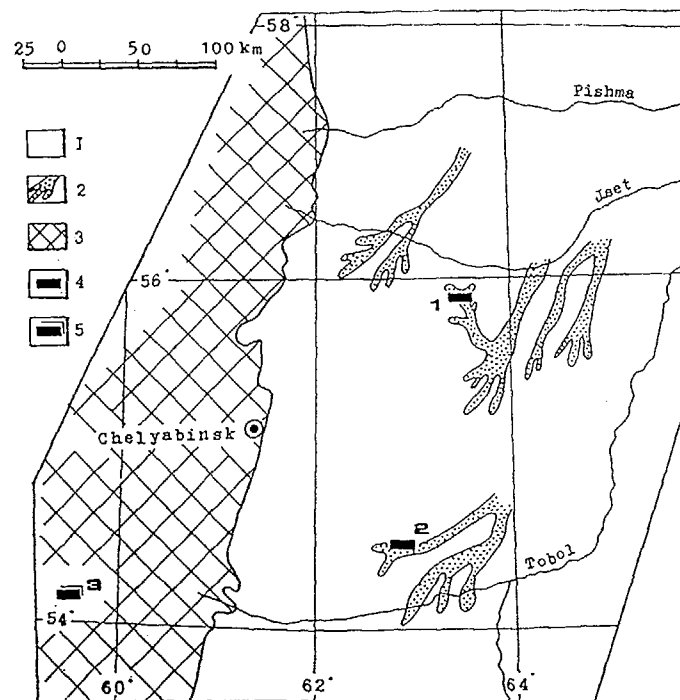
The upper Jurassic deposits are located in paleo-valleys incised in the basement with Paleozoic acid volcanics as the source for the uranium. The boundaries of the region are mainly economically determined by the thickness of the post-Jurassic cover, which increases in northern, eastern and southern direction. The deposit in recent sediments (Sanarskoye) is associated with carbonaceous sandy valley fills. This may justify the reclassification of this deposit as surficial deposit.

The upper Jurassic deposits are low grade, but amenable to underground ISL. In addition to U, the ore contains traces of Re, Sc and REE, partially also recovered. The Sanarskoye deposit the only one in recent sediments, is mined out.

The uranium resources of the area are compiled in Table 7.

TABLE 7
TOTAL URANIUM RESOURCES OF THE ZAURALSK AREA
(Tonnes U)

	< \$ 80/kg U	\$ 80 - 120/kg U
Known Resources	4 400	12 000
Additional Resources	100	2 500
Prognosticated Resources	---	15 000



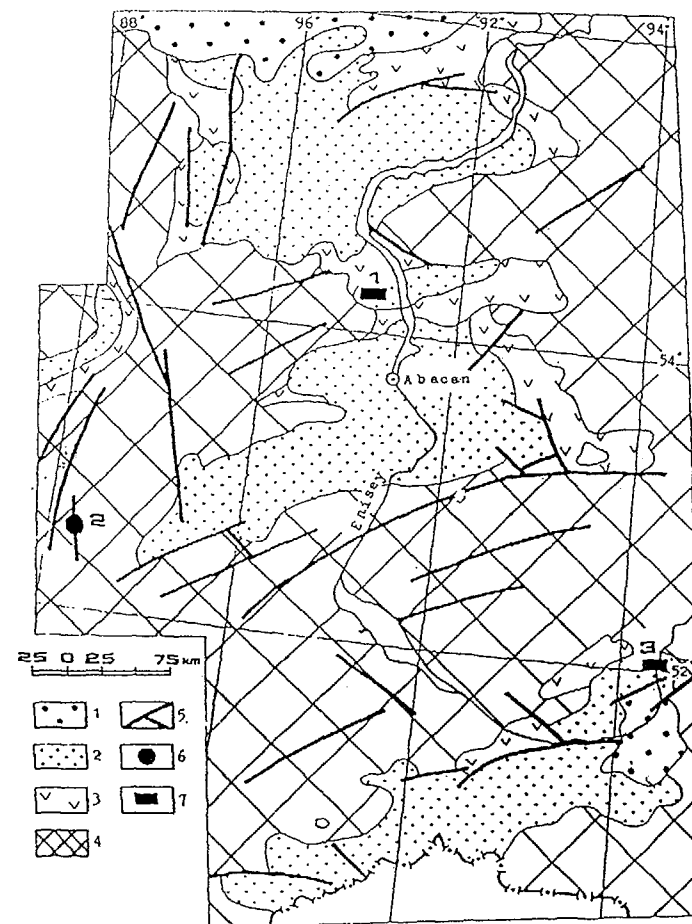
EXPLANATION

- 1: Platform complex (J - Q)
- 2: basal sediments of the platform complex including paleovalleys on the buried surface of the folded complex
- 3: folded complex (PZ) outcropping
- 4 - 5: uranium deposits of the exogenic type:
 - 4: in Jurassic sediments: 1: Dolmatovskoye, 2: Dobrovolskoye
 - 5: in recent sediments: 3: Sanarskoye

Fig. 9: Geological map of the Zauralsk area

The **Yeniseisk** area is located in the Yenisei river basin. The city Abakan lies approximately in the center of the area.

The geology of the area (Fig. 10) consists of anticlinal uplifts and synclinal depressions. The cores of the uplifts are made up of Precambrian granites and metamorphic complexes and the depressions are filled with oxidized sandy siltstones of upper Devonian-Carboniferous age, which overly lower Devonian volcanics of acid and basic composition.



EXPLANATION

- 1: Platform complex (J)
- 2 - 3: orogenic complex (D - C)
 - 2: late orogenic continental sediments
 - 3: early orogenic volcanogenic sediments
- 4: pre-orogenic complexes
- 5: fault
- 6: uranium deposit of the endogenic type: 2: Labyshskoye
- 7: uranium deposits of the exogenic type: 1: Primorskoye
3: Ust-Kuduk

Fig. 10: Geological map of the Yeniseisk area

TABLE 8
TOTAL URANIUM RESOURCES OF THE YENISEISK AREA
(Tonnes U)

	\$ 80 - 120/kg U	> \$ 120/kg U
Known Resources	7 600	7 200
Additional Resources	2 400	6 000
Prognosticated Resources	20 000	---

The known uranium deposits are vein type deposits in Devonian volcanics (Labyshskoye) and sandstone type deposits hosted in the upper Devonian sediments (Primorskoye, Ust-Uyuk).

The sandstone type desposits contain the majority of the resources of the area. In the Primorskoye deposit, the ore bodies are thin (0.3 - 0.5 m) but relatively high grade (0.2 - 0.3 per cent U), while the mineralisation in Ust-Uyuk is of lower grade. As regards the resource and cost categories, the resources of Primorskoye, totalling 7 600 tonnes U are classified as known resources of the \$ 80 - 120/kg U cost category, while the resources of the remaining two deposits are in the + \$ 120/kg U cost category. A summary of the resources is given in Table 8.

The **Vitimsk** uranium bearing area is located in the northern Transbaikalian district and covers a portion of the plateau between the rivers Vitim and Amalat. The town Chita lies approximately 200 km south of the Vitimsk area.

The area is covered by Quaternary basalts (Fig. 11 and 12) which overly a Proterozoic granitic-metamorphic basement. The mineralisation is associated with paleo-valleys below the basaltic cover (generally 10 - 30 m thick, but occasionally up to 200 m thick), which are filled with sediments containing abundant organic matter. The deposits are located in aquifers, but below the permafrost, which makes it possible to recover the uranium by underground ISL methods.

The resources of individual deposits are small, hundreds to a few thousand tonnes U, but the density of paleo-valleys and the large number of ore bodies make this area an economically viable district. Due to its remote location, however, the cost category of the resources is estimated to be between \$ 80 - 120/kg U (Table 9).

In the same general area as Vitimsk, but south of the town of Chita, lies the **Central-Transbaikalian** uranium bearing area, which does not have a uranium ore processing facility.

TABLE 9
TOTAL URANIUM RESOURCES OF THE VITIMSK AREA
(Tonnes U)

	\$ 80 - 120/kg U
Known Resources	23 700
Additional Resources	3 800
Prognosticated Resources	40 000

In this area there are known three diferent uranium deposits types (Fig. 12). These include vein-stockwork deposit, also referred to as Streltsovsk type, in upper Jurassic - lower Cretaceous volcanic complexes, vein deposits in highly radioactive Mesozoic granites, as well as stratiform sandstone type deposits in lower Cretaceous carbonaceous sediments. The deposits belonging to these three types include Olovskoye (vein-stockwork), Gornoye and Berezovoye (vein deposit in granites) and Stepnoye, a stratiform sandstone deposit.

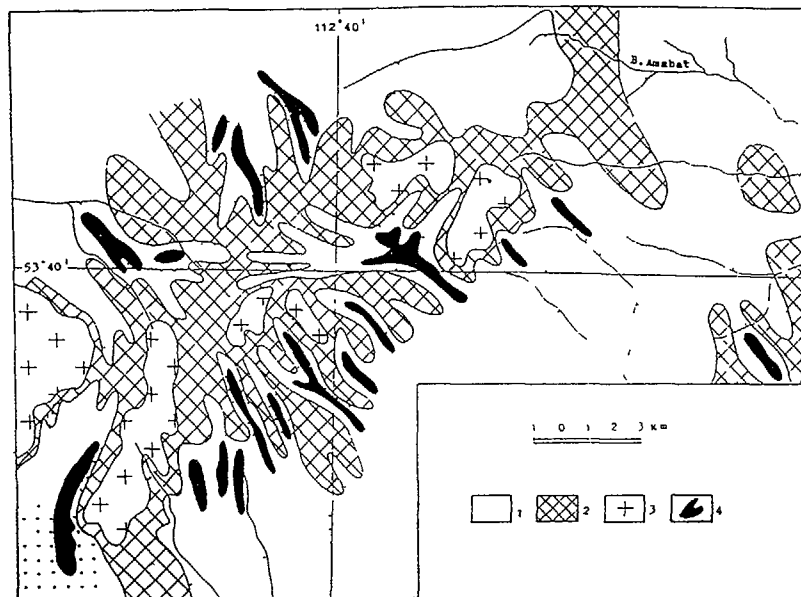
The ore grade in all deposits of the area is low. The largest deposit is Olovskoye with 15 000 tonnes U contained. All resources belong to the cost category \$ 80 - 120/kg U (Table 10).

The **Onezhsk** area covers the northern shore of the Lake Onega, northwest of St. Petersburg, and lies in the morphological Onega depression in an Archean - lower Proterozoic granite-gneiss basement (Fig. 13).

This depression is filled with Proterozoic sediments of the Karelian complex, including conglomerates, schists, partly as black schists, and dolomites cut by numerous diabase dykes and sills. This complex is subjected to severe folding and faulting.

TABLE 10
TOTAL URANIUM RESOURCES OF THE CENTRAL-TRANSBAIKALIAN AREA
(Tonnes U)

	\$ 80 - 120/kg U
Known Resources	20 700
Additional Resources	8 300
Prognosticated Resources	5 000



EXPLANATION

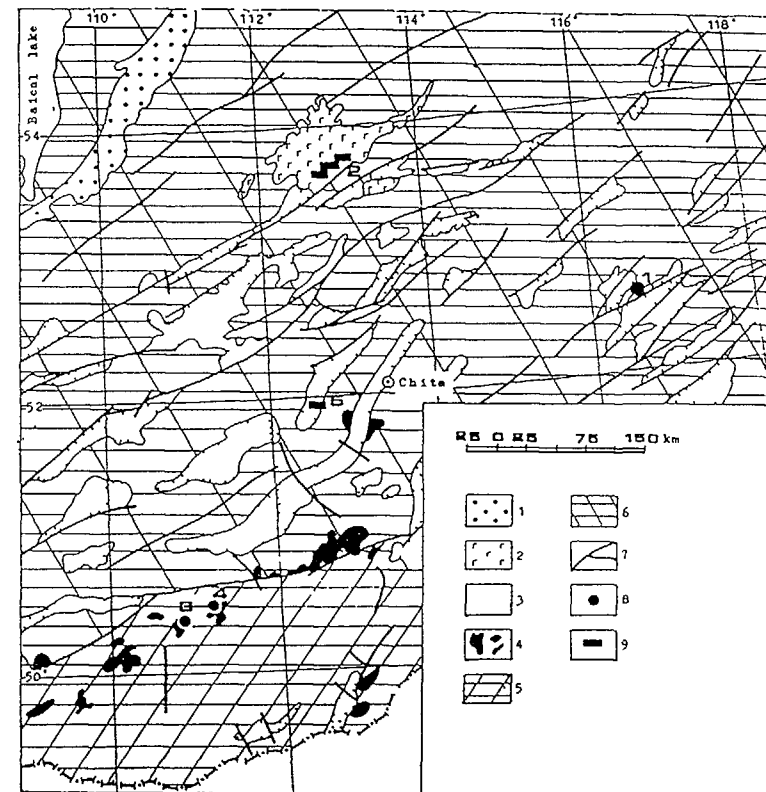
- 1: Sand and other surficial sediments (Oligocene - Miocene)
- 2 - 3: granitic and metamorphic rocks of the peneplained basement (PR)
 - 2: buried under basalt cover
 - 3: outcropping
- 4: uranium mineralisation

Fig. 11: Geological environment of the uranium mineralisation of the Vitimsk area

One uranium deposits, Padma, a stockwork type with small vertical extensions, is associated with this folded and faulted complex. The mineralization includes a series of elements including U, V, Bi, Se, Cu, Mo, Au and PGM. The uranium content is between 0.1 - 0.25 per cent, but can reach several per cent.

The resources in this area are limited (Table 11), but the potential of the area is still not completely assessed.

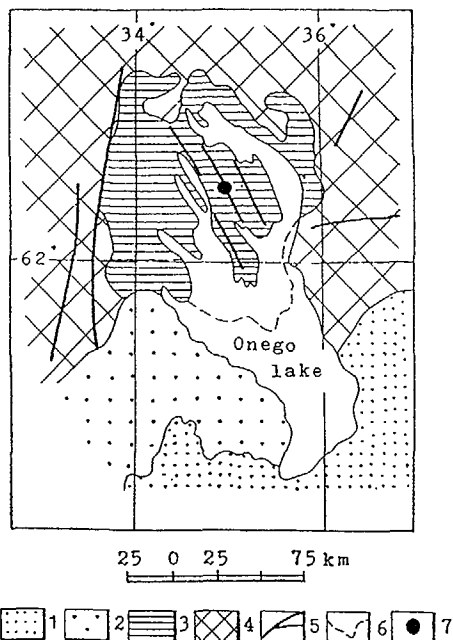
The Far Eastern area covers the Ussuri and Amur basins, the Lena basin, the Okhotskoye sea coast, as well as the basins of the Kolyma and Chukotka peninsulas.



EXPLANATION

- 1 - 2: Rift complex (Q)
 - 1: continental sediments
 - 2: basaltic rocks
- 3: platform complex. Carboniferous continental sediments locally overthrusts over volcanics of orogenic complex
- 4: orogenic complex
- 5 - 6: pre-orogenic complexes
 - 5: of Hercynian age
 - 6: of Caledonian age
- 7: fault
- 8: uranium deposits of the endogenic type: 1: Olovskoye, 3: Gornoye, 4: Berezovoye
- 9: uranium deposits of the exogenic type: 2: Vitimsk area, 5: Stepnoye

Fig. 12: Geological map of the Vitimsk and Central-Transbaikalian areas



EXPLANATION

- 1 - 2: Platform complex
 - 1: Paleozoic age
 - 2: Riphean (Eocambrian) age
- 3: protoplatform complex (shales, dolomites, diabase (PR₁))
- 4: metamorphic complex (AR - PR₁)
- 5: fault
- 6: boundary of protoplatform complex on lake bottom
- 7: uranium deposit: Padma

Fig. 13: Geological map of the Onegzhsk area

So far, this vast area has not been systematic covered by uranium prospection. The only deposit known in this area is the Lastochka vein-stockwork deposit, associated with a volcanic-tectonic depression of lower Cretaceous age on Paleozoic granites. This deposit has known resources of 3 900 tonnes U at an ore grade of between 0.1 - 0.2 per cent. In addition, the

TABLE 11
TOTAL URANIUM RESOURCES OF THE ONEZHISK AREA
(Tonnes U)

	\$ 80 - 120/kg U
Known Resources	2 000
Additional Resources	3 000

TABLE 12
TOTAL URANIUM RESOURCES OF THE FAR EASTERN AREA
(Tonnes U)

	\$ 80 - 120/kg U	> \$ 120/kg U
Known Resources	3 900	---
Additional Resources	---	---
Prognosticated Resources	---	300 000

Prognosticated Resources of the entire area is estimated to total 300 000 tonnes U of the plus \$ 120/kg U cost category (Table 12).

3.3. SUMMARY OF RESOURCES

After the uranium - ore areas and the uranium - bearing areas of the USSR and their resource base have been reviewed individually, an overview (Table 13, A, B, C) including the total of the three resource and cost categories is presented.

According to this summary, the total known resources of the three cost categories amount to 692.8 million tonnes U, the additional resources to 507.2 million tonnes and the prognosticated resources to 800.0 million tonnes U. The low cost portion of the resources, i.e. the one which is recoverable at costs of < \$ 80/kg U totals 465.0 million tonnes U known resources, 325.8 million tonnes U additional resources and 200.0 million tonnes U prognosticated resources.

4. THE SOVIET URANIUM RESOURCES BY DEPOSIT TYPES

With the quartz pebble conglomerate and unconformity type uranium deposits not yet discovered in the USSR, the resources are distributed somewhat differently as regards the deposit types from the other resource countries in the world.

TABLE 13
URANIUM RESOURCES OF THE USSR

A) KNOWN RESOURCES
(Thousand Tonnes U)

AREA	< \$ 80/kg U	\$ 80-120/kg U	> \$ 120/kg U
Kirovograd	44.0	38.4	---
Krivorozh	28.2	2.6	---
Stavropol	3.3	---	---
Zacaspysk	---	64.4	---
Kyzylkum	155.6	9.7	---
Karamazar	20.0	---	---
Pribalkhash	17.9	4.0	---
Kokchetavsk	72.4	26.8	---
Streltsovsk	119.2	4.8	---
Zauralsk	4.4	12.0	---
Yeniseisk	---	7.8	7.2
Vitmsk	---	23.7	---
Ctrl Transbaik.	---	20.7	---
Onezhsk	---	2.0	---
Far East	---	3.9	---
TOTAL	465.0	220.6	7.2

B) ADDITIONAL RESOURCES
(Thousand Tonnes U)

AREA	< \$ 80/kg U	\$ 80 - 120/kg U	> \$ 120/kg U
Kirovograd	71.5	111.6	---
Krivorozh	2.2	2.9	---
Stavropol	2.0	---	---
Kyzylkum	106.0	8.3	---
Pribalkhash	50.0	---	---
Kokchetavsk	9.2	30.4	---
Streltsovsk	84.8	2.2	---
Zauralsk	0.1	2.5	---
Yeniseisk	---	2.4	6.0
Vitmsk	---	3.8	---
Ctrl Transbaik	---	8.3	---
Onezhsk	---	3.0	---
TOTAL	325.8	175.4	6.0

C) PROGNOSTICATED RESOURCES
(Thousand Tonnes U)

AREA	< \$ 80/kg U	\$ 80 - 120/kg U	> \$ 120/kg U
Kirovograd	---	70.0	---
Kyzylkum	100.0	60.0	---
Pribalkhash	50.0	---	---
Kokchetavsk	---	70.0	---
Streltsovsk	50.0	20.0	---
Zauralsk	---	15.0	---
Yeniseisk	---	20.0	---
Vitmsk	---	40.0	---
Ctrl Transbaik	---	5.0	---
Far East	---	---	300.0
TOTAL	200.0	300.0	300.0

The uranium resource base in the USSR is associated with three main types of deposits; sandstone deposits, albititic stockwork deposits as well as volcanic vein - stockwork deposits. In addition, as reviewed in the above chapters, there are resources in other deposit types. In the following table, the distribution of the resources recoverable at costs of up to \$ 120/kg U (1.686 million tonnes U), by deposit types is compiled and shown in Table 14.

TABLE 14
DISTRIBUTION OF RESOURCES
BY DEPOSIT TYPE

DEPOSIT TYPE	KNOWN + ADDITIONAL RESOURCES RECOVERABLE AT COSTS UP TO \$ 120/KG U		PROGNOSTICATED RESOURCES RECOVERABLE AT COSTS UP TO \$ 120/KG U	
	Thousand t U	Per Cent	Thousand t U	Per Cent
1 vein - stockwork				
- in volcanics	273.6	23	70.0	14
- in folded complexes	152.8	13	75.0	15
2 albititic stockwork	290.9	25	70.0	14
3 sandstone type				
stratif, Cenozoic	315.1	26	210.0	42
stratif, pre-Cenoz	10.0	1	20.0	4
rollfront	57.0	5	55.0	11
4 bone detritus	64.4	5	---	---
5 black shales	23.0	2	---	---
TOTAL	1 186.8	100	500.0	100

5. CONCLUDING REMARKS

Based on the profound knowledge of the geological settings of the known uranium deposits in the USSR and the geological similarities of large areas still unexplored for uranium, the chances of finding additional deposits in the USSR are to be considered very good. Therefore, the prognosticated resources given in this report are very conservative.

COMPLEX OF GEOPHYSICAL METHODS FOR RECONNAISSANCE OF URANIUM DEPOSITS AND RADIOLOGICAL INVESTIGATION

I A LUCHIN

Ministry of Geology of the USSR,
Leningrad, Union of Soviet Socialist Republics

Abstract

An overview over the application of complex i.e. integrated geophysical methods in the USSR is provided. They are being used for exploration resource assessment as well as for the detection and monitoring of technogenic fall out of radionuclides. In detail these complex geophysical methods include aerogeophysical survey methods surface gamma/gamma spectrometer methods hydrolithochemical methods as well as down hole and ground gamma methods combined with instant fission electron detection and other nuclear methods.

1. Introduction

The unique properties of uranium ores among them its natural radioactivity are the precondition for the utilization of geophysical methods, radiometric methods being the most important among them. The discovery of practically all uranium deposits is associated with the detection of anomalous levels of radioactivity of geological formations on the surface, in mines and open pits, in drill holes, trenches, underground water sources, run offs of small rivers and/or stream sediments.

Also the search, mapping and quantitative estimation of uranium ore deposits, including calculations of the tonnage, are carried out using radiometric methods in all cases.

The integration of geophysical and geochemical methods which are used in the USSR for the survey and reconnaissance of uranium ore deposits includes the following types of survey.

Complex aerogeophysical survey Gamma spectrometry is the most important component of the survey, and in most cases is used together with magnetic and electrometric methods.

Aerogamma spectrometry is performed at different scales, ranging from 1:100 000 and 1:200 000 for regional survey, to more detailed survey in the scale up to 1:50 000 and 1:25 000. Practically all of the USSR territory accessible for this method, is covered presently by regional scale surveys.

In addition to the survey of radioactive anomalies which are the object for further detailed direct studies leading to the recognition of uranium ores, these methods are used more and more intensively for the determination of correlations between natural radionuclides for the purpose of selection of the so-called "specialized" rock complexes not only for uranium prospecting, but for the prospecting of gold, non-ferrous metals and other mineral resources.

The correlation of these data with magnetic data is used for more detailed mapping of rocks and structural unconformities. In addition, electrometric measurements allow to find electroconducting rock complexes. This method is realized in the USSR by means of instrumentation sets (CKAT-77, CTK, etc.), developed in "Rudgeophysica". Presently aerogamma spectrometry is widely used for radioecological investigations, mostly for mapping and definition of territories contaminated by technogenic radionuclides (Cs-137, etc.).

Surface gamma and gamma spectrometry survey: Up to now this is the most important method used for geological and radioecological mapping, for regional prospecting for different mineral resources, not only uranium, and for the assessment of technogenic contaminations associated with natural and man-made radionuclides.

The above-mentioned methods together with emanation surveys proved to be the most efficient for the survey of surface deposits or deposits overlain by porous and permeable sediments up to 5 m thick. All these methods were utilized in the discovery of the majority of deposits in Northern and Southern Kazakhstan, Middle Asia, etc.

These methods and corresponding instrumentation were developed by "Rudgeophysica" and are widely used for radioecological studies as well.

Hydrolithochemical survey along small rivers: This type of survey is performed in the scale 1:1 000 000 - 1 point per 100 km² - and includes the sampling of water and sediments. The efficiency of this type of survey is provided by using the "Angara" instrumentation set, which utilizes the laser-luminescence method with a unique sensitivity of $2 \cdot 10^{-8}$ g U/l. This work was carried out by "Rudgeophysica" at an area of 8 000 000 km² and leads to the detection of

some anomalies, some of them with an area of several tens of thousand km². It was found that all uranium provinces of the country are situated within the borders of these large anomalies. The rest of the anomalies found during the survey may be treated as one of the factors which guides to new undiscovered uranium provinces. The method provides for the possibility of quantitative prognosis of uranium in deposits. Lately positive results were obtained in using this method for the prospecting of ores other than uranium; mostly gold, for regional ecological survey and environmental monitoring of territories.

Down-hole gamma survey, gamma survey in mines and open pits; neutron survey by means of instant fission electrons detection, other nuclear-physical and integrated methods: Gamma logging is used in practically all drill holes for reconnaissance of any kind of mineral resources. One more feature of gamma logging at uranium ore deposits is the possibility for quantitative interpretation of anomalous concentrations and utilization of information thus obtained for the estimation of uranium reserves.

Gamma electrometric logging survey is mainly used for lithologic purposes, to determine the potassium (K) and thorium (Th) contents in the rock.

New problems of this logging method are associated with exploration for, and especially exploitation of hydrogenic type deposits with sharp equilibrium shear between uranium and radium. This phenomenon prohibits utilization of gamma survey for the assessment of drilled ore bodies.

Due to this, the method for direct uranium detection by instant fission electrons was developed using the excitation of electrons neutron generators with high output ($2 \cdot 10^9$ n/s). "Rudgeophysica" developed the methodology and corresponding instrumentation for this detection method. It is used in the survey of all hydrogenic deposits as well as for the control of the leaching process during uranium production with the ISL method.

The survey and reconnaissance of deep U-ore deposits, including hydrogenic types and unconformity-related types, etc., are carried out by a wide spectrum of geophysical methods.

Electrosurvey (mostly non-contact and utilizing neutral electromagnetic fields) proved to be most efficient. These methods were used to map paleo-valleys, to determine deep geological structures under traps, etc.

THE SUE URANIUM DEPOSITS, SASKATCHEWAN, CANADA

F EY, J.P. PIQUARD, D BAUDEMONT, J ZIMMERMAN

Minatco Limited,
Calgary, Alberta,
Canada

Abstract

The Sue uranium deposits occur in northern Saskatchewan at the eastern edge of the Athabasca Basin. They were discovered from 1988 to 1989 by Minatco Limited, operator of the former Wolly Joint Venture (Inco Limited, Canadian Occidental Petroleum Ltd and Minatco Limited). These deposits have been named in order of discovery: Sue A, Sue B (1988), Sue C and Sue CQ (1989).

Although the Sue orebodies belong to the Athabasca unconformity type represented on the property by McClean and Jeb (discovered previously by Canadian Occidental Petroleum Ltd and Inco Limited) they display numerous particularities in their geological setting. They are spatially related to the unconformity separating the Helikian Athabasca Sandstone Group from the underlying graphitic metasediments of the Wollaston Group and are clustered on the western edge of the presumed Archean Collins Bay Dome along a 2.9 km long NS trend. They occur at a shallow depth of 80 m below surface which should allow open pit mining methods.

Although all the Sue deposits belong to the same trend, their geological settings differ. Sue A is hosted in the sandstones at the unconformity with little to no mineralization in the basement. The particularity of the Sue B orebody is to display perched mineralization i.e. high up in the sandstones, to within 8 m of the surface. Sue C and CQ are basement hosted orebodies with the U mineralization occurring as a single lens in Sue C and forming a multiple lens system in Sue CQ. No mineralization occurs in the sandstones.

In all four deposits, the mineralization is related to extensive hydrothermal alteration overprinting the retrograde, regolithic and diagenetic alterations. The widespread hydrothermalism is indicated by intensive argillization and hematization in the basement and in the sandstones.

The high grade core of the orebodies is spatially related to the proximity of the graphite rich pelitic gneisses and is fault controlled. The U mineralization of Sue A and Sue B is characterized by an association of uranium oxides and Ni-Co arsenides, the Sue C and Sue CQ ore is monometallic.

All these deposits have been found using a combination of geophysical techniques. Sue A and Sue B were discovered using ground VLF and HLFM (Max Min) techniques whereas the Sue C and Sue CQ deposits located 100 m off the EM conductor are the result of DC resistivity and IP surveys.

INTRODUCTION

The Sue deposits occur in Northern Saskatchewan at the eastern edge of the Athabasca Basin (Fig. 1). They are located on the Wolly project, situated 6 kilometres west of the Rabbit Lake mill complex and 60 kilometres north-east of the Cigar Lake deposit.

The property consists of 7 claims covering a total of 26,604 hectares, at about latitude 54° 15' north and longitude 103° 55' west (Fig. 2). The property is accessible by an all weather private road branching off Provincial Highway 905. A well maintained year-round airstrip is located at Points North some 29 kilometres west of the property.

Eight deposits have been discovered to date on the property since exploration was initiated by the former joint-venture partners INCO and Canadian Occidental Petroleum Limited. These deposits have been named in order of discovery: McClean Lake North, McClean Lake South, JEB, Moonlight, Sue A, Sue B, Sue C and Sue CQ (Fig. 3).

The geological descriptions presented in this paper are based on diamond drill hole information, as minimal outcrop exists on the Wolly property, and on information gathered out of preliminary studies conducted by scientific laboratories.

The recent discoveries of the Sue deposits have generated new ideas with respect to general geological models. This paper should be considered as a case history description and as such a long term exploration success story.

The Sue deposits on the Wolly property belong to the uraniferous district of the Athabasca basin, flanked by the Rabbit Lake, Collins Bay and Eagle Point deposits to the east and the Dawn Lake, and Midwest Lake deposits to the west.

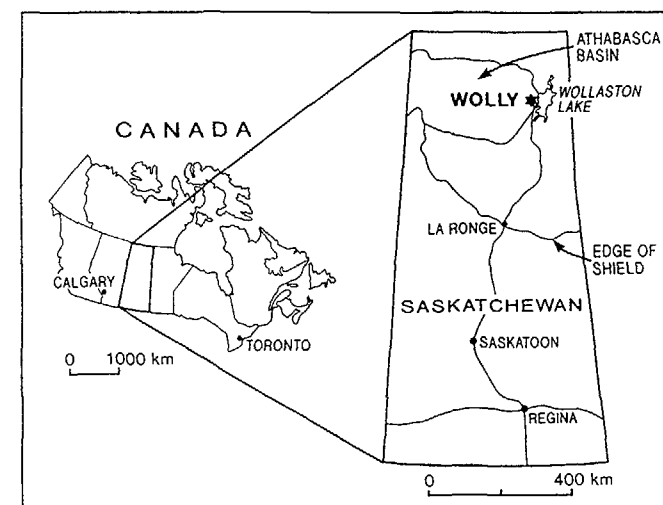


Fig 1 Location of the Wolly project in Saskatchewan, Canada

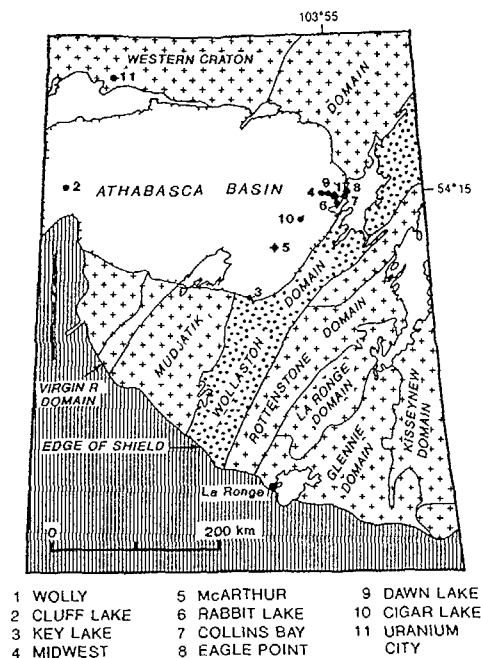


Fig 2 Lithostructural subdivisions of the Canadian Shield and major uranium deposits in Saskatchewan After Sibbald (1983), modified

Although the Sue deposits have many similarities with the major uranium orebodies in the Athabasca they display different characteristics.

The orebodies are covered by Athabasca Group sediments and glacial overburden, with a thickness of approximately 80 metres.

The first case history on the Wolly property describes the McClean deposits (Wallis et al , 1983)

REGIONAL GEOLOGY

CRYSTALLINE BASEMENT

The Saskatchewan Precambrian Shield, belongs to the Churchill Structural Province (Stockwell, 1970) and has been subdivided into several lithostructural domains (Fig 2, Lewry et al , 1978)

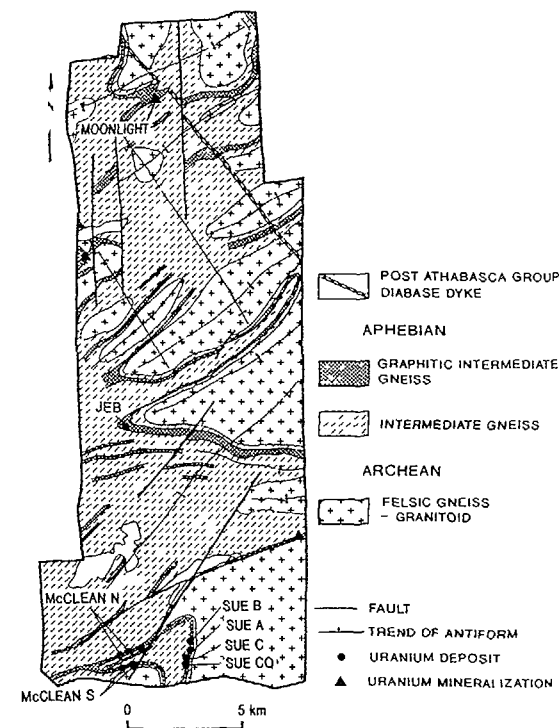


Fig 3. Wolly project geological compilation.

The Wolly property straddles the transition zone between the Mudjatik Domain to the west and the Wollaston Domain to the east (Lewry and Sibbald, 1977), the latter hosting most of the economical uranium occurrences of Saskatchewan

The basement geology of the Wollaston Domain on the Wolly project, interpreted mainly from aeromagnetic surveys, suggests that approximately one-half of the property consists of remobilized Archean rocks (Fig 3)

These rocks occur as domes and range from granitoids in the core to foliated granitoids and more gneissic rocks on the margins. They present a uniform texture and consist essentially of biotite-quartz-feldspar. Most of the domes have been named (eg McClean, Rainbow, Collins Bay etc). They show various shapes from circular to elongated and in many instances are wrinkles or bulges of much larger regional features.

Throughout the Wolly property there is a thin cover of Aphebian gneissic rocks, believed to be 200 - 300m thick, unconformably overlying the remobilized Archean granitoid gneisses. The

typical composition of the Aphebian gneisses is cordierite-biotite-quartz-feldspar with several anatectic layers. The lower member of the Aphebian cover contains one to several multiple graphitic layers. All the known uranium deposits on the property are directly associated with these graphitic gneisses.

There is a distinct change in the lower pelitic Aphebian gneisses from north to south on the property, mainly because of the higher grade metamorphic effects through the litho structural transition zone into the Mujatuk Domain. There also appears to have been a facies change within the original lithologies. The main variations in composition are the increase of aluminosilicates (garnets, sillimanite) in the north, the presence of meta carbonates (diopside rich and marbles) in the central part and amphibolites in the south (Tent Seal Lake).

ATHABASCA GROUP

The basement rocks and the post Hudsonian paleoweathering profiles are unconformably overlain by a flat lying Athabasca sandstone formation of Helikian Age (1430 Ma, Armstrong and Ramaekers, 1985). The sandstone cover on the Wolly property varies in thickness from 0 to 200 metres and is generally mantled with up to 30 meters of quaternary glacial till consisting of sand mixed with sandstone and felsic boulders.

The Athabasca sandstones belong to the Manitou Falls Formation (B and C members, Ramaekers 1981), a non marine fluvial sandstone with conglomeratic lenses in the basal B member. These sandstones were deposited on alluvial plains by braided streams and typically show abundant cross bedding, graded bedding and a general horizontal layering locally containing heavy minerals. The rocks are generally 90% quartz and are well cemented with silica, clay, and hematite.

Post-Athabasca tectonic activity (1350-900 Ma) apparently reactivated an earlier northwest trending fracture system into which diabase dykes were intruded. One of these dykes cuts the sandstones and extends for at least 10km across the northern part of the Wolly property in the vicinity of the Moonlight uranium occurrence (Fig. 3). Many authors relate this tectonic episode to the Mackenzie diabase dyke swarm (Fahrig and Wanless, 1963, Ruzicka and Lecheminant, 1986).

REGIONAL STRUCTURAL SETTING

The present structures of the Archean and Aphebian basement result from polyphasic Hudsonian folding and late Helikian brittle deformation (Table I).

The first Hudsonian event (D₁) described by Lewry and Sibbald (1977) produced isoclinal folds and flat lying foliation. It resulted in imbricated Archean and Aphebian zones coexisting at different structural levels.

The present narrow Aphebian synforms with multiple graphitic units enclosed near large granitic domes (Fig. 3) are probably inherited from this early deformation. In the Aphebian basement of the Wolly project, the early foliation has been transposed by later events.

TABLE I Structural Organization of the Main Deformation Events on the Wolly Property

	EVENTS	STRIKE	STRUCTURES CREATED	ORGANIZATION	DEFORMATION CONDITIONS
Hudsonian	D1	EW	Tight folds, foliation shear cleavage, mylonite	Lithological	Ductile Compressive
	D2	NW	Tight folds, foliation shear cleavage	New lithological strike (overprints in part totally the FW direction)	Ductile Compressive
	D3	NE	Open folds, dome and basin interference pattern	Concordant, transposes the previous foliation	Ductile Compressive
	Isotatic Basement Movements	NE NW EW NS	Fractures, influences of the basement topography	1 set parallel to lithology 2 cross cutting sets	Compressive Extensive
Helikian	Athabasca Sedimentation				Subsidence
	Post Athabasca		Major faulting, clay alteration, zone a boules, hydrothermal mylonite, Mineralization	NE + (NW transverse on EW lithology) NW + (EW) transverse on NE lithology	Compressive Extensive Hydrothermal Hydraulic fracturing
		NW	Diabase dyke Reactivation of older structures		Extensive

Two successive thermo-tectonic events, D₂ and D₃ resulted in NNW SSE folds (D₂) being refolded by NNE-SSW trending structures (D₃). In the Wollaston domain the later event is predominant and gives a general NNE-SSW orientation to the basement lithological units.

The interference of D₂ and D₃ folds is well illustrated by both the McClean - Sue periclinal structure which shows a steep plunge to the NE and the doubly plunging dome of Collins Bay (Fig. 4 and 5).

A general uplift of the late Proterozoic basement was followed by the deposition of the Helikian Athabasca sandstones.

Brittle deformation occurred during late Helikian tectonic events. Fracturing and faulting concentrated along reactivated late Hudsonian faults, commonly followed the trace of the graphitic unit. This tectonic event is contemporaneous to the uranium mineralization and the associated hydrothermal alteration.

The majority of the faults described in the present paper is likely to be part of this phase of deformation.

At the vicinity of the Collins Bay dome, three major sets of uranium bearing reverse faults have been identified. The Rabbit Lake fault, Eagle Point fault and Tent-Seal fault (north of the dome) strike ENE WSW. The Collins Bay 'A' fault is in a NNE SSW direction. The Sue fault and Collins Bay 'B' fault strike NS (Fig. 4).

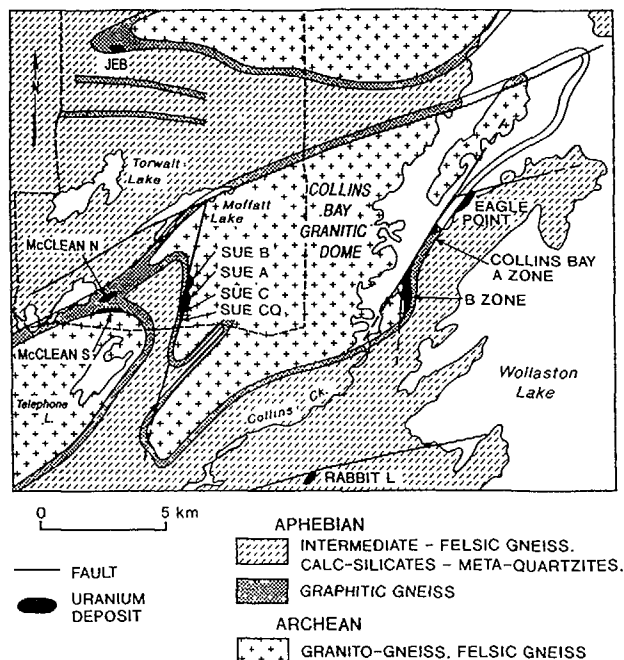


Fig 4. Uranium deposits surrounding the Collins Bay dome. General basement geology. After Sibbald (1983), modified.

DISCOVERY HISTORY

In the winter of 1988, a new grid was established in the Sue area and surveyed using Max-Min and VLF techniques. A program of 9 drill holes was scheduled on the 2.9km long Max-Min conductor near the end of the winter program.

The aim of the program was to test by a "first pass approach" the conductor over its complete length by drilling single holes spaced at 400m on the Max-Min conductor axis using a combined percussion - coring rig. The exploration strategy was successful as three holes, CS2, CS5, and CS9 scattered from north to south returned anomalous uranium values. Hole CS5 located on line 3+00S displayed the highest values (12.5m at 0.8% eU_3O_8) and was the discovery hole of Sue A.

During the summer of 1988, Sue A was drilled off on a 25 x 10m grid and exploration along the trend resumed.

The interpretation of the geophysical results emphasized a strong correlation between a VLF low and the Max-Min data for the position of Sue A on the trend. A similar anomaly was located 400 metres north of Sue A and was drilled in the summer of 1988. Hole CS44 was the discovery hole of the Sue B deposit and returned 16m at 0.2% and 4m at 1.5% U_3O_8 in two mineralized zones, one of which was just below surface. The Sue B deposit was immediately drilled on the same grid pattern as Sue A.

During the winter of 1989, a decision was taken to drill off Sue A and Sue B at a reduced spacing of 10 x 12.5m to avoid the "nugget" effect of localized high-grade pods and to define accurately the geological ore reserves.

Two holes CS34 and CS60 drilled on the southern limit of Sue A at the end of 1988, had identified anomalous uranium values (0.154% over 1.3m) in the basement contrasting with the unconformity mineralization of Sue A and Sue B. The location of these two holes was at the intersection of NE-SW trending VLF structures with the EM conductor.

In the summer of 1989, exploration resumed mainly south of Sue A. Emphasis was put on the holes previously mentioned (CS34 and CS60) and the anomalous basement-uranium zone was traced to the southwest until S209 was intersected. Hole S209 was the discovery hole of the Sue C deposit and returned 18m grading 21% U_3O_8 with the mineralization entirely hosted in the basement.

After the discovery of S209, the drill pattern was changed to match a near north-south structure revealed by a resistivity survey carried out along the Sue trend.

Then following confirmation of a steeply dipping vein attitude of the Sue C mineralization, the exploration methodology changed. A combination of angle holes and vertical holes was used to intersect the mineralization at depth and updip at the unconformity.

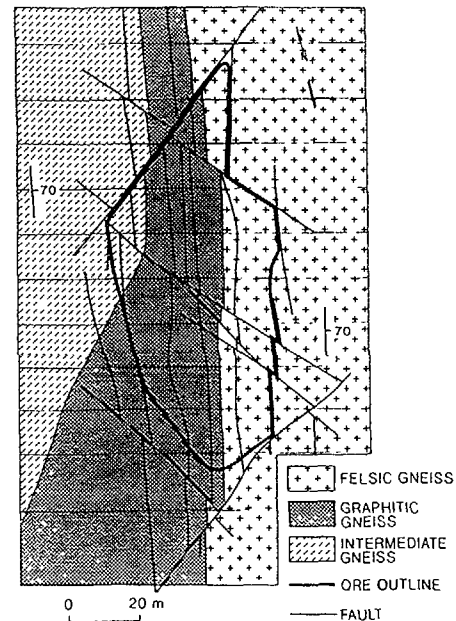
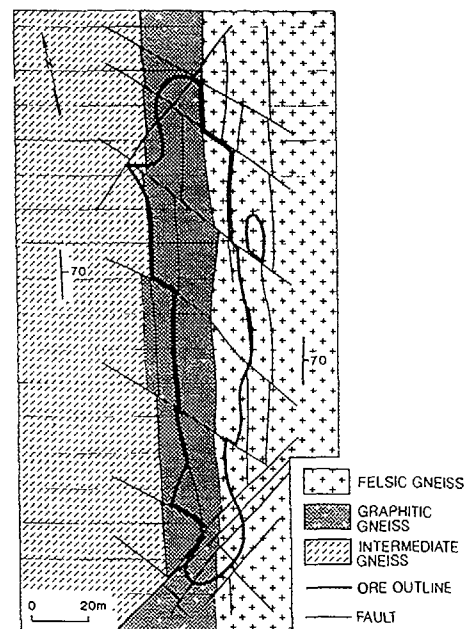
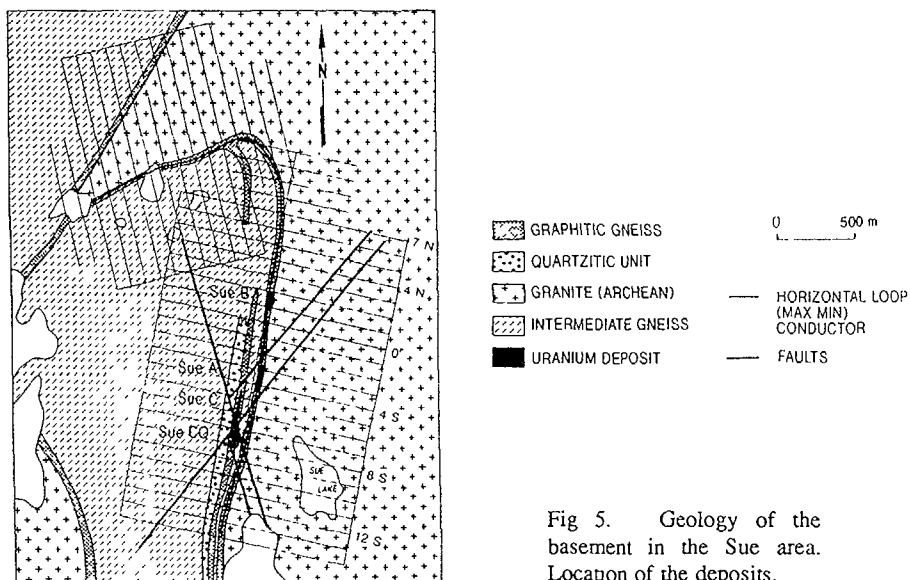
To date, Sue C is grading into the Sue CQ deposit, a zone of multiple mineralized lenses, to the south. Both deposits have been drilled using a pattern of 12.5 x 10m (1990 and 1991).

At the beginning of the summer of 1991, Sue C and Sue CQ remain open particularly at depth and numerous other occurrences have been discovered along this productive Sue trend (Sue D).

DEPOSIT GEOLOGY

The Sue deposits lie on the western flank of the Collins Bay dome, approximately 2.5 kilometres east of the McClean Lake deposits (Fig. 4). A 2.9 kilometre long basement Max-Min conductor coinciding with rock layers enriched in graphite was identified during the winter of 1988 and subsequently drilled. The deposits Sue A, B, C and CQ trend roughly north-south (N12°) along steeply east-dipping units of graphitic gneisses (Fig. 5).

All the Sue orebodies devoid of any surface expression were discovered above these conductors and are coincident with their geophysical expression.



The favourable graphitic gneiss extends south off the Wolly property. To the north it extends for one kilometre beyond Sue B where it is folded sharply to the west and eventually links up with the structure controlling the McClean Lake deposits (Fig. 4). In the Sue area, the graphitic unit is in fault contact to the east with feldspathic gneisses and granitoid/pegmatoid rocks whereas to the west it is gradational with intermediate gneissic units (Fig. 6, 7 and 8). The graphite content varies widely from 1% up to 70% and may occur as bands paralleling the foliation.

Combinations of recurring normal and reverse faulting parallel to the east dipping foliation in the graphitic gneisses produced a basement hump with 8 to 40 metres of relief. Reverse faulting has resulted in stepping the unconformity down to the west. It is on this basement high and its western flank that the Sue deposits occur. Northeasterly and northwesterly striking faults offset and modify the major NS structural controls, thus creating conditions which limit the extent of mineralization along trend.

The Sue trend is characterized by two main parallel graphitic units, 80 to 100m apart (Fig. 5). The eastern unit hosts the unconformity related deposits Sue A and Sue B, the western unit hosts the basement type Sue C and Sue CQ deposits.

The Sue deposits are located on, above or below the unconformity which lies 65 to 75 metres below surface. The bulk of the mineralization occurs either completely in the overlying sandstones or entirely in the basement. The eastern margin of the deposits is largely controlled by faulting and by the lack of graphitic rocks in the basement. Basement lithologies on the west appear to exert little control over the lateral extent of mineralization.

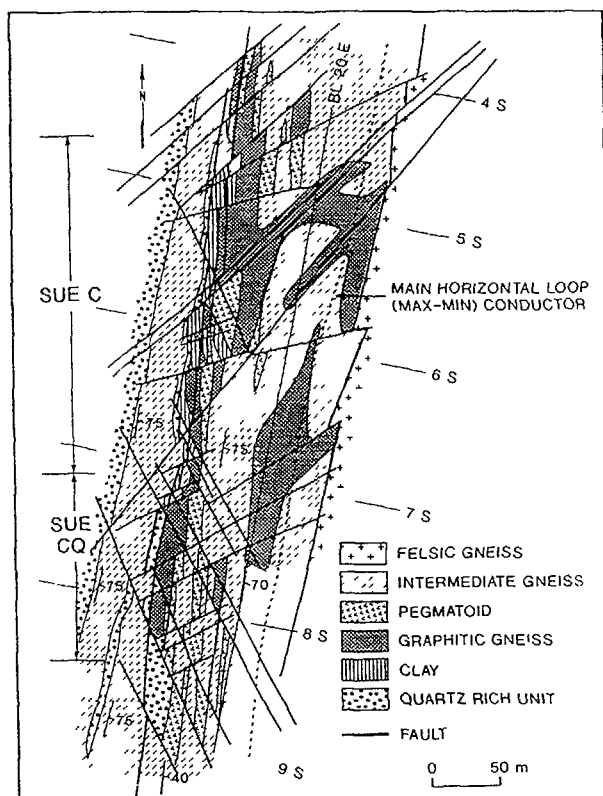


Fig 8. Sue C - Sue CQ. Geology at the unconformity.

The deposits are typically hosted in massive earthy-red clay extending above and below the unconformity. Laterally, alteration and mineralization diminish rapidly with the north-south faults providing sharp boundaries.

SUE A DEPOSIT

The Sue A deposit lies between 60 and 75 meters below surface (Fig. 9) and strikes north 12° east. At a 0.10% U_3O_8 cutoff, the deposit is 176 metres long with a width ranging from 8 to 30 metres and averaging 15 to 20 metres (Fig. 17). Its thickness varies from 1.5 to 12 metres with an average of 9 metres. The mineralization is extensively controlled by faulting, resulting in irregular cross-sectional shapes. In general, the deposit is flattened with a westerly dip conforming to the drowdropping of the unconformity. Mineralization terminates against two sets of faults, northeast and northwest in direction.

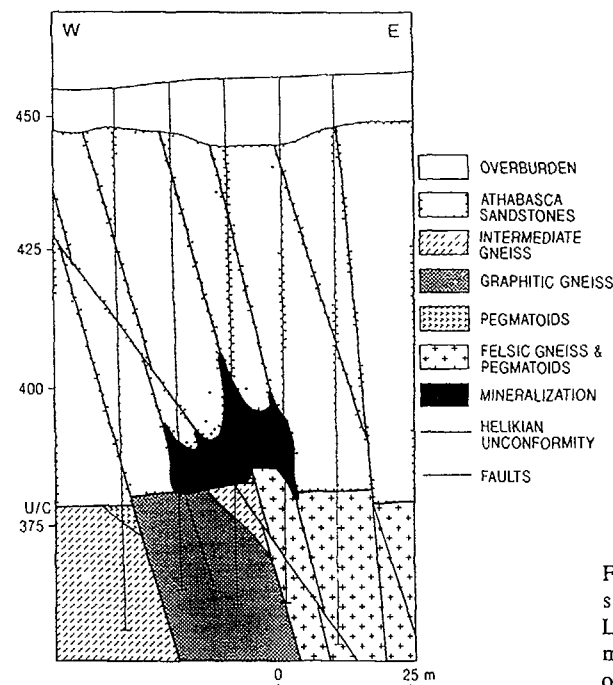


Fig 9 Sue A deposit Cross section Line 3+12.5S. Lithology and contour of mineralization at 500ppm cut-off.

The deposit lies on and immediately above the unconformity in an envelope of massive earthy-red clay (Fig. 13). Argillic alteration extends almost up to the sandstone subcrop along fault zones, leaving only scattered sections of silicification in the cap rock. An average of 9 metres of glacial overburden covers the sandstones.

Minor amounts of uranium mineralization extend downward into the basement as narrow roots along faults. Less than 2% of the Sue A deposit lies below the unconformity.

The distribution of uranium is confined to a few high grade ($>5\%$) pods, mostly in the south half of the deposit where some 70% of the total uranium is located. Grades exceeding 15% occur.

SUE B DEPOSIT

The Sue B deposit is located approximately 350 metres north of Sue A. The mineralized zone is 90 metres long and averages 40 metres in width (Fig. 18) but unlike Sue A, the mineralization occurs at two horizons within the sandstone (Fig. 10).

The upper horizon contains some 50% of the uranium mineralization. It lies at a depth of about 20 metres above the unconformity. Mineralization extends at one point to the subcrop of the

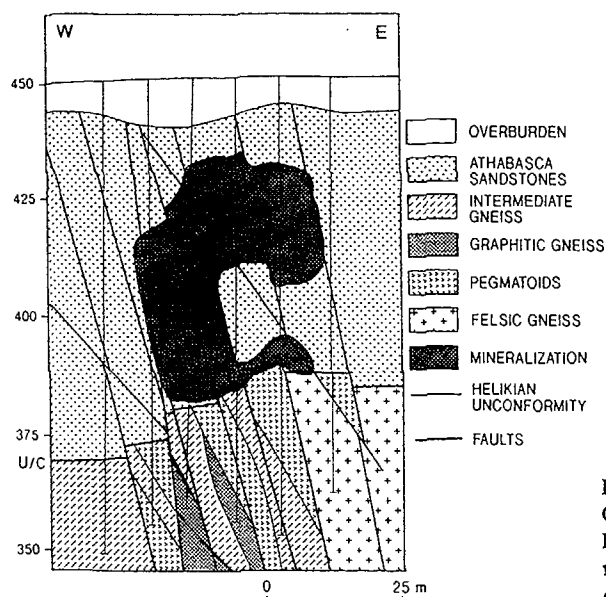


Fig 10. Sue B deposit. Cross section line 1+62.5N. Lithology and contour of mineralization at 500ppm cut-off.

Athabasca sandstones at a depth of 8 metres below surface. This upper zone is about 50 metres long, 26 metres wide, and 17 metres thick.

The lower zone of mineralization lies on and immediately above the unconformity at depths between 60 and 75 metres. In general the mineralization lies on the western flank of the basement high in the fault steps created by the drowdropping of the unconformity. The basement high has up to 20 metres of relief.

The upper and lower zones are connected by chimney-like zones of mineralization. Very little uranium occurs in the basement rocks (less than 1% of the deposit).

The Sue B mineralization is largely fault controlled. The upper zone may be a product of intersecting structures such as conformable and conjugate faults which created a zone of weakness and relatively high permeability. Uranium migrated upward along connecting faults and spread out into the highly fractured sandstone to form the upper deposit. Laterally, mineralization terminates rather abruptly against north-south trending faults. As in Sue A, northeast and northwest striking faults limit the extent of the deposit in the north-south direction.

The uranium mineralization is hosted in massive earthy-red clay, although the upper zone displays remnant silicifications. The sandstone between the upper and lower zones is slightly resiliified (Fig. 14).

The Sue B orebody is a medium grade deposit with very little high-grade mineralization. Grades usually do not exceed 5% U_3O_8 .

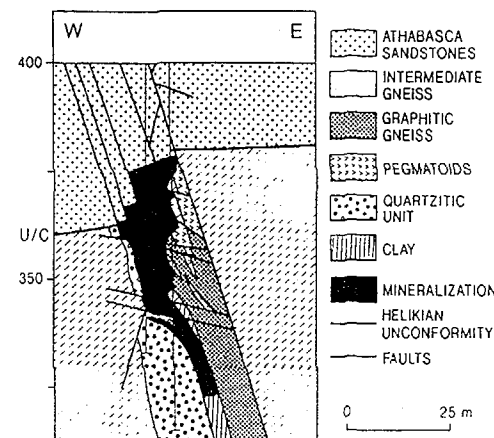


Fig 11. Sue C deposit. Cross section line 5+00S. Lithology and contour of the mineralization at 500ppm cut-off.

SUE C DEPOSIT

The Sue C area lies immediately to the southwest of the Sue A deposit.

The Sue C mineralized structure is a 10 to 15 metre wide NS subvertical structure dipping 70° to the east (Fig. 11), paralleling the graphitic/feldspathic gneiss lithological contact 100 metres to the east (Fig. 8).

The mineralization is continuous along trend from line 4+75S to line 8+50S (Fig. 19). The massive high grade ore which characterizes the northern part of the orebody clearly ends at line 7+25S where the subvertical vein ("C" type) coexists with widespread lower grade mineralization ("CQ" type).

The mineralized structure lies completely within the Aphebian pelitic metasediments in close proximity to graphitic gneisses.

From east to west, a typical lithological sequence across the Sue C deposit consists of non-graphitic gneisses intermixed with pegmatoids, graphitic gneisses (20 to 40m thick), massive ore and clay, a quartz rich unit and non-graphitic gneisses (Fig. 11).

The mineralization is hosted by reverse anastomosing faults, (the Sue C fault), striking N12° and parallel to the basement lithologies. It is located at the footwall of the graphitic gneiss, in a clay-rich zone as well as in the lower graphitic unit itself. It is typically underlain in sharp contact by a massive quartz rich lens. A second quartz lens was intersected 30m west of the ore.

The average grade is high, 7% U_3O_8 at 0.1% cut-off (Table III). The mineralization consists of massive pitchblende, pitchblende nodules and veinlets within a white or black clay envelope locally overprinted by intense hematitic-clay alteration. The mineralization contains minor amounts of arsenides.

At the scale of the deposit, the mineralization is typically exhibiting a vein geometry parallel to the remnant subvertical foliation. On a detailed scale, the high grade pods are distributed as vertically stacked, flat lenses bounded by the graphitic gneisses and the quartz lens (Fig 21). Therefore the main structural control of the mineralization is the concomitant action of steeply dipping faults and flat lying shears

The flat lying shears induce a thickening of the quartz lens at depth, thus controlling the downdip limit of the ore. The depth of the mineralization to date ranges from 115m in the north to 150m in the south

The unconformity is typically 75 to 80m below surface and is disrupted by the major reverse faults creating a hump or offsets of up to 42m. There is no evidence of the mineralization extending upwards into the overlying sandstones. Immediately above the ore, the sandstones are strongly argillized (chloritization and bleaching) with variable hematization (Fig 15)

The continuity of the mineralization in the Sue C deposit is perturbed by major NE and NW faults which have slightly displaced the vein over a few metres (Fig 19, 20). The NE trending faults are commonly strongly graphitic and display disseminated mineralization of low grade 0.05 to 0.1% with one layer at 3.3% U_3O_8 . The mineralization is discontinuous but may be traced for 120m especially south of Sue A towards Sue C

SUE CQ DEPOSIT

Although the lithology is consistent along trend, Sue C and Sue CQ are two distinct deposits. The geometry and the grade of the mineralization as well as the structural characteristics are clearly different

The Sue CQ deposit lies south of line 7+12.5S where a major NW-SE fault zone limits the southern extension of the Sue C mineralization

The mineralization is discontinuous and of lower grade (1.69% at 0.1% cut off, Table III) but spread over a large area

The ore is hosted completely within intensely altered graphitic to non-graphitic metapelites (Fig 12)

The overall mineralized volume is divided into multiple moderately dipping lenses for a total width of 40m over a strike length of 125m. The ore does not subcrop, with the upper limit at a depth of 120m i.e. 45 metres below the unconformity and is known to date to a depth of 165 metres (90m below the unconformity)

The bulk of the mineralization consists mainly of pitchblende nodules associated with an ubiquitous red-brown hematitic clay-rich envelope overprinting the pervasive illitization of the gneisses. The massive Sue C-type ore is not observed in the Sue CQ orebody

The structural controls of the mineralization remain similar to those observed in the Sue C deposit. Flat lying shears are inducing high-grade pods, vertically stacked in moderately dipping

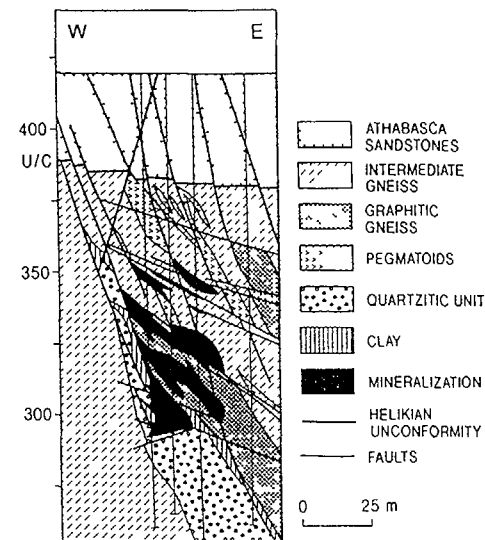


Fig 12 Sue CQ deposit
Cross section line 7+87.5S
Lithology and contour of the mineralization at 500ppm cut-off

lenses of 1 to 5m in thickness (Fig 22). However, the NW-SE fault corridor exhibits normal vertical offsets of the unconformity and controls a strong development of the silicification which invades almost completely the southern limit of the Sue CQ deposit.

ALTERATION

The Sue deposits have many common alteration characteristics also known in other Athabasca deposits

Most of the alteration pattern descriptions are based on macroscopic core observations. Detailed mineralogical and chemical studies on the ore are currently being carried out. Preliminary data using X-ray diffraction and electron microprobe are here provided (Severin, 1991)

All of the described rocks are altered, due to the fact that drilling to date has been confined to the orebodies

Five main types have been recognized

- 1 retrograde metamorphism during later stage of the Hudsonian event
- 2 pre-Athabasca paleoweathering profile
- 3 diagenetic alteration of the Athabasca sandstones
- 4 hydrothermalism
- 5 late alteration due to remobilization

RETROGRADE METAMORPHIC ALTERATION

Post Hudsonian retrograde metamorphic alteration is pervasive as fresh rocks are rare

In the Sue area, the retrograde metamorphism is a subordinate alteration phase in the vicinity of the deposits, and has been overprinted by all following events

The primary mineralogy of the basement rocks is transformed by two main phases of clay minerals, chlorite and illite. Quartz and graphite are preserved

Cordierite, sillimanite, garnet and biotite are transformed into mainly FeMg-rich chlorite whereas the feldspars are subject to sericitization with the presence of albite and calcite

PRE ATHABASCA PALEOWEATHERING

The paleoweathering has affected the basement rocks prior to the diagenesis of the Athabasca sandstones. It is well developed in the Sue area but away from the deposits. It is represented by a characteristic vertical gradation of colour zones. This was described by MacDonald's regolith study (1980) of the Wolly area

At the boundaries of the orebodies, the regolith or saprolite (Wilson, 1986) forms a discontinuous hematitic zone overlying pale to dark green metapelites extending 5 to 40m below the unconformity and within 15 metres of the deposit. The oxidized red zone is characterized by the absence of graphite and sulphides, presence of hematite and all metamorphic minerals are clay altered with chlorite, illite and kaolinite on top of the sequence. The original texture is often destroyed or completely transposed. In the green zones, the graphite and the original structuration of the rocks are preserved and the metamorphic minerals are altered into illite, Mg-rich illite and Fe-chlorite. The thicknesses of these alterations vary considerably according to the parent rocks. The prominent white zone as described by MacDonald (1980) is absent in the vicinity of the orebodies

The remnant pre Helikian weathering alterations are complicated by the later diagenetic and hydrothermal effects and even by the retrograde metamorphic event in the green zones

The paleoweathering profiles of the Sue deposits are typical and similar to those of other Athabasca orebodies (Halter, 1988, Bruneton, 1986, Ruhmann, 1986, Wallis et al., 1983)

DIAGENETIC ALTERATION

In the complex history of the Athabasca sandstones deposition, the diagenetic alterations are the most difficult to grasp as they are completely superimposed by late hydrothermal alterations affecting the entire stratigraphic column above the orebodies

A few remnants of diagenetic phases are preserved at distances of 15 to 25m from the mineralization where the sandstones range in colour from salmon pink to purple with

multicolored bands of dark hematitic layers (associated with heavy minerals) alternating with late limonitic sections

A silicified cap is preserved in the uppermost portion of the sandstones (down to 25m below the overburden) outside the faulted corridors controlling the orebodies

HYDROTHERMAL ALTERATION

In the Sandstones

The alteration in the sandstones is highly variable and does not form concentric haloes as in the Cigar Lake deposit (Bruneton, 1986, Fouques et al., 1986). It is more interfingering and each of the various deposits of the Sue trend has a unique distribution of alteration processes due to their structural setting and mineralization control

In Sue A and Sue B, the alteration extends to surface i.e. 60 to 70m above the unconformity and is mainly controlled and restricted to the fault corridor limiting the lateral extent of the deposits (Fig. 13 and 14)

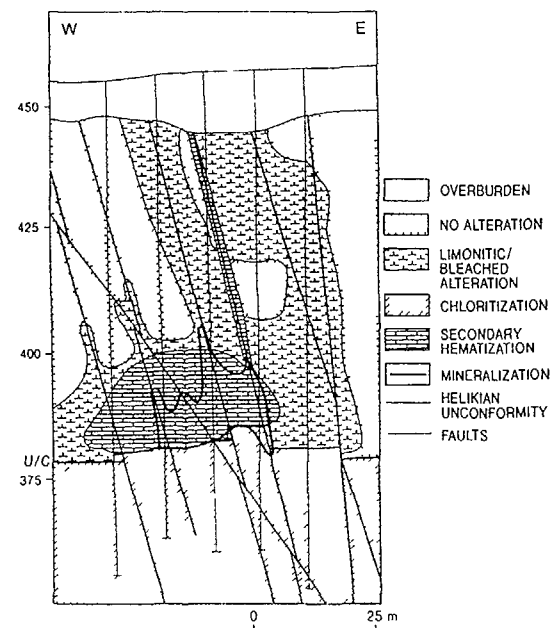


Fig 13 Sue A deposit
Cross section line 3+12 SS
Alterations and contour of the
mineralization at 500ppm cut-
off

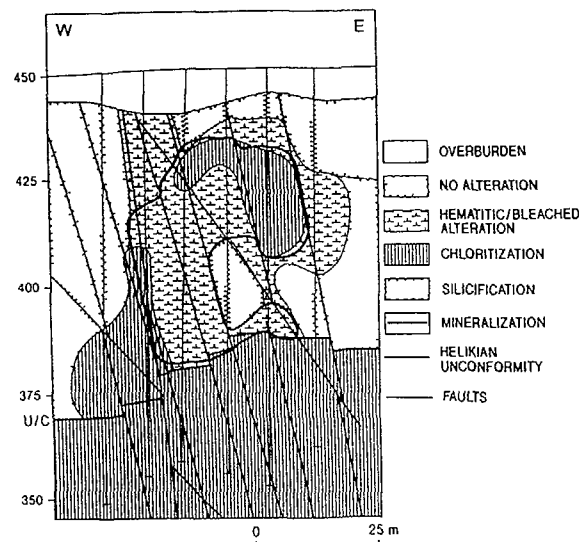


Fig 14 Sue B deposit
Cross section line 1+62 5N
Alterations and contour of the
mineralization at 500ppm cut
off

Two alteration processes are noticed beyond the fault zones and are represented by the following

- Overgrowths of euhedral quartz to milky translucent quartz which are infilling the subvertical fractures. They have been observed more than 50m away from the deposits along strike in the Sue area and can extend over 150m laterally in the McClean area
- A bleaching event with quartz dissolution and illite emplacement in the matrix and in complete leaching of hematite from the purple layers. This phenomenon is particularly well developed in the sandstones overlying the Sue C ore. In the Sue A and Sue B deposits, the bleaching has been observed 20m away from the mineralization

In the vicinity of the deposits the following zonation has been observed from the surface to the unconformity

- Destruction of the silicified cap, within the fault boundaries, thus creating a collapse of the surface of the sandstones well documented in the Athabasca basin (Wallis et al, 1983)
- Bleaching in the fault zone, with incomplete removal of hematite, increase of the illitic matrix and quartz dissolution. Hematite remains present in the sedimentary features (bedding planes, cross-beddings and Liesegang rings) although superimposed by several stages of limonitization along fractures and bedding planes. The uranium content varies from 1ppm to 3ppm in the fracture zones. Dravite occurs as spherule rich layers as well as overgrowths on green detrital tourmaline (Skupinsky, 1990). These accessory minerals occur as well in the barren zones and have been identified in several other Athabasca deposits (Ey et al, 1985, Bruneton, 1986, Ruhrmann, 1986)

A white zone occurs 20 to 40m above the mineralization with an increase of the illite content of up to 30% and an almost complete removal of hematite. The distribution of this alteration is coincident with a high density of fracturation and is characterized by breccias, micro-'zone a boules', unconsolidated sandy sections and several metres of thick clay sections where the bedding planes are well marked by a concentration of heavy minerals. Kaolinite and chlorite are generally absent as indicated by the low MgO content in the Sue A and Sue B ore zones (Table II). Uranium content is up to 10-20ppm. Several episodes of silicification have occurred as some previous recemented breccia zones have been refractured and present euhedral quartz infillings and are wrapped in that clay zone. Limonitization is locally strongly developed and expressed as brown soft clay layers of up to 30cm thick and as black coatings on the subvertical fractures (Fig 13). A carbonate overprint represented by euhedral siderite crystals infilling the fractures and by diffuse impregnations as stringers is nearly ubiquitous.

A red brown zone consisting of massive hematitic clay is completely wrapping around the mineralization. It extends only a few metres laterally from the ore but 5 to 10m in the vertical direction. A few oxidized sandstone remnants containing interstitial red hematite are still present although completely rounded and rotated. The uranium content is noticeably increased from 100ppm to 0.5%. Locally the red brown illitic and hematitic zone disappears, and is interfingering with a dark green to grey chloritic clay zone (Fig 14). As in Cigar Lake (Bruneton, 1986), the contacts between the clay zones and the high grade mineralization are very sharp as the uranium grade increases from 100ppm to several percent in less than a metre.

Table II Distribution of major elements in the ore zones of the deposits (all values in percent)

	SUE A	SUE B		SUE C
		Upper	Lower	
SiO ₂	43.98	79.12	52.42	28.55
Al ₂ O ₃	10.09	2.10	18.04	19.91
Fe ₂ O ₃	7.11	3.22	5.57	6.65
MgO	0.82	0.58	0.49	8.05
Na ₂ O	0.22	0.12	0.24	0.10
K ₂ O	2.72	0.41	5.43	3.96
CaO	0.11	0.05	0.09	1.73
TiO ₂	0.18	0.11	0.12	1.32
P ₂ O ₅	0.45	0.12	0.28	0.11
MnO	0.03	0.02	0.01	0.23

- A grey quartzitic zone, only observed in Sue B, is separating the two levels of mineralization (Fig 14). This alteration is characterized by a silicification episode with quartz overgrowths around the detrital quartz grains rimmed by hematite, iron sulphide dissemination mostly marcasite (Skupinsky, 1990) and presence of Fe-chlorite.

In the basement

The alteration patterns in the basement are similar for all four deposits. However, their spatial distribution is related to the structural setting of the mineralization. In the unconformity-related deposits, Sue A and Sue B, the zonation is vertically stacked as opposed to the basement-hosted mineralization where the alteration haloes are concordant to the steeply dipping ore lenses either as a single halo in Sue C or multiple repeated haloes in Sue CQ.

In the Sue A and Sue B deposits, the basement rocks below the ore are completely argillized and bleached, white to pale green in colour. The primary minerals and the original texture have disappeared and often have been transposed by flat structures parallel to the unconformity. This clay zone extends 1 to 2m below the unconformity and presents no evidence of graphite or sulphides.

The intensity of the alteration gradually decreases downwards. The mafic minerals are strongly altered to Mg-Fe chlorite and sudoite, illite represents 50% of the rock volume and quartz dissolution still occurs. The original structuration is preserved parallel to the foliation where graphite is present. Dravite and phosphates are ubiquitous in the basement metapelites.

Locally red brown hematization associated to mineralization extends into the basement along major faults concordant to the lithologies.

In the Sue C area, the distribution of the alteration patterns is controlled in part by the lithology and by the steeply dipping attitude of the mineralization (Fig 15). A quartz-rich unit is the footwall of the ore over the length of the deposit and represents a major silicification event. This unit is composed of 90% quartz and 10% accessory metamorphic minerals (plagioclase, biotite and sillimanite) mostly depending on the original composition of the rocks either gneissic or pegmatoidic. Two different "quartzites" have been differentiated primarily by their colour based on the proximity of the highest uranium concentrations, respectively indicated by black smoky irradiated quartz and white translucent quartz. The quartz-rich unit is usually 1m thick at the unconformity and thickens up to 20m downwards at the intersection of flat-lying shears induced by the major reverse Sue fault.

The development of clay is intense and two different ore settings are observed.

- The mineralization is enveloped by a white clay zone, 2 to 3m thick on each wall and therefore is separated by it from the graphitic gneisses and from the quartz-rich unit. This geometry occurs in the central part and is representative of the deposit (S209). In the white clay zone, no texture is recognizable other than an aspect of hydrothermal mylonite i.e. the remaining quartz grains are completely elongated, cataclased, corroded and have been transposed parallel to the structures controlling the mineralization (Ey et al., 1985).

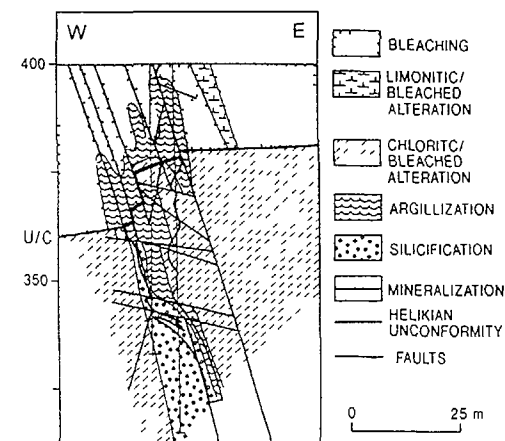


Fig 15 Sue C deposit. Cross section line 1+62.5N. Alterations and contour of the mineralization at 500ppm cut off.

The mineralization is in contact with the graphitic gneisses and a black massive clay zone is the host rock of both facies. A well-developed red-brown hematitic alteration is superimposed and extends a few metres beyond the graphitic gneisses. Pitchblende, hematization and limonitization are observed in conjugate or flat-lying fractures 30 metres away from the main ore.

At distances increasing away from the Sue C ore, the alterations of the gneissic lithologies are similar to those known below the typical unconformity-related deposits.

The basement alteration of the Sue CQ deposit is uniform and pervasive over the total width of all mineralized lenses (Fig. 16). It is of a typical texture, the mafic minerals altered to various chlorites with a predominant illitic matrix. The graphite is present and concordant to the rock foliation. A red-brown hematitic clay zone forms consistently the host of the mineralization and overprints the light green alteration. As below Sue A and Sue B, a bleached envelope is locally observed around some lenses. The hematitic and bleached facies extend only a few decimeters to one metre away from the ore. The silicification event mentioned above produced quartzitic lenses at the footwall of the ore veins. The mineralization pinches and swells at depth by the same mechanism of quartz thickening as in Sue C.

In general, during the quartzitic event, breccias in the fault zone have been resilicified, quartz is expelled gradually into the non-graphitic gneisses of the western block and the mineralized graphitic gneisses have been locally silicified.

The silicification episode is thought to be spatially related to and synchronous with the mineralization and argillization events, at the difference of Eagle Point, where the quartzites are classified as lithological units in the lower Aphebian stratigraphy (Eldorado Resources, 1986). In the sandstones and in the basement, illite is the main clay mineral observed. The index of crystallinity ranges from 3.75 to 6 indicating anchizonal or deep burial conditions in temperature.

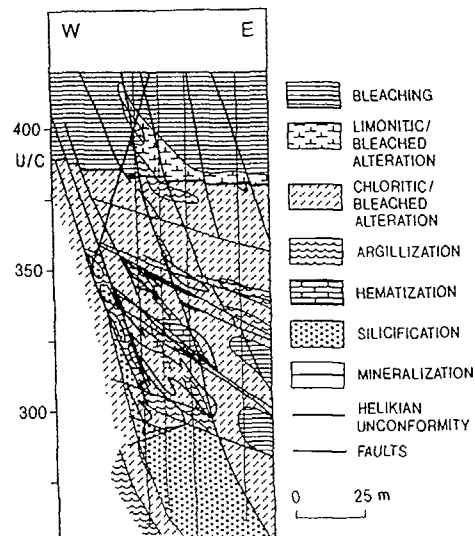


Fig 16. Sue CQ deposit. Cross section line 7+87.5S. Alterations and contour of the mineralization at 500ppm cut-off.

domains of 200 - 250°C (Dunoyer de Segonzac, 1969; Hoeve et al., 1981) and the 3T-polytype is characteristic of high pressure (Ey, 1983; Halter, 1988). The illite peak ratios (002/001) are varying from 0.35 to 0.52 confirming the aluminous composition observed in several other Athabasca deposits (Hoeve, 1982; Bruneton, 1986; Ruhrmann, 1986).

The illite content is decreasing in the basement for sudoite, a di-trioctahedral aluminous chlorite (Hoeve et al., 1981; Severin, 1991).

Late Alteration Due to Remobilization

Several late phases of alteration represented mostly by hematization and limonitization have been identified in the Sue deposits. Although not thoroughly documented to date on the Wolly project but known in the Athabasca basin these phases are probably related to the uranium remobilization with coffinite mineralization well illustrated in Sue B.

ORE MINERALIZATION

Each of the various deposits along the Sue trend displays a unique spatial distribution of the mineralization with however similarities in terms of paragenetic assemblages.

Uranium mineralization within the Sue A, B, C and CQ deposits are macroscopically described since only partial metallogenic studies on the orebodies are available to date.

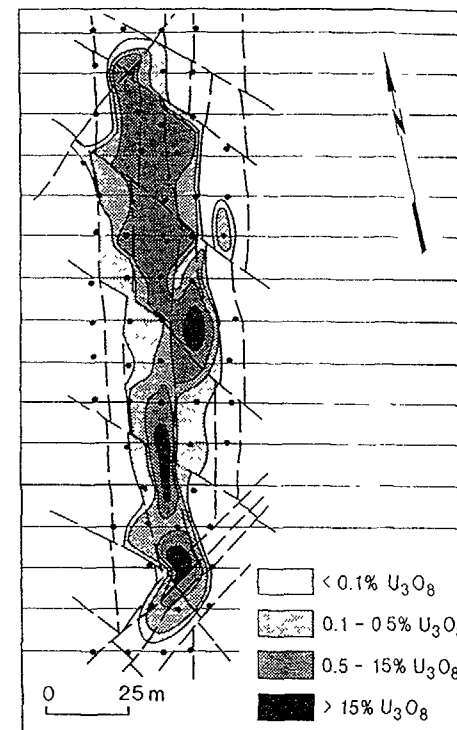


Fig 17. Sue A deposit. Plan map of the ore outlines.

The main types of uranium which have been identified are: 1) uraninite or pitchblende, massive or as nodules in clays, 2) pitchblende stringers and disseminations in fractures, faults or shear zones, 3) disseminated coffinite along bedding planes, foliation or fractures. The main phase of mineralization, at least the most economical, is the U-oxides of the first type.

Three different mineralization settings can be observed:

- mineralization at the unconformity, Sue A and Sue B
- "perched" mineralization, Sue B
- basement mineralization, Sue C and Sue CQ.

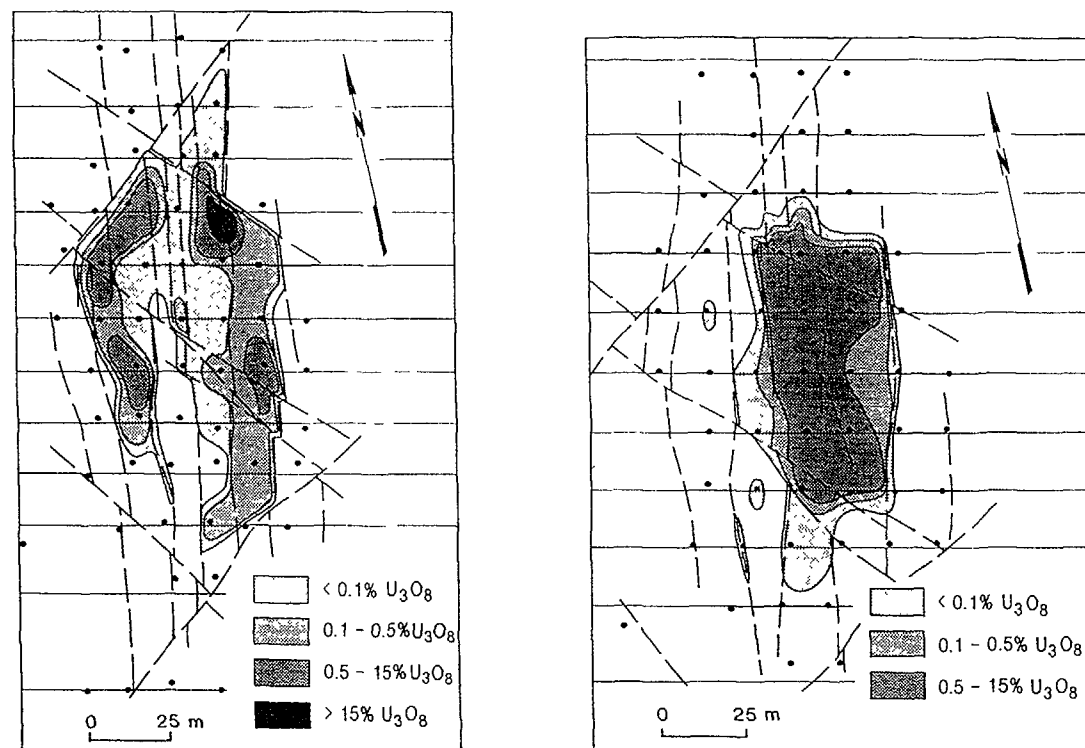


Fig 18. Sue B deposit. Plan map of the ore outlines. a) lower zone, b) upper zone

BASEMENT MINERALIZATION

Within the Sue C deposit, the bulk of the mineralization occurs as a vein parallel to the basement lithologies between graphitic clay rich gneisses and a quartz lens. Two distinct settings of the vein have been observed.

- The ore consists only of uranium oxides exhibiting a massive "metallic" aspect in the centre of the vein and progressively occurring as brecciated pitchblende at the contact and within the graphitic gneisses. In the latter case the host rock is a black clay zone. This distribution is the most common along the Sue C trend with samples commonly returning grades over 70% U_3O_8 . The uranium oxides consist of uraninite (euhedral) and pitchblende (radiating). They could represent the oldest uranium generation described as stage I in most of the Athabasca deposits and dated around 1300 Ma.
- The mineralization occurs in a hematitic clay zone as a vertical stacking of nodules elongated parallel to flat cross-cutting structures within the Sue fault. Locally the nodules

are coalescent to form patches of 10m x 5m in size with U_3O_8 values between 10 - 40%. The ore is hosted in a white clay zone overprinted by hematization, which in turn is superimposed by later limonitization. This setting is typical of the Sue CQ deposit and occurs in the Sue C orebody in a transition zone with the Sue CQ mineralization.

In both cases later stages of mineralization are represented by pitchblende impregnating all preexisting structures and can be observed 50m away from the main ore in the pale green graphitic gneisses. These infillings or disseminations are of lower grade and are associated with hematite. Carbonates have locally been observed cross-cutting and replacing the pitchblende. No major Ni-Co arsenides or sulpharsenides are associated to the primary generation of uranium oxides and as such Sue C can be qualified as a "clean orebody", similar to Eagle Point (Table III). High vanadium values forming a wider halo than U_3O_8 within clays both in the sandstones and in the basement rocks are observed in the Sue C-CQ area.

Table III Average composition of the main mineralization of the Sue deposits, (all values in percent at 0.1% U_3O_8 cut-off)

UNCONFORMITY MINERALIZATION								
	U	Cu	Ni	Co	Pb	Mo	V	As
SUE A	1.89	0.03	3.57	0.13	0.10	0.07	0.28	4.46
SUE B	0.91	0.05	1.67	0.02	0.06	0.01	0.13	1.86
BASEMENT MINERALIZATION								
SUE C	7.04	0.01	0.12	0.005	0.54	0.14	0.29	0.14
SUE CQ	1.69	0.006	0.14	0.002	0.13	0.05	0.19	0.18

UNCONFORMITY MINERALIZATION

In the Sue A and B deposits, the unconformity mineralization exhibits the following vertical zonation

A 'metallic' uranium oxide body (1m thick, 15m wide) consisting of uraninite and pitchblende

A 30cm thick cap of arsenides and sulpharsenides cemented by pitchblende. The dominant phase is niccolite exhibiting spherules of 2cm in size.

An earthy red-brown hematitic clay zone (20m thick), representing the bulk of the mineralization. The uranium oxides consist of pitchblende nodules, elongated parallel to the bedding planes and 0.5cm to 1cm in size. They are surrounded by a bleached rim up to 1mm in thickness. This type of mineralization has been well documented in the McClean area and is associated to arsenides and sulpharsenides minerals (Wallis et al., 1983).

The observed paragenetic minerals include gersdorffite, skutterudite, cobaltite, pyrite, marcasite, bravoite, with secondary inclusions of niccolite, siderite and calcite (Skupinsky, 1990). Sue A displays a definite higher Co content than Sue B due to the presence of brecciated cobaltite (Table III).

PERCHED MINERALIZATION

The remobilization of pitchblende to upper levels within the sandstones along faults is particularly evident in Sue B. The upper part offset of about 40m consists mainly of coffinite and pitchblende and is comparable to the "perched" mineralization of Cigar Lake (Bruneton 1986). Coffinite pseudomorphs after pitchblende and is structurally controlled as it occurs in

strong faulted zones as well as in the grey coloured sandstones with bedding planes rotated up to 70°.

Pyritic mineralization associated to Cu-sulphides is pervasive (Skupinsky, 1990). The U-mineralization is developed in green to dark grey sandstones, locally silicified and to a lesser extent in hematitic clay.

Although coffinite is the dominant mineral phase of the upper part of Sue B it is of economical interest as it represents the same tonnage as the lower massive part and occurs from surface to 40m below.

In all four deposits, lead and uranium show a strong correlation suggesting a radiogenic origin for the lead.

STRUCTURAL CONTROL OF THE MINERALIZATION

The Sue trend consists of a sequence of metamorphic units with a main NS trending foliation dipping steeply to the east.

NS FAULTS

The NS fault system parallel to the foliation is the predominant structural feature of the area. Strongly faulted trends are superimposed on the two main graphitic units (Sue A, Sue B and Sue C).

Except for the Sue CQ deposit, all other orebodies are elongated in a NS direction. The lateral extensions of the deposits are controlled by the width of these faulted zones (Fig 9, 10, and 11).

Structures are predominantly east dipping reverse faults which have induced a series of steps in the unconformity and local humps up to 40m similar to the ones observed in other Athabasca deposits (Midwest Lake, Cigar Lake, Key Lake). The reverse faults also created intense fracturing and tilting in the sandstones with local strong brecciation and clay development from the unconformity up to surface. In Sue B, they control the upward remobilization of the unconformity mineralization into fractured and brecciated perched blocks. Mineralization and hematitic and chloritic alterations extend for a few tens of metres up/dip or down/dip along these reverse faults.

In the basement, the intensity of deformation increases towards the Sue C, NS fault. Common mylonitic texture, fault gouges, intense microfolding and crenulation cleavage have been observed in the altered gneiss and the massive clay. Inside the mineralized zone and in close vicinity large tectonic lenses bound by shears and faults display a 'Z' shaped relict foliation. Coarse grained quartz breccias cemented by a clay or a quartz matrix are often encountered at the footwall of the mineralization.

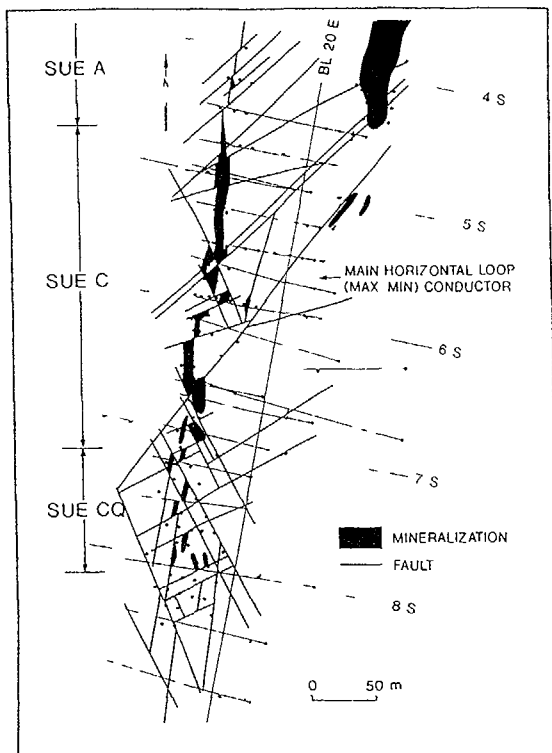


Fig 19 Sue C and Sue CQ deposits Outline of the mineralization at 500ppm cut-off projected at the 370m elevation (unconformity)

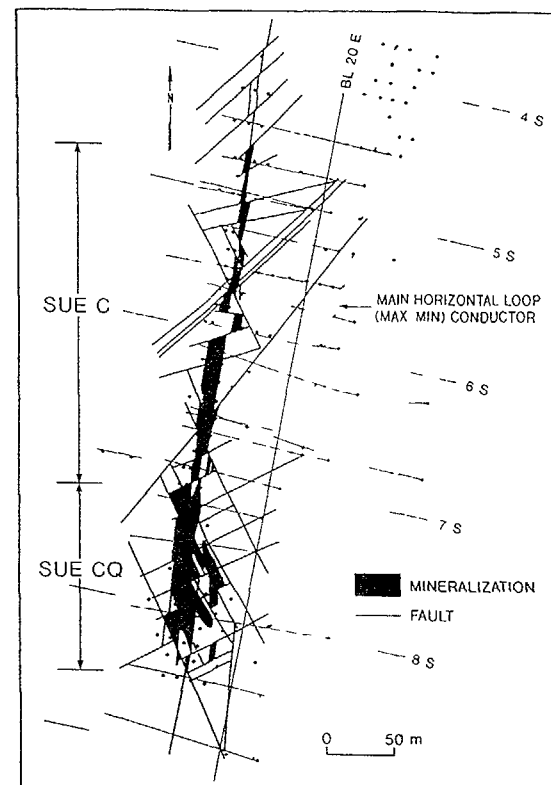


Fig 20 Sue C and Sue CQ deposits Outline of the mineralization at 500ppm cut-off projected at 340m elevation (30m below the unconformity)

The NS clay quartz faulted contact is the western edge of the major Aphebian "mobile" eastern block which extends to the contact with the Archean Collins Bay dome. At the footwall of the fault, the western block displays undisturbed foliations with weakly developed fracturing and faulting.

FLAT LYING STRUCTURES

Inside the steeply dipping Sue C fault zone, a well developed set of faults dipping at 0° to 25° to the east cross cut the footwall of the graphitic unit, the clay zone and the quartz rich unit (Fig 11). They are induced by the main reverse structures and therefore have a limited extension (10 - 20m) outside the tectonized zone. They can be considered as "Riedel" structures which are commonly observed in brittle shear zones. Inside the main ore envelope, they control the distribution of the high grade mineralization and the downdip extension of the orebody (Fig 21, 22). The flat lying structures are predominant inside the orebody by creating dilation zones infilled with pitchblende which can extend 30m away from the main mineralization.

In the upper part of the basement of all the Sue deposits, a flat lying "schistosity" develops in the clay altered zone, transposing and orienting the original foliation. In Sue C deposit, it is often controlling the extension of an upper mineralized zone which can be slightly shifted from the main vein.

NNE AND NNW FAULTS

The NS trending structures are cross-cut by two main sets of subvertical faults striking NNE and NNW inducing vertical displacements of a few metres.

Common sinistral and dextral displacements induced by the EW compressional event have been observed along these conjugate strike-slip structures (Figure 8).

These faults play an important role in the distribution of the mineralization inside the different ore-bodies, as shown by the ore outlines of Sue A and B.

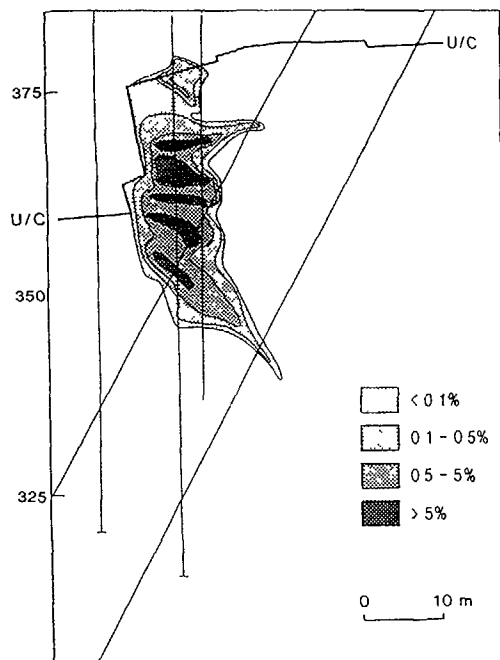


Fig 21. Sue C deposit. Cross section 5+00S. Ore outlines.

The Sue C deposit is characterised by maximum grade and vertical extension of the mineralization at the intersection of the NNE structures and the NS lithologies. South of the Sue A deposit, a major NNE - SSW fault extends southward into the Sue C area (Figs. 19, 20).

Weak basement mineralization has been encountered along this fault and led to the discovery of the Sue C deposit.

The NNW structures are predominant in the Sue CQ deposit. They induced a series of small steps with normal metric vertical displacements which contrast with the major reverse movements observed in Sue C (Fig. 12). These important changes in the structural pattern are accompanied by a modification in the attitude of the mineralization, with discontinuous low to medium grade lenses as opposed to a massive high grade vein. The NNW faults control the development of a wide silicified zone south of line 8+37.5S (Fig. 8). This zone appears to cut off the Sue CQ mineralization.

The fault patterns observed in the Sue area are the result of a major EW compression event which affected the western flank of the Collins Bay Archean Dome and the adjacent Aphebian

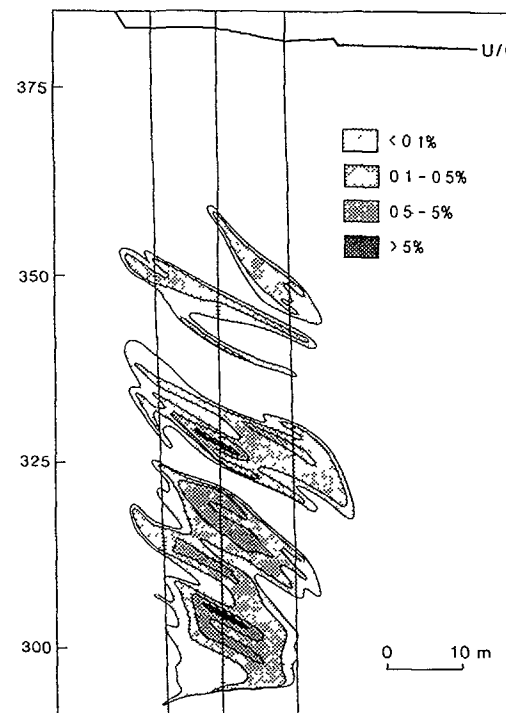


Fig 22. Sue CQ deposit. Cross section 7+87.5S. Ore outlines.

units. The predominant NS fault set appears to be the major structural control common to all the deposits. Contemporaneous NNE and NNW striking faulted trends cross cut the main set. They induced, inside large contiguous faulted blocks, important modifications in the structural settings which can be favourable to the development of mineralization.

GEOPHYSICAL SURVEYS

One of the first undertakings of Minatco, as new operator in 1985, was to prioritize exploration targets on the Wolly property, a property entirely blanketed by Athabasca sandstones. This was achieved by reassessing the large data base available and carrying out additional geological and geophysical work at the scale of the property.

Two models of mineralization were used in the process: the "unconformity type" and the "basement hosted, Eagle Point type" models. In both cases, the deposits are found close to "Archean" domes, in faulted and altered zones, associated with graphitic horizons in the Aphebian basement.

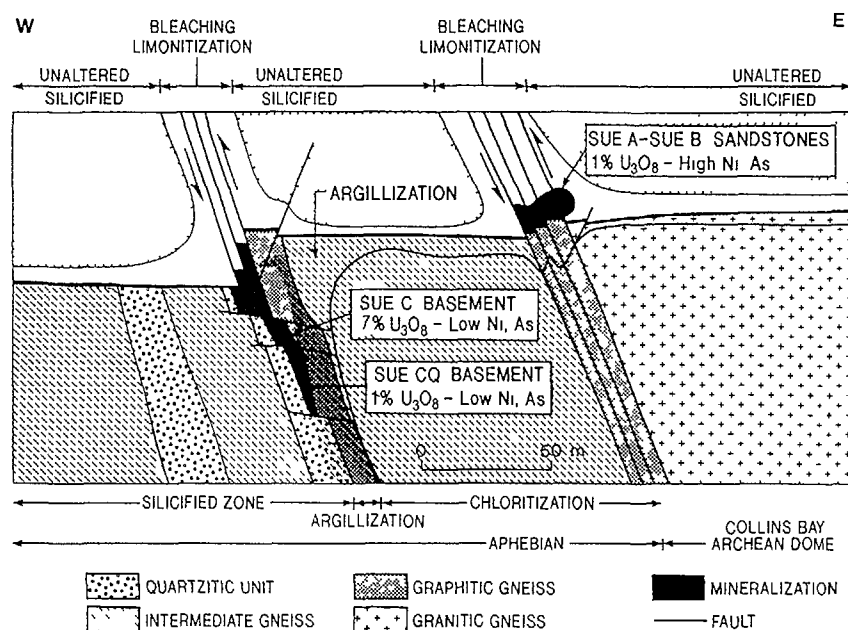


Fig 23 Schematic section across the Sue deposits

Because of the sandstone cover, geophysics is needed to outline such a favourable environment and the discoveries made on the property (McClean North 1979, McClean South 1980 and JEB 1982, Fig. 3), are essentially geophysical successes

By the end of 1986, several areas had been selected to carry out further exploration and prioritized. One of them was the Sue area, located along the western edge of the Collins Bay dome and 2.5 km east of the McClean Lake deposits

AIRBORNE GEOPHYSICS

A combination of airborne geophysical techniques usually allows to differentiate between different geological units and airborne surveys are well suited to provide rapidly and cost effectively a geological sketch of large expanses of land. In areas such as the Wolly project, information on the basement geology can be obtained from such surveys despite the fact that the basement rocks may lay under up to 200m of Athabasca sandstones. This is possible because the Athabasca sandstones are virtually magnetically transparent and because their usually high resistivity allows the EM systems to achieve depths of penetration well above average. Airborne techniques can thus be used to map basement lithologies according to their magnetic signature, their conductivity, or a combination of both.

Over the stretch of a decade the Wolly property had been covered by several airborne surveys including airborne spectrometer, magnetometer and EM surveys and in 1985 the available data was reviewed. Minatco concluded that the INPUT surveys provided an adequate EM coverage of the project but that it would be beneficial to resurvey the property to acquire high sensitivity magnetometer and gradiometer data. The survey flown in June 1986 by Aerodat with a helicopter borne system is characterized by a detailed grid (NS lines 100m apart), a high sampling rate (0.1s) and an accurate flight path recovery achieved through a combination of conventional techniques and radio navigation. This allows to correlate this data with other data acquired on the ground. VLF data was also recorded during the course of the survey.

By the end of 1986, the results from this new airborne magnetic and VLF survey, together with a new compilation and interpretation of the INPUT data were incorporated into the geological map of the property to outline the main basement lithologies and to indicate and determine the trends of the main graphitic horizons, the major faults and the areas of shallow fracturation in the sandstones.

In the southern sector of the project area, two prominent magnetic highs outline the northern edge of the McClean Lake dome and the west side of the Collins Bay dome (Fig 24). Both domes are flanked by well defined trends of INPUT anomalies. The anomalies, generally better defined around the Collins Bay dome because of a shallower basement (70 to 100m) outline mainly graphite horizons in the Apebian basement although there are indications that some of them may be related to faulting.

The NNE striking trend of INPUT anomalies located in a magnetic low along the southwest edge of the Collins Bay dome was one of the first regional signatures of the Sue trend.

Evidence of faulting is found in both the magnetic and VLF airborne data. The most obvious directions of faulting are:

- NE-SW (ubiquitous VLF and mag signatures)
- N70°E (the direction of the McClean Lake north mineralized trend and of the Seal Lake fault)
- EW (the direction of the McClean south trend, best seen as a straight, linear magnetic gradient marking the edge of the McClean dome)
- NW-SE (indicated by a magnetic low extending into the Collins Bay dome to the southeast of the airborne magnetometer survey)

Also, previous geological data indicated that the Sue graphitic conductor itself was associated with a major fault zone.

On the basis of the airborne geophysical responses and of other data gathered by Canadian Occidental Petroleum Limited as early as 1979, the Sue trend was designated as one of the top exploration targets on the property.

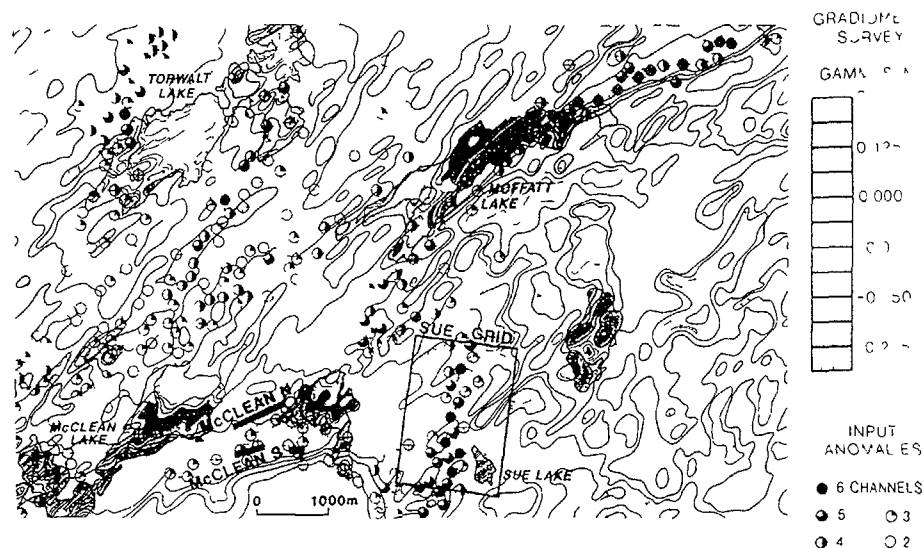


Fig 24 Airborne geophysical surveys on the south sector of the Wolly project

GROUND FOLLOW UP

GENERAL APPROACH

To better relate with Minatco's approach on the Wolly property, some basic principles will be restated only in a general way since several papers have already dealt in detail with the various geophysical responses encountered in the Athabasca basin (see references)

When exploring an area for the first time, Minatco relies on electromagnetic data and on the results of its high sensitivity airborne magnetometer survey to define potential drill targets. Magnetic data provides information about the geological environment while the electromagnetic methods outline conductive features within this geological environment.

The electromagnetic systems which are used fall into two categories: the low frequency and the high frequency systems. On the Wolly Project the Max-Min is operated at low frequencies and the VLF EM is the high frequency system.

The difference between the manner in which low and high frequency electromagnetic systems respond to conductive features is so distinct that a discrimination can be made which allows the mapping of two elements within the earth: the graphitic basement conductors best mapped by the low frequency systems and the non graphitic to weakly graphitic structural features such as faults and fractures best mapped by the high frequency systems.

The combined use of such EM systems to define drill targets has a proven record on the property. After an extensive test program carried out after the discovery of the McClean deposits, Canadian Occidental Petroleum Limited had observed that there is at McClean "an excellent correlation between uranium mineralization and locations where shallow VLF conductors coincide with, or cross-cut, interpreted horizontal loop basement graphitic conductors (Jagodits et al., 1986)". Subsequent exploration largely based on this observation resulted in the JEB discovery in 1982.

The Sue Grid

The ground follow up by Minatco in the Sue area started in the winter of 1987/1988 when the Sue grid was established by Coureur des Bois Ltd. in preparation for the ground geophysical surveys. The grid base line, cut in a N12°E direction on top of the INPUT conductive trend is 2.9 km long. Cross lines are 100m apart and the spacing between stations is 25m. They extend 750m on either side of the baseline and straddle the Archean Aphebian contact on the western flank of the Collins Bay dome (Fig. 5).

VLF EM AND HORIZONTAL LOOP EM SURVEYS

Data Collection

In March 1988, Walker Exploration Limited, contracted by Minatco Ltd. to carry out the geophysical surveys, acquired VLF EM and horizontal loop EM data across the Sue grid. The Horizontal Loop EM measurements taken with a Max-Min unit manufactured by Apex Parametrics Ltd. were digitally recorded with a KTP-84 data logger built by Rautaruukki Oy. The instrument was operated at two frequencies (444Hz and 1777Hz) with a coil spacing of 250 metres. The VLF EM survey was performed with a VLF EM16 built by Geonics Limited. The VLF primary field was generated by the station NLK located at Seattle (Washington, USA) which operates at 24.8 kHz.

Data Presentation

Data for both surveys was processed and plotted in the field using personal computers and peripheral equipment such as printers and plotters.

While the Max Min in-phase and out-of-phase responses for each frequency are presented on profile form, the VLF data is presented in two formats: in profile form and in a Fraser filtered contour form. Originally developed to reduce the background noise and to enhance the interpretation of very broad anomalies or of anomalies with a weak amplitude, the Fraser filter is a manipulation of the data that transforms the inflexion points of the in phase profiles into positive peak anomalies which indicate the location of the conductors. The contoured format of the VLF data is particularly useful in correlation with magnetic data that is contoured as well.

Interpretation

The interpretation of the well defined Sue information was straightforward. The Max Min conductor was a narrow, well confined graphitic conductor, dipping grid east and following

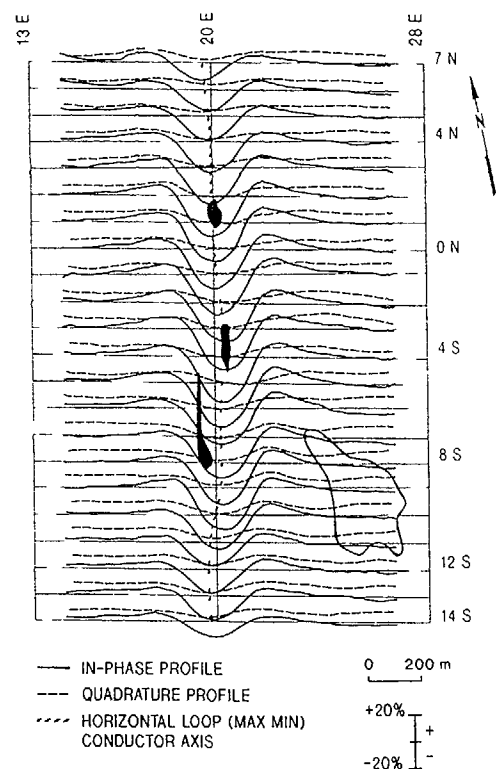


Fig 25 Sue grid Horizontal loop EM Survey (Max-Min)

roughly the baseline from line 14+00S to line 7+00N. Depths estimates to the top of the conductor were in the 73m to 108m range, the largest depths being calculated to the north and south of the grid. Interpreted dip values varied from 70° to 80° along most of the trend with the exception of lines 5+00S to 7+00S where lower values (55° to 70°) were found. Calculated conductivity thickness values were in excess of 40 mhos between line 9+00S and 4+00N. In the north the conductivity thickness values were significantly reduced but remained above 20 mhos, while south of line 9+00S the conductivity thickness dropped to about 15 mhos. There, this phenomenon is accompanied by a bend in the strike causing a shift in the conductor axis approximately 50 metres to the west (Fig 25).

Figure 26a and Figure 26b indicate a strong correlation between the main graphitic conductor as determined by the horizontal loop EM and the VLF. The strength of the VLF anomalies increases locally along the trend, particularly from line 5+00S to line 3+00S and on lines 1+00N and 2+00N. Northeast trends in both VLF components over the area east of the main conductor

are interpreted as the signatures of ubiquitous NNE faulting. It is worth noting that the 'bull's-eye' shaped VLF anomalies along the main conductive trend develop in the vicinity of the intersection of the two trends.

The Sue A and Sue B Discoveries

With all the geophysical signatures sought after outlined on the Sue grid, the Sue graphitic conductor was tested by drilling in March/April 1988. Hole CS5 drilled on line 3+00S, where one of the 'bull's-eye' VLF anomalies had been outlined, intersected 12.5m of ore grade mineralization. Sue A had been discovered.

Another VLF 'bull's-eye' on line 1+00N and 2+00N remained untested because a three hole fence had been drilled by the previous operator on the south edge of it. The geophysical signatures were however too similar to the anomalies outlining Sue A to be left alone and, in the summer of 1988, hole CS44 was collared on top of the HLEM conductor axis, in the centre of the VLF anomaly. Two zones with ore grade mineralization were intersected. Sue B had been found.

IP/RESISTIVITY SURVEY - SUE C AND SUE CQ

By the end of 1988, two holes drilled south of Sue A to test the Max Min conductor had intersected 0.154% U_3O_8 over 1.3m and 714ppm U_3O_8 over one metre. Both zones were intersected in the basement.

The low-grade mineralized holes had been spotted at the intersection of the basement graphitic conductor with a NNE trending VLF anomaly interpreted as being associated to faulting.

In addition another hole had been drilled off the HLEM conductor, 50m west of its axis, to test the VLF anomaly and gather geological data. Large amounts of graphite were intersected in this hole, a rather disturbing finding given the results of the HLEM survey.

Because no new information could be extracted from the existing geophysical data, it was decided to carry out an IP/Resistivity survey to add to the knowledge of the area. The objective was to map the basement rocks by detecting the changes in apparent resistivities, possibly combined with anomalous IP effects, associated with the changes in the basement geology.

The Survey

To carry out this electrical mapping preference was given to the modified Schlumberger array. With such an array, measurements are made within a rectangle whose dimensions are respectively one half and one third of the length of the transmitter dipole. The width of the rectangle is parallel to the transmitter dipole and the centre of the rectangle is located halfway between the current electrodes. For a given rectangle, the current electrodes are kept at the same location while the potential electrodes are moved along the profiles, parallel to the transmitter dipole.

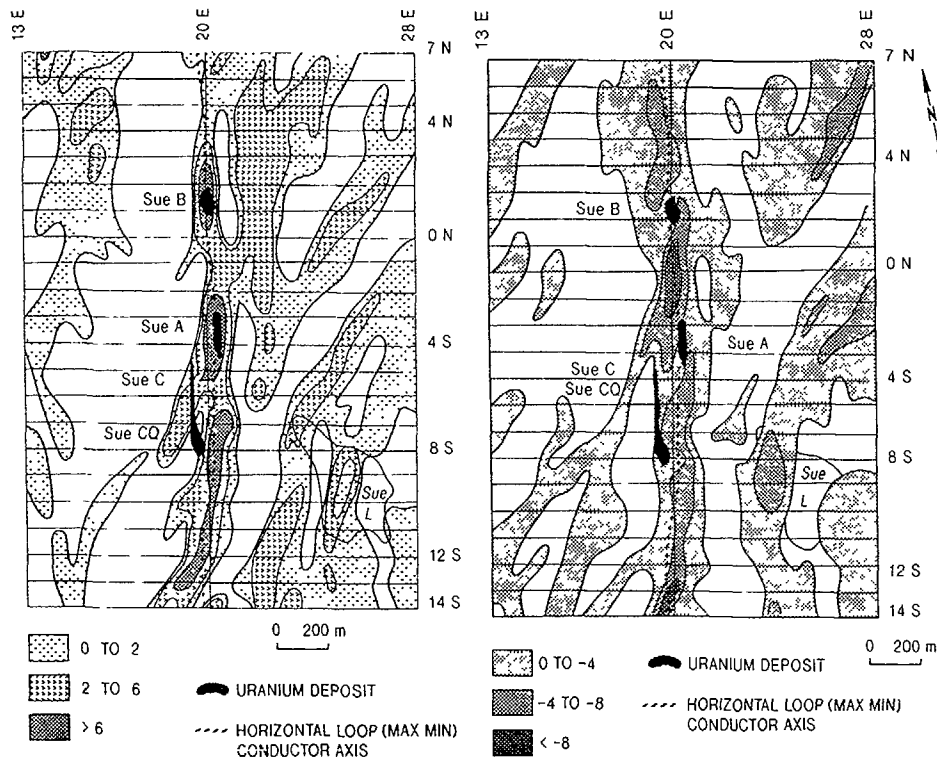


Fig 26. Sue grid. VLF EM survey. a) inphase, b) quadrature.

Prior to the survey, modelling and field tests had shown that accurate and meaningful measurements would be made using a 1.2km long transmitter dipole and a 12.5m spacing between the potential electrodes. With this array, readings could be taken within rectangles 600m long and 400m wide. Five rectangles straddling the Max-Min conductor were surveyed to map a strip of basement 2.6km long and extending 200m on either side of the HLEM conductor axis. Data was acquired at stations 12.5m apart along cross-lines established every 50m. Pacific Geophysical Ltd. carried out the survey between August 14 and September 2, 1989, with an EDA model IP 6 six channel time domain IP and resistivity receiver, manufactured by the Bureau de Recherches Géologiques et Minières, and a Huntex Mark 4 transmitter.

The Sue C Discovery

While the IP/resistivity survey was in progress, drilling was following the mineralization found at the end of 1988. The trend of the mineralization being NNE, parallel to the VLF anomaly,

the drill was moving away from the Max-Min conductor. On August 18, hole S209 drilled on line 6+50S, 75m west of the HLEM conductor axis intersected 18m of high grade mineralization. At the same time, interesting results were delivered by the IP/resistivity survey. It was decided to complete the geophysical survey in progress and to reassess the data available before drilling the next hole.

The Resistivity Map

Although IP measurements gave definite responses in some areas, the most interesting results, in terms of significance for the ongoing exploration program, were derived from the resistivity data (Fig. 27).

Apparent resistivities range from up to 12,000 ohm-metres down to less than 400 ohm-metres across the survey area. From west to east a zone of high resistivities maps the granite gneiss related to the Collins Bay dome. A sharp gradient marks the contact between these rocks and the more conductive graphitic Aphebian gneisses. The conductor outlined by the HLEM survey follows closely this contact. About 250m further west, the apparent resistivities rise again outlining different Aphebian lithologies. All the different directions of faulting interpreted from the airborne data are represented at the scale of the resistivity survey but the most important features of the resistivity map are the signatures of the orebodies already found and the location of the new discovery hole.

Sue A is located along the gradient mapping the contact between the granite gneisses and the graphitic metapelites and sits at the north end of an oval shaped zone of low resistivities within which even more conductive trends can be traced.

Sue B is readily outlined by a prominent resistivity low coincident with the axis of the HLEM conductor. Another outstanding signature is the well defined NNE trend of low resistivities which strikes into the Sue B area. This is the major fault zone affecting both the Athabasca sandstones and the basement rocks also indicated by the VLF survey.

Hole S209 is located within the oval shaped resistivity low already mentioned, at the intersection of two trends. One striking NE and drilled when chasing the mineralization identified south of Sue A, the other striking almost true north.

The Sue C deposit was to be defined by drilling along this N-S resistivity low.

The Sue CQ mineralization is found along the same trend in an area affected by another set of faults. The well defined resistivity low extending deeply into the high resistivities characterizing the granito-gneisses is a reflection of such faults.

At last, the somewhat higher resistivities flanking the resistivity lows marking the Sue C deposits seem to correspond to quartz enriched basement rocks identified by drilling. More could be said about the wealth of information extracted from this survey. It will only be observed that systematic drilling of the area displaying the lowest resistivities would have led to the discovery of all the orebodies found so far in the Sue area.

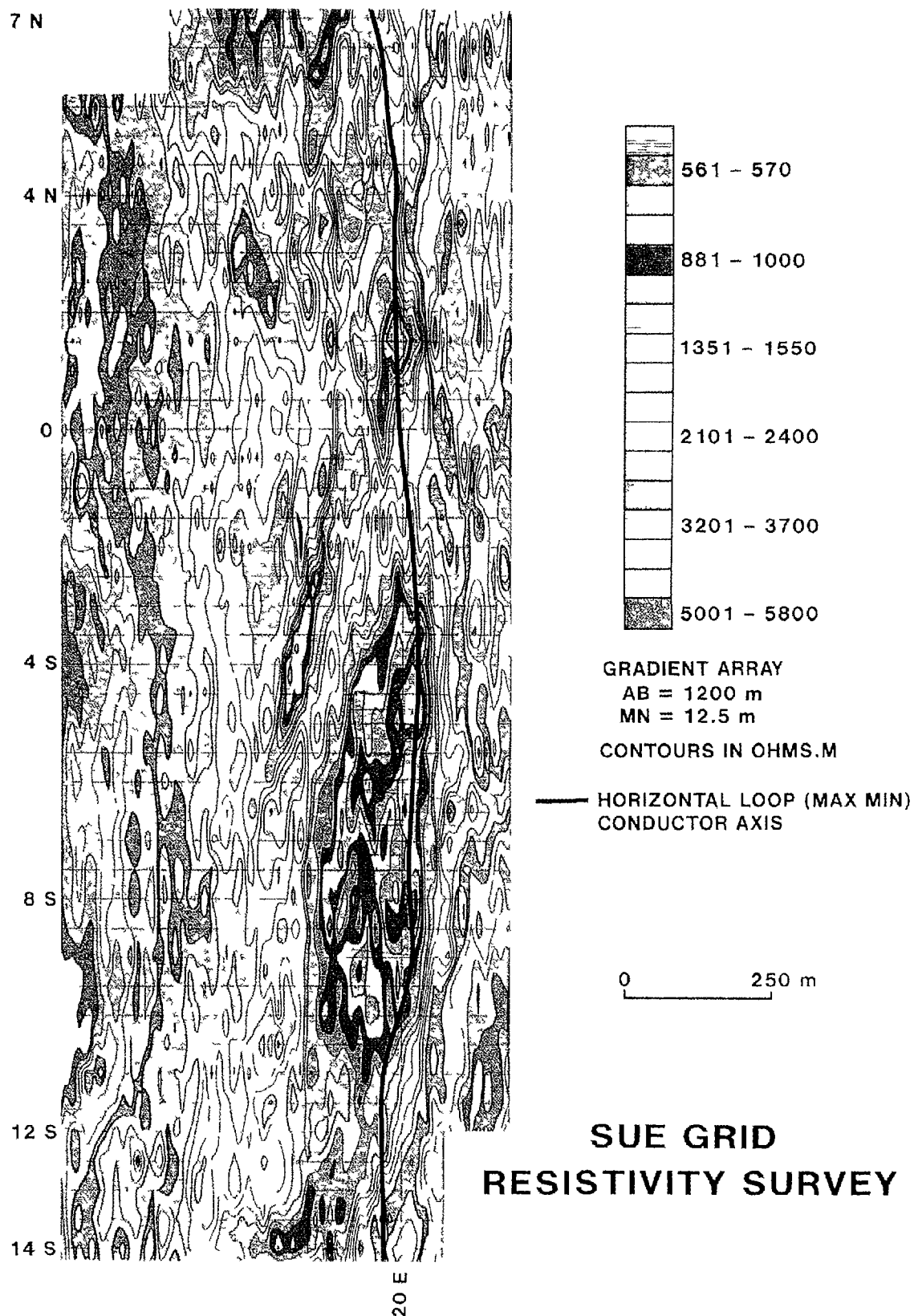


Fig 27 Sue grid Resistivity survey

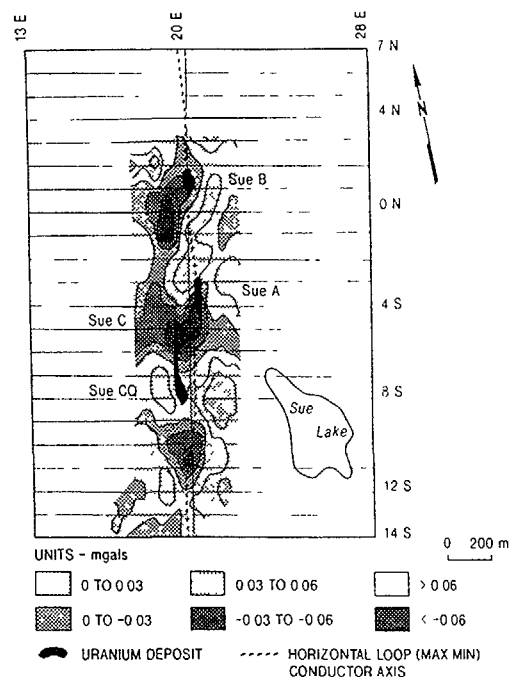


Fig 28. Sue grid. Residual gravity map.

ANOTHER SIGNATURE - THE GRAVITY SURVEY

Because the Sue C orebody lies a few tens of metres off the main Max-Min conductor axis which was so far the priority drill target on the Wolly project, it was felt that understanding and modelling the new orebody was a necessity. In 1990, the resistivity 1989 survey was extended and it was decided to check if the various orebodies along the Sue trend could generate measurable gravity anomalies. Because theoretical models indicated that the anomalies were likely to be in the order of a few hundredth of a milligal, it was difficult to pass a judgement on whether gravity could be added to the techniques already applied.

The Survey

The gravity survey was contracted to Diamond E Explorations Ltd. which used a Lacoste Romberg model G gravity meter for the acquisition of the data. The gravity stations were surveyed by Coastlines Ltd.

Because the anomalies to detect were expected to be very small the data had to be acquired with great care and techniques similar to those employed in civil engineering were applied.

The gravity stations were 50m apart along each profile, the profiles being cut at 50m spacing. A total of 792 independent gravity measurements were taken to complete a survey of 390 stations and the accuracy of the survey is better than 0.02 milligal.

Field data was reduced following standard procedures to arrive to Bouguer and residual anomaly maps. The data presented in figure 28 is the residual anomaly map obtained after removal of the gradient resulting from the density contrasts between the main lithologies.

The Results

As anticipated the amplitude of the residual anomalies are small and generally lower than 0.1 milligal. The anomalies are nevertheless definite and well defined due to the overall accuracy of the survey. Based on the geological knowledge of the area and the other geophysical surveys it appears that the significant anomalies are negative, suggesting that the survey is more sensitive to the alteration surrounding the orebodies than to an increase in metal content.

Although Sue B is located in a gravity low the amplitude of the anomaly is too minute to state that the orebody has definitely been seen.

To the south west of Sue B and striking into Sue B in a NNE direction a trend of low gravity values is outlined. This trend coincides exactly with a major resistivity low already documented as fault related. Along this trend a more localized gravity low is outlined which is the strongest anomaly in the survey area.

Sue A lies on a NNE gravity low trend and drilling its intersection with the Max-Min conductor axis would have led to the discovery of this orebody.

To the southwest the same trend abuts against a gravity high and a gravity low develops in a NS direction. This is the direction of the Sue C and Sue CQ orebodies. The gravity anomaly located at the intersection of the two trends lies on top of part of the C orebody. The correlation between gravity and resistivity data is here again striking.

CONCLUSIONS

The Sue deposits belong to the same trend and are clustered along the western edge of the Collins Bay Archean Dome. These recent discoveries emphasize the outstanding potential of the area.

They display many of the mineralization characteristics that are published on the Athabasca deposits (Fig. 23).

- Mineralization straddling the unconformity in a pencil shaped form (Sue A, B).
- Mineralization as a single tabular vein (Sue C) or multiple lenses (Sue CQ) in the metapelitic gneisses.

- Mineralization perched high up in the sandstones, 8m below surface, with no surficial expression (Sue B)
- All four deposits are genetically related to one or more graphitic metapelitic units ranging in graphite content from 1 to 70%

The Sue deposits occur in the sandstones or basement which have been considerably transformed in various clay minerals during a widespread hydrothermal event

Monometallic mineralization (Sue C) with uranium oxides associated with subordinate Mo and V to polymetallic ore (Sue A and Sue B) with uranium oxides paragenetically related to strong Ni-Co arsenides and sulpharsenides. The mineralization phases are well illustrated by massive uraninite (Sue C), nodules of pitchblende (Sue A, Sue CQ, Sue B) and coffinite (Sue B)

The Sue C and CQ deposits are characterized by a sharp contrast between a barren silicified compartment and the mineralized-argillized eastern block. The western block is the locus of strong silicification processes that have transformed the original textures into a massive quartz rich unit

Argillization, mineralization and silicification are synchronous events associated with the main reverse fault as part of a continuous tectonic-hydrothermal process

In general all Sue deposits are structurally controlled by reverse faults originating in and paralleling the graphitic metasediments. These faults control the lateral extent of the mineralization and create horsts and offsets of the unconformity surface. The strike length is controlled in part by a dense network of NE-SW and NW-SE faults, one of them linking Sue A to the Sue C deposit. In the basement hosted deposits, Sue C and Sue CQ, the distribution of the mineralization is controlled by flat lying shears which are vertically stacking the ore lenses.

The duplication of these structures at depth controls the down-dip extension of the mineralization and is the main ingredient for potential at depth

The Sue deposits are an excellent example of VLF EM mapping structures that rise to surface through the Athabasca sandstones

The discoveries of Sue A and Sue B are the direct result of drilling areas where these structures intersect graphitic basement conductors outlined by horizontal loop EM techniques

Electrical mapping of the basement rocks yields information regarding their nature and complements the VLF data when identifying fault patterns. The trend of the Sue C orebody and Sue CQ have been found by drilling resistivity lows which map graphitic horizons and alteration zones not detected by the HLEM surveys

To summarize,

The main features of interest about the Sue deposits are

- high to very high grade U_3O_8 mineralization, 0.9% to 7%

- location on a topographic high, with no swamps or muskegs
- shallow depth of the unconformity at 80m, where the bulk of mineralization occurs to date
- open-pit mining methods
- enormous potential at depth and along strike on untested HLEM, resistivity and gravity targets

ACKNOWLEDGEMENTS

Two of the most important factors to a successful mineral exploration effort are an adequate budget over a reasonable length of time and certainly the management of TOTAL has provided both to Minatco over the last five years in a very 'positive' and 'timely' manner

This paper is not only the recognition of the new Sue deposits but is also a 'reminder' of the efforts made since 1985 by the staff and management of Minatco under the leadership of J.P. Nicolet. Acknowledgement should be given to the former operator Canadian Occidental Petroleum Ltd. and their partner Inco Limited who explored the property from 1974 to 1985. The staff of both companies provided encouragement and support during the joint venture period (1985-1990).

Thanks to the Wolly Exploration team, past and present, especially R. Aumaitre, M. Grandprat, J. Harper, E. McGowan, J. Pukas, R. Sol and K. Wheatley for their dedication in the field

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FIELD TEST FOR IN SITU LEACH MINING OF URANIUM IN PAKISTAN

M.Y. MOGHAL

Atomic Energy Minerals Centre,
Lahore, Pakistan

Abstract

A number of small sized uranium ore bodies have been delineated in Siwalik group of rocks in Bannu District in the northwestern part of Pakistan. The host sandstone is very soft and unconsolidated which makes it difficult to mine the ore body by conventional mining methods.

The Kubul Khel ore body meets most of the suitability criteria for insitu leach mining. The results of laboratory leaching tests on the ore body were also encouraging. A field test for insitu leach mining of uranium in Pakistan has therefore been undertaken, using a mixture of sodium carbonate and sodium bicarbonate as lixiviant. The amenability of the uranium ore body to carbonate leaching has been proved and the leaching efficiency increases with the use of hydrogen peroxide as oxidant, alongwith the lixiviant. The overall cost of production can be significantly reduced by substituting hydrogen peroxide with oxygen as lixiviant.

1. INTRODUCTION

1.1 General

Insitu leach mining or solution mining, by way of definition, is that method, in which the ore mineral in the original geological setting is preferentially leached from the host rock by the use of specific leach

solutions, and the metal values are recovered. This technology virtually eliminates materials handling of the ore including crushing, grinding and hauling. It also eliminates the solid waste. Other advantages of this technology include minimum surface disturbance, fewer workers and safer working conditions. It also requires relatively lesser capital cost and shorter lead time for production in comparison to the conventional mining methods.

1.2 Technical Requirements for Insitu leach Mining

The viability of the insitu leach mining technique is subject to a number of stringent conditions. The ore body is considered to be amenable to insitu leach mining only if it has most of the following characteristics:-

- i, Adequate permeability
- ii, Adequate porosity
- iii, Located below the natural water table at reasonable depth
- iv, Good thickness
- v, Confined within the impervious layers both above and below
- vi, Ore mineral is amenable to chemical leaching
- vii, Shallow dip

Thus where the ore body meets the above criteria and practical difficulties are faced with conventional mining methods, the applicability of insitu leach mining method can be tested.

1.3 Justification for Insitu leach test in Pakistan.

A number of small sized uranium ore bodies have been delineated in Middle Siwalik group of rocks in Bannu District in the north western part of Pakistan. The host sandstone is friable, very poorly cemented

and lacks compressive and shear strength. The friable and unconsolidated nature of the host rock posed ground control problems during underground exploratory mining. Only "Cut and Fill" mining method was found to be applicable, but even that was slow, hazardous and expensive.

On the other hand, the host rock of the uranium ore body bears good porosity and permeability, the thickness of ore is good, the ore body is located below water table and has thicknesses of 2 to 15 meters averaging more than 7 meters and the uranium is easily leachable. Thus it meets most of the critical suitability criteria for insitu leach mining method. Although the dip of the strata is upto 30° and the ore body is not confined at the top, still in view of the other favourable factors, it was considered justified to conduct a field test to determine the applicability of this technique particularly because no other method was found suitable.

2. LOCATION OF TEST SITE

Fluviatile sediments of Siwaliks group belonging to Middle Miocene to Pliocene age are considered to be a potential host for uranium in Pakistan. These rocks extend from Kashmir in the east through Potwar Plateau and Bannu Basin to Sulaiman Range in the west and then continue southwards grading into its shallow marine equivalents (Fig.1). The test has been carried out on one of the uranium ore bodies hosted by Siwalik group of rocks. It is located in Surghar Range along the eastern flank of Bannu Basin, in north western part of Pakistan (Fig.2). The ore body is named Kubul Khel uranium ore body after the small village nearby. The area exhibits rolling topography with low altitude strike ridges. Climatically the area is arid with annual rain fall of less than 150mm. The day temperatures in summer can rise up to 50°C .

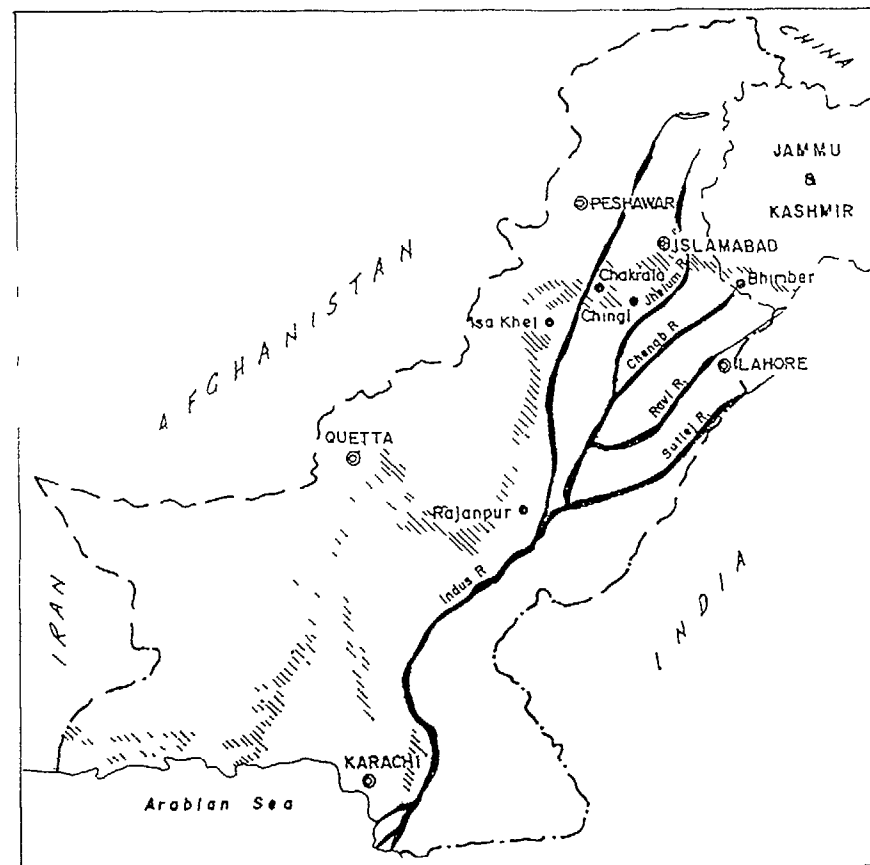


FIG 1 Map of Pakistan showing the Siwalik test sites

3. GEOLOGY

3.1 Dhok Pathan Formation

The uranium ore bodies are located in Dhok Pathan formation which forms the upper middle part of Siwalik group of rocks. It consists of alternating sand shale sequence, deposited in a cyclic fashion. In the vicinity of test area the sandstone beds are 40 to 50 meters thick, interbedded shales varying in thickness from 10 to 15 meters. The sandstones

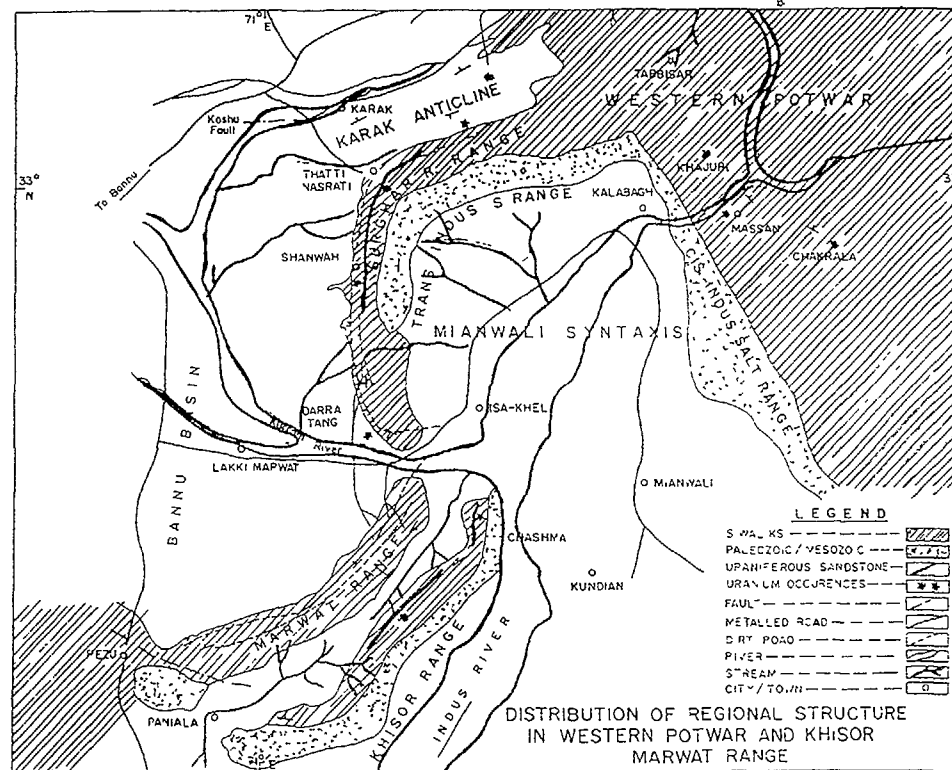


Figure 2

are grey, soft and friable. The dull brown and grey shales are silty in nature. The shales often contain varied amount of volcanic material in the form of bentonite and bedded ash with glass shreds.

3.2 Host Sandstone

The sandstone hosting the uranium ore body is also named after Kubul Khel village as Kubul Khel sandstone which is stratigraphically located near the top of Dhok Pathan Formation. It is commonly whitish grey, loose, poorly cemented and medium to coarse grained. The mineral

assemblage of the sandstone is typical of sublithic to subarkosic arenites, the major constituents being quartz, mica, feldspars, amphiboles and rock fragments. Some carbonaceous matter, micro fine humic material and diagenetic pyrite is also present in some of the core samples.

3.3 Characteristics of Ore Body

The uranium ore body is associated with the sandstone-shale contact. The ore body is of tape like configuration following the trace of water table with the underlying shale. Mostly the ore body follows the boundary of the sandstone with the shale, but at places, it tends to develop another leg parallel to the present day water table attaining maximum thickness near the confluence of both the legs (Fig.3). The minerals present in the ore body are a combination of uraninite (UO_2) and coffinite ($USiO_4$) $(OH)_4$, which occur as pore fillings. Pitchblende also occurs as micro fine globules (1).

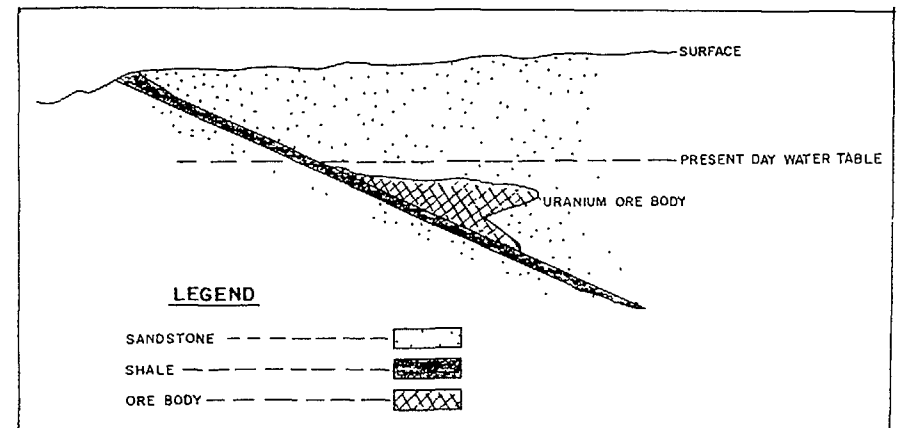


FIG. 3. Sketch map showing the shape of the Qubul Khel ore body.

T A B L E - I

RESULTS OF URANIUM LEACHING TESTS

Lixiviant	Concentration grams/litre	RECOVERY PERCENT ON LEACH BASIS		
		in first stage (42 hours)	in second stage (54 hours)	Accumulated (96 hours)
1. $(\text{NH}_4)_2\text{CO}_3$	5	59	25	84
2. $(\text{NH}_4)_2\text{CO}_3$	10	74	21	95
3. $(\text{NH}_4)_2\text{CO}_3$	15	80	18	98
4. $\text{Na}_2\text{CO}_3 +$ Na HCO_3	3.5 1.5	42	21	63
5. $\text{Na}_2\text{CO}_3 +$ Na HCO_3	10.0 4.5	55	28	83
6. H_2SO_4	5	55	30	85

4. LABORATORY STUDIES

4.1 Laboratory Leaching Tests

In order to select a suitable lixiviant for the field test, a number of shake flask leach tests on the ore samples and several column packed tests were conducted in the laboratory. Lixiviants that are normally usef for insitu leach mining of uranium, include $(\text{NH}_4)_2\text{CO}_3 + \text{NH}_4\text{HCO}_3$, $\text{Na}_2\text{CO}_3 + \text{NaHCO}_3$ and H_2SO_4 . Different concentrations of ammonium carbonate, sodium carbonate and bicarbonate and sulphuric acid were therefore used for leaching out the uranium from the ore. The results of two stage leaching tests carried out with different lixiviants are presented in table I. It can be seen from the table that while the higher concentration of all the lixiviants affects the leaching positively, the recovery of uranium also increases with increase in leaching time.

The results of leaching in column packed tests were similar.

However it was noted that there was some reduction in the permeability of sandstone when $\text{Na}_2\text{CO}_3 + \text{NaHCO}_3$ was used as lixiviant, while with ammonium carbonate, there was no change in permeability.

4.2 Adsorption and Elution Test

Laboratory tests were also conducted to select suitable resin for uranium adsorption. Various uranium loading and elution tests at different pH of leach solution were conducted to find out the loading and elution curve using highly basic anion exchange resin SBR-P - a product of M/s. Dowex Chemicals.

The results revealed that at pH 9-10, about 60 grams of uranium could be loaded on one litre of the said resin.

5. FIELD PREPARATION

Subsequent to the injection of the lixiviant into uranium ore body, several chemical reactions take place underground and the chemical and geological parameters effect the flow rates and reaction rates. As the physical and chemical characteristics of any two deposits cannot be similar, the rate of uranium production therefore varies considerably from one ore deposit to another under similar operating conditions. It is therefore difficult to predict the performance of any insitu leach mining operation, without actually carrying out a field test at a particular place. To examine the applicability of this technique to Kubul Khel ore body, and to check the results of laboratory one well field pattern was prepared.

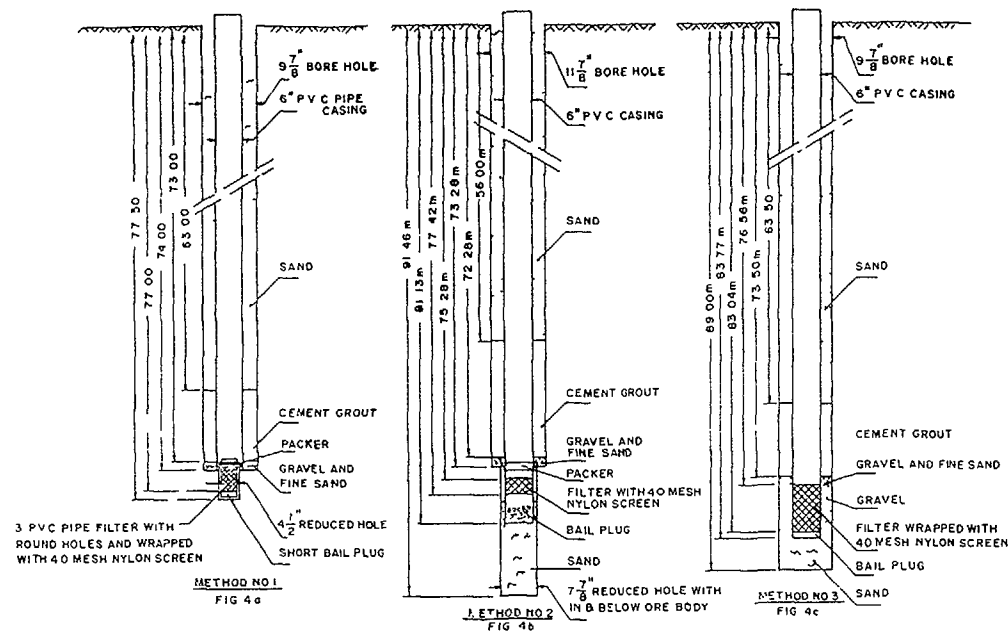


FIG 4 Construction pattern of injection well

5.1 Well Field Pattern

From amongst the various well field patterns generally employed for insitu leach mining, five spot pattern was adopted for the test. It comprised of four injection wells on the corners of a square with the production well in the centre. The injection holes on one side of the square were located 19 meters apart. One monitor well to monitor any excursion of the lixiviant or leach solution was also drilled on the down dip direction of the well field outside the leaching area.

5.1.1 Well Construction

Well construction is the most important part of the insitu leach operation. As the strata is very soft and easily caves in the hole, the

normally used cement basket method has not been used for sealing the drilling hole. Rather simpler techniques have been used.

In one of the methods (fig.4^a) the well consists of two segments. In the top portion extending from surface to 1 meter above the top of ore body, $9\frac{7}{8}$ " dia hole is drilled, and $4\frac{1}{2}$ " dia hole is drilled through the ore zone. In the top segment a string of high quality 6" dia PVC casing was emplaced, while PVC pipe of 3" dia of appropriate length, sealed at the bottom and selectively screened, is fitted with a PVC packer at the top and lowered into hole of $4\frac{1}{2}$ " dia, positioning the screened portion against the ore zone.

With slight variation, the well in the second method (fig.4b), also consists of two segments. In this method in the top portion extending to 1 meter above the top of ore body, the hole is of $11\frac{7}{8}$ " dia, while hole of $7\frac{7}{8}$ " is drilled through the ore body. A high quality PVC pipe of 6" dia, of appropriate length is sealed at the bottom and is fitted with filter and a PVC packer, and lowered into the hole, positioning the filtered portion against the ore zone and resting the packer at the top of $7\frac{7}{8}$ " dia hole.

In both the methods fine sand is filled in the annular space above the packer for about 1 meter length, and then sealed with cement slurry upto one meter above the water table. The remaining annular space upto the top is filled with fine sand.

In the third method (fig.4c) still simpler technique is used. A hole of $9\frac{7}{8}$ " diameter is drilled upto the bottom and PVC pipe of 6" dia, with screens equal to the thickness of ore zone, is lowered in the hole, adjusting it so that the screened portion faces the ore zone. The 2" wide annular space between the pipe and hole wall below the ore zone is filled with fine sand, and then fine gravel is packed in the annular space facing the ore zone. About one meter thick layer of sand is filled on top of gravel and later like in the first two methods, cement slurry is filled upto 1 meter above the water table. The annular space above the cement slurry is filled with sand.

In the first two methods/^{of} well construction, the filter is in direct contact with the hole wall, while in the third method, gravel is packed between the filter and the hole wall. It has been noticed that the holes without gravel shrouding, suck in sand with water, while those with gravel shrouding, pump out clean water. Therefore the technique

using gravel shrouding is considered to be better than the other techniques.

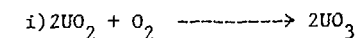
6. LEACHING OPERATION

After all the holes of the well field were completed, stainless steel submersible pump was lowered in the production hole, and pumping started. Lixiviant comprising of 10g/l Na_2CO_3 and 5g/l NaHCO_3 was injected keeping the total injected volume slightly less than the volume pumped out so as to ensure the maintenance of a positive hydraulic gradient to the production well. The pH of the water from the production well started increasing after 7 days. Simultaneously the uranium values started picking up and stabilized at 25 ppm(fig.5). The uranium pregnant solution was recovered and collected in a surge tank at the well site and pumped to ion exchange recovery plant. This process is schematically shown in fig.6.

6.1 Use of Oxidant

Oxidants that normally have been used to convert insoluble U^{+4} to soluble U^{+6} , include oxygen, hydrogen peroxide and sodium chlorate(2). The cost advantages and disadvantages of the oxidants provide only a general guide for oxidant selection and thorough laboratory and field testing is necessary to determine the suitability of an oxidant for a particular deposit.

Following reactions take place on the addition of oxidant alongwith the lixiviant:



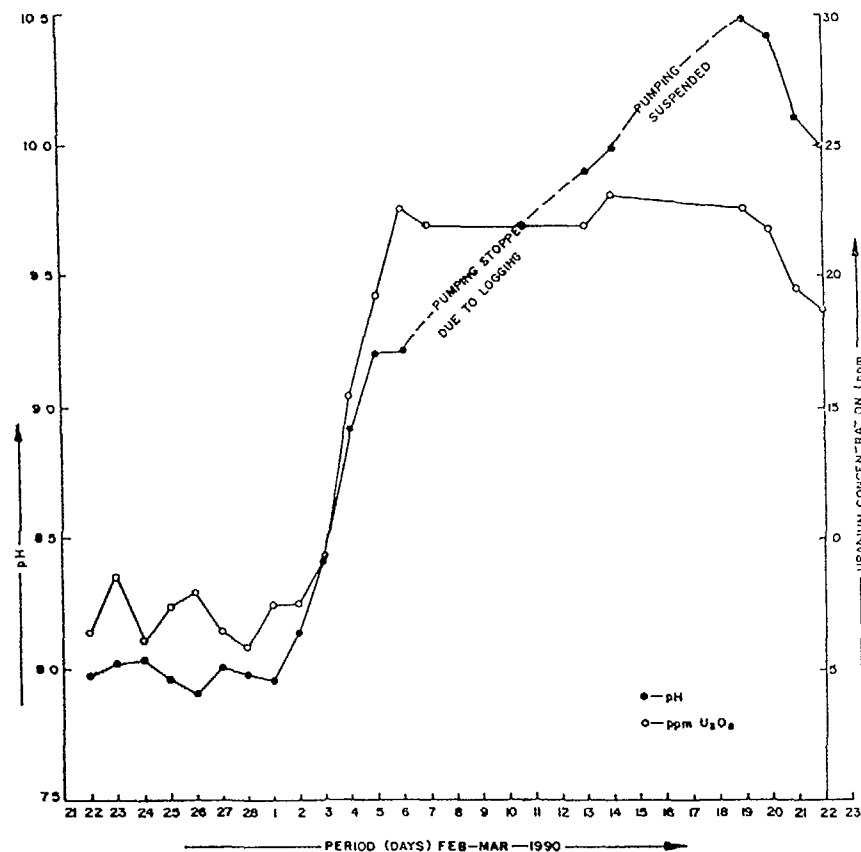


FIG 5 Time versus pH and uranium concentration during first month of leaching

The uranyle tricarbonat complex ion is soluble in water and can be recovered.

6.1.1. H_2O_2 as Oxidant

Amongst the oxidants mentioned above H_2O_2 was selected because of no complexing ion and easier handling. Initially H_2O_2 was injected along with the lixiviant at a rate of 1.5 g/l. The concentration of uranium which had stabilized at 25 ppm, subsequently increased to 65 ppm.

When the injection of H_2O_2 was stopped, the uranium values in the leach liquor started coming down again and within a period of two months decreased to 30 ppm, i.e. almost to the level from which it had started. The addition of H_2O_2 was resumed @ 0.6 g/l and then successively increased to 3 g/l. The uranium values in pregnant solution, subsequently increased to a maximum value of about 160 ppm. The enhancement of uranium concentration in the leach liquor, due to addition of H_2O_2 is shown in fig.7.

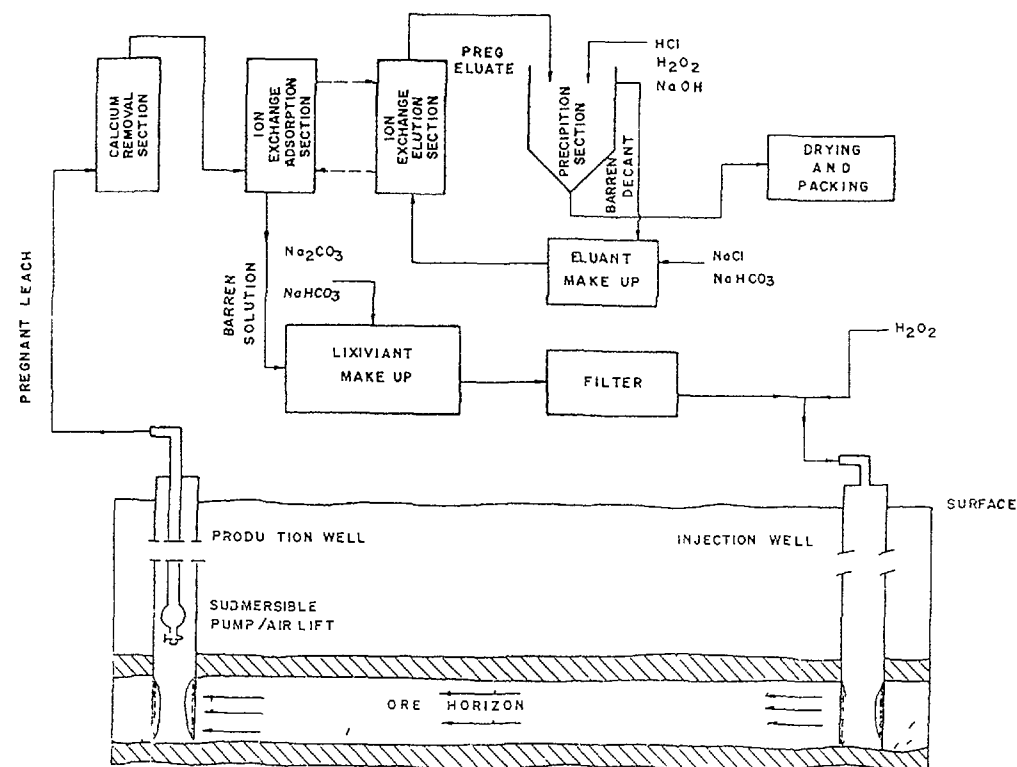


FIG 6 Schematic uranium leaching and recovery system

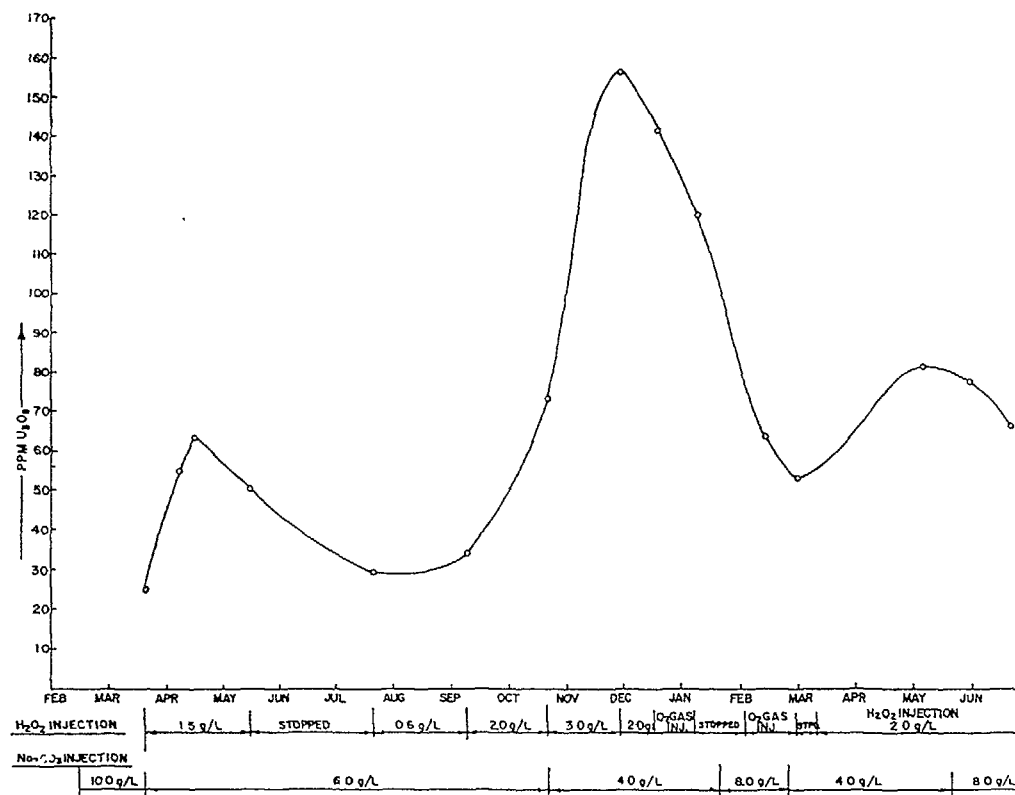


FIG. 7. Effect of H₂O₂ on uranium concentration.

At enhanced rate of leaching, the precipitation of calcium carbonate considerably increased, which caused frequent clogging of pump and the same had to be repeatedly cleaned with nitric acid to maintain the required pumping rate. Ultimately, however, the mode of pumping was changed to airlift technique using the Airman compresor PDR 600. This resulted in the decrease of production flow rate from 6 to 2.5 M³ per hour. The rate of injection was adjusted accordingly.

6.1.2 Oxygen as Oxidant

H₂O₂ is quite expensive, hence attempts were made to replace H₂O₂ with oxygen. The oxygen was injected at the bottom of the hole under almost 20 metre column of water, but the required quantity still could not be injected perhaps due to gas blockage.

Low solubility of oxygen in the lixiviant required special mode of injection, for which the knowhow was not available, hence the use

of oxygen as oxidant was discontinued and the injection of H_2O_2 was resumed. Alternate procedure(s) for using oxygen as an oxidant are however being studied.

6.2 Role of Carbonate Ion Concentration in Leach liquor

The concentration of sodium carbonate and bicarbonate in the lixiviant was also reduced gradually from 10/5 to 4/2 g/l respectively. It was found that optimum concentration of uranium was obtained if the unutilized carbonate in the leach liquor was maintained within the range of 600 to 1000ppm. As soon as the unutilized carbonate became less than 600 ppm the uranium values also started decreasing. The concentration of the lixiviant had, therefore to be made up to 8/4 g/l of sodium carbonate/ bicarbonate respectively, to maintain carbonate ions in the leach liquor within the range, as shown in fig.7.

6.3 Contaminants in Leach Liquor

At the maximum uranium concentration of 160 ppm attained in the pregnant leach liquor, the level of various contaminants is given in table II.

It is observed from table II that the level of most of the constituents in the pregnant leach liquor are lower than those present in natural ground water in the area except sodium carbonate, bicarbonate and uranium. Due to concentration of sodium carbonate in the leach liquor, the pH rises to 9.9 from base level of about 8. Hence the quality of ground water can be easily brought to it's original level simply by pumping out water at restoration stage.

7. PROCESSING OF LEACH LIQUOR

The leach liquor, received from the production well is passed through a sand bed filter to remove undesirable insolubles, before it

T A B L E - II

COMPARISON OF LEACH LIQUOR WITH BASE LEVEL

<u>Constituents</u>	<u>Concentration(ppm)</u>	
	<u>Leach Liquor</u>	<u>Base Level</u>
Sodium	1020	332
Carbonate	620	ND
Bicarbonate	1390	230
Chloride	217	389
Sulphate	43	281
Aluminium	1	1
Silicon	8	82
Magnesium	16	20
Calcium	2	22
Potassium	12	42
U_3O_8	160	4

is fed to the processing plant. The plant consists of 4 numbers of resin ion exchange columns arranged in series in which uranium tri carbonyl anion complex is adsorbed on the resin beads. The loaded column is eluted with NaCl solution. The eluant containing 12-15 g/l U_3O_8 is subjected to uranium precipitation, using hydrogen peroxide as precipitating agent. Uranium is precipitated as per oxide ($UO_4 \cdot 2H_2O$), which is then filtered and dried.

The barren solution obtained after the adsorption of uranium in resin ion exchange columns, passes through a surge tank. The raffinate is analysed for sodium carbonate/bicarbonate content, which afterwards is refortified with the appropriate quantity of leaching agents. The reconstituted solution is passed through a gravel/sand bed filter and then reinjected as lixiviant into the formation for a new leach cycle.

8. COST

The cost of well construction, pumping and the reagents used, constitute the major portion of the cost of the insitu leach operation. No estimates have yet been made for the drilling and pumping cost, but the consumption of the reagents per kg of U_3O_8 , estimated at an average of 100ppm uranium in the leach liquor, is given in table III.

As it is obvious from table III sodium carbonate and hydrogen peroxide account for more than 90 percent of the total cost of the reagents. With little more optimization, the consumption of sodium carbonate may be reduced marginally, but the reagent cost can be reduced by about 30 percent if hydrogen peroxide can be replaced by oxygen as oxidant.

T A B L E - III
CONSUMPTION OF CHEMICAL REAGENTS

Reagents Commercial grade)	Consumption kg/kg U_3O_8	Cost (Pak Rs)
Sodium Carbonate	50	300
Sodium Bicarbonate	5	50
Hydrogen Peroxide	22	330
Sodium Chloride	4	4
Hydrochloric Acid	4	4
Sodium Hydroxide	0.1	2
Total Cost of Reagents per kg of U_3O_8		690

9. PROBLEMS ENCOUNTERED

The most serious problem encountered during insitu leaching test in Pakistan was the precipitation of calcium carbonate in the system. The calcium carbonate caused the scaling in submersible pump, and the delivery pipes, and also affected the efficiency of the resin columns.

The main source of calcium in the leach liquor is from ground water which contains 22 ppm calcium and the ore bed, which contains upto 8% calcium. Due to high pH(9.5-10) of lixiviant, the calcium generated from the ore bed by ionic reaction with sodium on clay particles, reacts with carbonate ions and produces fine calcium carbonate. In the existing pattern, the problem has been partially overcome by the use of anticlogging reagent in the lixiviant.

Also difficulty has been faced in the use of oxygen as oxidant, as the required quantity of oxygen could not be solubilized in the lixiviant under normal working pressures. This requires complicated mode of injection, for which the knowhow is being developed.

10 CONCLUSIONS

The amenability of the Kubul Khel ore to carbonate leaching has been proved and various parameters like oxidant type, concentrations of oxidant and lixiviant have been optimized.

Packing of gravel between the screened pipe and the hole wall is necessary, otherwise the sand is sucked into the pumping system.

The consumption of leaching reagents and oxidant are comparable with the figures normally reported for similar operations.

The process does not prohibit the recycling of raffinate as no build up of undesirable contaminants is observed. Therefore so far no bleed stream in the system is found to be necessary.

The ion exchange resin losses are very nominal due to efficient working of bottom fed ion exchange columns in the uranium adsorption circuit.

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URANIUM MINING IN THURINGIA AND SAXONY

H. RICHTER, P. MUHLSTEDT

SDAG Wismut,
Chemnitz, Germany

Abstract

Its cumulative production of 216,500 t of uranium made the former GDR rank prominently among the world's leading uranium-producing countries. Prospection, exploration and exploitation were carried out by a Soviet-German stock corporation, SDAG Wismut. Five different types of deposits were developed. Of a major economic importance were deposits of the 'Ronneburg' type in Lower Palaeozoic shales, limestones and diabases (Eastern Thuringia) as well as hydrothermal vein deposits in granite exocontact (Western Erzgebirge). More than 86 percent of the uranium was mined underground. Among the methods used were both conventional mining methods (shrinkage stoping, room and pillar, roomwork, sublevel caving and sublevel stoping with cemented fill) and underground in-situ leaching as well as surface heap leaching. Due to low prices for uranium on the world market and to cessation of uranium imports by the USSR, the mines will be closed, decommissioned and the sites reclaimed.

In what formerly was the GDR, uranium mining was carried out exclusively by the Wismut Corporation on the territories of the new States of Thuringia and Saxony. The corporation was founded after World War II, in 1946, as a Soviet stock company (SAG Wismut) to produce uranium for the manufacture of nuclear weapons within the framework of German reparation payments to the USSR.

In 1954, the corporation was converted into a joint Soviet German stock company (SDAG Wismut), with equal shares held by the USSR and the GDR. Uranium concentrates produced by the two State corporations SDAG Wismut were delivered for further processing to the USSR on the basis of agreed prices until the end of 1990.

As shown in Table I, the Wismut Corporation, and likewise the former GDR, was one of the world's most important uranium producers with a total production of 216,500 t of uranium over the period between 1946 and 1990 (see /1/,2/).

Production comprised both concentrates from pitchblende (81,500 t U) produced by radiometric ore sorting and/or density separation from high grade ores from vein deposits of the Erzgebirge and chemical concentrate (135,000 t U) from relative low-grade ores. Uranium production reached its peak from the middle of the sixties to the middle of the seventies, with a maximum annual output of 7,100 t of uranium in 1967.

By 1990, production had decreased to about 3 000 t of uranium per year. At the end of 1990 SDAG Wismut ceased uranium mining. In the course of initial rehabilitation work, about 1 200 t of uranium will be produced in 1991. The cessation of uranium production was caused on the one hand by high unit costs of an average of more than 300 DM / kg U and by the cancellation of uranium imports by the USSR in the second half of 1990 on the other.

TABLE I URANIUM PRODUCTION OF SELECTED COUNTRIES

country	cumulative uranium production by end of 1990 t U	uranium production in 1990 (estimate) t U
USA	334 400	3 846
Canada	242 000	10 400
South Africa	140 200	2 600
France	65 400	2 757
Niger	51 000	2 900
Namibia	49 100	3 300
Australia	48 100	3 600
Gabon	20 400	900
rest of WOCA	43 400	708
WOCA total	994 000	31 011
GDR (SAG/SDAG WISMUT)	216 500	2 972

The entire southern part of the former GDR south of the central German main fault (Mitteldeutscher Hauptabbruch) was prospected and explored for uranium by the Wismut corporation

That work involved annual costs of up to 200 million GDR marks, which represent a total of 5,500 million GDR marks since 1946

Exploration was focussed on areas with outcrops of Variscan rocks or areas with thin cover of Meso-Cenozoic platform sediments. All known methods for uranium exploration were used, e.g.

- geological mapping and regional surveying,
- radiometric and gamma-ray spectrometric surveys as airborne, carborne or manborne,
- radon and helium surveys in soils as well as in ground and surface waters,
- seismographic, gravimetric, magnetic and electric ground geophysical exploration
- surface and underground drilling up to a well depth of 2,000 meters with and without coring
- trenching and underground work (prospecting shafts, adits, mining drifts and raises)

Table II shows the main features of prospecting and exploratory works

Costs incurred for the exploration of 1 kg U averaged approximately 20 GDR marks. Geological prospecting for uranium by SDAG Wismut was stopped in 1989

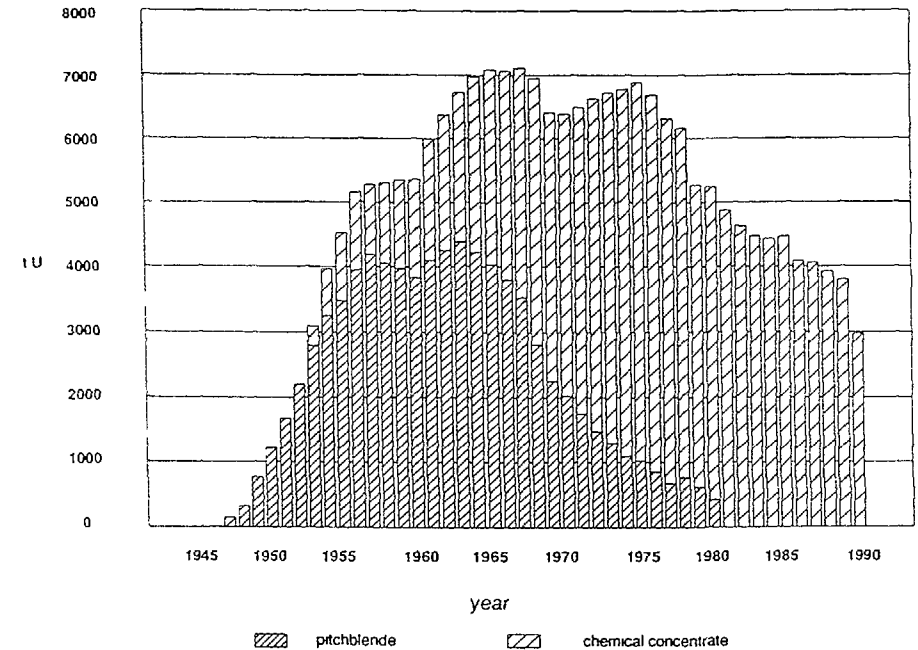


FIG 1 Uranium production by Wismut Corporation, 1946–1990

A number of medium size and large uranium deposits occur at the intersection between the Saxo-Thuringian zone of the northern edge of the Bohemian massif with regional depth seated faults trending NW SE (Hercynian), i.e. the Neidek-Crimmitschau fault zone and the Elbe lineament (see Figure 3)

The Wismut Corporation explored, developed and mined five major uranium deposits with a uranium content of each deposit exceeding 5,000 t. These are the Ronneburg and Culmbitzsch deposits in Thuringia and the Schlema, Zöbe and Königstein deposits in Saxony. The Ronneburg and Schlema deposits were by far the most important, as they stand for 80 percent of the uranium mined in the former GDR.

Uranium was further mined in Saxony and Thuringia at six deposits of a content ranging between 500 and 5,000 t of each deposit, and at 16 smaller deposits of less than 500 t uranium content each.

Presently five deposits – Ronneburg, Schlema, Tellerhäuser, Königstein and Freital – are still fully or partly accessible.

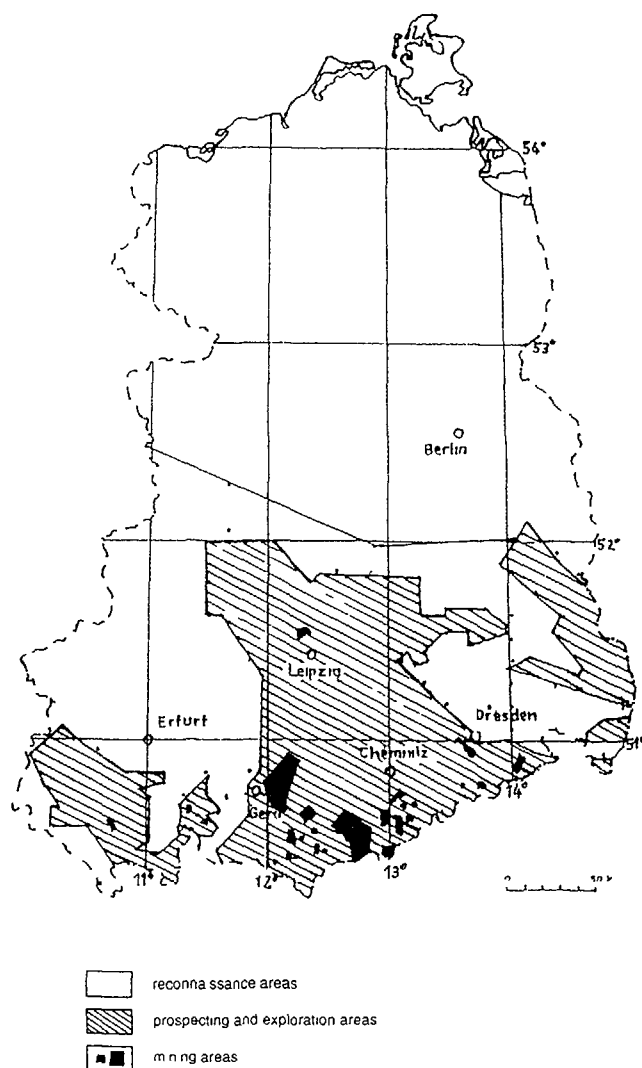


FIG 2 Areas of geological and geophysical exploration and uranium production by Wismut Corporation

TABLE II VOLUME AND RESULTS OF URANIUM PROSPECTING AND EXPLORATION BY SDAG WISMUT, 1954-1989

financial expenditure	5 000 million GDR marks
surveyed surface	55 000 km ²
drilling total	24 000 km
surface	8 000 km
underground	16 000 km
drifting	1 600 km
trenching	more than 1 million m ³
reserves (RAR + EAR I)	264 200 t U
additional resources (EAR II + speculative)	61 500 t U

Uranium deposits of major economic importance that were mined by Wismut Corporation can be subdivided into five geological types according to their morphology, their structural bonds and genesis /3/

- 1 lenticular and stockwork deposits in Palaeozoic shales, limestones and diabases (Ronneburg type)
- 2 hydrothermal vein deposits in the exocontact of Variscan granites,
- 3 peneconcordant and stock type deposits in Upper Cretaceous sandstones
- 4 seam like deposits in fluvio lagoonal carbonate sediments of Upper Permian age (Zechstein),
- 5 uraniferous bituminous coals in molasses of Lower Permian beds

Figure 4 illustrates the contribution of the different types of deposits to the uranium reserves of Wismut Corporation notably the dominating role of the Ronneburg type /4/ and the vein deposits of the Erzgebirge. The Schlema deposit is presumably the biggest uranium vein deposits in the world /5/ /6/

Table III contains a concise characterisation of the different types of deposits

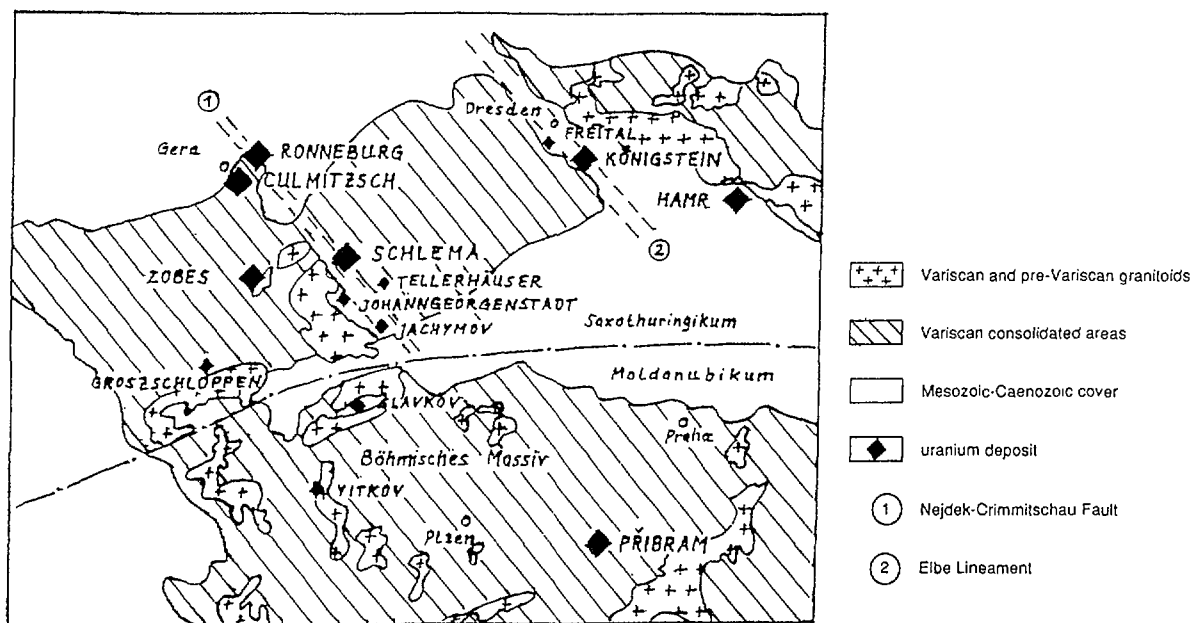


FIG. 3. Occurrence of uranium deposits at the edge of the Bohemian massif.

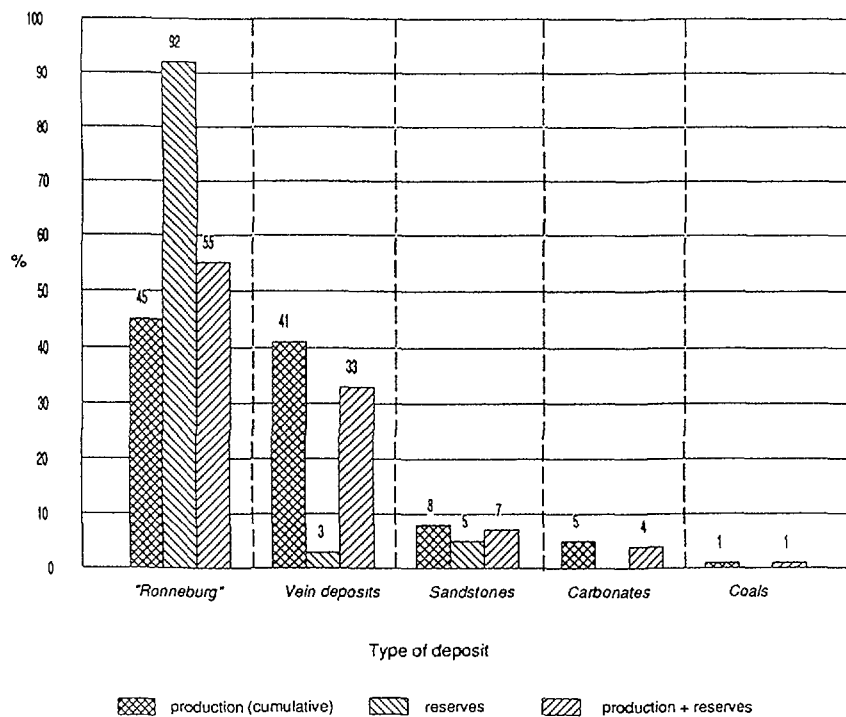


FIG. 4. Contribution of the different types of deposits of the uranium production and reserves of Wismut Corporation.

TABLE III GEOLOGICAL CHARACTERISTICS OF TYPES OF DEPOSITS MINED BY WISMUT CORPORATION

	Type of deposit				
	Ronneburg	Veins	Sandstones	Carbonates	Bituminous coals
Uranium content (t)	n·10 ² n·10 ⁵	n·10 n·10 ⁴	n·10 ⁴	n·10 ² n·10 ³	n·10 ² n·10 ³
Host/wall rocks	shales limestone diabase	shales amphibolite	sandstone siltstone	dolomite shaly clay	bituminous coal carbonaceous shale
Age of host/wall rocks	Ordovician Devonian	Precambrian Devonian	Upper Cretaceous	Upper Permian	Lower Permian
Uranium minerals	pitchblende sooty pitchblende (Coffinite)	pitchblende (uraniferous mica)	pitchblende sooty pitchblende	sooty pitchblende	sooty pitchblende
Average uranium content in run-of mine ore (% U)	0.08 0.09	0.4 n·1	0.11	0.07	0.09 0.10
Associated minerals		Co Ni Bi Ag			coal
Genesis	polygenic (supergene hydrothermal?)	hydrothermal	epigenetic (infiltrated)	epigenetic (infiltrated)	syngenetic
Example	Ronneburg	Schlema Tellerhauser Zobes Johanngeorgenstadt Annaberg	Königsstein	Culmitzsch	Freital (Bannewitz Gittersee)

By January 1st, 1991, the Wismut Corporation held identified geological reserves of approximately 48,000 t U ¹⁾ and additional speculative resources of at least 61,000 t U ²⁾

Uranium production consisted mainly of conventional mining uranium ore, followed by hydro metallurgical beneficiation. A small portion came from underground sulphuric acid leaching and surface heap leaching of low-grade ores.

Underground mining is the dominant method. If one considers the total volume of mined rock ore and waste - of 900 million t, only one third is attributable to underground mining and two thirds to open pit mining.

In the fifties there was open pit mining employed at the Culmitzsch deposit, and in the Ronneburg ore field between 1958 and 1976. From the large open pit at Lichtenberg (160 million m³) came a production of about 11,000 t U in chemical concentrate. Its depth reached

240 meters, today, after it has been partly refilled it represents a hole of 1,600 meters by 900 meters. Exploitation used drilling and blasting, loading and haulage of ore and waste was by shovels (1-4 m³ bucket capacity) and dump trucks (3-12 t maximum load) /7/.

Underground mining was used at 19 mining sites of different sizes with annual outputs ranging from tens of thousand to 1.2 million tons of ore (see Figure 6). Development work of the entire period resulted in approximately 8,900 km of shafts, winzes, rises, adits, drifts and inclines. Some 1,200 km of these workings are still open.

About 18 million t of material was broken in 1967, the year of maximum production. Different mining methods were used, such as room and pillar, shrinkage, sublevel caving and sublevel stoping with cemented fill. The latter method (see Fig. 7) dominated in the Ronneburg ore field over the past 20 years and has produced between 70 and 75 percent of the uranium mined in recent years.

In the early years jackleg drills, drilling columns and scrapers were standard equipment, in later years jumbos and loaders, driven by compressed air, predominated. In 1990, uranium production was going on at 6 mines. Table IV indicates their main characteristics.

On January 1st, 1991, the Schmirchau and Paitzdorf mines were merged to form the Ronneburg mine, and the Beerwalde and Drosen mines were combined to form the Drosen mine.

¹⁾ Official reserves of national classification categories C₁ + C₂ (approximately RAR + EAR I by NEA/IAEA classification) converted into uranium in concentrate.

²⁾ Estimated resources of national classification groups Delta 1 + 2 (approximately EAR II + SR by NEA/IAEA classification), converted into uranium in concentrate.

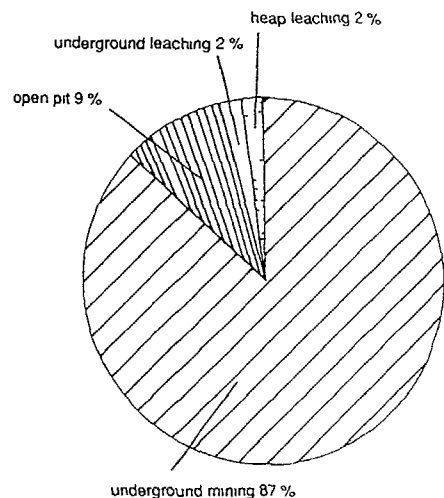


FIG 5 Contribution of the different mining methods to the cumulative uranium production by Wismut Corporation

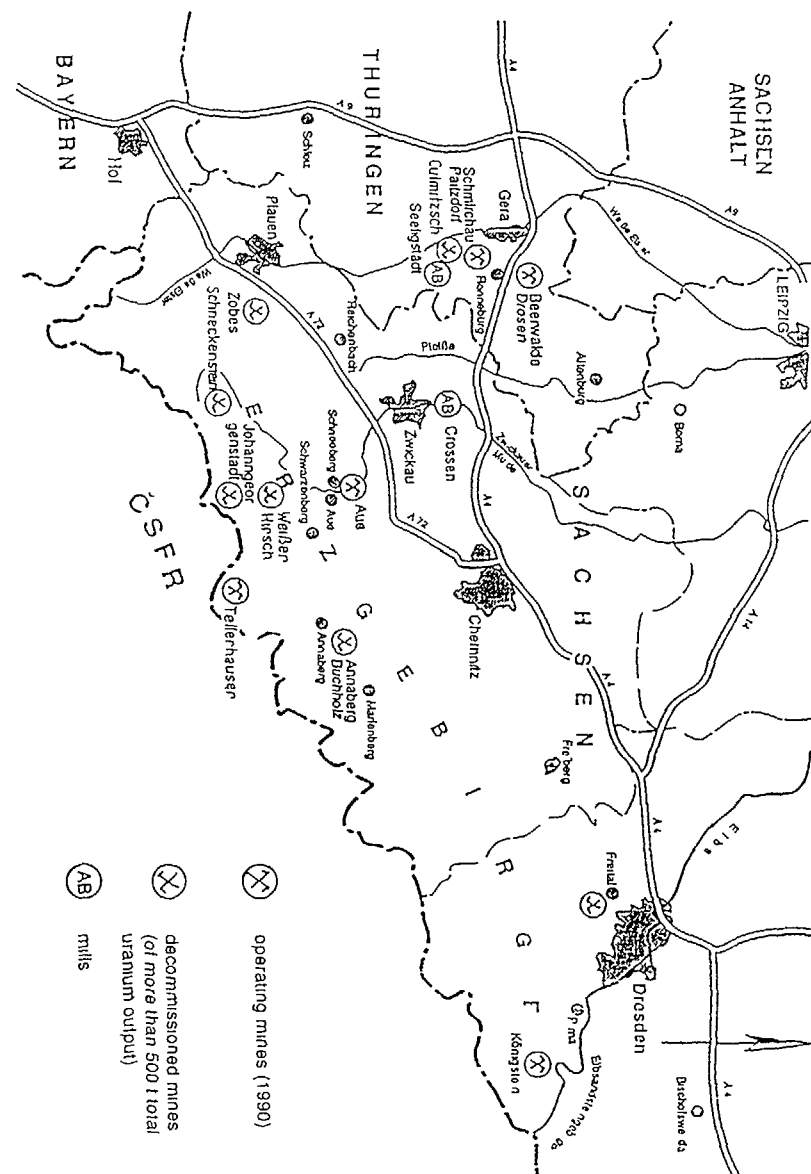
Beneficiation of uranium ores in the beginning was mainly by radiometric sorting or by density separation (tables), from the mid fifties onwards hydrometallurgical processing by soda alkaline and sulphuric acid leaching was also used. In the early stages of chemical processing the recovery was between 70 and 80 percent, it later improved to approximately 90 percent. The two most important among the 9 milling sites were those of Crossen near Zwickau (annual capacity of 2.5 million t ore) and Seelingstadt near Gera (annual capacity of 4.6 million t ore). In 1990, only the Seelingstadt milling facility was still operating and processed approximately 3.3 million t of ore by the soda alkaline process.

Fig 8 shows the flowsheet of the Seelingstadt mill in 1990.

The crushing of the ores and the separation of pyrites by flotation are followed by soda alkaline pressure leaching and by atmospheric leaching. The dissolved uranium is then sorbed from the pulp by ion exchange resin in a countercurrent process, before the resin is separated from solids. After the stripping of uranium, concentrate ($(\text{NH}_4)_2\text{U}_2\text{O}_7$) containing approximately 70 % U is obtained by ammonia precipitation. The tailings are discharged into the slime ponds. Average specific soda consumption is about 21 kg per t of ore.

In May 1991, on the terms of government agreement, the USSR renounced its 50 percent shareholding in the Wismut Corporation.

FIG 6 Mining and milling sites operated by Wismut Corporation



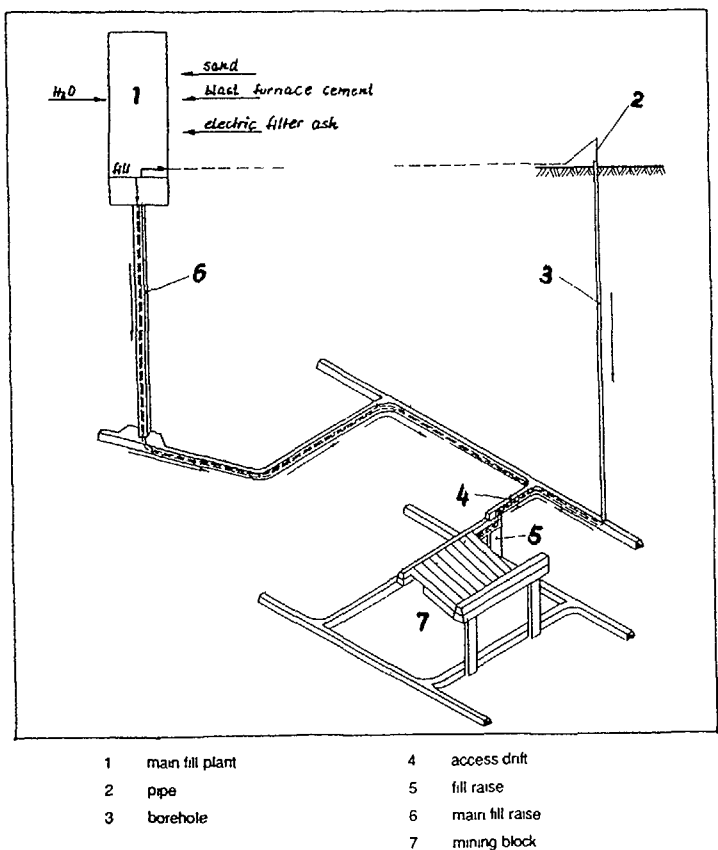


FIG 7 Graphic representation of sublevel stoping with cemented fill

Since that time, the Federal Republic of Germany, represented by the Federal Ministry of Economics, is the sole proprietor of the corporation which held assets of approximately 400 million DM (without mining facilities) in mid-1990. It is intended for "Wismut" to retain its traditional name. The corporation is to be converted into a company under German law, and its existing divisional branches are to be developed into five market oriented profit centers.

- Consulting and Engineering
- Mechanical engineering and structural steelwork
- Civil engineering
- Logistics
- Services.

Mine	Schmirchau	Paltzdorf	Beerwalde	Drosen	Aue	Königsstein
Deposit	Ronneburg	Ronneburg	Ronneburg	Ronneburg	Schlema/ Tellerhäuser	Königsstein
Max. mining depth	525 m	480 m	600 m	780 m	1 755 m	300 m
Mining method 1)	TmV	TmV	TmV	TmV	FS	UTL, HL
Employees	3 850	2 400	1 540	2 250	3 540	1 640
Total ore-waste-prod	(kt)	1 120	1 180	540	1 120	420
Ore	(kt)	731	935	420	760	87
Uranium content in ore	(% U)	0,082	0,085	0,089	0,073	0,431
Uranium in ore 2)	(t U)	601	794	371	558	375
						385

- Legend
- 1) TmV = sublevel stoping with cemented fill
FS = shrinkage
UTL = underground leaching
HL = heap leaching
- 2) Königsstein uranium in pre concentrate (200 gU/t)

TABLE IV URANIUM MINES OPERATED BY SDAG WISMUT IN 1990

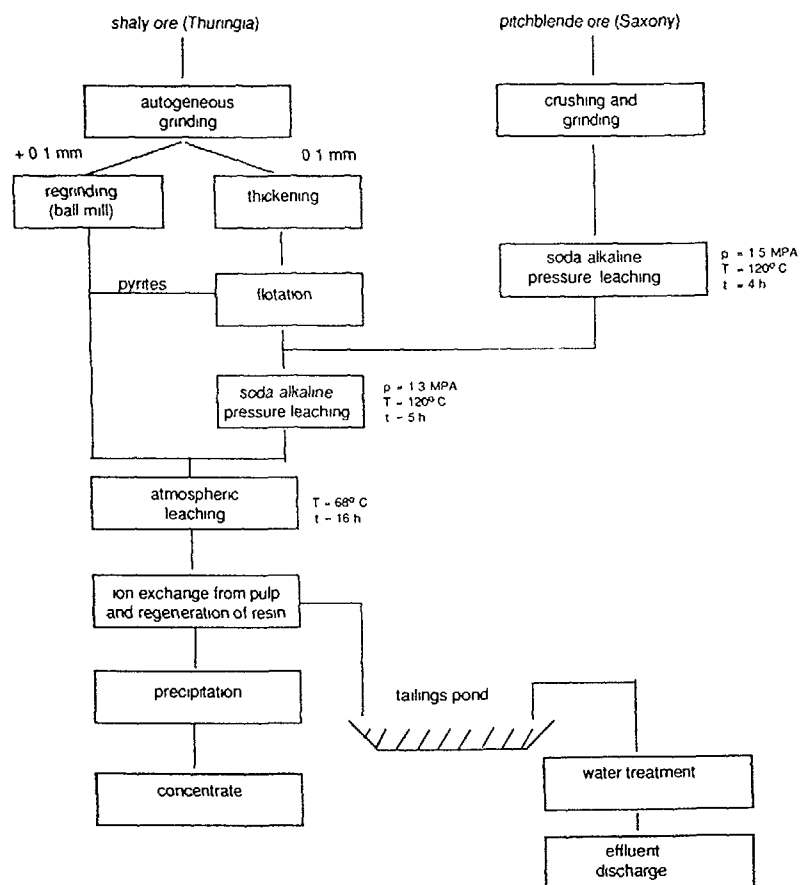


FIG 8 Flowsheet of soda-alkaline uranium processing at the Seelingstädt milling facility, 1990

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- /6/ SCHRÖDER, B. & LIPP, U., "Der Uranerzbergbau der SDAG Wismut im Raum Schneeberg - Aue - Schlema und seine Mineralien" (II), Mineralienwelt, 1 (1990), 3, p 21 - 44
- /7/ PROKOP, R., HOFFMANN, M. et al., "SDAG Wismut Der Uranerzbergbau in der ehemaligen DDR", Glückauf 127 (1991), 9/10, p 345 - 357

Remediation of uranium mining and milling sites will take many years to complete. The total contaminated surface is approximately 2,325 ha. Estimates of the costs for remediation work vary between 5,000 and 15,000 million DM, depending upon the future use of the sites to be reclaimed. In addition we are not certain, what reclamation technologies will be used at various sites.

Research will be intensified in order to establish reclamation technologies that will address the consequences of uranium mining in densely populated areas. In 1991, reclamation work will provide jobs for 9,000 people.

ENVIRONMENTAL ISSUES RELATED TO THE DECOMMISSIONING OF MINE SITES AND TO THE REHABILITATION OF SITES OF THE WISMUT CORPORATION, GERMANY

R. HÄHNE
SDAG Wismut,
Chemnitz, Germany

Abstract

The environmental impact of the extensive uranium mining and processing activities of the Wismut Company in Eastern Germany on the biosphere are described. The total area contaminated by these past and present operations is estimated at 1000 km², located in the Federal Provinces Saxony, Thuringia and Saxony-Anhalt. In addition to the environmental damages caused by the uranium mining industry there is the impact of the silver mining activities which throughout the Middle Ages were carried out in the Erzgebirge.

More than forty years of uranium mining by SDAG Wismut in the Eastern part of Germany produced a total of 220 000 tons of uranium and a steady increase in contaminated surfaces, overburden dumps and mill tailings.

These burdens of the past constitute long-term damaged sites. Their safe disposal, which must be scientifically founded, technologically safe and in conformity with regulatory standards, is, therefore, one of the most urgent tasks that face the new Wismut AG today. In that effort, we are interested in profiting from experience garnered by other uranium producers.

Rehabilitation, today, covers a total surface of 32 km² to which stretched Wismut activities. The areas affected in the early years of uranium mining and which today are labelled as burdens of the past are indeed much greater; as they cover some 10 000 km². That

includes, among others, 16 suspect sites situated in Saxony, Thuringia and Saxony-Anhalt having 280 shafts and adits, 180 uranium ore loading sites, 15 tailings impoundments and more than 3 000 dumping sites and residual holes. That represents a surface of some 1 000 km² to be rehabilitated.

Long-term damaged sites such as dumps, tailings impoundments, tips and abandoned underground mines in the aeration zone are the principal sources of pollutant release to soil, biosphere, water and atmosphere. The release of pollutants is however retarded by continuously changing geochemical barriers. The aquatic dispersal of contaminants may continue for several hundred, up to 1 000 years, the atmospheric dispersal even for geological periods of time.

The location and remediation of these sources of contamination as well as the landscaping of dumps and residual holes in Eastern Thuringia, for example, is therefore an ecological challenge for more than one generation, and it is of great political significance.

One will probably have to admit that nature will hardly be made again what it looked like before mining started. It has also to be taken into account that background levels due to aquatic dispersals of uranium and radium in the past were higher in the Ronneburg and Aue/Schneeberg areas as compared to other regions.

This was evidently due to the surface discharges of one of the biggest uranium deposits in the world, and it was additionally enhanced by the burdens left behind by silver mining in the Ore

Mountains throughout the Middle Ages. That is especially true of the western part of the Ore Mountains where measurements made in houses that were built on overburden dumps or with uranium-containing materials taken from silver mining wastes have indicated levels of Rn-exhalation of up to 20 000 Bq/m³

The actual exposure due to atmospheric emission by Wismut is determined by the return air from the mines on the one hand and by the waste dumps on the other. Radon and its daughter products in particular are released into the atmosphere. They have an almost equal share in the annual Radon activity of a total of 10¹⁵ Bq. Ra-content in the dumps is between 0.4 and 1.2 Bq/g, that in the tailings amounts to 10 Bq/g.

An further component of exposure by inhalation, i.e. the long-lived alpha-emitters, represent less than 10 percent of the Radon-exposure. The aquatic pathway is essentially determined by Uranium and Radium. Ra-contents at discharges are between 0.18 Bq/l and 1.5 Bq/l; uranium-contents vary between 0.4 mg/l and 1.2 mg/l according to the different deposits.

Today the total area of mining dumps stacked up by Wismut is of more than 1 700 ha: It is made up of some 600 ha in the Ronneburg area, some 630 ha of tailings ponds and 300 ha of milling wastes, some 150 ha of uranium mining sites in the Ore Mountains and of some 30 ha at the Konigstein mine.

Mining dumps contain up to 70 per cent of Palaeozoic sedimentary metamorphous rock and eruptives from Ordovician and Devonian periods with high portions of siliceous schist and alum slate in Eastern Thuringia, up to 25 per cent of crystalline rocks from the

Western Ore Mountains and up to 5 per cent of sediments dating from Cretaceous and Permian beds.

In the Ronneburg region, there is important aquatic dispersal of contaminants. Seepage waters from tailings, which are collected underground, partially show water hardness of more than 1 000 °dH on the German hardness scale, and they contain 20 g/l sulphate, 2 g/l Fe, 4 mg/l U, 200 mBq Ra/l, and have fairly high temperatures. Although seepage waters represent only 9 per cent of the drained mine waters, they contain 39 per cent of the hardeners, 46 per cent of the sulphate, 83 per cent of the iron and 24 per cent of the uranium discharge. Their separate collection and purification is part and parcel of in-plant ecological measures.

The phenomenon of contaminants release from dumping and mining sites is due to microbial leaching processes. At relatively high stability constants, metals form freely soluble complexes with ligands such as SO₄ and CO₃. The dissolution of hardeners is pH-related. It was also demonstrated that the formation of sulphate is limited by O₂ Sulphate concentration, on the other hand, determines bacteria activity.

The lowering of the ground-water table in the mining areas led to a general increase in the circulation in deposit areas and zones of active infiltration. The original fissure circulation turned into filter circulation, especially in roof-fall exploitations and their hanging zones.

Seeping distances grew longer, and the influx of oxygen stepped up natural leaching processes. The content of harmful substances

increased. This leach-stimulating two-phase circulation is particularly recurrent in the neighbourhood of mine openings, in subsidence areas and in dumps. As a prevention and in order to limit the process of contaminants release from dumps, which are the principal sources of contamination, the following approaches may be considered.

- 1 encapsulation by additional sealing,
- 2 in-situ remediation, e.g. neutralization
- 3 accelerated lixiviation/washing;
4. dewatering and waste water treatment,
- 5 capping and rehabilitation,
6. checking of microbial processes.

All these measures require vast technical means and funding. In the case of neutralization of dumps with lime, for instance, costs would probably amount to 100 Dm/m² without giving lasting results.

Besides that, the dumps would be rendered only partially inert, in particular along the principal paths of seepage transport. Checking of microbial processes, e.g. by CaF₂, would require expenses to the same extent. Accelerated washing at a rate of 1 000 m³/h, the equivalent of the entire mine waters in the Ronneburg region, would mean a discharge of contaminants for some 150 years. During that period of time, treatment would be required for these quantities of water. The same holds true for the purposeful dewatering which technically will be extremely difficult to achieve, given the uneven permeability of the dumps. The only thing left as a practicable solution to the immense dumps is, in principle, the additional sealing of the surfaces. The use of cohesive soils of the thickness required to minimize

infiltration and Radon exhalation rates would entail very high costs in the Ronneburg region alone.

Therefore, surface treatment must be efficient, resistant to erosion and as low-cost as achievable. In that approach, the following premises would have to be taken into consideration:

- a) Attenuation of the rate of Radon exhalation by extended retention periods in capping materials, and accumulation of daughter products before release to the atmosphere,
- b) Minimization of infiltration through sealing, collection or drawing-off of precipitation, increase of evaporation by means of vegetation and intermediate storage;
- c) Chemical blocking of circulations and pH-increase through appropriate alkaline waste products to slow down ion exchange and to increase buffer capacity of surface zone;
- d) Rapid revegetation and landscaping, probable later use by communities, prevention of rooting in cover;
- e) Sufficient stability

In order to identify adequate means of covering, a 3 ha testing field was established on a 240 ha plateau dumping site in order to carry out tests related to covers of different types of soil and of waste materials of different thicknesses.

The choice of appropriate wastes, even of types having a low potential of contamination, to be used as capping materials for highly contaminated dumping sites, can help to enormously cut the costs for the remediation of dumps. Until completion of these operations, the treatment of mine waters is compulsory till the

beginning of the flooding of the underground mine workings in order to respect regulatory limits in the receiving waters. When, 3 years ago, natural leaching of dumps and cavings resulted in rapidly increasing water contamination, a quick countermeasure and remedy was the surface and underground collection of highly-contaminated seepage waters and their subsequent spraying on ash dumps.

Ashes are stacked up to 15 meters high on sealed ground, partly mixed with lime and then are sprayed on in a ratio of up to 2 m³ of water per ton of ashes. Hardeners are removed by 70 per cent, Fe, SO₄ and U by more than 80 per cent.

An analysis carried out regarding flooding processes in the Ronneburg region showed that the level of -240 m will be reached after 5 to 6 years, and the 60 meters level after 10 to 14 years; that means that during that period no mine water treatment will be required. After the abandonment of mining, one has to be prepared for fundamental changes in the hydrodynamic and hydrochemical conditions. After an initial phase of elevated discharge of contaminants due to the replacement of highly contaminated pore waters and forced natural leaching processes by acid mine waters, quasi-hydrostatic conditions will prevail. Groundwaters will slowly flow through the deposit area, through the abandoned mine workings and through the 240-m-deep residual hole of the open cut, that might be backfilled with waste material, and will then discharge into surrounding valleys.

The overflow of some 150 m³/h could be treated in a water treatment facility, so that there would be no contamination of surface waters. In case that the worked-out open cut would be

backfilled with waste material, one will have to consider that deposited below the ground water table this material is likely to become a source of contamination of its own, and that on the other hand, the removal of radioactive, pyritic and flammable wastes from the dumps may entail new hazards to the environment that need yet to be intensively studied.

The decommissioning and remediation of Wismut milling site tailings ponds of more than 160 million tons and up to 70 meters of thickness will constitute a major challenge in the field of protection against contamination.

Here, intensive scientific work will have to determine whether the best approach will be the backfilling of the ponds, to be followed by dewatering, water treatment, silting-up and covering, or the preservation of the water lamella. Technological preparatory studies will play a key role, given the enormous expenses related to rehabilitation.

When eliminating the sources of Radon exhalation, it will be very difficult to separate natural background from man-made contamination. Prior to expensive technical measures, extensive surveys will have to be carried out in this field to determine the degree of contamination of the soils, in particular in the vicinity of dumping sites, plant sites, return air shafts, ore transport lines etc. It will be only then that limit values can be established and measures of remediation can go ahead.

The future ecological situation at the Königstein mining site presents itself in a very complicated way.

Mining activities and subsequent leaching operations have contaminated more than 12 million m³ of rock and aquifers. 1.8 million m³ of sulphuric-acid solution are currently circulating. Two aquifers, which initially were separated from each other, are now connected through mining work. Abandonment and flooding would be followed by long-term natural secondary lixiviation of technically leached rock. Ground waters carrying U, Ra, SO₄, Fe and heavy metals would discharge into the Elbe river or other receiving waters or drain off into the partly tapped Cretaceous aquifer of the Elbe valley.

Rewashing of leached rock over a period of several years and an intermediate period of neutralization are envisaged to provide remedy for this situation. Water treatment plants should also continue to run until the required limit values will have been reached.

The uranium deposit of Niederschlema/Alberoda, mined in 1946, will also be abandoned and flooded in 1991. It is estimated that it will take some 8 years to flood the mine workings of approximately 40 million m³.

During the period of active mining, some 8 million m³ of mine waters were pumped out annually and discharged into the Zwickauer Mulde river after mere mechanical treatment. In that way, up to 12 tons of uranium and 3.5 tons of arsenic were annually discharged into the receiving waters. Seepage from dumps is adding further contaminants to the pollutants carried by mine waters.

Once the dewatering system is turned off, there will be no discharge of water from the deposit into the receiving water for a

period of approximately 8 years. During that time, receiving waters will be polluted essentially by seepage from dumps and former mines. Once the cavings will be flooded, the discharge is estimated to amount to 500 - 700 m³/h. In the beginning, the discharge will have considerably higher concentrations of arsenic and uranium than the presently pumped out mine water. In the medium-term, one can expect a water composition similar to that of dump seepage, i.e. of approximately 0.3 ... 1.2 mg/l arsenic and 1.4 ... 2.7 mg/l uranium. An extensive study is under way concerning the technology of future water treatment which will be based on experience garnered so far in treating mine waters by tubular settlers, BaCl₂ and lime.

In the forthcoming years, Wismut experts and their partners will therefore face the following main tasks:

1. Prognosis concerning the flooding processes at the Ronneburg, Aue and Konigstein mining sites and deduction of technological measures in order to prevent environmental contamination.
2. Comprehensive geological and ecological research in order to determine the degree of contamination of soils, water and air
3. Conception of technological solutions for the safe disposal, remediation and rehabilitation of dumps, tailings ponds and plant sites with a view to minimize Rn exhalation and natural lixiviation
4. Establishment of ecologically sound solutions for the backfilling and remediation of underground mine workings and of the worked-out open cast at Lichtenberg

5. Conception and implementation of a monitoring system that will integrate the soil, atmospheric, biological and aquatic pathways.

These tasks will only be accomplished by national and international co-operation and through the participation of authorities and of the population. To that end, intensive contacts have been established with institutions and corporations in Canada, USA, Australia, France and CSFR. In the forthcoming years, the abandoned operations of SDAG Wismut will be an interesting testing laboratory both in the scientific and technical fields, and will yield experience that will benefit other countries, too.

PRESENT STATE AND FUTURE OF THE CZECHOSLOVAKIAN URANIUM INDUSTRY

O PLUSKAL
Faculty of Sciences,
Charles University,
Prague, Czechoslovakia

Abstract

The situation as regards uranium exploration and production, carried out by the state-owned Czechoslovak Uranium Industry (CSUI) is briefly described. Within the geological environments of the CSFR, the Bohemian Massif has proved to have the best uranium potential, concentrated mainly in the Variscan basement. The uranium potential of the Bohemian Massif is estimated to be in the range of 250 000 to 300 000 tonnes U, of which about 100 000 tonnes U have been recovered. About 80 percent of the total potential is in vein type deposits, the remainder in sandstone type deposits. In the past, six uranium provinces in the Bohemian Massif have been exploited: Příbrán, Western Moravia, Western Bohemia, Jáchymov, Horní Slavkov and the Northern Bohemian Cretaceous basin. In the future, it is expected that the only district which remains in production, will be the Northern Bohemian Cretaceous basin.

Czechoslovakia belongs to the countries participating in a wide range of IAEA activities, with varying intensity according to branches of work. A marginal sphere was prospecting, exploitation and processing of uranium ores, though Czechoslovakia was one of the important, however not biggest, producers. It belongs to the characteristic features of the past, that even not long ago, exploitation and processing of uranium ores were performed independent of the remaining sectors of the Czechoslovak nuclear industry.

The only institution which dealt with exploration, exploitation and processing of U ores in the whole territory of the CSFR until the end of 1990, was the present state enterprise of Czechoslovak Uranium Industry (CSUI) in Stráž pod Ralskem. The enterprise, protected by the state in the past, worked entirely independently in some spheres and was isolated from the economic development of the Czechoslovak mining industry. Also, its collaboration with the Czechoslovak nuclear industry was insignificant and the co-operation with the Czechoslovak Academy of Science was negligible.

Recently, the CSUI is directing its efforts towards a considerably wider range of fuel cycle issues, however, the exploitation and processing of U ores from indigenous sources maintain their important position. As a new development, this activity is now closely associated with ecological tasks. The former isolated and monopolistic position of the CSUI in exploration, exploitation and processing of U ores influenced by a lack of environmental concern typical of socialist economy mainly in the 50-ies and 60-ies left in the country some serious ecological problems which have to be solved now. A certain advantage is that the majority of data, not only on the uranium deposits, but also on natural radioactivity are available in the CSUI. This forms a good base for a large variety of future activities for the company. Similarly, both the experience and need to solve mining and technological problems in various complex geological terrains in depth of more than 1000 m places the CSUI in the position to offer spent fuel storage possibilities.

A large portion of the knowledge currently available within the CSUI is mainly associated with geological problems. As far as uranium resources are concerned Czechoslovakia belongs to the best explored countries in Europe, of not even of the world.

The geological structure of the CSFR comprising structural units of the two youngest European tectogenes allows the comparison of the two environments as regards the uranium mineralization and their industrial importance from quantitative and qualitative points of view.

The Bohemian Massif, classified as Variscan, is characterized by the presence of three types of uranium deposits of economical significance. These include vein deposits, sandstone deposits and surficial deposits. The economical importance of surficial deposits, however, is minute in comparison with those deposits associated with the Western European Variscan units. The territory of the Bohemian Massif can be divided into the Variscan basement and its platform cover. In the Variscan basement, ranging in age from lower to upper Permian, the highest concentration of uranium resources in vein deposits occurs, while the sandstone deposits are typically associated with the basement cover.

In the Western Carpathians, belonging to the European part of the Alpine Himalayan system, the only uranium deposit type of a certain importance are sandstone deposits. Other occurrences of uranium mineralization, including vein deposits, appear economically irrelevant. The exploration of uranium mineralization in sandstones of the Western Carpathians disproved the potential for this type of mineralization especially in the Alpine part of the tectogene. A

small amount of this ore, not exceeding a few hundred tonnes U was mined in this environment.

As regards the present state and future development of uranium mining in the CSFR, it appears most likely that the sandstone deposits of Bohemian Massif will continue to be an important source for uranium. The vein deposits, however, which have contributed the biggest share of the uranium production through the beginning of 70ies, are losing their important position.

The complex structure and development of the geological units of the Bohemian Massif still permit a certain speculation on the possible existence of undiscovered resources in different deposit types, which so far with the applied exploration methods have not been found.

All of this is apart of a number of complex problems which in the CSFR are considered in relationship to other possibilities to secure the resource base for nuclear fuel. The review of the present data on uranium deposits in the CSFR is needed to recognize that their evaluation and development in the past were done under criteria of the planned economy, and moreover, heavily influenced by political considerations. Therefore, most of data about quality and quantity of uranium resources and the estimated reserves are not comparable with conditions prevailing in a market economy. Yet, the data on vein and sandstone deposits of the Bohemian Massif contribute to the profound knowledge of uranium metallogeny of Variscan Europe and to the economical importance of both deposit types. Generally said, the lower limit of grade of mined ores was 0,08 to 0,10 % U and was mostly dependent on geological, mining and technological conditions of a certain deposit or district.

The estimate of the entire uranium resource potential of the Bohemian Massif totals about 250 000 to 300 000 tonnes U. Roughly 100 000 t have been mined. Of this cumulative production, 85% were produced from vein deposits and the remaining 15 % from sandstone deposits. The majority of data on uranium resources refers to the vein type deposits and deals with their geological position, spacial distribution, morphology of ore bodies and ratio between disseminated and massive mineralization. The evaluation of this information allows a good comparison with deposits and districts of other parts of Variscan Europe.

Roughly 90 % of the resources in vein deposits was concentrated in five districts. Further two deposits had cumulative resources of about 5 000 tonnes U, twelve deposits had

resources in the range of 100-500 tonnes U and eight deposits had from 10 to 100 tonnes U. A number of uranium occurrences appeared in all crystalline units of the Bohemian Massif, except in the entirely barren Solencan unit at the NE part of Massif. A special genetic importance for the uranium deposits of the Bohemian Massif seem to have the Variscan granitoids as evident in four districts and in the majority of individual deposits and occurrences. Beside that, a certain number of smaller deposits and occurrences, however, lack this relationship to granites and it must be assumed that there is an additional uranium source. The relationship between morphology and type of mineralization (whether disseminated or massive) on one hand, and the development of fracture tectonics in various rock complexes, on the other hand, was clearly shown. In the whole of Bohemian Massif, the exocontact type uranium mineralization is prevailing. However, endocontact mineralization is also present, but its volume and the density of occurrences are considerably lower (about 5%). Endocontact mineralization cannot be associated with or limited to certain bodies or massifs of granitoid rocks, because it is known in different scales from the Central Bohemian pluton, Bor massif or Karlory Vary massif and some others. Fracture tectonics, mainly the existence of deep seated faults, are considered important elements controlling distribution of vein deposits. Certain geological units, their confinement together with shape and orientation of igneous bodies and massifs, play a significant role in fracture tectonics in the geological history of the Bohemian Massif. The geological position of many occurrences or even uranium ore districts, however, do not indicate any controlling role of structures, or deep seated faults. In view of these considerations about the role of deep seated faults, there was not great interest in relating the distribution of zones or districts to mechanically disturbed rocks to the stage of cataclastites or ultramylonites. A relationship of vein mineralization distribution with certain units or rock complexes, with the exception of the Variscan granitoid bodies, was not proved in the framework of the Bohemian Massif.

The vertical range of vein type uranium deposits in the Bohemian Massif was verified to a depth of 1500 m for the massive mineralization (Příbram deposit) and more than 1000 m for the disseminated mineralization of both, endocontact and exocontact deposits. Changes in intensity of mineralization with depth have various developments, different in disseminated and in massive ore deposits. In larger deposits one or two peaks with successive decrease to depth can often be observed. A steadier development is typical of disseminated ore bodies, forming broad columns with an area $n \times 10^6 \text{ m}^2$ within lining of faults.

The most common minerals in the uranium deposits in Bohemian Massif are pitchblende, coffinite and uranium blacks in deposits with disseminated mineralization. Organic U complexes

are known from the deeper parts of the Příbram district and from some smaller deposits in Eastern Bohemia. The mineralogy of all districts and individual deposits in the entire Bohemian Massif is well described.

The biggest uranium district in CSFR was the Příbram district. It is situated directly on the exocontact of the Central Bohemian intrusive, which is surrounded by low grade metamorphics of Upper Proterozoic age, on the contact with granites affected by contact metamorphism. Systems of mineralized structures of NW-SE and N-S strike, plunge to NE. In the SW part of the district the veins are almost outcropping, while in the NE part the mineralization appears deeper than 1000 m. The district's potential of 50 000 tonnes U formed about 50% of total vein type mineralization, almost exclusively as massive vein mineralization. As regards to the frequency, thickness, variety of mineralogical composition and vertical extent of ore mineralization in Příbram, there is nothing comparable to this district among Czechoslovakian vein districts and deposits. On the other hand, the morphology of the ore bodies, texture and depth development of the Příbram veins indicate number of similarities with base metal deposits in the Barrandian Proterozoic.

The second biggest vein type uranium district was Western Moravia. It lies at the contact between highly metamorphosed rock of Moldanubian age, and lower metamorphosed rocks of the so-called "micaschists zone". Its potential of 25 000 tonnes U is mainly in disseminated ores which fill thick N-S faults, often containing graphite in mylonitized rocks. With these faults are associated smaller and shorter dislocations and joint sometimes containing lenses and veins with massive ore. It is the only district of the Bohemian Massif, where the relation to Variscan granitoids is not so clear. But in the ore district and its surroundings, granitoids, similar to rocks of large Třebíč massive which is situated 15-20 km W of the district, occur in dikes and small bodies. An identical morphology of ore bodies, texture and mineralogical composition of ores as in the Western Moravian district have been observed in other uranium deposits and occurrences, which are, however, much smaller in the highly metamorphosed complexes of the Moldanubian zone (Ohrouhlá Radouň, Hermanický, Jásenice).

The third district, Western Bohemia, had resources of 8 500 tonnes U. It contains various deposit types, located at the endocontact and exocontacts of the Bor granite massif. The deposit of Zdobův has mineralized structures and a vertical extension similar to the deposits of the Western Moravian district. The deposit Vítkov at the endocontact of the Bor massif is similar to the Variscan deposits of Western Europe, with ore vertically extending to about 1000 m. The smaller deposit Světa Anna with mostly massive ores is similar to deposits

of the Jachymov district as regards its mineral assemblage. Numerous reports were published on the geological setting and economical aspects of the Jachymov district. Its whole potential was not more than 8060 tonnes U. The Horn Slavkov district, S of Jachymov, was considerably smaller with a potential of 2600 tonnes U. Both districts are characterized by massive ore as vein fillings with a thickness rarely over 1,0 m. They form a type of deposit which can be termed "a deposit of Variscan granitoid mantle". Beside a network of huge barren fault structures striking at several directions, developed in both districts, it was typical that the mineralized wedges near the contact between schists and granites, penetrated the granites only to minor extents. A separate deposit of "granitoid mantle", called Predbořice in the Central Bohemian plateau contained also some gold, not directly related to the U mineralization. Its potential of uranium was to 250 t. Apart from the mentioned districts and deposits in the Bohemian Massif, probably two pronounced fault lines can be delineated which show a relationship to smaller deposits with mostly massive ores. This includes the Zelezné Hory fault, passing the N margin of the Western Moravian district towards NW and plunging under the Cretaceous sediments SE of Prague (deposits Slavkovice, Štrdlovice, Chloteber, Licomerice, Bernardov). The other tectonic line, strikes roughly WNW-ESE occurs at the S margin of the Central Bohemian plateau (Dlázov, Ustalec, Dametice, Nahosín, Měchov). The deposits and ore occurrences along the Zelezné Hory fault belong together with ore occurrences in Northern Bohemia and the deposit of Zálesí in Western Moravia to those lacking close spatial relation to any Variscan granitoids. The role of granites in metallogenic processes, however, may have been replaced by some types of migmatites or orthogneisses.

In the near future, a hypothesis or model of vein type U mineralizations genesis will be made. It is based on the knowledge about the role, which played the geological position, role of granitoids or similar rocks, especially fracture tectonics, circulating fluids and solutions. Using data of the Bohemian Massif, the results of research from other regions of Variscan Europe are taken into consideration. For the development of this model, simulations of ore-forming processes are attempted. It cannot be ruled out that despite the high level of prospecting and exploration in the Bohemian Massif, some interesting areas and districts have not yet been sufficiently evaluated.

Today the other most important deposit type is the sandstone type in Bohemian Massif. Higher concentrations of uranium in molasse sediments in Permian Carboniferous basins were due to Variscan tectogeny. They follow some parts of coal seams, filling Paleo-river courses or horizons with tuffitic admixtures. Higher or economic concentrations of uranium were briefly

exploited in the Lower Silesian basin in NE Bohemia, and less significant U occurrences appear in the Kladno-Rakovník basin in Central Bohemia.

The deposits of the Bohemian Cretaceous basin have been exploited from the early 70s and remain the main source of uranium for the future. Similar deposits in Tertiary basin were explored and to a small extent, they were mined by open pits. The total U-potential in Cretaceous sediments is estimated up to 200 000 tonnes U, while the potential in Tertiary environments does not exceed 1000 tonnes. However, it should be taken into account that the assessment of the Cretaceous potential has been based on criteria of the planned economy system. Today it would be more appropriate to include into an assessment additional criteria such as ecology and the utilization of by-products.

The geological setting of U mineralization in the Upper Cretaceous sediments is actually simple. Finely dispersed U minerals and U-Zr complexes form mainly flat ore bodies in siltstone and sandstone horizons of fresh water facies as well as of marine Cenomanian transgression on mainly crystalline highly weathered basement. Parts of the basement are Cadomian and Variscan granites as well as Permian rhyolites or their tuffs. The geological situation is complicated by fracture tectonics and intrusions of Tertiary basic effusives as dikes and plugs. All of this affects the hydrological setting and it played an important role in the genesis of the deposits. The district currently mined in so-called Straz block is characterized by mineral assemblage of U-Ti-Zr. In addition, many details have been researched including the mineralogical composition of ores, the development of oxidation within the sedimentary strata, as well as the role of organic matter and iron sulfide. While investigating the genesis of the Cretaceous deposit, as well as Permian and Tertiary U mineralization and occurrences it is obvious, that all the occurrences in the Bohemian Massif are controlled by endogenous processes which are marked by a presence of effusive acid or basic rocks. The comparison of geological setting of different parts of the Cretaceous and Tertiary units of the Bohemian Massif, with completely analogical lithological and structural conditions, revealed that the U mineralization is confined only to occurrences of effusive rocks in the local geological situation. This idea is well documented by prospecting in both Cretaceous and Tertiary terranes of the Bohemian Massif.

A brief summary of economic and the geological setting of U mineralization of special interest to the CSÚI should comprise the fact that a profound processing of data on prospecting and exploitation will be used as well to influence activity in the sphere of ecology (protection of the environment). Here the CSÚI is going to develop a highly qualified activity together with

the activity for the preparation of disposal sites for spent fuel CSUI is also prepared for a direct international cooperation in all fields of the nuclear fuel cycle And this is one of the aims of the participation of the CSFR Delegation at this IAEA meeting

STRUCTURE OF OWNERSHIP AND URANIUM PRODUCTION, 1970-1990

A P KIDD
RTZ Limited,
London, United Kingdom

Abstract

The aim of this paper is to illustrate one aspect of the uranium industry namely the structure of ownership of production and how this has evolved The data is based on equity shares on a mine by mine basis Data for four years have been chosen spanning a twenty year period from 1970 to 1990 The data are summarised under four categories of ownership, private domestic, Government domestic, private foreign and Government foreign

Most of the discussion in this paper centres on data which has been summarised in two slides which I shall show presently More detail is given in the Appendix to the paper The aim is to illustrate one aspect of the uranium industry, namely the structure of ownership of production and how this has evolved over time The data has been built up from considerable detail on equity shares on a mine by mine basis and the larger part has been reviewed and edited by individual mining companies in the country concerned This has been possible due to the help of members of the Uranium Institute and I am most grateful to everyone who has contributed

Data for four years have been chosen, spanning a twenty year period from 1970 to 1990, the most recent year for which data is available Production data for the early years across all countries by production centre proved difficult to obtain so nominal production capacity has been used in some cases Where this applies it is noted at the bottom of the summary tables in the Appendix

For this meeting, I have summarised the data under four headings or categories of ownership By definition, a summary masks the complexity and much of the nature of shareholdings, there are grey areas and there may well be overlap, foreign private sector investment sometimes includes an element of State equity What is true, is that ownership has become more fragmented over the twenty year period

The first category is private domestic, that is private sector companies of the country concerned This of itself covers a very wide variety of companies, both large and small, some producing only uranium and perhaps established for a single deposit Nufcor in South Africa is a private sector company which was established to process and market uranium on behalf of the mining companies, largely gold mining companies producing uranium as a by-product Other companies may produce uranium as part of a portfolio of other minerals, there are others whose income is derived from a range of industries which include mining Some are utility companies,

with interests in other energy minerals. Finally there is an, admittedly small, portion of non-institutional shareholdings, that is, shares owned by individual private investors.

The second category of ownership is Government domestic, where the public sector owns and operates the uranium mining company. This may be for reasons of security of supply or quite simply because it is the Government which has the financial resources to invest in the mining sector. It is not uncommon for energy resources to be under the aegis or control of Government. In the case of nuclear power, this may be for reasons of nuclear non-proliferation. In some countries the Government may own and operate not just uranium mining facilities but all industries involved in the nuclear fuel cycle. Good examples here are both Argentina and Brazil where it is only relatively recently that the Government has begun to encourage private sector shareholding, perhaps explained by their need to reduce public sector debt. In India the Government owns and operates all sectors of the nuclear fuel cycle, production is relatively small but wholly committed to the domestic nuclear program so is not intended for trade. Uranium production at the mine in the Republic of Slovenia, Yugoslavia, was solely to match domestic uranium requirements. Portugal has no nuclear program at present and as late as 1981 uranium mining was a state-monopoly with all output for the export market.

The military origins of the uranium industry explain much of the early Government involvement; in the USA only the Federal Government owned significant amounts of domestic production up to the mid-1960's. Examples elsewhere include the Rum Jungle deposit in Australia and the mine in the Beaverlodge area of Saskatchewan, Canada. By 1970 however the transition to a civil nuclear program and private sector ownership was well underway.

The third category of ownership is private foreign, that is private sector companies registered outside the country where the mine is located and the fourth is Government foreign, which is public sector investment overseas. As the charts will show this latter category is not common, the clearest examples are in Africa, in Gabon, Namibia and Niger. In Gabon the French Government holds a share in the Comuf Mounana deposit, originally through the Commissariat a l'Energie Atomique and now through Cogema. In Namibia the Industrial Development Corporation of South Africa has a shareholding in the Rossing mine. This has been diluted and the Namibian Government now has a share in the mine.

Perhaps the clearest example of Government ownership is in the Eastern bloc countries where uranium production and other nuclear fuel cycle industries have long been exclusively under State control, here reflecting the nature of Government which also controlled other sectors of the economy. Statistical information on the uranium industry in most Eastern Bloc countries is still relatively poor so no production figures for these countries have been included in the tables or charts.

Twenty-one years ago, 1970 was just at the start of the development of nuclear power as a source of electrical energy. In September 1970 the European Nuclear Energy Agency/IAEA estimated demand for uranium in that year of 9231 tonnes U. This was based on an installed nuclear generating capacity of just 18 GWe, with almost one third being in each of the UK and USA. 1980, ten years later marks the peak to date in uranium production although, again please note that this takes no account of production in Eastern bloc countries. Output in that year was just over 44000 tonnes U, more than double production in 1970. Since 1980, production has fallen steadily, reaching 35500 tonnes U in 1985 and a little over 29000 tonnes U in 1990. The 18 GWe installed capacity in 1970 referred to above compares with 278.8 GWe last year, when there were 423 reactors in operation worldwide, 355 excluding the Eastern Bloc countries.

So, what has been the pattern of ownership across these four years, 1970 one of the earliest in the history of civil nuclear power, 1980 the year of peak uranium production, 1985 and 1990.

This first chart shows a breakdown of production for each of the categories described. As already noted, the size of the 'pie' has dropped substantially, by about one-third, from the peak in 1980.

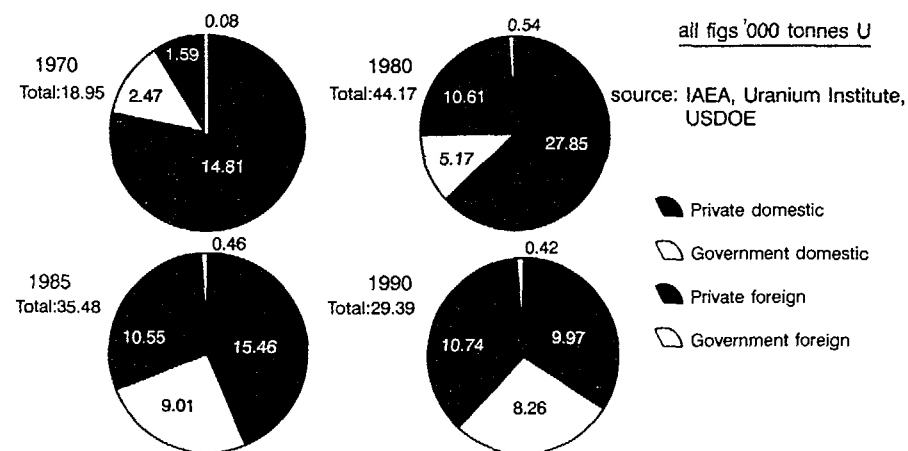


Chart I. Uranium — structure of ownership.

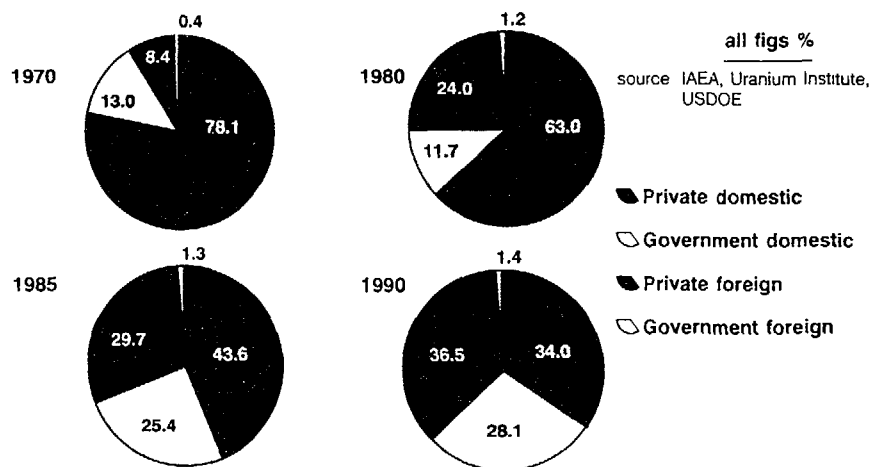


Chart II. Uranium — structure of ownership.

This second chart shows the same data, but expressed as a percentage of total production in each year. It shows very clearly the dominance of the private sector in the two early years, accounting for around 87% of output in both 1970 and 1980. Investment was encouraged by forecasts of generating capacity and rising prices for uranium.

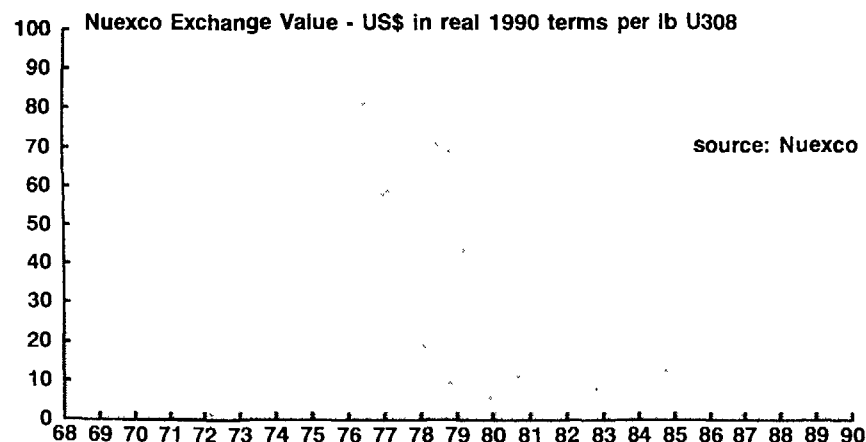


Chart III. Trend in spot prices.

This chart shows the trend in the Nuexco Exchange Value, expressed in real 1990 terms, over the twenty year period. As you can see the peak in prices preceded that in production by only a few years. Some of this was by oil companies, particularly in the USA. In 1980 they included Chevron, Continental Oil, Exxon, Getty, Gulf Oil, Mobil and Standard Oil.

As uranium output fell in the past decade, the decline in the shareholding of the domestic private sector has been matched by an increase in foreign private sector investment. One explanation for this would be the need or wish to diversify sources of supply, particularly on the part of utilities. There has been an increase in utility company shareholdings in the period under review. The United Kingdom for example, has no indigenous source of primary feed uranium and diversification of sources of supply is a stated utility objective. Through a US subsidiary it has a majority shareholding in the Highland in-situ leach project which began operation in 1988. An earlier example, but also in the USA, is the Schwartzwalder mine in Colorado which was operated by Cotter Corporation for the US utility Commonwealth Edison. An example of utility shareholdings elsewhere is the Ranger mine in Northern Australia. German, Japanese and a Swedish utility all held shares at the start of mine operation in 1981.

The public sector share was very similar in both 1970 and 1980, around 13%. By 1985 the public sector share had risen to 25.4% and was just a little higher, 28% last year. To some extent this increase quite simply reflects the change in the composition of those countries producing and the quantities produced.

Is an analysis of ownership useful, can it provide any guidance to market behaviour? The answer is only a qualified yes. Governments, whether they have a share or not, can restrict supply to the market. There may be limitations on the amount of foreign equity, restrictions on exports or approvals required so production may be more Government controlled than appears from the numbers. There have been several examples of this in the uranium industry. They include the complete embargo on imports into the USA for domestic use up to 1977, more recently the imposition of sanctions on imports of uranium from South Africa, lifted earlier this summer. The Australian Government place a ban on exports of uranium to France between 1983 and 1986, and Federal Government policy in Australia restricts uranium mine development to three. The possibility that this kind of restriction may arise is of itself an incentive for producing companies to diversify the countries in which they invest.

The nature of ownership can provide little guide to security of supply in a poor market. In response to weakening prices and revised expectations of growth in consumption, many companies, worldwide and irrespective of ownership, have announced cutbacks and closure of operations.

Shareholdings represent essentially a financial commitment. On the part of utilities they can provide an alternative means of purchasing the uranium they need, prospectively for the longer term. Investment share may not however equal

production share and owners may have no obligation to purchase output, it may be just a first option. Allocations may be agreed quite separately, perhaps on an annual basis. Even more important, there is nothing inherent in the term shareholding which prevents output excess to requirements coming onto the market. The increased fragmentation of shareholdings over the period and particularly the greater investment by electrical utilities does indicate an awareness of potential benefits in sharing the risk and management of investment in the uranium supply required to fuel a nuclear power reactor.

APPENDIX

1970

COUNTRY	PRIVATE	GOVERNMENT	PRIVATE	GOVERNMENT	TOTAL*
	DOMESTIC	DOMESTIC	FOREIGN	FOREIGN	tU
1. Argentina	-	122.0	-	-	122.0
2. Australia	-	254.0	-	-	254.0
3. Belgium	-	-	-	-	-
4. Brazil	-	-	-	-	-
5. Canada	1643.2	587.0	1289.8	-	3520.0
6. France	75.0	1175.0	-	-	1250.0
7. Gabon	-	-	305.6	76.4	382.0
8. India	-	200.0	-	-	200.0
9. Namibia	-	-	-	-	-
10. Niger	-	-	-	-	-
11. Pakistan	-	-	-	-	-
12. Portugal	-	66.0	-	-	66.0
13. South Africa	3167.0	-	-	-	3167.0
14. Spain	-	51.0	-	-	51.0
15. Germany	24.0	-	-	-	24.0
16. Japan	-	1.0	-	-	1.0
17. Sweden	-	14.0	-	-	14.0
18. U.S.A.	9900.0	-	-	-	9900.0
19. Yugoslavia	-	-	-	-	-
20. EBloc ²	na	na	na	na	na
TOTAL	14809.2	2470.0	1595.4	76.4	18951.0

Source: IAEA, Uranium Institute and USDOE

1. Federal Republic

2. Bulgaria, Czechoslovakia, GDR, Hungary, Romania, USSR and PRC - 100% state-owned

* All figures are for production except Argentina which is production capacity.

COUNTRY	1970 (%)				
	PRIVATE	GOVERNMENT	PRIVATE	GOVERNMENT	TOTAL
	DOMESTIC	DOMESTIC	FOREIGN	FOREIGN	
	%	%	%	%	%
1. Argentina		100.0			100.0
2. Australia		100.0			100.0
3. Belgium	-	-	-	-	-
4. Brazil	-	-	-	-	-
5. Canada	46.7	16.7	36.6	-	100.0
6. France	6.0	94.0	-	-	100.0
7. Gabon	-	-	80.0	20.0	100.0
8. India		100.0	-	-	100.0
9. Namibia	-	-	-	-	-
10. Niger	-	-	-	-	-
11. Pakistan	-	-	-	-	-
12. Portugal	-	100.0	-	-	100.0
13. South Africa	100.0	-	-	-	100.0
14. Spain	-	100.0	-	-	100.0
15. Germany	100.0	-	-	-	100.0
16. Japan	-	100.0	-	-	100.0
17. Sweden	-	100.0	-	-	100.0
18. U.S.A.	100.0	-	-	-	100.0
19. Yugoslavia	-	-	-	-	-
20. EBloc	-	100.0	-	-	100.0
TOTAL	78.1	13.0	8.4	0.4	100.0

COUNTRY	1980				
	PRIVATE	GOVERNMENT	PRIVATE	GOVERNMENT	TOTAL*
	DOMESTIC	DOMESTIC	FOREIGN	FOREIGN	tU
1. Argentina	-	244.0	-	-	244.0
2. Australia	1063.0	250.0	187.0	-	1500.0
3. Belgium	40.0	-	-	-	40.0
4. Brazil	-	-	-	-	-
5. Canada	3545.3	421.0	3183.7	-	7150.0
6. France	395.1	2238.9	-	-	2634.0
7. Gabon	10.23	255.67	767.1	-	1033.0
8. India	-	200.0	-	-	200.0
9. Namibia	-	-	3496.33	545.67	4042.0
10. Niger	-	1250.04	2657.96	-	3908.0
11. Pakistan	-	30.0	-	-	30.0
12. Portugal	-	82.0	-	-	82.0
13. South Africa	6048.8	-	97.2	-	6146.0
14. Spain	-	190.0	-	-	190.0
15. Germany ¹	35.0	-	-	-	35.0
16. Japan	-	5.0	-	-	5.0
17. Sweden	-	-	-	-	-
18. U.S.A.	16714.0	-	217.0	-	16931.0
19. Yugoslavia	-	-	-	-	-
20. EBloc ²	na	na	na	na	na
TOTAL	27851.43	5166.61	10606.29	545.67	44170.0

Source: IAEA, Uranium Institute and USDOE

1. Federal Republic

2. Bulgaria, Czechoslovakia, GDR, Hungary, Romania, USSR and PRC - 100% state owned

* All figures are for production except Argentina, Australia and Belgium which are production capacity ECON/APK/MC/IAEA

COUNTRY	1980 (%)				
	PRIVATE	GOVERNMENT	PRIVATE	GOVERNMENT	TOTAL
	DOMESTIC %	DOMESTIC %	FOREIGN %	FOREIGN %	%
1. Argentina	-	100.0	-	-	100.0
2. Australia	71.0	17.0	12.0	-	100.0
3. Belgium	100.0	-	-	-	100.0
4. Brazil	-	-	-	-	-
5. Canada	49.6	5.9	44.5	-	100.0
6. France	15.0	85.0	-	-	100.0
7. Gabon	0.99	24.75	74.26	-	100.0
8. India	-	100.0	-	-	100.0
9. Namibia	-	-	86.5	13.5	100.0
10. Niger	-	31.99	68.01	-	100.0
11. Pakistan	-	100.0	-	-	100.0
12. Portugal	-	100.0	-	-	100.0
13. South Africa	98.42	-	1.58	-	100.0
14. Spain	-	100.0	-	-	100.0
15. Germany ¹	100.0	-	-	-	100.0
16. Japan	-	100.0	-	-	100.0
17. Sweden	-	-	-	-	-
18. U.S.A.	98.72	-	1.28	-	100.0
19. Yugoslavia	-	-	-	-	-
20. EBloc ²	-	100.0	-	-	100.0
TOTAL	63.0	11.7	24.0	1.2	100.0

1. Federal Republic

2. Bulgaria, Czechoslovakia, GDR, Hungary, Romania, USSR and PRC - 100% state-owned

Note: percentage shares are based on production excepting Argentina, Australia and Belgium which are based on production capacity.

COUNTRY	1985				
	PRIVATE	GOVERNMENT	PRIVATE	GOVERNMENT	TOTAL*
	DOMESTIC %	DOMESTIC %	FOREIGN %	FOREIGN %	tU
1. Argentina	42.5	182.5	-	-	225.0
2. Australia	2695.5	-	704.5	-	3400.0
3. Belgium	40.0	-	-	-	40.0
4. Brazil	-	420.0	-	-	420.0
5. Canada	3495.1	3838.9	3546.0	-	10880.0
6. France	446.46	2742.54	-	-	3189.0
7. Gabon	9.39	232.4	697.21	-	939.0
8. India	-	200.0	-	-	200.0
9. Namibia	-	-	2926.3	456.7	3383.0
10. Niger	-	1009.89	2171.11	-	3181.0
11. Pakistan	-	30.0	-	-	30.0
12. Portugal	-	119.0	-	-	119.0
13. South Africa	4758.1	-	124.9	-	4883.0
14. Spain	-	201.0	-	-	201.0
15. Germany	31.0	-	-	-	31.0
16. Japan	-	7.0	-	-	7.0
17. Sweden	-	-	-	-	-
18. U.S.A.	3941.0	-	378.0	-	4319.0
19. Yugoslavia	-	30.0	-	-	30.0
20. EBloc ²	na	na	na	na	na
TOTAL	15459.05	9013.23	10548.0	456.7	35477.0

Source: IAEA, Uranium Institute and USDOE

1. Federal Republic

2. Bulgaria, Czechoslovakia, GDR, Hungary, Romania, USSR and PRC - 100% state-owned

* All figures are for production except Argentina, Australia, Belgium and Brazil which is production capacity.
ECON/APK/MC/IAEA

COUNTRY	1985 (%)				
	PRIVATE	GOVERNMENT	PRIVATE	GOVERNMENT	TOTAL
	DOMESTIC	DOMESTIC	FOREIGN	FOREIGN	
	%	%	%	%	%
1. Argentina	19.0	81.0	-	-	100.0
2. Australia	79.3	-	20.7	-	100.0
3. Belgium	100.0	-	-	-	100.0
4. Brazil	-	100.0	-	-	100.0
5. Canada	32.1	35.3	32.6	-	100.0
6. France	14.0	86.0	-	-	100.0
7. Gabon	1.0	24.75	74.25	-	100.0
8. India	-	100.0	-	-	100.0
9. Namibia	-	-	86.5	13.5	100.0
10. Niger	-	31.75	68.25	-	100.0
11. Pakistan	-	100.0	-	-	100.0
12. Portugal	-	100.0	-	-	100.0
13. South Africa	97.44	-	2.56	-	100.0
14. Spain	-	100.0	-	-	100.0
15. Germany ¹	100.0	-	-	-	100.0
16. Japan	-	100.0	-	-	100.0
17. Sweden	-	-	-	-	-
18. U.S.A.	91.25	-	8.75	-	100.0
19. Yugoslavia	-	100.0	-	-	100.0
20. EBloc	-	100.0	-	-	100.0
TOTAL	43.6	25.4	29.7	1.3	100.0

1 Federal Republic

Note percentage shares are based on production excepting Argentina, Australia, Belgium and Brazil which are based on production capacity

COUNTRY	1990				
	PRIVATE	GOVERNMENT	PRIVATE	GOVERNMENT	TOTAL*
	DOMESTIC	DOMESTIC	FOREIGN	FOREIGN	tU
1. Argentina	72.5	72.5	-	-	145.0
2. Australia	2691.0	-	1409.0	-	4100.0
3. Belgium	40.0	-	-	-	40.0
4. Brazil	-	420.0	-	-	420.0
5. Canada	2145.0	3462.32	3091.68	-	8699.0
6. France	253.44	2562.56	-	-	2816.0
7. Gabon	7.09	175.48	526.43	-	709.0
8. India	-	150.0	-	-	150.0
9. Namibia	-	112.4	2777.5	321.1	3211.0
10. Niger	-	926.32	1904.68	-	2831.0
11. Pakistan	-	-	-	-	-
12. Portugal	-	110.0	-	-	110.0
13. South Africa	2432.0	-	37.0	-	2469.0
14. Spain	-	212.0	-	-	212.0
15. Germany	-	-	-	-	-
16. Japan	-	-	-	-	-
17. Sweden	-	-	-	-	-
18. U.S.A.	2325.6	-	991.8	102.6	3420.0
19. Yugoslavia	-	60.0	-	-	60.0
20. EBloc ¹	na	na	na	na	na
TOTAL	9966.6	8263.6	10738.1	423.7	29392.0

Source: IAEA, Uranium Institute and USDOE

1 Bulgaria, Czechoslovakia, Hungary, Romania, USSR and PRC, 1990 GDR production committed to USSR - 100% state-owned
* All figures are for production except Argentina, Australia, Belgium and Brazil which is production capacity.
ECON/APK/MC/IAEA

COUNTRY	1990 (%)				TOTAL %
	PRIVATE	GOVERNMENT	PRIVATE	GOVERNMENT	
	DOMESTIC %	DOMESTIC %	FOREIGN %	FOREIGN %	
1. Argentina	50.0	50.0	-	-	100.0
2. Australia	65.6	-	34.4	-	100.0
3. Belgium	100.0	-	-	-	100.0
4. Brazil	-	100.0	-	-	100.0
5. Canada	24.7	39.8	35.5	-	100.0
6. France	9.0	91.0	-	-	100.0
7. Gabon	1.0	24.75	74.25	-	100.0
8. India	-	100.0	-	-	100.0
9. Namibia	-	3.5	86.5	10.0	100.0
10. Niger	-	32.72	67.28	-	100.0
11. Pakistan	-	-	-	-	-
12. Portugal	-	100.0	-	-	100.0
13. South Africa	98.5	-	1.5	-	100.0
14. Spain	-	100.0	-	-	100.0
15. Germany	-	-	-	-	-
16. Japan	-	-	-	-	-
17. Sweden	-	-	-	-	-
18. U.S.A.	68.0	-	29.0	3.0	100.0
19. Yugoslavia	-	100.0	-	-	100.0
20. EBloc	-	100.0	-	-	100.0
TOTAL	34.0	28.1	36.5	1.4	100.0

Note percentage shares are based on production excepting Argentina, Australia, Belgium and Brazil which are based on production capacity

TAXATION IMPACT ON URANIUM MINING IN CANADA AND AUSTRALIA

R T WHILLANS

Electricity Branch,
Energy, Mines and Resources Canada,
Ottawa, Ontario,
Canada

Abstract

The impact of taxation on the economic viability and competitive position of the uranium industry was studied using four model deposits types as they occur or could occur in different provinces in Australia and Canada. On the basis of project characteristics and assumed uranium prices, before tax indicators such as net revenue, undiscounted cash flow, net present value and rate of return were determined and subjected to the tax regimes in the different jurisdictions using assumed varying levels of cost of capital. Under the assumed conditions, it was found that among the internal factors the uranium prices have a very significant impact on the economics of the model operations. Looking at external factors, the tax systems evaluated in these model operations were found to have a decisive influence on the viability of the operation. Among the fiscal regimes tested, the Saskatchewan royalty system and Australia's Northern Territory tax system generate the highest government revenue but provide the least incentive for investment. However, despite having the highest effective tax rate, the Saskatchewan royalty was found to be attractive at the investment margin and flexible as its progressive nature provides for increasing tax rates as profitability increases. In contrary, all the other systems are regressive, with relative high tax burdens at the investment margin and decreasing burdens as a function of profitability.

Introduction

As principal competitors in the uranium export market and major players in the world's uranium industry, Canada and Australia have often invited comparison. The impact of taxation on the economic viability and competitive position of uranium mining industries in the two countries was examined in detail in a recent Canadian study. As you will see, taxation can be an important factor in making or breaking a uranium mining project in any country.

Reference is made to a report prepared jointly by the Centre for Resource Studies (CRS) Queen's University at Kingston, Ontario, and the Uranium Division - Electricity Branch of the Department of Energy, Mines and Resources, Ottawa. The report will be published as a CRS Monograph in late August 1991, and those interested are invited to examine its findings.

Without going into great detail, the approach and the results of this study will be briefly summarized, and then some specific details from the report will be provided. It should be pointed out that the study focused solely upon the effects of taxation. While the tax structure is undoubtedly an important component in determining economic viability and competitive

position, there are other factors of equal importance, such as geological potential, economic conditions, and political climate, that were not taken into account in the study.

Method

To keep the study manageable, the evaluation was based on the four types of uranium deposits of primary economic interest in the two countries. As Canada and Australia have similar geological settings, the provinces/states most likely to host one or more of these four deposit types were identified, three in each country, and the tax jurisdictions of each taken into consideration.

Table 1 lists the four deposit types, and indicates in which of the six jurisdictions such deposits have been, or are likely to be, discovered. The names of some deposits or occurrences already discovered are also noted. Profiles were constructed around hypothetical deposits assumed to have been discovered and delineated, and awaiting a mine development decision. The

TABLE 1

URANIUM DEPOSIT-TYPES BY LOCATION

Uranium Deposit-types by Location	Deposit-Type			
	High-Grade Unconformity	Quartz-Pebble Conglomerate	Conventional-Grade Unconformity	Uranium-Polymetallic
Saskatchewan	Key Lake, Cigar Lake, Collins Bay, Eagle Point	Not Favourable	Dominique-Peter, Maurice Bay	Not Favourable
Ontario	Not Favourable	Elliot Lake	Sibley Basin	Not Favourable
Northwest Territories	Perhaps Favourable	McInnes Lake (occurrence)	Kiggavik	Port Radium, Bear Province
Northern Territory	Nabarlek I & II	Favourable	Ranger, Koongarra, Jabiluka I & II	Coronation Hill
South Australia	Not Favourable	Not Favourable	Not Favourable	Olympic Dam
Western Australia	Perhaps Favourable	Mentheena Basin	Kintyre	Not Favourable

four project profiles were synthesized from actual situations to provide scenarios as realistic as possible.

Table 2 summarizes the characteristics of the four representative uranium projects, providing such data as ore reserves and grade, plant capacity, preproduction expenditures etc. etc. As market price is both the most important and most uncertain variable in the assessment of cash flows, the evaluations were carried out as a function of price.

As shown in Table 3, the expected or mean price outlook is bounded by an upper- and a lower-limit price for uranium, with appropriate exchange rates for each limit. These limits were required to evaluate the time distribution of revenues for the projects. With cost of capital as the other principal variant for sensitivity analysis, a base case was established for comparison

TABLE 2

SUMMARY CHARACTERISTICS OF REPRESENTATIVE URANIUM PROJECTS

Summary Characteristics of Representative Uranium Projects	Uranium Projects			
	High-Grade Unconformity	Quartz-Pebble Conglomerate	Conventional-Grade Unconformity	Uranium-Polymetallic
Recoverable Ore Reserves (tonnes & grade)	3,500,000 2.45% U ₃ O ₈	40,000,000 0.14% U ₃ O ₈	5,000,000 0.50% U ₃ O ₈	75,000,000 0.16% U ₃ O ₈
Mining Method	Open pit	Underground	Open pit	Underground
Capacity (tonnes/year)	250,000	1,500,000	500,000	2,500,000
Preproduction Period (years)	3	4	3	4
Preproduction Capital Expenditures (\$ million)	619	513	340	872
Employment during Production (number of persons)	350	600	250	1,050
Production Costs (\$/tonne)	305	50	102	70
Mill Recovery (%)	98	94	96	94
Recoverable Product (tonnes)	84,000	56,600	24,000	110,400

TABLE 3

MARKET PRICE PROJECTIONS

Market Price Projections	Lower-Limit Prices	Expected Prices	Upper-Limit Prices
Uranium (\$US/lb U ₃ O ₈)	20	30	40
Exchange Rate (\$US/\$C)	0.90	0.80	0.70

purposes using an expected price of \$US 30/lb U₃O₈, and a 10 per cent cost of capital. To be most suitable for tax analysis, the profiles were designed to give a good economic spread of before-tax returns, varying under base case conditions from marginally economic to highly profitable.

It should be noted that to achieve the desired make-up, the project profiles tend to portray superior examples of the respective deposit types in terms of ore reserves and costs. By normal mining standards all four are relatively large. To get a feel for these projects, they were first evaluated on a potential value or before-tax basis, that is, to measure their attractiveness from a mining company perspective.

Before-tax Evaluation

Table 4 shows the before-tax evaluations of net revenue, undiscounted cash flow, net present value at 10 per cent, and rate of return for the four project profiles. Using these criteria, the minimum acceptable condition for an economic project is a net present value equal to zero and a rate of return equal to the cost of capital.

Total sales revenue varies from \$2 to \$13 billion, rate of return from 14 to 45 per cent, and net present value from \$151 to \$1,820 million. Despite there being roughly a six-fold difference in the revenue-generating attribute of these potential mining projects, all are economic before tax under the assumed base case conditions. Ranging from marginally economic to highly profitable, all four projects have the potential to create substantial amounts of new wealth. The economic indicators evaluated are highly sensitive to future market prices, especially net present value.

After-tax Evaluation

The four project models, initially appraised on a before-tax basis, were then subjected to taxation in the six noted jurisdictions, selected as being geologically prospective environments for the types of deposit evaluated. In each jurisdiction, several taxation criteria were assessed

TABLE 4

BEFORE-TAX ECONOMIC INDICATORS
UNDER BASE CASE CONDITIONS

Before-tax Economic Indicators Under Base Case Conditions	Uranium Project			
	High-Grade Unconformity	Quartz-pebble Conglomerate	Conventional-grade Unconformity	Uranium-polymetallic
Net Revenue (\$ million)	6,947	4,352	1,984	12,763
Undiscounted Cash Flow (\$ million)	5,281	1,856	1,149	6,682
Net Present Value at 10% (\$million)	1,817	151	404	928
Rate of Return (%)	45	14	31	21

from a government policy viewpoint, including after-tax measures of investment incentive, discounted tax revenues, effective tax rates, intergovernmental tax shares, and comparative tax levels. Not only were the impacts of taxation shown to be both high and variable, but the ranking of the six tax jurisdictions changed somewhat as a function of the deposit type, market price, and cost of capital variants.

Table 5 illustrates the effective tax rate ranges in the six jurisdictions for the expected study conditions. Generally, the taxation systems in Saskatchewan and Australia's Northern Territory generate the most government revenue and provide the lowest incentive for investment.

This should not be surprising, as their tax systems have obviously been designed in consideration of the relatively profitable unconformity-related uranium deposits discovered and developed in these jurisdictions. At the opposite extreme, Canada's Northwest Territories and Ontario offer the best investment incentive and collect the least amount of taxes. Western Australia and South Australia fall in the middle of the spectrum.

Despite having the highest effective tax rate, Saskatchewan's uranium royalty system is the only progressive regime, capturing an increasing proportion of before-tax net present value as project profitability rises. As such, the development of uranium deposits in Saskatchewan is attractive at the investment margin. In all the other jurisdictions, the tax system is regressive, placing the greatest tax burden at the investment margin and decreasing it as a function of profitability. In Western Australia and South Australia, revenue-based taxation has a particularly distorting effect on investment.

TABLE 5

EFFECTIVE TAX RATES BY JURISDICTION

Effective Tax Rate Ranges for Expected Study Conditions (%)	Jurisdictions
43 to 51	Northwest Territories
48 to 59	Ontario
50 to 64	Western Australia
52 to 64	South Australia
52 to 70	Northern Territory
68 to 71	Saskatchewan

General Conclusions

Although the study focuses on the impact of taxation on uranium mining in Canada and Australia, important inferences may be drawn that are of global relevance. By choosing projected market price conditions that are significantly above current spot prices and are viewed by many as being on the optimistic side even in the longer term, and by selecting model projects with superior ore reserve and cost characteristics, an attempt was made to give a good economic range of before-tax returns.

However, as the price of uranium and the cost of capital were varied, even under the idealistic conditions assumed, a large number of project situations became uneconomic when taken from a before- to an after-tax status. The study demonstrates quite clearly that only the best uranium mining projects have a chance of being developed under present market conditions. As those conditions will likely become more stringent in the future, as governments and the public demand that all projects be environmentally neutral, the economic justification of uranium exploration will require quite exceptional targets.

Interesting insights were also gained on the circumstances under which these uranium projects would not be economic to develop. Potentially economic mining projects were rendered non-viable as a result of inefficient taxation policies. When the actual value to be realized from the project falls short of the potential value which exists, the wealth-creating, employment, income, and foreign exchange benefits associated with uranium mining are thereby diminished.

One might ask: "to what extent does taxation of uranium mining distort mining company investment decisions?" Projects that are barely economic before tax, and become uneconomic after tax, are victims of a system providing investment disincentives; this represents an important type of government inefficiency. The draft report of the Australian government's Industry Commission for its inquiry into the mining and minerals processing industries in Australia is

particularly critical of the inefficiency and economic distortion associated with existing revenue-based state royalty arrangements.

As noted previously, the Saskatchewan royalty system, the only progressive tax regime evaluated, captures an increasing proportion of before-tax net present value, but provides for more incentive at the investment margin. This illustrates that the progressive tax structure offers a competitive advantage, which is of particular importance considering today's depressed uranium prices.

Specific Details

The different combination of assumptions under the market price projections and various economic indicators applied to the six tax jurisdictions makes it difficult to summarize easily the advantages of a particular deposit type or a certain jurisdiction. However, some specific details arising from this study can be illustrated with the help of two sets of tables.

In the first set (Tables 6a-6d), the effects of taxation under varying uranium price conditions using a 10 per cent cost of capital are compared on a before-tax and after-tax basis for the four deposit types. Clearly, there are circumstances where some deposit types are uneconomic even on a before-tax basis, and many do not become economic on an after-tax basis until uranium prices reach \$US 30/lb U_3O_8 . As might be expected, quartz-pebble conglomerate deposits are the most vulnerable, whereas high-grade unconformity deposits are the least affected.

In the second set (Tables 7a-7b), the effects of taxation under varying costs of capital using an expected uranium price of \$US 30/lb U_3O_8 are compared on a before-tax and after-tax basis for the four deposit types. Again, varying the cost of capital has a dramatic effect on the economic viability of developing the four deposit types. With a uranium price of \$US 30/lb, all four deposit types are economic in all jurisdictions at 5 per cent cost of capital. However, when the cost of capital reaches 10 per cent, quartz-pebble conglomerate deposits become uneconomic in four of the six jurisdictions, and at a 15 per cent cost of capital, uranium-polymetallic deposits become uneconomic in the same four jurisdictions. With \$US 30/lb uranium, it comes as no surprise that conventional-grade and high-grade unconformity deposits are economic in all jurisdictions, even with a 15 per cent cost of capital.

Although not shown in tables in the report, or in this paper, conventional-grade unconformity deposits do become economic in all but one jurisdiction if a 5 per cent cost of capital is assumed with \$US 20/lb uranium; interestingly, high-grade unconformity deposits remain economic in all jurisdictions even assuming a 15 per cent cost of capital with \$US 20/lb uranium.

In conclusion, it has been observed that only the very best uranium projects have a chance of being developed under present market conditions, especially with prices below \$20/lb U_3O_8 . If regulatory and environmental conditions lead to higher operating and capital costs, and if uranium prices do not recover, the economic justification to explore for the deposit types studied here means that quite exceptional targets will be required. Perhaps unconventional sources of uranium, for example *in-situ* leachable deposits, will be examined with renewed interest in the near future.

TABLE 6a

TAXATION EFFECT ON NET PRESENT VALUE & RATE OF RETURN
UNDER VARYING URANIUM PRICES AND A 10% COST OF CAPITAL

QUARTZ-PEBBLE CONGLOMERATE DEPOSITS				
Net Present Value at 10% (\$ millions)		Uranium Prices		
		\$US 20/lb U ₃ O ₈	\$US 30/lb U ₃ O ₈	\$US 40/lb U ₃ O ₈
<i>Before-Tax Results</i>		-267	151	689
<i>After-Tax Results</i>	Ontario	-276	1	307
	Northwest Territories	-274	27	359
	Saskatchewan	-304	-21	160
	Northern Territory	-324	-52	256
	South Australia	-293	-28	276
	Western Australia	-303	-34	277
Rate of Return (%) <i>Before-Tax Results</i>		1	14	24
<i>After-Tax Results</i>	Ontario	-1	10	18
	Northwest Territories	0	11	19
	Saskatchewan	-2	9	15
	Northern Territory	-3	8	16
	South Australia	-1	9	17
	Western Australia	-2	9	17

TABLE 6b

TAXATION EFFECT ON NET PRESENT VALUE & RATE OF RETURN
UNDER VARYING URANIUM PRICES AND A 10% COST OF CAPITAL

URANIUM-POLYMETALLIC DEPOSITS				
Net Present Value at 10% (\$ millions)		Uranium Prices		
		\$US 20/lb U ₃ O ₈	\$US 30/lb U ₃ O ₈	\$US 40/lb U ₃ O ₈
<i>Before-Tax Results</i>		-222	928	2,434
<i>After-Tax Results</i>	Ontario	-303	379	1,214
	Northwest Territories	-289	454	1,375
	Saskatchewan	-357	285	674
	Northern Territory	-404	277	1,139
	South Australia	-338	333	1,157
	Western Australia	-350	337	1,219
Rate of Return (%) <i>Before-Tax Results</i>		6	21	34
<i>After-Tax Results</i>	Ontario	4	16	27
	Northwest Territories	5	17	27
	Saskatchewan	3	15	21
	Northern Territory	2	14	24
	South Australia	3	15	24
	Western Australia	3	15	25

TABLE 6c

TAXATION EFFECT ON NET PRESENT VALUE & RATE OF RETURN
UNDER VARYING URANIUM PRICES AND A 10% COST OF CAPITAL

CONVENTIONAL-GRADE UNCONFORMITY DEPOSITS				
Net Present Value at 10% (\$ millions)		Uranium Prices		
		\$US 20/lb U ₃ O ₈	\$US 30/lb U ₃ O ₈	\$US 40/lb U ₃ O ₈
<i>Before-Tax Results</i>		31	404	884
<i>After-Tax Results</i>	Ontario	-28	190	467
	Northwest Territories	-21	208	501
	Saskatchewan	-43	116	227
	Northern Territory	-75	139	413
	South Australia	-55	155	415
	Western Australia	-62	155	433
Rate of Return (%)				
<i>Before-Tax Results</i>		12	31	47
<i>After-Tax Results</i>	Ontario	8	22	36
	Northwest Territories	8	18	36
	Saskatchewan	7	18	26
	Northern Territory	4	19	31
	South Australia	6	20	32
	Western Australia	5	19	32

TABLE 6d

TAXATION EFFECT ON NET PRESENT VALUE & RATE OF RETURN
UNDER VARYING URANIUM PRICES AND A 10% COST OF CAPITAL

HIGH-GRADE UNCONFORMITY DEPOSITS				
Net Present Value at 10% (\$ millions)		Uranium Prices		
		\$US 20/lb U ₃ O ₈	\$US 30/lb U ₃ O ₈	\$US 40/lb U ₃ O ₈
<i>Before-Tax Results</i>		698	1,817	3,256
<i>After-Tax Results</i>	Ontario	319	951	1,760
	Northwest Territories	358	1,040	1,913
	Saskatchewan	248	589	869
	Northern Territory	225	865	1,688
	South Australia	266	875	1,675
	Western Australia	263	912	1,745
Rate of Return (%)				
<i>Before-Tax Results</i>		27	45	63
<i>After-Tax Results</i>	Ontario	20	35	49
	Northwest Territories	20	35	48
	Saskatchewan	18	28	35
	Northern Territory	16	30	44
	South Australia	18	31	44
	Western Australia	17	31	45

TABLE 7a

TAXATION EFFECT ON NET PRESENT VALUE UNDER VARYING COSTS OF CAPITAL USING AN EXPECTED URANIUM PRICE OF \$US 30/LB U₃O₈

QUARTZ-PEBBLE CONGLOMERATE DEPOSITS				
Net Present Value at Varying Costs of Capital (\$ millions)		Cost of Capital		
		5%	10%	15%
<i>Before-Tax Results</i>		606	151	-35
<i>After-Tax Results</i>	Ontario	260	1	-107
	Northwest Territories	324	27	-96
	Saskatchewan	224	-21	-123
	Northern Territory	191	-52	-147
	South Australia	231	-28	-131
	Western Australia	224	-34	-137

TABLE 7b

TAXATION EFFECT ON NET PRESENT VALUE UNDER VARYING COSTS OF CAPITAL USING AN EXPECTED URANIUM PRICE OF \$US 30/LB U₃O₈

URANIUM-POLYMETALLIC DEPOSITS				
Net Present Value at Varying Costs of Capital (\$ millions)		Cost of Capital		
		5%	10%	15%
<i>Before-Tax Results</i>		2,405	928	324
<i>After-Tax Results</i>	Ontario	1,178	379	44
	Northwest Territories	1,379	454	74
	Saskatchewan	994	285	-11
	Northern Territory	1,071	277	-40
	South Australia	1,159	333	-3
	Western Australia	1,188	337	-5

TABLE 7c

TAXATION EFFECT ON NET PRESENT VALUE UNDER VARYING COSTS OF CAPITAL USING AN EXPECTED URANIUM PRICE OF \$US 30/LB U₃O₈

CONVENTIONAL-GRADE UNCONFORMITY DEPOSITS				
Net Present Value at Varying Costs of Capital (\$ millions)		Cost of Capital		
		5%	10%	15%
<i>Before-Tax Results</i>		683	404	232
<i>After-Tax Results</i>	Ontario	356	190	88
	Northwest Territories	390	208	97
	Saskatchewan	244	116	36
	Northern Territory	295	139	44
	South Australia	317	155	57
	Western Australia	318	155	56

TABLE 7d

TAXATION EFFECT ON NET PRESENT VALUE UNDER VARYING COSTS OF CAPITAL USING AN EXPECTED URANIUM PRICE OF \$US 30/LB U₃O₈

HIGH-GRADE UNCONFORMITY DEPOSITS				
Net Present Value at Varying Costs of Capital (\$ millions)		Cost of Capital		
		5%	10%	15%
<i>Before-Tax Results</i>		3,039	1,187	1,114
<i>After-Tax Results</i>	Ontario	1,627	951	557
	Northwest Territories	1,803	1,040	601
	Saskatchewan	1,063	589	314
	Northern Territory	1,552	865	473
	South Australia	1,563	875	482
	Western Australia	1,625	912	505

LIST OF PARTICIPANTS

Aikas, O	Geological Survey of Finland P O Box 1237 SF-70701 Kuopio Finland
Arkimaa, H	Geological Survey of Finland Betonimiehenkuja 4 SF-02760 Espoo Finland
Ayatollahi, M.-R	Atomic Energy Organization of Iran North Karegar Avenue P O Box 14155 1339, Tehran Islamic Republic of Iran
Ballery, J -L	C E A./D.D AMN CEN de Saclay Bât 476 91191 Gif-sur Yvette, Cedex France
Barthel, F.H (Session Chairman)	Bundesanstalt für Geowissenschaften und Rohstoffe Stilleweg 2 P.O. Box 510153 D-W-3000 Hannover 51 Germany
Bejenaru, C.	Regia Autonoma Pentru Metale Rare Bucarest Romania
Bhasin, J	Uranium Corporation of India, Ltd Hyderabad, India
Bobe, M	Regia Autonoma Pentru Metale Rare Feldioara, Romania
Bojkov, I B.	Economic Enterprise Rare Metals 1830 Sofia Bulgaria

Carisey, J.-C.

COGEMA
BP 4
2, rue Paul-Dautier
F-78140 Vélizy-Villacoublay Cedex
France

Civin, V

MEGA Research and Development Inst
471 27 Stráz pod Ralskem
Czechoslovakia

Delorme, D

COGEMA
BP 4
2, rue Paul Dautier
F-78140 Vélizy-Villacoublay Cedex
France

Ey, F

MINATCO Ltd
1240 - 202 - 6th Avenue S W
Calgary, Alberta
Canada T2P 2R9

Feng, Shen

Bureau of Geology
China National Nuclear Corporation
P.O. Box 1436
Beijing 100013
Peoples's Republic of China

Finch, W.I.
(Session Chairman)

U S. Geological Survey
Office of Energy and Marine Geology
Branch of Sedimentary Processes
Box 25046, M S 939
Denver Federal Center
Denver, Colorado 80225
United States of America

François, A P

Avenue du Hoef, 26 bte 1
B-1180 Bruxelles
Belgium

Gatzweiler, R.

Uranerzbergbau-GmbH
P O Box 1638
Kolner Strasse 38-44
D-W-5047 Wesseling
Germany

Giroux, M	COGEMA BP 4 2, rue Paul-Dautier 78140 Vélizy-Villacoublay Cedex France	Kidd, A	RTZ Limited 6 St. James's Square London SW1Y 4LD United Kingdom
Hähne, R	SDAG Wismut Jagdschankenstrasse 29 D-O-9030 Chemnitz Germany	Lapin, A L	Ministry of Geology of the USSR Concern Geologorazvedka 4, Marshal Rybalko str Moscow 123436 Russian Federation
Hambleton-Jones, B B	Earth and Environmental Technology Atomic Energy Corporation of South Africa Limited P O Box 582 Pretoria 001 South Africa	Lightner, J	Nuexco Europe Limited The Exchange 71 Victoria Street London SW1H 0HW United Kingdom
Hampel, H	Western Mining Corporation Holdings Limited, Melbourne Frankfurt Representation Junghofstrasse 16 D-W-6000 Frankfurt am Main 1 Germany	Lindholm, I	Swedish Nuclear Fuel and Waste Management Co Box 5864 S 102 48 Stockholm Sweden
Hogg, G	Permanent Mission of Australia to the IAEA Mattiellistrasse 2-4/III A-1040 Vienna Austria	Luchin, I A	Ministry of Geology of the USSR Rudgeophysika 20, Fayansovaya str Leningrad 193019 Russian Federation
Hussein, H A M	Nuclear Materials Authority P O Box 530 El Maadi, Cairo Egypt	Matos Dias, J. (Session Chairman)	Empresa Nacional de Uranio, S.A. URGEIRIÇA 3525 Canas de Senhorim Portugal
Ishido, A	Power Reactor and Nuclear Fuel Development Corporation International Division 4/8 rue Sainte Anne F-75001 Paris France	Matthews, R	CAMECO Corporation 2121-11th Street West Saskatoon, Saskatchewan Canada S7M 1J3
Jansky, J	The Czechoslovak Uranium Industry ul 28. října 184 261 02 - Příbram VII Czechoslovakia	Moghal, M Y	Atomic Energy Minerals Centre Ferozepur Road P O Box 658 Lahore - 16 Pakistan

Mühlstedt, P	SDAG Wismut Jagdschankenstrasse 29 D-O-9030 Chemnitz Germany	Pluskal, O	Faculty of Sciences Charles University Albertov 6 128 43 Praha 2 Czechoslovakia
Naumov, S S	Ministry of Geology of the USSR Concern Geologorazvedka 4, Marshal Rybalko str Moscow 123436 Russian Federation	Pollock, G	CAMECO Corporation 2121 11th Street West Saskatoon, Saskatchewan Canada S7M 1J3
Navarra, P R	Comisión Nacional de Energia Atómica Azopardo 313 5501 Godoy Cruz, Mendoza Argentina	Puustinen, K	Geological Survey of Finland Betonimiehenkuja 4 SF 02150 Espoo Finland
Nguyen, Van Hoai	Geological Division No 10 Geological Department Ministry of Heavy Industries of Vietnam Liên đoàn địa chất 10 Cau Dien, Tu'liem, Hanoi Viet Nam	Pype, J	SYNATOM Avenue marnix 13 B 1050 Brussels Belgium
Nikitin, M	Ministry of Atomic Power and Industry Staromonetny pereulok 26 Moscow 109180 Russian Federation	Sayyah, T A	Nuclear Materials Authority P O Box 530 El Maadi, Cairo Egypt
Nikodem, Z D	Nuclear and Alternate Fuels Division Office of Coal, Nuclear, Electric and Alternate Fuels Energy Information Administration U S Department of Energy Washington, DC 20585 United States of America	Simov, S D (Session Chairman)	Geological Committee of Bulgaria Bldg "G Dimitrov" 22 Sofia Bulgaria
Nouza, R	Ministry for Economy Policy and Development Vrsovicá trida 65 101 60 Praha 10 Czechoslovakia	Solaimani, Gh	Atomic Energy Organization of Iran North Karegar Avenue P O Box 14155 1339, Tehran Islamic Republic of Iran
Parslow, G R	Department of Geology University of Regina Regina, Saskatchewan Canada S4S 0A2	Spross, W	Urangesellschaft mbH Solmstrasse 2-26 P O Box 900980 D-W-6000 Frankfurt 90 Germany
		Tassinari, C C G	Universidade de Sao Paulo Rua do Lago 562 Caixa Postal 20 899 SP 01498 - Sao Paulo Brazil

Vels, B.	Uranerzbergbau-GmbH P.O. Box 1638 Kölner Strasse 38-44 D-W-5047 Wesseling Germany	Whillans, R.T.	Energy, Mines and Resources Canada Electricity Branch Ottawa, Ontario Canada K1A 0E4
Vetrov, V.I.	Ministry of Atomic Power and Industry Staromonetny pereulok 26 Moscow 109180 Russian Federation	Wilde, A.R	BHP Minerals 801 Glenferrie Road P.O. Box 619 Hawthorn, Victoria 3122 Australia
Volkman, Y.	Israel Atomic Energy Commission Negev Nuclear Research Center P.O. Box 9001 Beer-Sheva, Israel	Joosten, J (Co-sponsor)	OECD Nuclear Energy Agency
Weber, L.	Federal Ministry Economic Affairs Supreme Mining Authority Sektion VII/3 Landstrasse Hauptstrasse 55-57 A-1030 Vienna Austria	Muller-Kahle, E (Scientific Secretary)	International Atomic Energy Agency Division of Nuclear Fuel Cycle and Waste Management