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COMISION INTERAMERICANA DE ENERGIA NUCLEAR Y
COMISION NACIONAL DE ENERGIA ATOMICA DE LA REPUBLICA ARGENTINA

CURSO LATINOAMERICANO DE CAPACITACION PARA LA PROSPECCION Y EXPLORACION DE YACIMIENTOS URANIFEROS

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RECURSOS URANIFEROS MUNDIALES
EVALUACION DE YACIMIENTOS DE URANIO

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COMISION NACIONAL DE ENERGIA ATOMICA

BUENOS AIRES
OCTUBRE 1978

URANIUM: AN INTRODUCTION

Uranium is a silvery white metal with an atomic weight of 238 but actually consiting of three semistable radioactive isotopes U238 (99.3%), U235 (0.7%) and U234 (0.005%). It is an important energy source because fission of isotope U235 releases large amounts of energy.

Uranium was discovered in 1789 by Marton Klaproth in pitchblende from a mine in Germany and was first isolated in 1842. Radioactivity was discovered in 1896, and radium a daughter of uranium decay was discovered by the Curies and Bemont in 1898 in pithcblende from Joachymov (Joachimstal), Czechoslovakia, where the mineral had been known since 1727. From the early 1900's radium became important in medical therapy and thus led to the search for and development of uranium deposits in all parts of the world.

The average percentage (abundance) of uranium in the Earth's crust (the clarke) is 0.0002.[1/] Uranium in the Earth's crust is approximately twice as abundant as molybdenum or tungsten and ten times more abundant than antimony and bismuth and four hundred times more abundant than gold. The relative abundance of uranium and typical occurrences is illustrated in Figure 1 taken from a paper by P.H. Dodd[2/].

The matter of importance to the economic geologist and the mining industry is the location of concentrations of uranium with economic tenors. The formation of deposits has been attributed to a limited number of well recognized geological processes, physical and/or chemical controls. In the case of uranium, one major factor in the distribution of deposits appears to be association with the early evolution of the Earth's crust.

Over 90% of the known low-cost uranium deposits of the world occur either in Precambrian rocks or in Phanerozoic rocks immediate overlying the basement. (Figure 2 and reference 3/). The really large contributions to ore reserves are made by only a few well defined areas of the world (Figure 3). Such areas are known as metallogenic provinces and such provinces have, of course, been defined for many different metals.

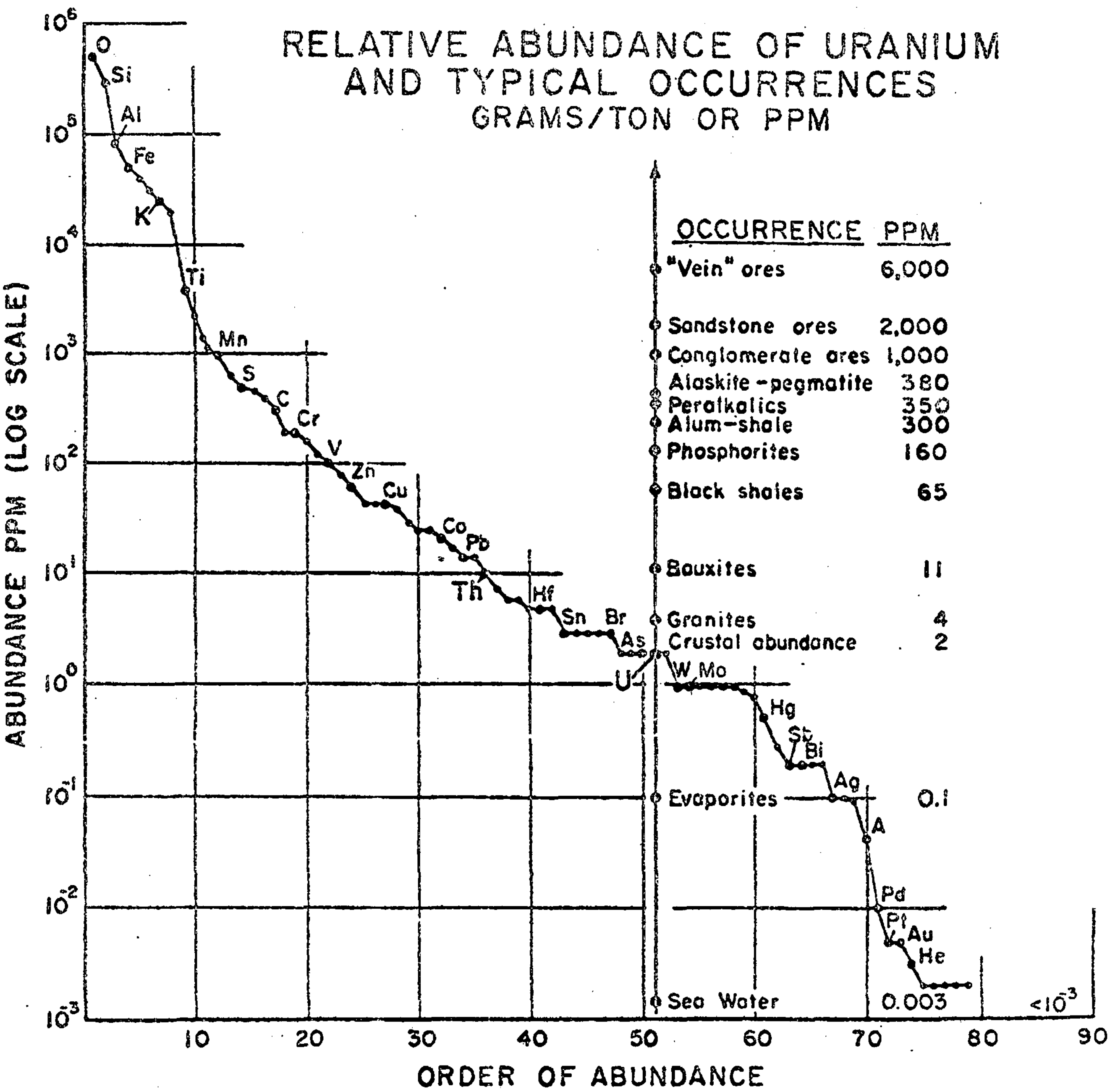

Fig. 1. Relative Abundance of Uranium and typical Occurences (Grams/Ton or PPM)

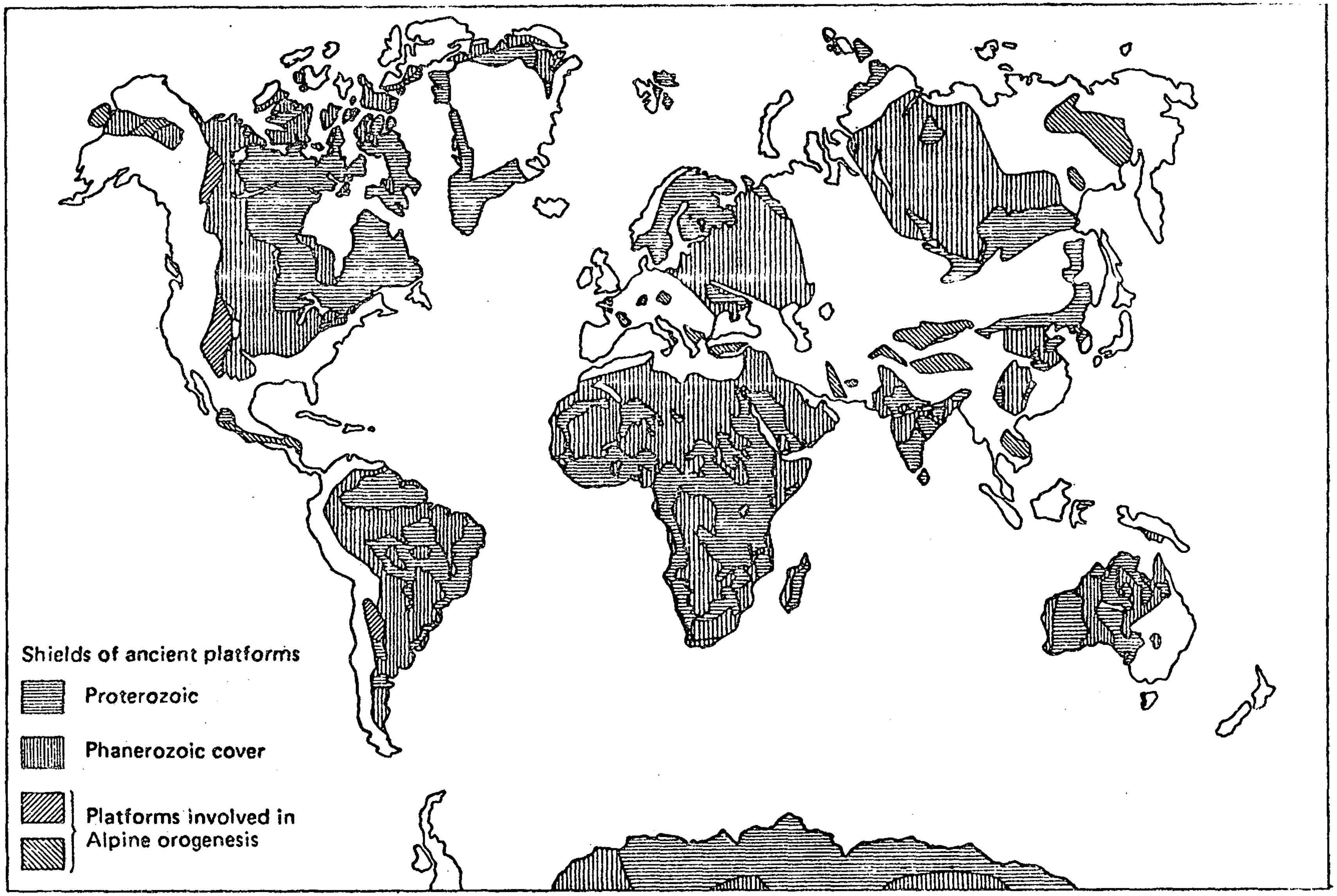

Fig. 2. Tectonic map showing Shield areas of world with areas of sedimentary cover and regions involved in Alpine orogenesis. After Khain V. E. and Muratou M. V. Crustal movements and tectonic structure of continents. In "The earth's crust and upper mantle" Hart P. J. ed. (Washington, D. C.: American Geophysical Union, 1969), 523—38 (Geophys. Monogr. 13)

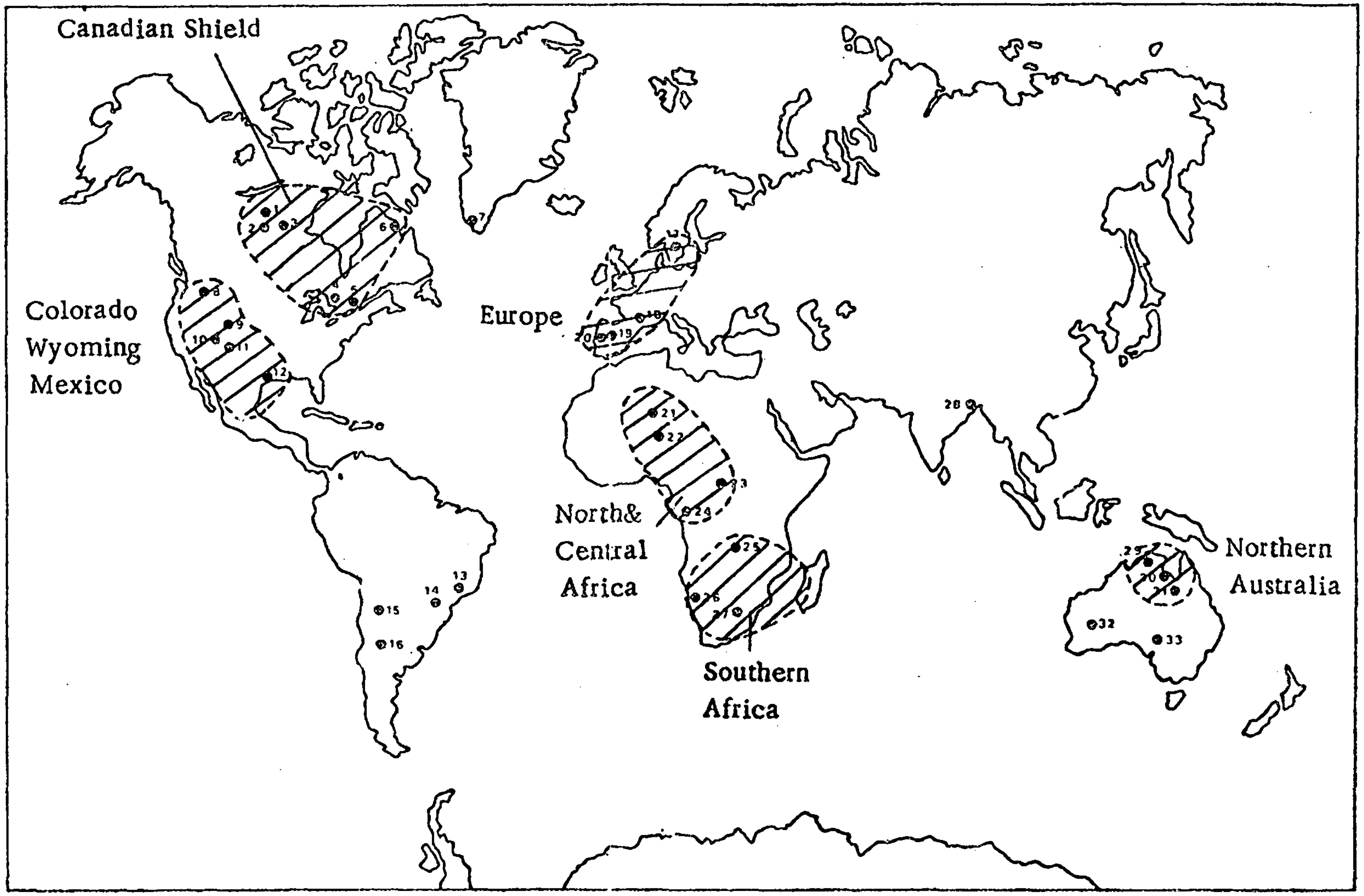

Fig. 3. Major uranium deposits and provinces (non-Communist countries): 1, Beaverlodge; 2, Cluff Lake; 3, Wollaston belt; 4, Elliot Lake-Agnew Lake; 5, Bancroft; 6, Makkovik; 7, Ilimaussaq; 8, Spokane; 9, Wyoming basins; 10, Uravan; 11, Grants; 12, Texas; 13, Pocos de Caldos; 14, Figueira; 15, Salta; 16, Malargue; 17, Ranstad; 18, Massif Central; 19, Salamanca; 20, Urgeirica; 21, Hoggar; 22, Agades; 23, Bakouma; 24, Mounana; 25, Shinkolobwe; 26, Rössing; 27, Witwatersrand; 28, Singhbhum; 29, Alligator Rivers; 30, Westmoreland; 31, Mary Kathleen; 32, Yeelirrie; 33, Frome-Yarramba

Major uranium provinces are roughly six in number.

Table 1

Major Uranium Provinces

	Relative abundance of present low cost uranium reserves (see Table 6) Tonnes	%
Colorado Plateau, Wyoming, New Mexico area, U.S.A.	523	31.7
Witwatersrand Basin, etc. South Africa	306	18.6
Northern Territory, Australia	289	17.5
Algeria, Niger, Gabon and Central African areas	217	13.2
Precambrian Shield, Canada	167	10.1
Hercynian Area, Europe	60	3.6
Others	88	5.3
	1650	100.0

The Australian, Canadian and South African provinces are all Precambrian or Precambrian/Phanerozoic. In the Colorado-Wyoming-New Mexico province in the U.S.A. the actual deposits range in age from 210 to 10 m.y. but an important feature is their proximity to the Precambrian basement which is a southwest extension of the Canadian Shield. In the Central African province the deposits are either in Precambrian or closely associated with the edges of the crystalline Precambrian massifs.

The main apparent exception is Europe where many of the deposits are apparently associated with the Hercynian orogeny, and even in this case it has been questioned (Figure 4 and reference 4/) whether there might not be a relationship to the ancient Precambrian orogenies.

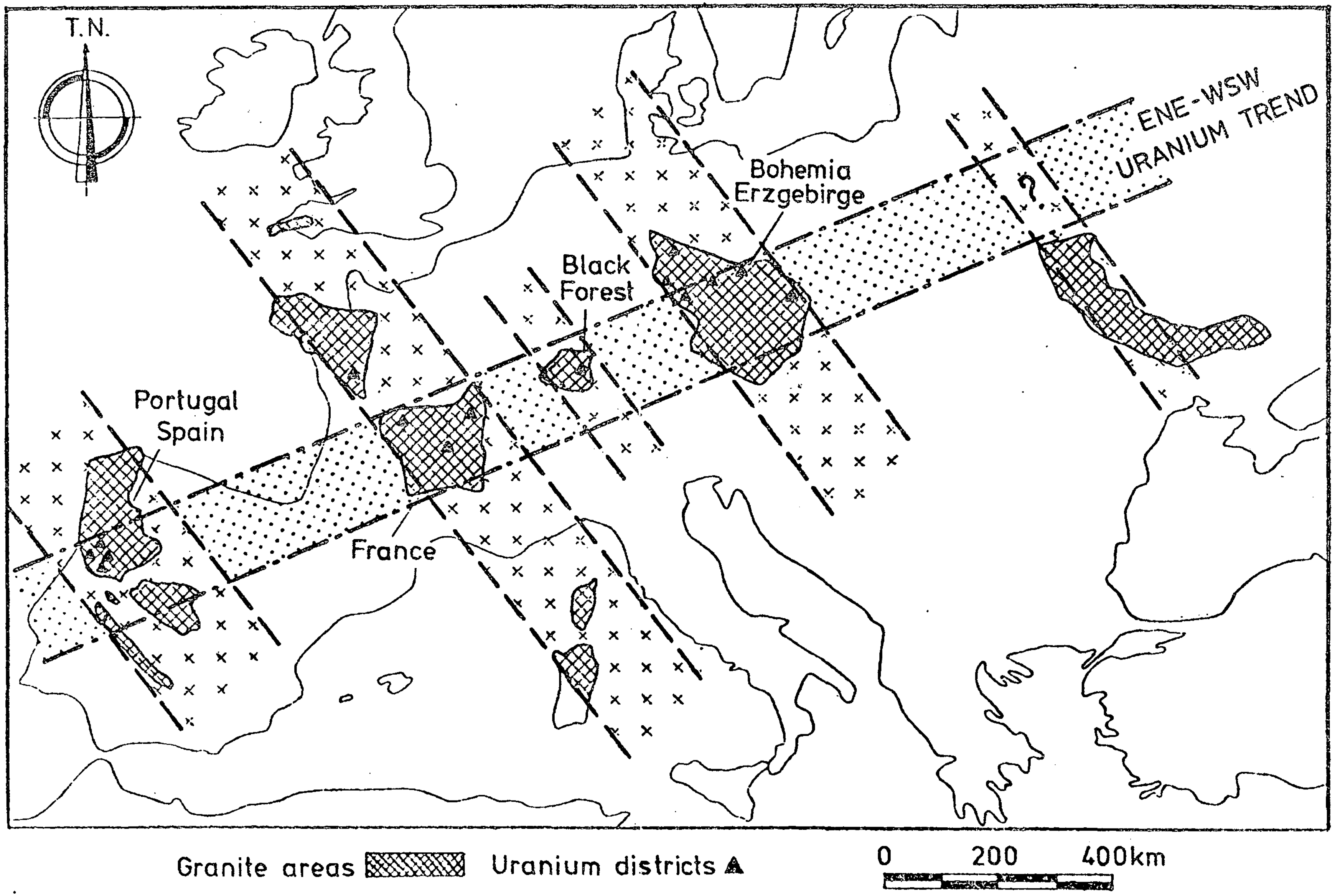

Fig. 4. Map of Europe showing principal uranium districts, ENE-WSW Uranium trend and NW-SE oriented Hercynian granite masses

BRIEF HISTORY OF THE URANIUM INDUSTRY AND ITS POSITION IN THE GENERAL MINERAL INDUSTRY

Prior to 1942, uranium was used chiefly for colouring glass and ceramic glazes and an ample supply was obtained by recovering some uranium from the ores which, after the beginning of this century, were mines for radium in four principal source areas, Eldorado, Canada; Shinkolobwe in the then Belgium Congo; the Colorado Plateau in the U.S.A. and at various locations in Europe.

In the preiod 1942 - 1945 the envisaged military uses brought an entirely new outlook. Increased and secure resources were sought but the greatest hope of the geological thinking at that time was to find new Shinkolobwe type deposits and to revive and increase production on known deposits. The hoped for supply was not satisfied and thus, from 1946 onwards, government stimulated uranium exploration programmes were initiated in many parts of the world.

The desired results did not appear in the immediately following years but as incentives were increased, very large new discoveries were made in the early 1950's. Indeed, exploration was so successful that by the late 1950's there was a surplus of uranium and this, co-inciding with a diminution in the military requirement, led to a period of depression in the uranium producing industry which lasted from about 1959 to 1966.

In the 1960's nuclear reactors were developed from the experimental to the commercial stage and another new phase in the history of uranium started. The apparent reduction in nuclear power costs, the assurance by certain sections of the nuclear power industry of a rapidly increasing potential market led to what has been called the false boom in uranium exploration from 1966 to 1969. Particularly in the United States, the resurgence of exploration activity seemed even more vigorous than the boom of the 1950's.

It soon became evident, however, that reactor construction schedules were longer than expected, completion dates were continually postponed, reactor orders accumulated and thus nuclear power utilities were reluctant to commit themselves to large fuel investments for the long term.

Except for continuing exploration in Australia another period of depression in the uranium exploration industry, particularly in North America ensued from 1970 to 1973.

By 1972 a more cautious but possibly a more soundly based assessment of the uranium requirements for the long term nuclear power programmes was taking place among uranium mining and power utility companies but despite the very substantial forecast of future demand and the exhortations by national and international experts that a huge uranium exploration programme was required, no marked acceleration in exploration was apparent in 1972 - 73 and the sales price of uranium remained low (US \$ 6 - 8 per lb U_3O_8)

Into this slow resurgence of interest came the energy crisis of the winter of 1973 - 74 and the substantial rises in petroleum prices. While this undoubtedly had the effect of making nuclear power economically more attractive, it also had contrary side effects and contributed significantly to world-wide recession, unprecedented inflation and a shortage of investment capital. The overall result was that these negative influences were in the short term almost as powerful as the advantages that the nuclear industry was likely to gain from the changed structure of energy prices. However, the medium and long term trends appeared to become more sharply upwards and as this view became dominant in the power utility industry, uranium prices moved rapidly upwards from the 1973 base of about US \$ 6.00/lb U_3O_8 to recent spot contracts of around US \$ 43.00/lb U_3O_8 (Figure 5).

In the last few years there have, however, been further problems for the uranium industry, slow-down of world economic growth, resulting in rescheduling of nuclear reactor programmes, environmental considerations, and, in many countries, government policy in regard to nuclear power and uranium development has been indeterminate. Thus it can be noted that while the uranium price rose from US \$ 7.00 per pound U_3O_8 on 1st January 1974 to US \$ 40.00 per pound U_3O_8 in March 1976 it rose only from \$ 40 to \$ 43.25 between March 1976 and March 1978.

Although uranium is a highly publicized metal it is not yet among the major metals in the overall mineral industry income. The 1977 production of uranium of approximately 28,600 tonnes U at an average value of \$ 20.00/ lb U_3O_8 was worth something of the order of 1250 million dollars whereas the

HISTORICAL U_3O_8 EXCHANGE VALUE

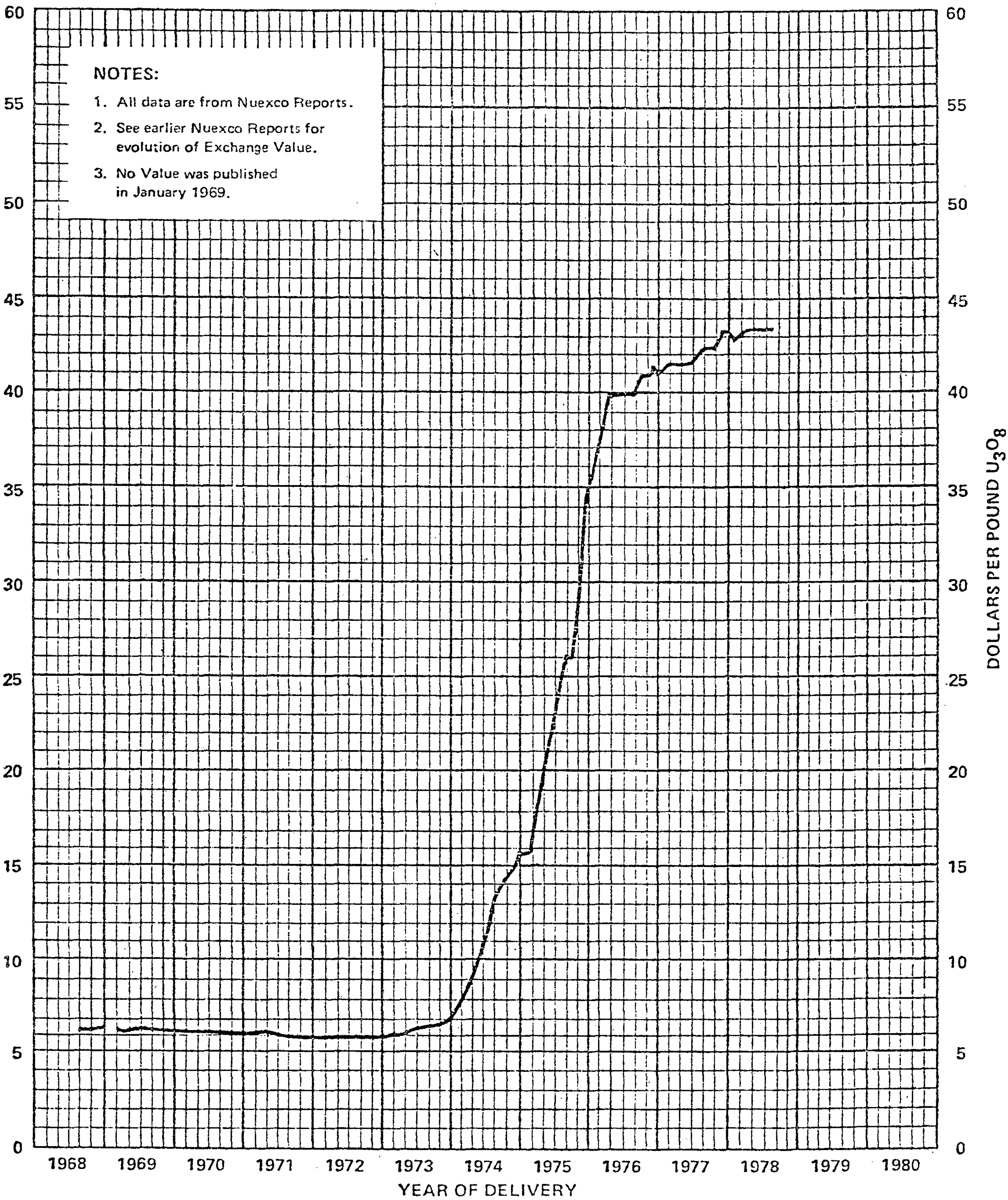

FIGURE 5

world's aluminium production in 1977 was valued at some 12,450 million dollars, copper at 8360 million dollars, gold at 4600 million dollars, zinc at 3160 million dollars, tin at 2175 million dollars and lead at 1700 million dollars.

Table 2

Approximate values of the production of some principal metals in 1977

	US millions of dollars
Aluminium	12,450
Copper	8,360
Gold	4,600
Zinc	3,160
Nickel	2,520
Tin	2,175
Lead	1,700
Molybdenum	1,700
Uranium	1,250

In terms of ore mined, uranium also occupies a relatively modest position in relation to the other principal metals.

If the projected demand for uranium rises as described in the next section and there is an accompanying discovery rate, the next twenty-five years are likely to change the importance of uranium in the mining industry relative to other metals. A study by the US Bureau of Mines which is illustrated in Figure 6 shows that the projected growth rate for uranium will exceed that of any other metal, being comparative only to aluminium in its rate of growth. There is no way of predicting prices of any of the metals over these twenty-five years but the probability is that uranium will occupy a much more prominent position in the league table of annual values of production in the year 2000 in relation to many of the other metals.

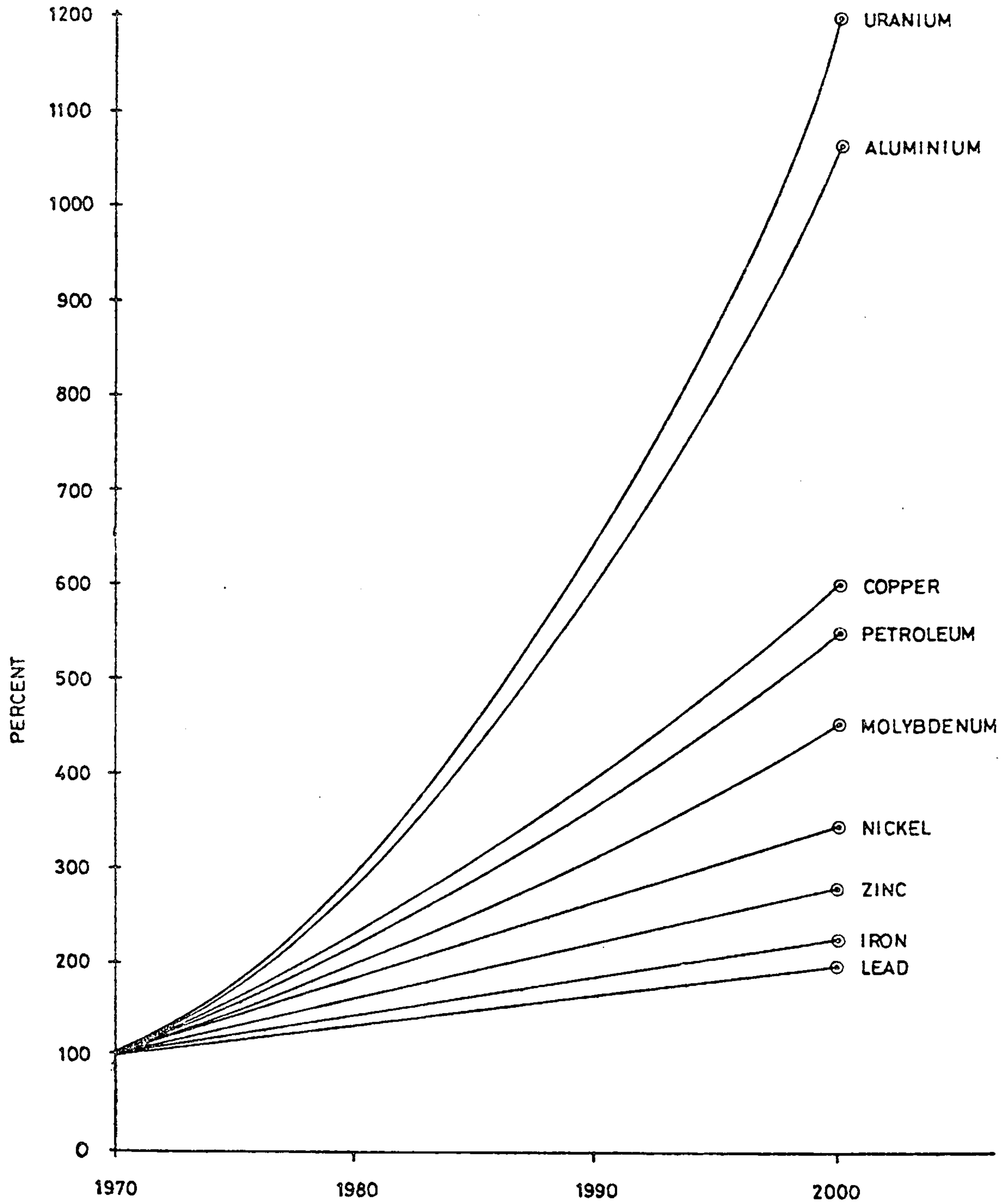

Fig. 6 World annual mineral production projected growth comparisons as from 1970 production rates

FUTURE DEMAND FOR URANIUM

Nuclear power production began on a commercial scale approximately 17 years ago and up to the end of 1977 about 87,000 MWe generating capacity has been installed in the world's nuclear power stations. This represents about 5% of the world's total electricity generating capacity.

Further projections show that nuclear is expected to produce between 35 - 50% of all electric energy in the world in the year 2000.

From the present figure of about 87,000 MWe, it is estimated that there will be an installed capacity of between 500,000 and 700,000 MWe by 1990.

It should, however, be kept in mind that there are considerable uncertainties in present estimates. While we can make fairly precise forecasts up to 1985, based on orders and national commitments, the later estimates are obviously much less precise and should be used only to show trends.

It must also be noted that estimates have been revised downwards in the last few years. This is due to the world economic crisis, and some public opposition to nuclear power which has led to longer licencing procedures, greater safety precautions and greater capital costs.

Thus the corresponding annual and cumulative world requirements for uranium up to 2000, based on a world survey by the joint NEA/IAEA Working Party*/ on 'Uranium Resources, Production and Demand', have had to be modified in recent months. The medium range estimate now shows a cumulative requirement between 1977 and 1990 of approximately 900,000 tonnes U and about three million tonnes by the year 2000. Table 3 and Figure 7.

From a 1977 lelvel of about 23,000 tonnes U per year (Table 3), the annual demand will rise under relatively conservative assumptions to an annual

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*/ Because of lack of information on the long term plans for nuclear power in the countries with centrally planned economies and a total lack of data on their uranium resources, the present analysis does not cover these countries. They are assumed to be more than able to meet their uranium requirements from their own sources and may even become potential exporters.

Table 3 WORLD URANIUM REQUIREMENTS*
(1,000 tonnes U)

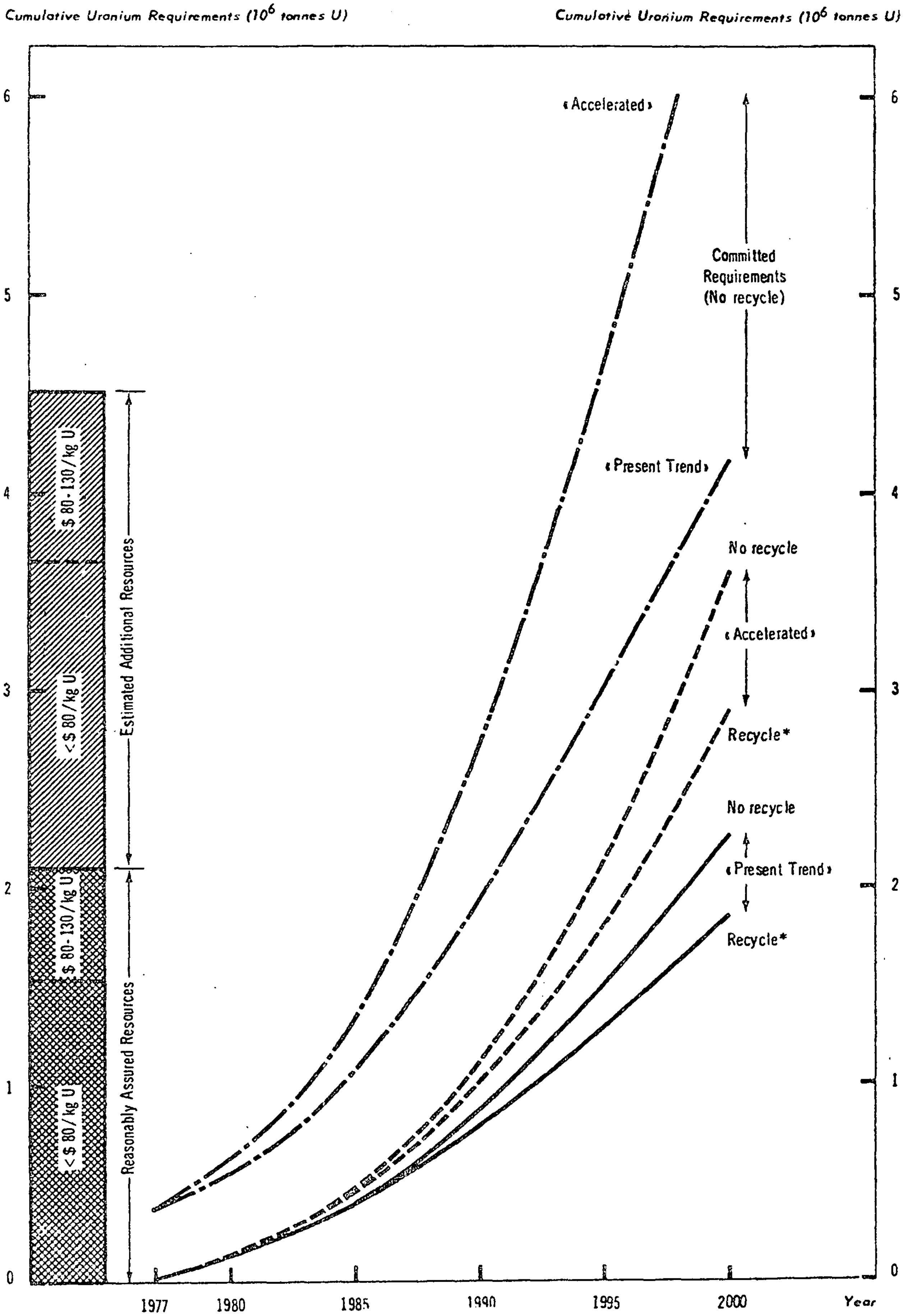
YEAR	WITHOUT RECYCLE				WITH RECYCLE+			
	"ACCELERATED" POWER GROWTH		"PRESENT TREND" POWER GROWTH		"ACCELERATED" POWER GROWTH		"PRESENT TREND" POWER GROWTH	
	ANNUAL	CUMULATIVE	ANNUAL	CUMULATIVE	ANNUAL	CUMULATIVE	ANNUAL	CUMULATIVE
1977	23	23	23	23	23	23	23	23
1978	29	52	29	52	29	52	29	52
1979	35	87	35	87	35	87	35	87
1980	43	130	41	128	43	130	41	128
1981	51	181	47	175	51	181	47	175
1982	60	241	53	228	60	241	53	228
1983	69	310	59	287	67	308	57	285
1984	79	389	65	352	74	382	61	346
1985	88	477	71	423	82	464	65	411
1986	98	575	78	501	90	554	69	480
1987	111	686	84	585	98	652	73	553
1988	125	811	90	675	106	758	77	630
1989	140	951	96	771	115	873	81	711
1990	156	1,107	102	873	126	999	85	796
1991	171	1,278	108	981	137	1,136	89	885
1992	188	1,466	114	1,095	149	1,285	93	978
1993	206	1,672	121	1,216	160	1,445	96	1,074
1994	221	1,893	127	1,343	171	1,616	100	1,174
1995	234	2,127	134	1,477	181	1,797	104	1,278
1996	247	2,374	142	1,619	193	1,990	108	1,386
1997	267	2,641	150	1,769	205	2,195	112	1,498
1998	295	2,936	160	1,929	218	2,413	116	1,614
1999	317	3,253	169	2,098	230	2,643	120	1,734
2000	338	3,591	178	2,276	242	2,885	125	1,859

* Based on 0.25% U-235 enrichment plant tails assay and other data shown on Tables 8, 9 and 10.

(+) Beginning in 1985.

Figure 7

WORLD CUMULATIVE URANIUM REQUIREMENTS
(1977-2000)



* Recycle begins in 1985.

production requirement of 42,000 tonnes U by 1980, 76,000 by 1985, 117,000 by 1990 and 220,000 by 2000. Few mineral production industries have been called upon to plan for a four to five-fold increase in production in a space of about 15 years as these forecasts imply. This might possibly mean that, depending on their capacity, possibly 5 times the present number of uranium mines will have to be planned and engineered by 1990. As already noted, a comparison with the projected growth rate of eight other principal minerals as shown in Figure 6 gives a dramatic view of the situation. Due to the lead times for mine development and construction, it may already be too late to satisfy the production needs of the early 1980's and again another temporary unbalance period for uranium may occur in these years.

Looking further ahead, it has been estimated that the requirement up to the year 2000 will be approximately three million tonnes U and an IAEA study group on reactor strategies has indicated the probability that up to the year 2025 a cumulative quantity of as much as 10 to 20 million tonnes U might be required.

Demand will continue to climb for some years after 2000 even if there is a successful commercial introduction of breeder reactors in the 1980's or 1990's. Light water reactors, if installed in the last two decades of the century will, according to the present projections, require uranium supplies for their life times of 25 - 30 years. The forecast of cumulative total uranium demand of about 3,000,000 tonnes U up to the year 2000 and close to 10,000,000 by 2025 gives only the cumulative totals of consumption up to these dates and not integrated lifetime requirements, which would be substantially greater and thus the total requirement into the middle of next century will be well in excess of 10 million tonnes uranium. The annual demand in the first 15 years of the 21 century will, however, depend on the concentration of reactor type and installation rate in the 1980's and 1990's. If the presently predicted figures are approximately correct and even assuming a steady fall-off in uranium requirements for non-breeder reactors there could still be an increase in total uranium demand around the first 10 years of the 21st century. In order to satisfy the estimated cumulative figures, annual demand might rise from 220,000 to nearly 350,000 tonnes pursuant to the full takeover by breeder reactors. If such forecasts were even approximately correct, they would again have serious implications for the uranium mining industry with the familiar and unacceptable 'boom' and 'bust' situation which neither the financial structure nor the physical nature of mines or mills would be geared to deal with. The decreasing uranium demand beyond 2010 would make the possibly short boom period of the first 10 years of the 21st century a difficult period for the mining industry.

URANIUM RESERVES AND RESOURCES

Reports of world surveys of uranium resources have been issued at roughly two year intervals since 1965 by a Joint Working Party organized by the OECD Nuclear Energy Agency, Paris, and the International Atomic Energy Agency, Vienna. As already noted, the most recently published study is entitled "Uranium Resources, Production and Demand" and, while issued in December 1977 contains data as of 1st January 1977. In the rapidly evolving uranium situation, considerable changes have occurred since that date and the two Agencies are presently engaged in the preparation of a new report which it is hoped will up-date the figures to 1st January 1979 and be issued about September 1979.

The Reasonably Assured Resources and the Estimated Additional Resources in two cost ranges as stated in the December 1977 report are given in Tables 4 and 5. The changes which have occurred since the first NEA/IAEA report in 1965 are shown in Figure 8.

The definitions used by the NEA/IAEA Working Party should be stated and are as follows:-

Reasonably Assured Resources (RAR) refers to uranium which occurs in known ore deposits of such grade, quantity and configuration that it could be recovered within the given production cost range, with currently proven mining and processing technology. Estimates of tonnage and grade are based on specific sample data and measurements of the deposits and on knowledge of ore-body habit. Reasonably Assured Resources in the cost category below $ 30/lb are considered as Reserves for the purpose of the present report.

Table 4. REASONABLY ASSURED RESOURCES
(1,000 tonnes U)
Data available 1st January, 1977

COST RANGE	< \$ 80/kg U (< \$ 30/lb U_3O_8) RESERVES	\$ 80-130/kg U (\$ 30-50/lb U_3O_8)
Algeria	28.	0
Argentina	17.8	24
Australia	289	7
Austria	1.8	0
Bolivia	0	0
Brazil	18.2	0
Canada[1]	167	15
Central African Empire[2]	8	0
Chile	0	0
Denmark (Greenland)	0	5.8
Finland	1.3	1.9
France	37	14.8
Gabon[2]	20	0
Germany, F.R.	1.5	0.5
India	29.8	0
Italy	1.2	0
Japan	7.7	0
Korea	0	3
Madagascar	0	0
Mexico[3]	4.7	0
Niger	160	0
Philippines	0.3	0
Portugal	6.8	1.5
Somalia[4]	0	6.2
South Africa	306	42
Spain	6.8	0
Sweden	1	300
Turkey	4.1	0
United Kingdom	0	0
United States	523	120
Yugoslavia	4.5	2.0
Zaïre	1.8	0
Total (rounded)	1,650	540

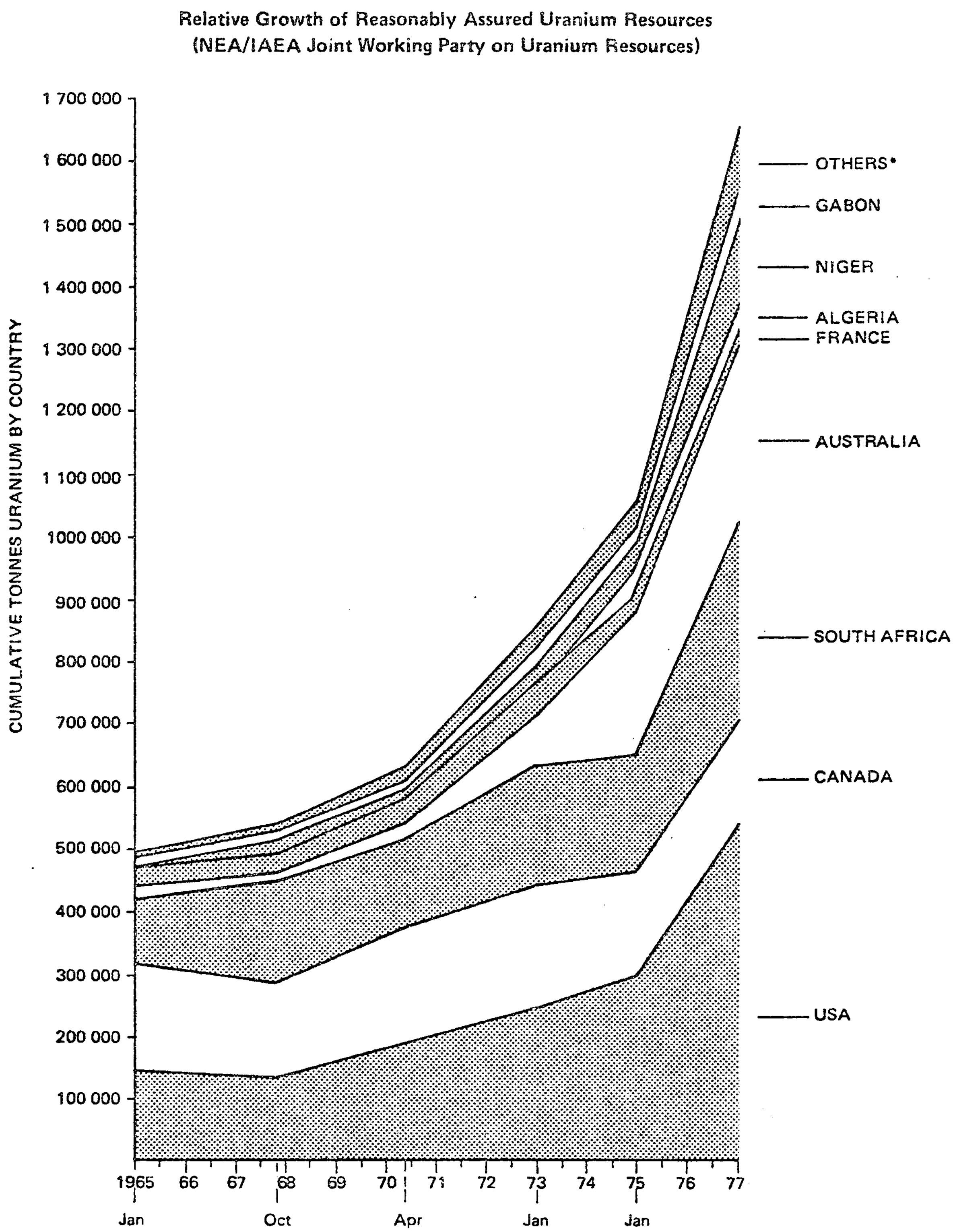
1. The material reported as Reserves is minable at prices up to \$ 104/kg U and the other Reasonably Assured Resources are minable at prices between \$ 104 and \$ 156/kg U.
2. Source of data: Uranium Resources, Production and Demand; Paris 1975.
3. Data refer to resources "in-situ", rather than recoverable.
4. Costs of recovery are not known so the resources are arbitrarily assigned to the higher cost category.

Table 5 | ESTIMATED ADDITIONAL RESOURCES
(1,000 tonnes U)
Data available 1st January, 1977

COST RANGE	<80 US $/kg U (< 30 $/lb U_3O_8)	80-130 US $/kg U (30-50 $/lb U_3O_8)
Algeria	50	0
Argentina	0	0
Australia	44	5
Austria	0	0
Bolivia	0	0.5
Brazil	8.2	0
Canada[1]	392	264
Central African Empire[2]	8	0
Chile	5.1	0
Denmark	0	8.7
Finland	0	0
France	24.1	20.0
Gabon[2]	5	5
Germany, F.R.	3	0.5
India	23.7	0
Italy	1	0
Japan	0	0
Korea	0	0
Madagascar	0	2.0
Mexico[3]	2.4	0
Niger	53	0
Philippines	0	0
Portugal	0.9	0
Somalia[4]	0	3.4
South Africa	34	38
Spain	8.5	0
Sweden	3	0
Turkey	0	0
United Kingdom	0	7.4
United States	838	215
Yugoslavia	5.0	15.5
Zaïre	1.7	0
Total (rounded)	1,510	590

1., 2., 3., 4. - As in footnotes to Table 3.

NB: A number of occurrences of uranium are not well enough defined to be included in Tables 3 and 4 but are described in Part II, the country reports.



* Others: Argentina, Brazil, Central African Republic, Congo, F.R. Germany, Denmark (Greenland), Spain, India, Italy, Japan, Morocco, Mexico, Portugal, Yugoslavia, Zaire.

Figure 8

Estimated Additional Resources (EAR) refers to uranium surmised to occur in unexplored extensions of known deposits or in undiscovered deposits in known uranium districts and which is expected to be discoverable and could be produced in the given cost range. The tonnage and grade of Estimated Additional Resources are based primarily on knowledge of the characteristics of deposits within the same districts.

The definition of "cost" as used in the stated cost ranges include, in general, not only the direct costs of mining, milling and extraction but also the cost of capital spent in providing and maintaining the production unit. Exploration costs are not included. Some 87.5% of the low cost reserves exist in five countries - Australia, Canada, Niger, South Africa and the United States.

This is a remarkable situation and immediately prompts the questions - is there a freak geological distribution of uranium which favours these countries? - or: is it merely that heavy expenditure has been made on exploration in these countries? Almost all geologists will deny the first possibility and agree with the second. Uranium exploration virtually started in 1945, and thus, despite the tremendous efforts of the 1950's and in recent years it is still very young and it has not yet been possible to make a real world wide uranium resources inventory. However, the IAEA and the NEA are currently engaged in a project named "International Uranium Resources Evaluation Project" (IUREP) which is designed to make a world estimate of uranium resources and potential. The first phase of IUREP has now been completed and the Speculative Resources of the world defined as "undiscovered resources in addition to RAR and EAR, which are thought to exist mostly on the basis of indirect indications and geological extrapolation in deposits discoverable with existing techniques and exploitable up $ 130/kg U" are estimated to lie between 6 - 5 and 14.8 million tonnes uranium in some 175 countries of the world.

PRODUCTION

Estimated uranium production capacities in 1978 and attainable to 1990 are shown in Table 6.

Table 6

Estimated uranium production capacities attainable 1978 and 1990 1000 tonnes uranium per annum

	Estimated 1978	Attainable 1990
Argentina	0.28	0.60
Australia	0.50	20.00
Canada	6.45	11.25
Gabon	1.20	1.20
Niger	2.40	9.00
South Africa	8.80	12.00
U.S.A.	19.30	47.00
Western Europe	3.22	6.30
Other	0.25	2.65
Total (rounded)	42.40	110.00

The above table illustrates capacities. The actual production in 1977 is estimated to have been 28,617 tonnes uranium. The table suggests that on the basis of presently known reserves, the production capacity could be increased three.fold between 1977 and 1990 but the availability of reserves must be taken into account. For example, Australian uranium policy was under review for some years as to whether their resources should come into production or not and if so, when and upon what conditions. Possible constraints on exports by Canada have also been in evidence and there are physical constraints due to the nature of ore bodies and dependence on other metals in such countries as South Africa.

There are, therefore, considerable problems and questions related to any achievement of a 110,000 tonne uranium production capacity by 1990 but even if this level could be achieved, it would only be possible to sustain it for a very short time after which it would decline due to the depletion of some deposits and the need to mine lower grade material. In order to maintain or increase capacity beyond 1990, substantial additional reserves will have to be identified.

RECENT PRICE TRENDS

During the earlier period of uranium exploration, i.e. in the 1950's discovery rates were very sensitive to sufficiently attractive price stimulation. This was followed by the period of the development of nuclear power from the experimental to the commercial stage and for many years, essentially during the 1960's the price of uranium remained low and stable within a price range of approximately \$ 5-7/lb U_3O_8. In the early 1970's when the substantial future requirements for uranium were already evident, and despite recognition of the fact that a huge exploration programme would be required, the price continued to remain static at an average of just over \$ 6/lb U_3O_8. It was not until the energy crisis of the winter of 1973-74, which caused very substantial rises in petroleum prices, that the stimulation for a rise in the uranium price also occurred. Starting from the end of 1973, the price rose dramatically from a base of approximately \$ 7/lb U_3O_8 to prices for spot contracts of \$ 40/lb U_3O_8 in 1976. The recent high spot contract prices of \$ 43/40/lb U_3O_8 has been much publicized (Figure 5) but it should be remembered that the average price including existing long term contracts is much lower. The latest figure for 1977 is an average of US \$ 19.75/lb U_3O_8.

It is hoped that the price increases over the last few years will provide the necessary stimulation for the major exploration effort that will be required in the next decades. However, extreme price increases are likely to be counter-productive as indicated in a study of the economics of nuclear power by Davis[7] and it was felt that the important growth in nuclear generating capacity could not be achieved unless there were confidence in its technological maturity as well as in its economics. Table 7 shows a typically representative comparison of costs in the European Community: for power stations coming into operation in the early 1980's nuclear generation costs are 33% less than for oil-generated electricity. This calculation assumes U_3O_8 at US \$ 18.50/lb U_3O_8 but if it had to be assumed that prices of US \$ 40/lb U_3O_8 have to be paid, the cost of the nuclear kWh increases to 20.7 mills u.a. and its cost advantage would have shrunk from 33% to 15%.

Table 7

Comparison of Electricity Generation Costs between Nuclear and Fuel Oil

(assuming \$ 11/bbl fuel oil; 16.7 u.a. or \$ 18.50/lb U_3O_8)

	Nuclear		Fuel Oil	
	mills u.a./kWh	%	mills u.a./kWh	%
Capital	10.0	61	7.0	29
Fuel	3.8	23	14.9	61
Operation	2.5	16	2.5	10
Total	16.3	100	24.4	100

There is thus an important warning. Should the price rise continue, it is likely to curtail the ordering of new nuclear power stations and thus the market for uranium. Unless uranium prices are maintained in reasonable relation to real costs so that there is an economic incentive to select nuclear, utilities will start to reduce their new investment in nuclear power. Uranium producers must not try to secure prices that are exorbitantly in excess of profitable operation. On the other hand the rate of increase in uranium prices between 1976 and 1978 has not been keeping pace with world inflation and thus prices in the range of US \$ 60 to 80/lb U_3O_8 may well be expected five to ten years from now.

EXPLORATION ACTIVITIES

Worldwide uranium exploration declined after 1970 as a consequence of the soft uranium market. However, since 1974, there have been positive signs that prospecting activities are increasing again at least in some areas such as North America.

The renewed effort in uranium prospecting has been initiated by market forces, i.e. the substantial increase in uranium prices during the past five years. However, market forces alone may not be sufficient to stimulate adequate exploration efforts on a worldwide scale. As already noted, 85% of reserves are situated in four countriés, (USA, South Africa, Canada and Australia), and these are the countries which have undertaken the greatest exploration efforts in the past. It might be concluded that a comparable effort in other geologically favourable areas would give similar results.

However, as exploration expenditure is heavy and considerable experience and technical skill is needed for location and development of new deposits, close collaboration is required between the industrialized nations and developing countries where these unexplored areas are mainly situated.

It is felt that increased international co-operation in the field of uranium exploration could make a contribution towards avoiding a tight situation on the uranium market, with all its negative consequences.

This has been one of the principal objectives of IUREP, to identify the areas, geological trends and countries of the world where the best geological information indicates that the future uranium resources are most likely to be found and to stimulate international co-operation in investigating these areas.

THE LONG TERM URANIUM RESOURCES SITUATION

The information made available by contributing nations to the NEA/IAEA Report has enabled presently known uranium reserves and resources to be quantified under two categories, 'Reasonably Assured Resources' and 'Estimated Additional Resources'.

It is, however, obvious that only the 'Reasonably Assured Resources' can be considered for specific planning and forecasting in the short and medium term and that even the availability of much of these resources is contstrained by technical mining schedules. Even if it were assumed that the present 'Estimated Additional Resources' could be confirmed and developed, the total of the two categories is still inadequate to meet the long term uranium requirement.

There is, therefore, an obligation to comment on the world uranium potential and the problems involved in the future discovery of adequate uranium to meet requirements.

The three principal factors and constraints are generally considered to be:-

(1) Physical: i.e. the existence of deposits

(2) Economic: i.e. the availability of adequate funds for exploration, development and capital investment. (This includes funds for research and development of new methods and techniques)

(3) Political: i.e. availability of search areas and production and export facilities

The future potential may be considered under two cost categories; i.e. under \$ 30/lb U_3O_8 and lower grade uranium with a cost of above \$ 30/lb U_3O_8.

A. Potential Uranium Ore under \$ 30/lb U_3O_8

Physical availability

The type of material considered in this category is equivalent to that now being examined in the USA under the D.O.E. National Uranium Resources Evaluation (NURE) programme and referred to as 'possible' and 'speculative' potential resources. These would be within postulated deposits in formations or geologic settings that have not been previously productive within a productive province or alternatively in new deposits postulated in geologic provinces that have not previously been productive.

Informed programmes to examine speculative potential through detailed examination of geologic provinces and recognized uranium favourability criteria have been initiated both in the USA and in Canada. However, very little had been done elsewhere in the world and it was for this reason that the two Agencies, Nuclear Energy Agency of OECD and IAEA jointly initiated the International Uranium Resources Evaluation Project (IUREP) to estimate world's 'Speculative Resources'.

It is recognized that very large areas of the world remain to be explored and that there is a probability that much more uranium remains to be discovered in the upper part of the Earth's crust. However, any attempt to make a quantitative appraisal of the ultimate size of the world's uranium resources, based on existing information must remain extremely speculative. Despite the fact that the joint NEA/IAEA Steering Group in their IUREP bibliographic study came to the conclusion that in 175 countries there may be Speculative Resources of between 6.5 and 14.8 million tonnes uranium, there remains a range of technical opinion on whether low cost uranium deposits to the amount of 10 million tonnes actually physically exist in occurrences in the upper part of the Earth's crust from which it could be economically produced. The wisest course will be to expand exploration and exploration technology as rapidly as possible coupled with continuous intelligent interpretation and extrapolation of the geological data obtained. If this is done, there is a reasonable expectation of successful discoveries which may, or may not, be adequate to identify the required tonnages. However, as there is a risk that adequate uranium will not be found, all avenues should be explored to make the best economic use of the available uranium resources (e.g. ensuring the rapid commercialisation of short doubling time breeder reactors and the production of more efficient reactor systems).

Economic factors

One of the biggest challenges of the future might appear to be the financing of the required exploration, development and construction effort, particularly against the background of a frequently unpredictable relationship between financial sources and potential producer countries of all stages of development.

The amount of necessary finance is very large. Taking into account a ten year forward reserve requirement, the funds which will be necessary to cover the exploration requirement up to the year 2000 are likely to be of the order of 20 billion dollars, while the necessary mine and mill construction finance would be of the same order of magnitude.

This had appeared to be a major requirement but at a 1976 meeting in Geneva of the Atomic Industrial Forum, Mr. W.E. Pelley, First Vice President, Bankers Trust New York, presented a paper 8/ entitled "A Banker Looks at Uranium Financing" and came to the conclusion that the financial requirements of the uranium industry will amount to less than two-tenths of one percent of total capital funds likely to be raised in the period up to 1990. He remarked in conclusion "— the financing problem of uranium mines in the United States and throughout the rest of the world is not one of capital availability, Capital is available. The real problem is one of the uranium industry being able to attract the capital. The problems lie in market stability, government approvals and public recognition of a world energy shortage".

This expert view appears to minimize the problem of the existence of adequate funds but points up the semi-political problems which are also covered by the third factor.

Political factors

The main problems of ensuring future uranium supplies do not, therefore, appear to be purely geological nor of the technical capability of the geological and mining profession to discover and develop deposits nor of the existence of adequate funds but rather of politico-economic constraints limiting the availability of search areas and the freedom to develop, produce and export from identified deposits. Even in the main uranium reserve countries, governments are involved in re-assessing their policies in regard to uranium exploration, development, export and import and some of these policies are, or have been restrictive to a greater or lesser degree. While fully understandable from national viewpoints this has, in some cases, tended to limit exploration and development of uranium resources.

For the governments of the less well developed countries, the problems are equally complex. In many countries even the speculative potential for uranium has barely been considered. Governments which are faced with problems such as whether there will eventually be a national requirement, the extent of national control of uranium exploration and development, environmental problems, and possible financial sources for development, have frequently been unwilling or unable to initiate uranium exploration on a major scale.

It is recognized that problems such as these constitute perhaps one of the greatest challenges which lie ahead in trying to ensure adequate uranium supplies. Increasing dialogue between producer and consumer nations will become more and more important.

The principal factors which should be taken into account by governments of both advanced and developing countries when defining their future policies in regard to uranium are considered to be:-

(1) Present uranium reserves are about 1.65 million tonnes.

(2) There will be a requirement of up to three million tonnes by the year 2000 and of the order of 10 million by the year 2025.

(3) The commercial importance of uranium may be limited to the next 40 - 60 years and the requirement thereafter may decrease sharply.

(4) The lead time between initial exploration in a new area (particularly in a developing country) and initial production may be as much as 15 years.

(5) The scale of the finance required for major exploration and development programmes is likely to be only available from commercial or national organizations in the advanced countries or through international development funds.

(6) The price structure and rewards over the next decades are likely to be attractive.

B. Lower Grade Uranium Ores

Because the risk that production of lower cost uranium may be insufficient, the availability and the need for uranium in the category with a cost higher than \$ 30/lb U_3O_8 has been considered.

At the present time (1977) there is no clear lead as to how intensive the search for low grade resources to cover the requirements of the next fifty years shoould be. To a great extent this will be dependant on the intensity and the success of the exploration effort for lower cost uranium, i.e. uranium at a cost of less than \$ 30/lb U_3O_8 (present dollar values).

For the rest of this century, presently identified conventional type uranium resources at cost levels no greater than \$ 30/lb U_3O_8 may be able to provide for forecast requirements, but because of the rise in petroleum prices, uranium up to \$ 100/lb U_3O_8 could be competitive for nuclear power generation and thus it will be necessary to examine higher cost uranium resources more closely.

Up to the present, relatively few countries have attempted to quantify higher cost uranium resources. The United States and Canada have made estimates of some of their resources in this category and general estimates for the uranium content of phosphates, etc., in other parts of the world have been attempted.

Past exploration has generally been directed to deposits with average grades greater than 0.1% U_3O_8 and at the other extreme, a good deal is known about the characteristics and problems involved in recovery from very high cost material such as shales, granites, phosphates, etc. There is, however, a considerable gap in knowledge about uranium between these two extremes and much future exploratory effort will need to be directed towards sources of uranium in the range 0.1 - 0.01% U_3O_8.

It is well known that enormous quantities of uranium, probably in excess of any visualized nuclear power requirement exist in marine black shales, marine phosphates, granites, sea water and other unconventional sources. The problem to be considered is how much of this material can be made available at a cost acceptable to the nuclear power industry within a useful time period. The limiting factors on the utilization of low grade material of this nature will be the price which the nuclear power industry will be prepared to bear, the environmental constraints on mining, the huge ore tonnages necessary to recover the required uranium tonnages and the development of the technologies necessary for the recovery of such uranium.

It is unlikely that very low grade material, that is, under 100 ppm U_3O_8, can provide any substantial part of the presently envisaged requirement. This is principally due to the environmental problems that the development of such material would cause but also because the time scale required for the development of techniques to exploit such deposits is so great that the uranium requirement may be diminishing before substantial tonnages can be produced and also there is presently not enough pressure to start up such programmes.

To be able to select the economically most favourable material in grades greater than 100 ppm and to produce from it, it will be necessary to fund a major research and development programme on exploration, mining and milling techniques and with improvements in technology and higher prices some contribution from higher cost material of this type might be expected in the medium and long term future.

ROLE OF INTERNATIONAL ATOMIC ENERGY AGENCY IN URANIUM RESOURCES

Since its inception in 1957, the International Atomic Energy Agency has carried out various programmes related to nuclear raw materials. These have covered many aspects of the subject including geological studies through exploration, radiometric measurements and the production of uranium concentrate-

General Programmes

The Agency acts as medium for the interchange of information mainly through organizing scientific meetings and administering research contracts and by publishing the proceedings of these meetings and the results of the research work for world circulation. These meetings consist of symposia, seminars, technical committees, advisory groups and consultants meetings. Published proceedings of symposia include such titles as "Exploration for Uranium Ore Deposits" (1976), "The Oklo Phenomenon"(1975), "Formation of Uranium Ore Deposits"(1974), "The Recovery of Uranium"(1971), "Nuclear Techniques and Mineral Resources"(1969) and "Radiological Health and Safety in Uranium Mining and Milling"(1964).

In recent years, advisory groups have resulted in publication of "Processing of Low Grade Uranium Ores"(1966), "Uranium Exploration Geology" (1970), "Uranium Exploration Methods"(1973), "Radon in Uranium Mining"(1973), "Uranium Ore Processing"(1976 and "Recognition and Evaluation of Uraniferous Areas"(1976).

Consultants meetings may discuss and recommend action on specific problems or assist in writing a report or preparing a manual. For example, Technical Report No. 158, "Recommended Instrumentation for Uranium and Thorium Exploration"(1974), Technical Report No. 174, "Radiometric Reporting Methods and Calibration in Uranium Exploration". Technical manuals on "Geochemical Methods for Uranium Exploration" and on "Evaluation Methods in Uranium Ore Deposits" are being prepared by two consultant groups for publication in 1979.

Working Groups are normally convened for the purpose of advising the Agency on specific problem areas. As already metnioned, one group of experts, jointly sponsored by the Nuclear Energy Agency of the OECD and the IAEA has reviewed the world's uranium resources, production capacity and demand at roughly two-year intervals since 1965. Other joint Working Groups with NEA are concerned with research and development problems in uranium exploration, uranium resources (IUREP) and uranium extraction.

Six Working Groups on uranium geology were set up as a result of the 1970 Agency advisory group and were re-convened in 1972 at the International Geological Congress in Montreal and in August 1976 at the International Geological Congress in Sydney, Australia.

The research contract programme is a means of providing financial support to research institutes in Member States. Contracts are normally awarded for a period of one year and are renewable up to a total project period of three years. Publication of results, either by the research institute or the Agency is normally done at the end of the contract.

The IAEA is building a Uranium Resources Information File which will include Reasonably Assured Resources, Estimated Additional Resources, Speculative Resources, Production and Exploration data as presented on a country by country basis.

Technical Assistance Programmes to Developing Countries

The second major activity of the International Atomic Energy Agency in uranium resources is Technical Assistance.

All Member States are eligible for technical assistance provided under the Agency's Regular Programme. Both Member and non-Member States are eligible from the Agency from United Nations Development Programme funds provided that they are economically developing and are members of either the United Nations or one of the UN specialized agencies. Technical assistance is provided at the request of Member States, but the content of the projects requested may be finalized in consultations with the Agency and, where necessary, the United Nations Development Programme. The kinds of assistance available under these programmes may include expert services, equipment and supplies, UNDP large scale projects, fellowships and regional projects such as training courses and study tours.

The IAEA programme on nuclear raw materials is at present expanding in response to the increase in uranium exploration activities in all parts of the world.

CONCLUSIONS

The past history of uranium exploration has been one of slump and boom, perhaps even more spectacular and more marked than in the exploration history of any other metal. The stimulation of the booms and the origin of the slumps have mainly come from the nuclear power industry's own problems. A new confidence in nuclear power forecasts is required as a sound basis from which to establish a secure uranium supply programme. The fulfilment of the presently predicted demand for uranium is not likely to be an impossible task for a well organized, well stimulated uranium exploration and mining industry but the building up of confidence in the now predicted nuclear power timetable will be vital in assisting the mining industry in actually achieving the targets envisaged.

There is a strong probability that adequate conventional type, relatively low cost uranium resources (up to \$ 30/lb U_3O_8) do exist in the world. The problem is in identifying where they are and whether they can be exploited. Considerable potential remains to be explored in developing countries but whether such potential can be realized will depend on the building up of confidence between the Governments concerned and the sources of finance and technical skills which are mainly available in the more advanced countries, and, most important of all, in the establishment of a sufficently attractive and stable price level to stimulate the exploration and development of uranium resources in all countries.

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C O N T E N T S

FIGURES

THE EVALUATION OF URANIUM DEPOSITS

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INTRODUCTION

Factors involved in the Evaluation of Ore Deposits

The objective of reserve evaluation is to determine the tonnage and grade of the ore recoverable in a given technical and economic environment. The factors involved are many, both natural and human, and no two ore deposits are identical. Thus, except for some general factors, each orebody has to be evaluated in detail with consideration of its individual characteristics.

Some of the variable factors involved in evaluating uranium deposits might be considered under the following headings:

Geological. Physical nature of the deposit, its size and shape, tonnage and grade distribution of the mineralised material, etc.

Mineralogical-Metallurgical. Chemical and physical characteristics of the mineralisation affecting the extraction and the degree of recovery of the final product.

Geographic. Location of the deposit in respect to sources of supply and delivery of the product and conditions affecting working operations.

Human. Availability of labour and staff, efficiency of management and workers, efficiency of designed working plan and methods, taxation, legal and political factors.

These are just some of the variable factors involved in the evaluation of a deposit. In the earlier stages of evaluation it is mainly the geological variables which are involved. Uranium does, however, have special characteristics which affect the evaluation process.

Special Characteristics of Uranium Deposits

Uranium deposits are metalliferous deposits and, in general, the principles involved in their exploitation are essentially similar to that for other metals. All metals do, however, exhibit their own distinguishing characteristics which can be utilized in their discovery, evaluation, mining and milling. The only justification for presenting a paper on the evaluation of uranium is to emphasise the characteristics which distinguish it from other metals and which affect the techniques of evaluation and mining.

The main characteristics are threefold:-

(1) Although a metal, uranium is a fuel and thus has very distinctive marketing and utilization characteristcs;

(2) Radioactivity, and the production of radioactive daughter elements including radon gas;

(3) The relative ease of solubilization of uranium from its principal ore minerals and the differential solubility between the uranium and some of the daughter elements such as radium.

These three characteristics strongly affect the mining of uranium and distinguish it from other metals.

I. Marketing

In the large mines of the main uranium producing countries, the sales contracts and marketing of uranium is subject to agreements and contracts between the mines and the nuclear power utilities. This may be direct or subject to some measure of governmental control. The principal purchasers are the power utility companies or the governments of the advanced countries.

In regard to developing countries, uranium resources may be utilized (a) in a national nuclear power programme or (b) may be developed purely for commercial export to the advanced industrial countries. In the first case, while low cost national uranium would be desirable, strict adherence to the world market price may not always be necessary because in the interests of saving foreign currency, utilizing national resources, establishing an industry to provide employment and for general politico-strategic reasons a government may be prepared to pay a premium above world prices for nationally produced uranium.

In the second case, that is, sale on the world market, the product must be competitive and furthermore, purchasers are likely to be only interested in larger quantities and secure long term delivery contracts.

As the ultimate basis of evaluation is a cost/price target, the evaluation of uranium deposits in developing countries may depend on national policies in regard to these two options.

II. Radioactivity

(a) Advantages

(i) Evaluation Techniques

In the evaluation of a uranium ore body, radioactivity is an asset because calibrated gamma-ray logging equipment can give acceptable evaluation data in non-core drill holes. More expensive diamond core drilling may be avoided in many cases where it would be essential for other metals.

In the sampling of ore bodies at surface and within a mine, particularly in relatively homogeneous ores, a differential face scanner provides a rapid in-situ radiometric assay of a discrete yet adequately large sample, thus reducing requirements for conventional sampling and assaying.

(ii) Mining Operations

In marginally economic uranium mining operations it is highly important that, at all stages up to milling, ore selection be made as efficient as possible. Radiometric measurements can greatly assist this objective. Delimination of ore prior to blasting by using face scanners or even simple GM counters is an obvious first step. Broken ore selection within a stope can also be done using the same type of instruments or portable T-handle probes. Radiometric picker belts can also be used.

In the transport system it will also be normal to have a fixed station truck or carload radiometric analyser system. The sampling of bulk loads is therefore cheap and stockpiles of different grades can be made for future blending to produce a consistent feed grade to the mill. Low grade ore storage for heap leaching systems can also be arranged.

(b) Disadvantages

The principal disadvantage of the radioactive character of uranium ores lies in the hazards arising from radioactivity and principally in the decay of the uranium series and the production of radon gas. The radon daughter products found in uranium mine atmosphere have been assumed to be the cause of the higher than expected incidence of lung cancer among underground uranium miners: All mines require that they are ventilated to assure the comfort of the workers and their freedom from dust and fume hazards, but in underground uranium mines the costs of controlling radiation hazards arise from the need to ventilate active working places more extensively than would be necessary to control the normal mine air pollutants. Mining costs may therefore be higher than in equivalent mines for other metals and thus must be taken into account in the evaluation of the ore deposit.

III. Solubility of Uranium Ore Minerals

(a) Advantages

Uranium is bi-valent and produces two series of components, uranous and uranyl which differ greatly in their solubility, the former being relatively unsoluble and the latter highly soluble. Most of the common uranium ore minerals such as pitchblende and uraninite are easily soluble particularly in a distinctly acidic or alkaline environment. Many of the common ores of uranium also contain iron pyrites which in normal weathering may be relatively easily broken down, producing an acid environment which brings the uranium ore minerals into solution.

This property has been taken advantage of in many uranium ore districts to recover uranium in what has been called a "natural leaching" or "heap leach" system. Sub-mill grade ore, which may be regarded as having no mining cost, is loaded on to a bay with an impervious base surface and a controlled drainage system. On completion of the heap, water and pyrite and/or sulphuric acid is sprayed on the surface and the run-off liquors collected, probably re-circulated but ultimately drained to the main mill or to a simple small precipitation system. This possibility can have great economic advantages in uranium mining and should be taken into account during the evaluation of a property.

(b) Disadvantages

As radium and some daughter elements are far less soluble than uranium, differential solubility may leave detectable radioactivity in a deposit but with depleted uranium. The opposite situation of enriched uranium and low radioactivity can also occur. The deposit is said to be "out of equilibrum". This factor can cause erroneous evaluations and must be determined if radiometric measuring systems are used.

If the ore is proved to be susceptible to easy leaching then an expense must be incurred to cover any ex-mine ore stock completely either by plastic sheeting or by a roofed construction.

The other problem is that within a mine the same process is at work on all wet exposed surfaces and mine water, either pumped or flowing from adits is likely to be acid and contain uranium. Pollution of local streams and agricultural land can occur and protection costs may be very much greater than in equivalent sized mines producing other metals.

All of these features, related to both radioactivity and solubility must be taken into account when uranium deposit evaluation is done.

Each uranium occurrence has its own cost characteristics as well as its geological and mineralogical characteristics, and it is the economic geologist's duty to consider and evaluate all variable factors and to arrive at as good an estimate as possible of the size, average grade and economics of the deposit so as to eventually arrive at an estimated "cost of product" figure. Such a figure can then be judged against the market value or 'price' of the product; the difference being either profit or loss. For the purpose of this talk, the current (mid 1978) spot price of US$95,70/kilogramme U_3O_8, or 115 \$/kg U, or US$43,40/lb U_3O_8 is used as the market value or "price".

DISCOVERY AND PRELIMINARY EVALUATION OR "PROVING"

Exploration and Discovery

Uranium presents a unique record of successful exploration over the last thirty years. For no other metal in world history was such a high rate of discovery achieved in such a brief period. The success was based on two factors; firstly, the important utilisation of the metal which prompted high rewards, and secondly on the natural radioactivity and high solubility of most of the uranium ore minerals, which were aids to discovery.

Probably in no other type of mineral exploration is there a wider range of technical methods than those which are at the disposal of the geologist working in uranium. Full advantage must be taken of these special properties but it must also be remembered that otherwise uranium is a metal occurring in mineral deposits which are subject to all the other normal techniques of discovery, evaluation, and production.

As this talk is intended to deal with evaluation, and as discovery techniques will have been dealt with by other speakers, consideration will immediately be given to the first stage of evaluation.

Preliminary Evaluation of "Proving"

Preliminary Evaluation of radiometric anomalies is based on accurate examination and measurement of surface features only. The main principles of judging and assessing a surface anomaly are the same as for all minerals, but for the uranium geologist, measurement of radioactivity provides a marked advantage.

In preliminary evaluation, a system of elimination is involved by which anomalies are classed as not appearing to warrant any further attention and others as meriting some type of further examination.

It is well known to persons experienced in prospecting work in metals generally that most occurrences and discoveries cannot even be classed as prospects and that few prospects eventually become producing mines. Because of the ease of detection of radioactivity the ratio is even more unfavourable in the case of uranium.

One example of such statistics from a country which had been completely surveyed by aerial radiometric methods, the anomalies proved, prospected and developed will illustrate this.

		Number	% of total
1.	Number of aerial radiometric anomalies discovered (two and a half times normal background)	1,192	100
2.	Number of anomalies recommended by "proving" group for further work	114	10
3.	Number of anomalies prospected and developed	24	2
4.	Number of anomalies proved to have economically exploitable uranium ore reserves	4	0.3

This is probably a very typical set of figures and illustrates the importance of the "proving" stage.

The element of experienced judgement is so important in preliminary evaluation that this is work which must be done by experienced men. The "proving" geologist should have wide and considerable knowledge not only of geology but of all the evaluation stages. He should also have at least some knowledge of production methods and costs.

A first essential in proving work is the full and complete recording of all relevant information about an anomaly. No anomaly, no matter how apparently insignificant, should be dismissed with no record of it kept. Time after time it is later found that some information about such anomalies is required after all. It is also important that the directors of the campaign should have all the factual information available so that they can judge whether, in their opinion, the proving team has come to the correct decision. A standard reporting form for anomalies should be designed and used by the proving group and kept as permanent record at headquarters. A suggested type of form is illustrated, but variations can be made where necessary.

ANOMALY RECORD SHEET

LOCATION ______________________ SERIAL NO. ______________

Local Name ______________________ AERIAL SURVEY NO. ______________

District ______________________ DATE DISCOVERY ______________

Map Reference ______________________ DATE GROUND PROVED ______________

Aerial Photograph ______________________

HOST ROCK ____

CAUSE OF ANOMALOUS RADIOACTIVITY ______________________________

MAXIMUM RADIOACTIVITY (a) Aerial Sur. ____________ (b) Ground ____________

HOST ROCK RADIOACTIVITY ______________________________________

EXTENT OF ANOMALOUS ZONE: Maximum Length ______________________

Maximum Width: ____________ Average Width: ____________

TYPE OF STRUCTURE: ____

Strike ____________ Dip ____________ Length (max) ____________

Width (a) Maximum ______________________ (b) Average ____________

URANIUM MINERALS ____

ASSOCIATED HYDROTHERMAL EFFECTS ______________________________

____

METALLISATION ____

GANGUE MINERALS ____

CONCLUSIONS ____

____

____

This suggested form should be altered in whatever way necessary to satisfy the objectives of the project and the needs of the organisation.

At this stage no accurate estimate of resources can be made. A discovery is uranium mineral in place (an outcrop, in a trench, intersected in a drill-hole). The evaluation of a discovery is to determine its dimensions.

Size of Potential Orebody

The "proving" geologist must be sufficiently knowledgeable about structural geology to interpret the possible extension and continuity of a structure in which the anomaly occurs and thus be able to estimate its potential size and the ore tonnage which it could contain. He is not expected to make an accurate estimate, the margin of error may be very wide but at least he must be able to estimate whether there is any possibility of a reasonable tonnage in the structure or not. However, it is no good just making a guess, such measurements as are possible at the outcrop must be taken and a mental calculation of volume and tonnage made.

For example, in "Anomaly A", the geologist has located the anomaly and found that it is caused by an occurrence of radioactive minerals in a small fracture vein. He measures the vein and finds it is only 10 metres long and has no possible extension and that the average width is 0.20 metres. The radioactivity is confined to a two metre length of the vein. A rapid mental calculation shows that the vein tonnage per metre of depth is likely to be

$$\begin{aligned} \text{Volume} &= 10 \times 0.20 \times 1.00 \\ \text{Tonnage} &= 10 \times 0.20 \times 1.00 \times 2.50 \\ &= 5.0 \text{ Tons vein rock per metre of depth} \end{aligned}$$

(2.5 is used as the Specific Gravity of the vein).

Only one fifth of the outcrop length was mineralised so the available ore tonnage may only be one ton per metre of depth. Even if the structure extended to 50 metres depth this would only provide 50 tons ore and if the calibrated scintillometer had indicated a grade of 0.10% U_3O_8 this could mean only one kilogram U_3O_8 per metre of depth. The geologist knows that in vein structures the economic lower limit is likely to be about 0.25% U_3O_8 grade, in a vein width approximating to one metre and with a substantial ore tonnage (certainly some tons of thousands of tons), he can therefore safely dismiss this anomaly as being of no further interest.

In the case of "Anomaly B", the geologist finds a vein structure 2000 metres in length and with an average vein width of 2.50 metres. He can calculate that this structure could contain per metre of depth

$$\begin{aligned} \text{Volume} &= 2000 \times 2.50 \times 1.00 = 5000 \text{ m}^3 \\ \text{Tonnage} &= 5000 \times 2.50 = 12{,}500 \text{ Tons} \end{aligned}$$

Even assuming that the structure only extended to 200 metres depth this would provide 250,000 tons. The calibrated scintillometer indicates that about half the length might be mineralised to 0.20% U_3O_8 grade that is 125,000 tons at 0.20% U_3O_8 = 250 tons U_3O_8.

The geologist does not require to go any further, he can recommend with confidence that the anomaly is worth further investigations.

In considering the potential size of a vein structure, the "proving" geologist is faced with the problem of how the vein behaves in depth. It may take many forms, of which the straight continuation is only one; it may pinch out,

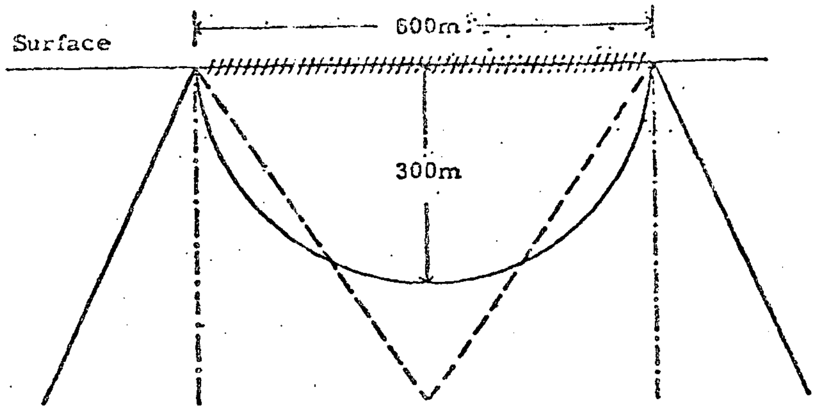


Figure 1

it may broaden, it may be a lens. Lacking any other evidence and simply for the purpose of the preliminary estimate of potential tonnage one useful method is to assume that the outcrop is the diameter of a circular lens. (Figure 1). This is usually a safe estimate for small to moderate length vein outcrops. It would not be applicable, or necessary, in vein outcrops of some kilometres.

In this case the estimate of potential size for a vein outcrop of, say, 600 metres and average width one metre would be as follows:

$$\text{Area of half circular lens} = \frac{(\pi r^2)}{(2)}$$

$$= \frac{3.14 \times 300^2}{2}$$

$$= 141{,}000 \text{ m}^2$$

$$\text{Volume} = 141{,}000 \times 1.00 \qquad = 141{,}000 \text{ m}^3$$

$$\text{Tonnage} = 141{,}000 \times 2.5 \qquad = 350{,}000 \text{ Tons}$$

On outcropping horizontally bedded deposits, the principles are the same, but additional evidence on continuity may be available.

Anomalies are therefore divided on the basis of surface evidence and mental or rough calculations into those which are rejected and those which are recommended for further work. Determination of whether the deposit could be economic is left to a later stage of the investigation.

The following Table I attempts to illustrate in very broad terms the progressive development of an exploration programme from the discovery phase through all the evaluation phases to the development and production phase. It relates numbers of anomalies at each phase to the relative approximate cost and possible accuracy of the phase.

Table I Relationship of Exploration Phase to Number of Anomalies, Relative Cost of Effort and Possible Margin of Error

Phases of Exploration & Evaluation Programme	Number of Anomalies and Prospects at each phase (the others having been eliminated	Relative Cost of Work done on each Anomaly	Margin of Error
1. Identification of anomalies	1000		
2. "Proving"	1000	x	1000%
3. Surface mapping	100	5x	
4. Trenching	75	15x	500%
5. Drilling	50	1500x	100%
6. Underground Mining Prospecting	20	6000x	20%
7. Mining Development and Production	3	800,000x	10%

This does not include the cost of discovery. It should also be noted that it is often possible to eliminate some of the phases. For instance, after preliminary evaluation it is not unusual to go almost directly to a drilling programme.

PHYSICAL EXPLORATION AND INTERPRETATION OF DATA

Policy The phases which succeed the "proving" phase may be referred to as physical exploration and include surface mapping, sub-surface techniques such as geophysics and detailed radon surveys, trenching, drilling and underground mine exploration. The detailed methodology of the various techniques require detailed description and demonstration and cannot be dealt with in a brief paper. It is only relevant in this paper to discuss briefly the policy and philosophy of the employment of the various techniques.

The development programme for an anomaly, occurrence or prospect consists of a series of steps or phases, each designed to eliminate an element of risk. The expenditure which is committed at each step should never exceed an appropriate fraction of the potential prize. A rough guiding target figure for the whole physical exploration programme up to the development of probable and proven ore reserves on which production could be based should not be more than 10% of the total content value of the orebody.

As will be seen from Table I the big cost jump comes as soon as investigation in depth has to be done, therefore, it is essential policy that the maximum possible information be first obtained from the cheaper surface exploration phases.

Surface Mapping All anomalies or occurrences which have been passed by the "Proving" group must be surface mapped. The requirement is to obtain complete information about the geological structures, the extent and degree of the mineralisation and the topography. In flat or moderately undulating ground, grid mapping is the best method as it is systematically complete in covering the ground, quick and sufficiently accurate for the purpose in view. Plane table mapping can be reserved for ground where the topography makes grid mapping impossible. Enlarged aerial photographs may also be used as a base.

These methods will pr̂ovide sufficient information to decide whether the prospect should go to a following stage or not. An exact triangulation survey will not usually be necessary until it has been decided that the prospect warrants drilling or mining prospection. When that is done, the information obtained in the grid or plane table survey can be incorporated in the final map.

Trenching The objective of trenching will be to delimit the general geology of the occurrence, possibly under shallow soil or alluvial cover, to delimit the extent and to sample the mineralised occurrence and to study changes in the mineralisation within the sub-surface depth explored by the trenches. Systematically taken samples may be used to estimate near surface blocks of ore.

Drilling With drilling, exploration work is now entering a more expensive phase and thus the necessity for drilling must have been fully proved in the previous phases and the objective of the proposed drilling must be clearly envisaged. Drilling may be designed to test the grade and width of an orebody or it may be used to seek geological information from which to plan other exploration methods such as prospect mining. Most drilling programmes are planned in phases, each phase having a limited objective. The results of each phase must be assessed and work will continue to the next only if the results justified it. Figure 2 might be an example in a small near vertical vein occurrence in which the surface information was inadequate to determine the true dip of the structure.

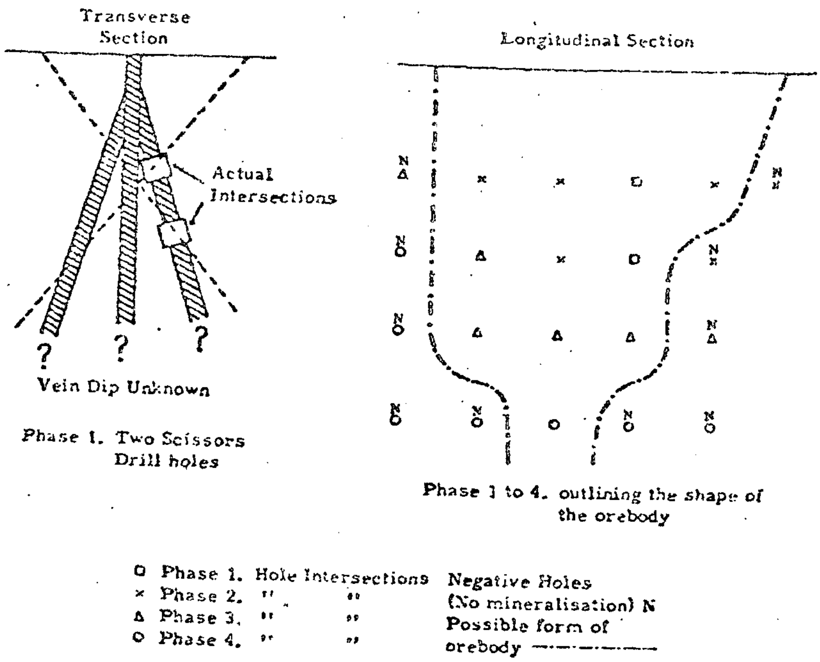


Figure 2

Drilling or Mining Prospection Problems may arise in deciding where and when to use drilling and where and when to use mining prospection.

Drilling is cheaper than mining prospection and is preferable whenever the nature of the ore can give decisive results. With a well planned drilling programme which discovers ore, the subsequent investment in underground work is subject to a much lower risk factor and the workings can be planned to open up the ore in the most economical manner. If, on the other hand, drilling finds no ore, the more expensive underground work is saved. Drilling can often be used intentionally to delimit barren ground and narrow down the zones that will have to be tested by the more expensive mining methods.

In stratiform deposits, chiefly of the sedimentary type, ore continuity and grade range is usually sufficiently well known so that no underground mining is required prior to development because complete evaluation can be done on drill hole evidence.

Mining Prospection Mining prospecting work is different from mining development work. The objectives in mining prospecting are to determine the size and grade and value of the orebody. With good and well planned work the mine prospection works may later be incorporated into the mining development plan of an economic orebody. This should always be attempted but it is not always possible. Mining prospection on an orebody where there has been no prior drilling is going into the unknown, except for such guidance as the geologist can give. False starts along veins which pinch out may be made, faults may cut off the orebody and mining exploration must be done in several directions to pick up the faulted section. Prospection policy must take all these possibilities into account and be prepared for a considerable cost of meterage done in barren ground. The old maxim for shallow mining prospecting should be adhered to where it can be employed "start in the ore, follow it and stay with it". This is the ideal but it is not always possible.

Records and Plan System A sound and efficient system for recording all the information required in the evaluation of ore deposits must be established.

Daily, weekly and monthly progress reports should be kept of the work done. The drill hole logs, gamma logs, assay records, surface maps, sample tickets, etc., should be sorted into the appropriate sections concerned, cross referenced and filed at regular and short intervals.

The whole system of recording mapped information, not only for underground work but for all phases should be standardised within an organisation. Geologists and prospectors should not be allowed to choose various sizes and scales of maps and different methods of reporting. If that were allowed to go on, the accumulation of unclassified, variously reported and stored map information could become a serious problem to the extent of making the efficient evaluation of mineral deposits difficult or even impossible. A systematic map reporting system should be decided upon and rigidly enforced throughout all programmes.

Several map recording systems are available and reference can be made to them in McKinstry's "Mining Geology" p. 162-197. (1).

CALCULATION OF ORE RESERVES

Averaging Assays In drilling programmes, the average grade and the true width of the mineralised structure represented are found and designated for each drill hole intersection. If the samples are multiple as is frequently the case in drilling, then one average grade value must be first found for the total width. Having obtained the single pair of figures for average grade and total width, these then represent a certain block of ground surrounding the drill hole.

In combining assay values, if the channel lengths of the sub-samples or the drill hole lengths are all equal then the average grade is simply the arithmetic average of the grades over the true total width.

Unequal Widths If, however, it is necessary to combine samples of unequal width (thickness) from either drill hole data or other types of samples, each sample must be weighted by its length as follows:

(a) Drill hole lengths

	%	m
A	0.20	0.20
B	0.50	0.80
C	0.20	0.50

(b) Mine level sub-channels

		%	m
	A	0.20	0.20
(4)	B	0.50	0.80
	C	0.20	0.50
(Samples)		(4)	(5)

The average value is a simple weighted average as follows:

Sample	Assay % U_3O_8	Width Metres	A. x. W.
A	0.20	0.20	0.040
B	0.50	0.80	0.400
C	0.20	0.50	0.100
Totals	-	1.50	0.540

$$\text{Average Assay} = \frac{A \times W}{W} = \frac{0.540}{1.50} = 0.36\%\ U_3O_8$$

The final figures which are therefore used to represent the average grade over the full width of structure at these points are 0.36% U_3O_8 over a width of 1.50 metres.

Effects of Density Since grade is expressed in percent uranium content by weight, in cases of extreme density differences further corrections will have to be made by weighting density and grade.

Averaging Assays in Sections Very few uranium deposits are of uniform width, and since a sample across a wide portion represents a larger tonnage than across a narrow portion, it is necessary to weight each assay in accordance with its width.

Sample Nos.	Average Assay Assay % U_3O_8	Width Metres	Assay x Width
1	0.27	0.85	0.229
2	0.15	0.65	0.097
3	0.35	0.70	0.245
(4) Sub-divided Calculated	0.36	1.50	0.540
5	0.22	0.85	0.187
6	0.25	0.60	0.150
Totals	-	5.15	1.448

$$\text{Average Assay} = \frac{\text{Total Width x Assay}}{\text{Total Width}} = \frac{1.448}{5.15} = 0.281\%\ U_3O_8$$

$$\text{Average Width} = \frac{\text{Total Width}}{\text{Number of samples}} = \frac{5.15}{6} = 0.86 \text{ metres}$$

The result for this section of equally spaced samples can therefore be stated as 0.28% U_3O_8 over an average width of 0.86 metres along a vein length of 6.0 metres.

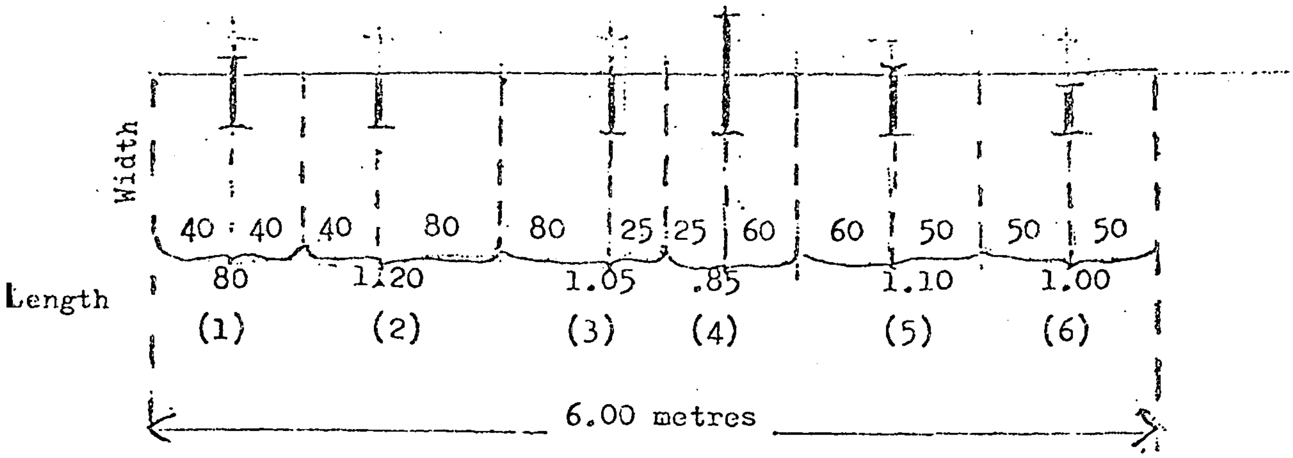
Uneven Spacing If the samples are not at evenly spaced intervals, another variable, that of the length of influence represented by each sample must be taken into account. Each sample must therefore be weighted by the length of influence which it represents, that is, a length equal to half the distance to the next sample on one side plus half the distance to the next sample on the other side. The average grade will then be given by

$$\frac{\text{width x length x assay}}{\text{width x length}}$$

and the average width by $\dfrac{\text{width x length}}{\text{length}}$

If it is assumed that the same six samples are spaced as follows (Figure 3) then the calculation will be made as over.

Figure 3



Sample No.	Assay % U_3O_8	Length Metres	Width Metres	W x L	W x L x A
1	0.27	0.80	0.85	0.680	0.183
2	0.15	1.20	0.65	0.780	0.117
3	0.35	1.05	0.70	0.735	0.257
4	0.36	0.85	1.50	1.275	0.459
5	0.22	1.10	0.85	0.935	0.206
6	0.25	1.00	0.60	0.600	0.150
Totals	-	6.00	5.15	5.005	1.372

Average Grade $= \frac{W \times L \times A}{W \times L} = \frac{1.372}{5.005} = 0.274\%\ U_3O_8$

Average Width $= \frac{W \times L}{L} = \frac{5.005}{6.00} = 0.83$ metres

The final result for this section of unequally spaced samples would therefore be stated as 0.27% U_3O_8 over an average width of 0.83 metres along a vein length of 6.00 metres.

The figures are slightly lower than in the situation where the samples were equally spaced because of the lesser weighting given to the higher grade sample No.4 because of its shorter length.

Erratic Assays Erratic assays occur infrequently but are considerably higher than other values. Erratic high assays can occur: 1) because of improper sampling methods; 2) because of assaying errors; 3) accidental salting of the sample; 4) a naturally occurring high grade spot; 5) sample volume is too small.

Most methods of calculating averages are based on the assumption that from each channel to the next the grade of ore changes at a uniform rate (a) or, that each assay represents the value of the ore for an interval extending halfway to the next sample on each side. (b) Although such an assumption yields a perfectly satisfactory approximation, it is rarely, or ever strictly true, and can lead to serious error if one or a few samples are notably richer than the rest. (c) This is illustrated in the Figure 4 where it can be seen that in the case of three samples 1, 2 and 3 the true situation (c) is not correctly averaged by the basic assumptions of the methods above (a) and (b).

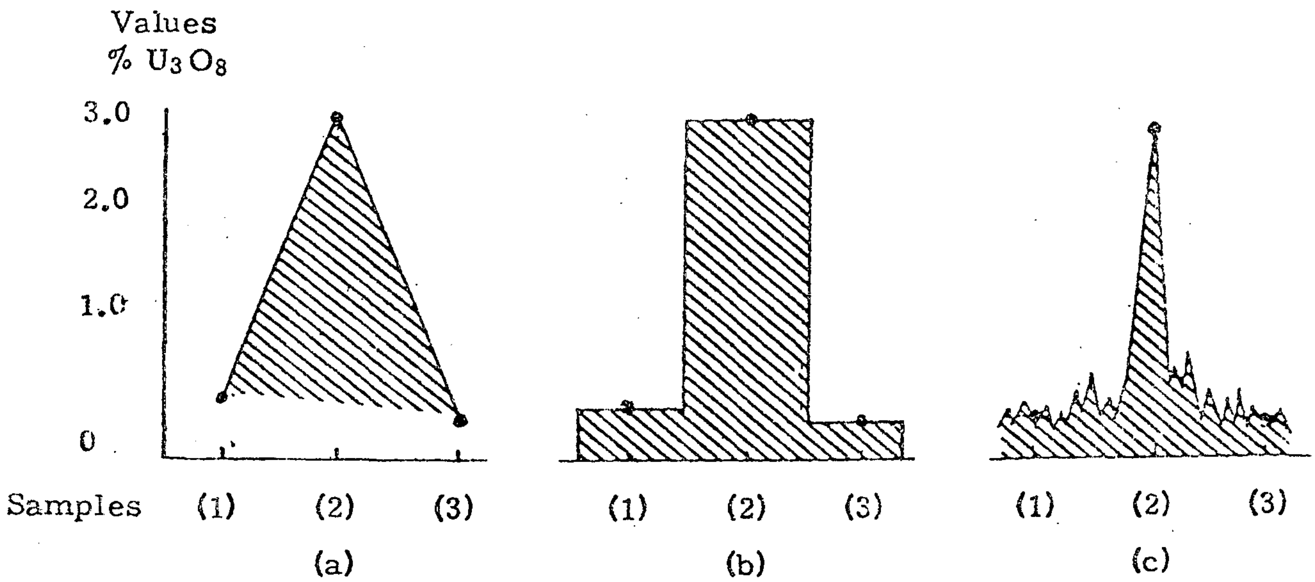


Figure 4

There is no easy way to determine whether or not to adjust for erratic samples. Reducing the grade of the erratic to the average of surrounding samples is preferred in some cases.

TONNAGE AND GRADE DETERMINATIONS

Having first computed average figures for samples it is then possible to calculate average grade for any particular length of mineralised structure exposed in a mine.

Calculating the Average Grades and Widths in Ore Blocks in Mines

The average grade of a block of ore is calculated from the average grades of the sampled openings which bound it. In typical vein deposits the openings consist of levels and raises that form the top, bottom and sides of the block.

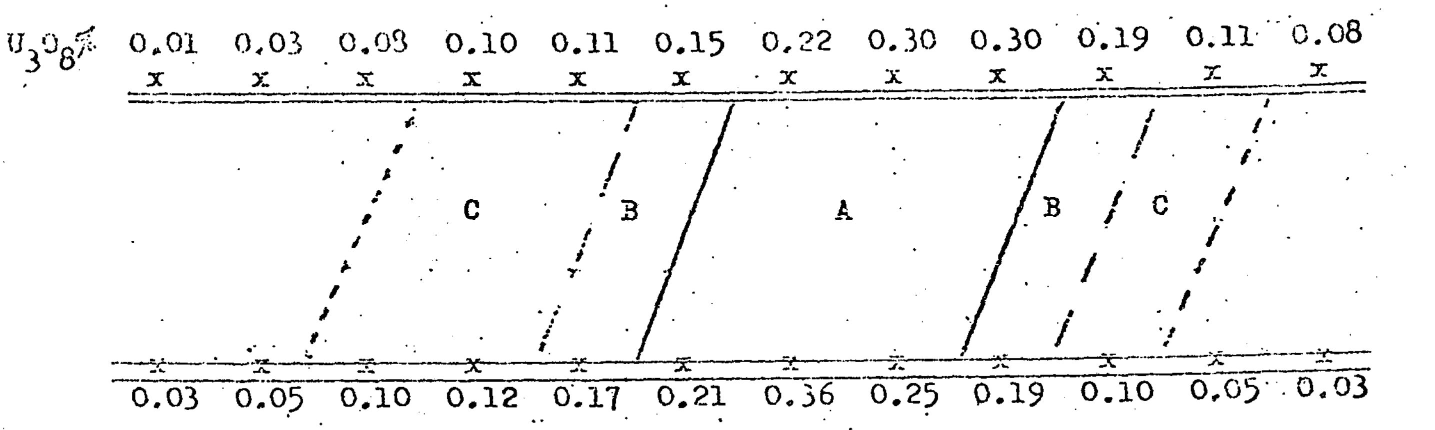
Cut-off Grade In order to outline ore a dividing line must be drawn between ore and waste. Just where to draw this line may be a very difficult problem in any orebody but is especially so in a new orebody. In effect, the cut-off grade is the lowest possible grade which can be included in an ore reserve estimation and yet maintain the ore as economic. Clearly, the lower the cut-off grade used, the greater will be the ore tonnage, but the lower will be the average grade of ore. The recoverable U_3O_8 does not therefore rise in proportion with the ore tonnage and as mining costs increase because of the greater tonnages of ore, there will be some point at which the operation becomes uneconomic. On the other hand higher tonnages usually mean lower unit costs so this is a factor working in the opposite direction.

What is ore and what is not ore depends on the cost, but the cost in turn depends on the scale of production; the scale of production depends on the amount of ore and the amount of ore depends on the cost - a vicious circle. The only way out of this is to calculate ore reserves using two or more cut-off grades. The whole process of provisional cost of product valuation is then followed through using the ore reserve tonnages calculated from these various cut-off grades and the one which most nearly matches the desired economic situation in regard to the final product is found.

In working mines this becomes a matter of adjustment depending on changing working costs, changing price of product or changing desired level of profit.

However, in new properties it is necessary to follow out the whole process, guessing initially as best one can from the evidence of comparable deposits, the cut-off figures which are most likely to be applicable.

The problem may be summarised in the following diagram:



Two mine levels 25 metres apart, all samples one metre wide and five metres apart.

Figure 5

Area	Cut-off grade % U_3O_8	Tons Ore	Average grade % U_3O_8	Tons U_3O_8
A	0.20	930	0.24	2.240
A+B	0.15	1.560	0.21	3.280
A+B+C	0.10	2.500	0.17	4.250

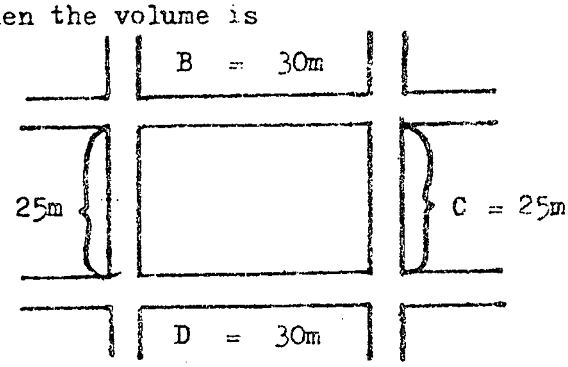
The complete provisional cost of product estimate would then have to be done for each case. Many conflicting factors would have to be taken into account; the lower percentage recovery of U_3O_8 in the treatment plant with lowering of average grade; the lower unit costs with higher tonnages and the higher total costs.

Calculation of Volume The weight of a block of ore is estimated by first calculating the volume and then applying a factor to convert volume to tonnage. Volume is the average thickness times the area. Area can be directly measured but it must be the true area and thus if a dipping vein has been horizontally projected in to a longitudinal section plan then the apparent area must be divided by the sine of the dip in order to correct the foreshortening of the projection. More simply a direct measurement can be made on vertical transverse sections of the area.

Areas of other irregular shapes, such as S curves on dip must be properly measured to give true volume.

Assuming an example of a vertical vein then the volume is

$$\frac{A + C}{2} \times \frac{B + D}{2} \times \text{Av. Width}$$
$$= 25 \times 30 \times 0.85 \text{ m}^3$$
$$= \underline{637.5 \text{ cubic metres}}$$



Calculation of Tonnage Tonnage is obtained by multiplying the volume by the specific gravity of the ore. A problem may arise in the correct determination of the specific gravity of the ore and it will always be necessary to establish the specific gravity of the ore by empirical methods in any orebody which is being examined.

Assuming that a specific gravity of 2.5 (or 2.5 dry tons per cubic metre) has been determined then the dry ore tonnage in Block A.B.C.D. will be = 637.5 m^3 x 2.5

= 1,594 dry metric tons

Moisture Content Although the ore reserves must be expressed in terms of dry tons it is also necessary to know the moisture content of the ore as it will be wet tons which will be mined and transported. This will obviously call for greater handling capacity and will increase the cost of mining and transport.

It is usually necessary to determine moisture content in each part of the mine separately as geological conditions may cause this factor to vary.

If in determining the specific gravity the experimental figures were as follows:-

Tons per cubic metre wet ore = 2.78
Tons per cubic metre dry ore = 2.50
Then the moisture content of the ore is 10%
The wet tonnage of Block ABCD will therefore be $\frac{1594}{90} \times \frac{100}{1}$

= 1771 wet tons and the ore reserve tonnage should be stated as follows:-

	Wet Tons	Dry Tons
Block A B C D	1,771	1,594

Tonnage U_3O_8 The content of U_3O_8 in the block is simply the product of the dry tonnage and the average grade of the block.

In the case of the example it would be

$$\left(\frac{1594 \times 0.35\% \ U_3O_8}{100}\right) = 5.580 \text{ tons } U_3O_8$$

Total U_3O_8 from Multiple Mine Blocks In reporting ore reserves, each block will be given with its wet ore tonnage, dry ore tonnage, average grade in % U_3O_8 for the whole mine are the simple sum of all the blocks. If the average grade figure for the whole mine is required as is usually the case then it must be found by weighting the grades of each block by the dry ore tonnages as follows:-

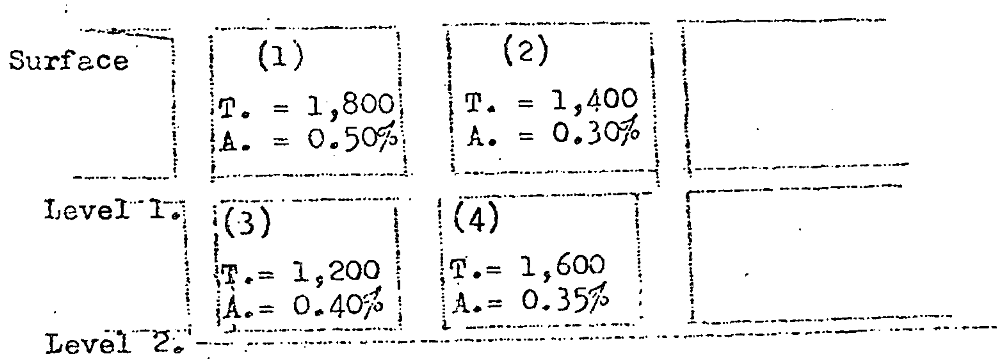


Figure 6

It will be assumed that block 4 is the same as the block A B C D calculation above. For purposes of ore reserve estimates, the degree of uncertainty is always such that it will be correct to round off numbers.

Block	Dry Tons Ore	Average Assay % U_3O_8	Tons U_3O_8 (T x A)
1	1800	0.50	9.00
2	1400	0.30	4.20
3	1200	0.40	4.80
4	1600	0.35	5.60
Totals	6000	-	23.60

Average Grade $\frac{T \times A}{T} = \frac{23.60}{6000} = 0.39\%\ U_3O_8$

The mine section as illustrated would contain 6,000 dry tons of ore with an average grade of 0.39% U_3O_8 and a U_3O_8 content of 23.60 tons.

Ore Reserve Estimation from Drill Holes In drilling programmes, the average grade and the true width of the mineralised structure represented are found and designated for each drill hole intersection. If the samples were multiple, as is frequently the case in drilling, then one average grade value must first be found for the total width as was explained in the section on Sub-divided Samples above. Having obtained the single pair of figures for average grade and total width these then represent a certain block of ground surrounding the drill hole. Areas, volumes and tonnages calculated for each block and then tonnages and average grades and widths are obtained by the methods described above.

Flat-Lying Orebodies All other factors being equal it is convenient and more satisfactory to design drilling programmes in flat lying orebodies so that the drill hole intersections are regularly and equally spaced. This may be on a square grid, any rectangular pattern or at the corners of 60° triangles.

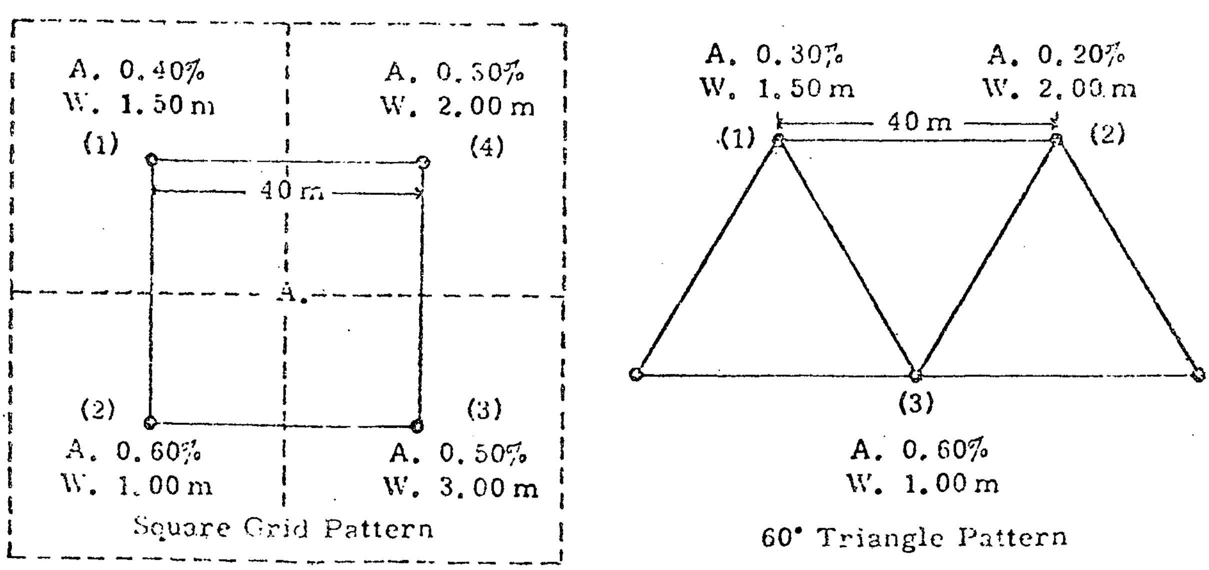


Figure 7

In the square grid pattern there are two ways of calculating the averages and tonnages. The first is simply to assign the values of each hole to a square of ground, (broken lines, squares 1,2,3 and 4). The calculations are simple and straightforward. This is the quicker method but it may not be as accurate as the other method which is to compute an average grade and width value for each block (A, full lines) from the four bounding drill holes.

Drill Hole	Assay % U_3O_8	Width m	A x W
1	0.40	1.50	0.60
2	0.60	1.00	0.60
3	0.50	3.00	1.80
4	0.30	2.00	0.60
Totals		7.50	3.60

Average Grade $= \frac{A \times W}{W} = \frac{3.60}{7.50} = 0.48\%\ U_3O_8$

Average Width $= \frac{W}{\text{No. of samples}} = \frac{7.50}{4} = 1.87$ metres

Area $= (40 \times 40) = 1600\ m^2$

Volume $= 1600 \times 1.87 = 2992$

Tonnage (S.G. = 2.00) $= 2,992 \times 2.00 = 5,984$

Say = 6,000 tons

U_3O_8 Tonnage $= 6,000 \times 0.48\%\ U_3O_8$

= <u>28.80 Tons</u>

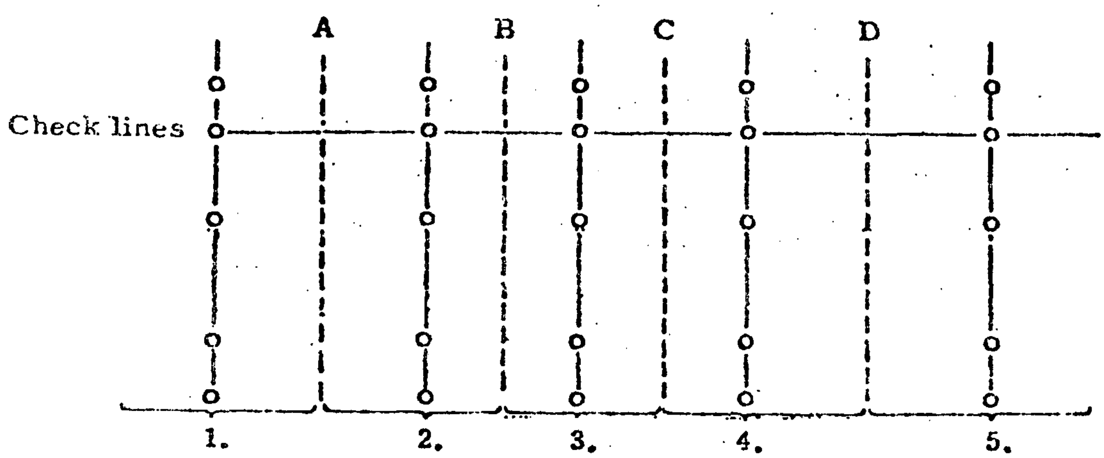
In the triangular patterns the block 1,2,3 is calculated in an exactly similar manner using the three sets of values and the area of the 60° triangle.

The whole reserves are calculated from the simple sums of all the blocks of ore and U_3O_8 tonnage. If an average grade is required this must be weighted by the ore tonnages. As the areas are all the same the average width is the straight arithmetic average.

Where rectangular but not square pattern drilling has been done it is possible to draw a series of cross-sections through parallel rows of holes and calculate the average grade and width for each cross section.

This method can be used even if the rows are not at equal distance apart nor the holes uniformly spaced along the rows. The average grade is calculated by combining the average grades of the respective cross sections, weighting each by its area (if the intervals between the cross sections are unequal) by the sum of half the distances to the adjoining cross-sections. (Similar to the case of the unequally spaced channel samples described above). A set of cross sections at right angles to the first gives a good means of checking the result.

In all these cases, extensions beyond the extreme ends are either assumed or omitted depending on the probable shape of the orebody.



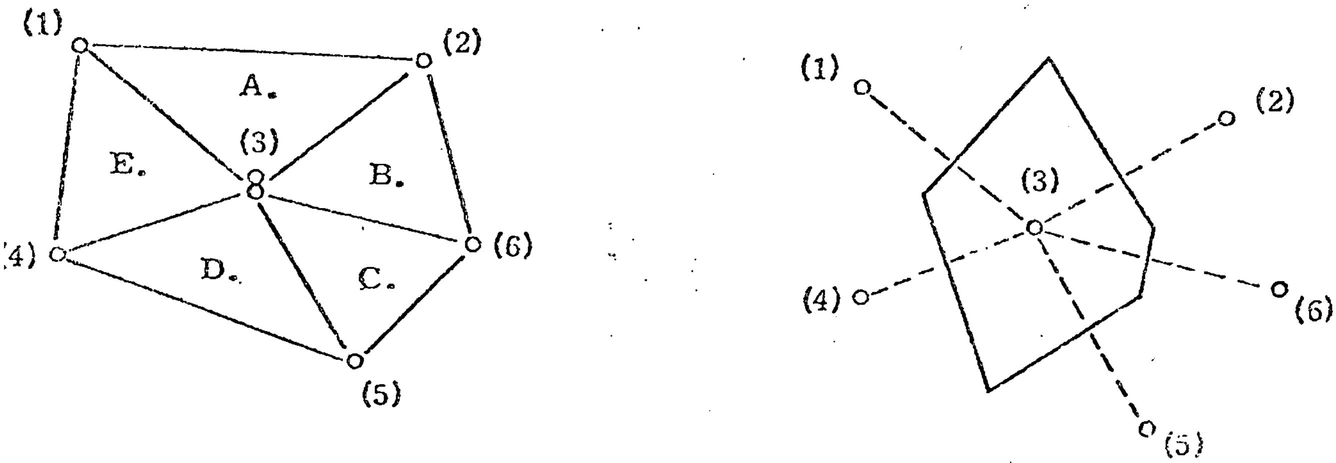
Rectangular Pattern Drilling

Figure 8

When the hole intersections are irregularly spaced, either on flat lying or in vertical, or near vertical vein deposits the calculations are more complex because the areas represented by each intersection will be of all different sizes.

The most common method is to divide the area into a series of triangles with a drill hole at each apex. The values of the three apex drill holes are then averaged as indicated above and assigned to that area. The measurement of the area represented may be calculated by geometry or measured by planimeter. Ore tonnage and U_3O_8 content are then found in the normal way and the averages for the whole orebody found by weighting the tonnages and the areas represented.

Figure 9



(a) Triangular Method

(b) Polygon Method

In example (a) the triangular method, the averages for triangle A are computed from the three drill holes, 1,2 and 3, triangle B from holes 2,3 and 6 and so on.

In method (b) a polygon is constructed around each hole by drawing its bounding lines perpendicular to, and bisecting the line connecting each pair of holes. In this method the values found in each hole are assigned to the whole polygon and the area of the polygon measured by planimeter. The polyogons are averaged together weighting each for its tonnage to get average grade for the whole area.

<u>Conclusions</u> The above is a summary of the general principles and the main methods of calculating ore reserves. It will be possible to use them directly or in modified versions for all uranium deposits.

Several matters concerning ore reserves have still to be discussed, such as ore limits and dilution.

PROVISIONAL COST OF PRODUCT ESTIMATE

Recoverable Reserves   Recoverable reserves are those that will ultimately be mined and must take into account the constraints of the mining method. These constraints are primarily dilution and recovery (mining loss).

Dilution   Dilution of ore may occur by intentionally including low grade sections in the ore reserve calculations which would be impractical to mine around.

An orebody can rarely be mined so cleanly that no waste is broken with the ore. To varying extents this applies to both vein and flat-lying deposits. To some extent it is dependant on the thickness of the ore-bearing structure - the thinner the ore structure the greater is liable to be the dilution. If a vein is narrow, say 25 centimetres, it will be physically impossible to mine only this width and a stope of perhaps one metre width will have to be made to follow the vein, thus cutting down 75 centimetres thickness of barren wall rock. In an extreme case like this, every endeavour would be made to separate the ore and waste, either by "reusing", that is breaking the waste and the ore in separate operations or by hand selection in the stopes. (The selection of ore in the stopes is another point at which radioactivity detectors particularly beta-window probes are especially useful in uranium ore mining).

The dilution factor has the effect of raising the ore tonnage to be mined and reducing the average grade of ore while leaving the total U_3O_8 content unaltered (provided the waste is truly barren).

Calculated Ore Reserve Content

Wet Tons	Dry Tons	Average Grade % U_3O_8	U_3O_8 Content Tons
6,670	6,000	0.39	23.600

Dilution Factor 15%

Mineable Ore $= \dfrac{(6{,}000 \times 115)}{100} = 6{,}900$ Grade $= \dfrac{(23{,}600)}{6{,}900} = 0.34\%$

Wet Tons	Dry Tons	Average Grade % U_3O_8	U_3O_8 Content Tons
7,670	6,900	0.34	23.600

This, therefore, concludes the factors involved in the calculation of ore reserves and in the measures which enable a statement of the mineable ore to be made. In reporting, a clear distinction should be made between ore reserve content and mineable or extractable ore. In the example, the two statements would be as follows:-

1. Ore Reserve Content

Dry Tons Ore	Average Grade % U_3O_8	Tons U_3O_8
6,000	0.39	23.600

2. Mineable Ore Reserves

Wet Tons Ore	Dry Tons Ore	Average Grade % U_3O_8	Tons U_3O_8
7,670	6,900	0.34	23.600

It will be the second statement which will be required for further evaluation steps.

Amenability Testing and Recovery Factor The next step in the evaluation process is to make preliminary tests of the ore in order to determine the most efficient process for the recovery of the U_3O_8 and how much of the U_3O_8 can be recovered economically. This latter is known as the mill recovery factor.

For uranium ores, the normal treatment process is a chemical one. The metallurgical laboratory must study the most suitable treatment process and also provide a preliminary cost estimate of the process. The laboratory must also find out and indicate the amount of the uranium in the ore which can be economically recovered. This recovery factor depends on the nature of the ore and the grade of the ore.

In evaluating the deposit it is therefore the recoverable uranium which must be taken into account. If the recovery factor for the above example of ore was 92.5% then the statement of ore reserves for evaluation purposes would be as follows:-

Mined Ore to Treatment Plant

Wet Tons	Dry Tons	Average Grade % U_3O_8	Tons U_3O_8
7,670	6,900	0.34	23.600

Recovery Factor 92.5%

$$\text{Recoverable } U_3O_8 = \frac{23,600 \times 90}{100} = \underline{21.830}$$

It is this 21.830 tons U_3O_8 which are recoverable which gives the value of the deposit (or section of the deposit). In this case, 21.830 kgs. U_3O_8 x 95.70 US$/kg. = 2,089,000 US dollars.

Rate of Production The possible or probable rate of production must first be estimated as this is a major factor in determining the capital and operating expenditure required. The nature of the orebody is likely to be the dominant influence in determining the rate of production. Expected rate of sale of product will also have an influence. In underground mines the tonnage which can be produced each day is governed by the number and size of stopes which can be worked and this is limited by the time required to develop the levels and prepare the ore for mining. A rough estimate of daily rate for a small mine is one tenth of the number of tons of ore reserves in each vertical metre.

In the case of flat lying ore deposits the rate of production may be partially determined by the operational problems in the open pit but in general the rate of production is more flexible in such operations.

The planned rate of production is also determined by the most economically correct size of capital installations and treatment plant for that particular orebody. It would be wrong, for example, to try to build a very large plant to treat the whole of the known ore in six months and then leave the plant with nothing further to do. This could only be possible if the whole of the cost of the plant could be written off in these six months. Normally a number of years are allowed for the writing-off of the cost of the installations and plant. This is often about ten years.

"Life" The life of a mine is difficult to predict unless the full extent of the ore is definitely known. For this purpose, the "Proven" and "Probable" ore reserve categories may not give a complete and accurate picture of the future situation. "Possible" ore and perhaps even "Potential" ore may have to be taken into account.

Some estimates for proposed rate of production and life must be made at this stage so that the costing of the whole process and the evaluation of the orebody can be computed.

Plan of Operations A complete plan of operations must then be worked out on a provisional basis. This must include a mining method with appropriate equipment, a treatment plant method and equipment, auxiliary requirements such as transport, stores, etc.

Cost Estimate of Mining and Treatment Processes The cost estimating for the actual construction of plant and purchase of equipment and for operations is a skilled job which must be done by specialists but for the provisional cost of production valuation, the economic geologist or mining engineer should have sufficient knowledge of the main capital and operational costs as to be able to conclude an intelligent estimate.

(a) Capital Expenditure The "rate of production" and the "life" of the orebody will determine the size of the capital installations necessary for the mine, the treatment plant, the auxiliary services, transport, etc. All of these items have to be listed and costed and a total capital expenditure arrived at.

All exploration expenditure on the property made up to the time of this estimate should also be included in the capital expenditure total.

The provisional cost of product is therefore:

$$= \frac{\text{Total Capital} + \text{Operating Expenditures}}{\text{Total kilograms } U_3O_8 \text{ produced}}$$

At the present time (1977-1978) the capital cost of a uranium mine, treatment plant, auxiliary services, etc. is estimated to lie between US$40,000 and US$100,000 per ton ore per day. This is based on recent installations in developing countries and on whether it is an open cast or an underground mine.

SUMMARY

Using the earlier example but increasing the tonnage by 100 times to conform to approximately the minimum size of a viable uranium mine. The various steps in evaluation may be summarised as follows:-

Step I Establish a provisional "cut-off" grade from comparative evidence.

Step II Calculation of Ore Reserve Tonnage and U_3O_8 Content and average grade in the ground.

Wet Tons Ore	Dry Tons Ore	Average Grade % U_3O_8	Tons U_3O_8
	600,000	0.39	2,360.00

Step III Estimate the moisture content. Assume a 10% Moisture Content to give Wet Ore Tonnage

667,000	600,000	0.39	2,360.00

Step IV Consideration of Dilution Factor of 15% to give Mineable Ore for the Treatment Plant

767,000	690,000	0.34	2,360.00

Step V Amenability testing and consideration of a Recovery Factor of 92.5% to give recoverable U_3O_8

-	-	-	2,183.00

Step VI Consideration of a value of 95.70 US$ per kilogram to give Recoverable Value of the orebody

= US$209,000,000

Step VII Estimate probable rate of production at 220 tons U_3O_8 per year for a life of 10 years. This would mean handling 76,700 wet tons per year from the mine and 69,000 dry tons per year in the plant (or 190 tons per day on a 365 day year).

Step VIII Plan a provisional mining method, mine, treatment process and plant.

Step IX Cost the provisional operational method, estimated for this exercise at 12.75 million US$ per year and capital expenditure at 13,300,000 million US$ (i.e. 190 x $70,000 = $13,300,000).

Step X Find the total estimated expenditure for the life (10 yrs) of the mine = (12.75m x 10) + 13.3m US$ = 141m US$.

Step XI Divide this total estimated expenditure (141m$) by the total estimated recoverable U_3O_8 (2,183,000 kilogrammes) to obtain a provisional cost of product.

$$= \frac{141,000,000}{2,183,000} = 64.59 \text{ US\$/kilogramme } U_3O_8$$

Step XII Compare this provisional cost of product (64.59$/kg U_3O_8) with the known market price of the product (95.70$/kg U_3O_8) and the difference is either profit or loss. In this example a profit of 31.11$/kg U_3O_8 is shown but if the result of the exercise was an apparent loss it might be worthwhile to adjust the cut-off grade and carry out the whole estimate again.

This concludes the brief summary of ore reserve and cost of product estimation methods. Clearly, actual practice is likely to be much more complex and specialists would be required for several phases of the operation, nevertheless the above paper gives a general outline of the processes which must be followed in evaluating a uranium orebody.

It is the intention of the I.A.E.A. to bring out a technical report or manual on the Evaluation of Uranium Ore Deposits during 1979 and which will go into the subject in more detail.

REFERENCE

McKinstry, H.E. MINING GEOLOGY, Prentice-Hall, Incorporated, New York, 1948.

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