

# **Nuclear Power: the Energy Balance**

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## **Chapter 2**

**From ore to electricity**  
**Energy production and uranium resources**  
sixth revision

## From ore to electricity

### Exploration

The energy costs of exploration for uranium ore resources are not taken into account in this study.

### Mining and milling

Uranium is a metal which has to be chemically extracted from uranium ore, much as iron has to be produced from iron ore. The uranium ore is mined, ground to a fine powder and treated with chemicals, such as sulfuric acid, to extract the uranium compound from the rock. The uranium is shipped to the next process in the nuclear chain as a compound called 'yellow cake'.

Most of the published calculations of the net energy production of a nuclear power plant - and hence the CO<sub>2</sub> emission produced by the whole nuclear chain - are based on a single, fixed value of the energy use of the production of uranium from the ore. Very rarely values dependent on the ore grade are used. Obviously, to produce one kilogram of uranium oxide U<sub>3</sub>O<sub>8</sub> from an ore grade of 0.1% U<sub>3</sub>O<sub>8</sub>, ten times the ore mass has to be processed compared to an ore with 1% U<sub>3</sub>O<sub>8</sub>. In addition, the extraction yield decreases as the ore grade falls. So, the specific energy expenditure of uranium extraction from uranium ore is strongly dependent on the ore grade. This fact determines the total extent of the world uranium reserves available for net electricity production. One can determine a physical cutoff grade, namely the grade when the mining and milling process consumes as much energy as can be produced from the uranium in the nuclear power plant, after deduction of the energy expenditure of all other supporting processes. In this chapter we examine this question. See further on in this chapter for graphic representations of the energy balance.

In the following we treat first the energy cost of mining the uranium ore, and then the cost of milling it to obtain the "yellow cake" (U<sub>3</sub>O<sub>8</sub>).

### Energy costs of mining

Uranium ore is mined in open pit mines when the ore bodies are not deeper than about 200 metres and the stripping ratio is not more than about 30. A stripping ratio of 30 means that for every Mg uranium ore 30 Mg overburden has to be removed. This ratio varies widely among open pit mines. In 2004, 51% of the world uranium ore was mined in underground mines 28% in open pit mines, and 20% in *in situ leaching* mines.

Besides on the ore type and grade, the specific energy requirements per Mg ore mined for mining are also dependent on the amount of overburden in case of open-pit mining and the depth of the ore body in case of underground mining. The stripping ratio can make differences in specific energy requirements of a factor of five, with the same ore type [Chapman 1976-2].

**Table 2. Energy requirements for mining uranium ore.**

Open = open pit mining, Und = underground mining, av = average (60% O + 40% U)

reference	type of mine	specific energy, $J_e + J_{th}$ GJ/Mg ore	thermal to electric energy ratio, $R = J_{th}/J_e$	remarks
Franklin et al. 1971	Open	0.079	-	only thermal input
Rombough & Koen 1974	av	1.004	6.78	direct energy only mining + milling
Rombough & Koen 1974	av	1.564	4.61	direct + indirect energy mining + milling
Rotty et al. 1975	av	0.530	6.2	direct energy only

Rotty et al. 1975	av	0.526	11.0	indirect energy only
Rotty et al. 1975	av	1.056	8.00	direct + indirect
SRI 1975	Open	0.222	4.85	
	Und	0.361	2.49	
Kolb et al. 1975	av	0.357	1.94	based on coal mines
Chapman 1975	Open	1.21	-	primary input, no details
ERDA-76-1 1976	av	1.056	8.00	from Rotty et al. 1975
Mortimer 1977	Open	0.0031 - 3.50	1.07 - 0.58	direct energy only
	Und	0.50 - 4.60	14.2 - 3.55	direct energy only
Orita 1995	Open	3.9 - 160 **	35 - 552	
	Und	3.9 - 77 **	0.83 - 2320	
WNA 2005	open	0.41	-	only thermal input mining + milling combined

One sees that there are many different values given. In many references it is not clear what the range of values refers to.

The figures from Rotty et al. 1975 are based on an unpublished survey of energy consumption in the US mining and milling operations, conducted by the US Bureau of Mines in 1973. At that time virtually all uranium in the USA was extracted from high grade sandstone deposits or other soft ores. The figures represent the average of 60% open pit and 40% underground mining and includes energy embodied in chemicals and and equipment, according to the authors.

The figures from Mortimer 1977 represent only the direct energy input and energy embodied in explosives and are the result of an elaborate process analysis. Data on fuel consumption were taken from actual equipment, such as excavators, bulldozers, trucks, etcetera, and actual mines. Mortimer developed a mathematical model of the energy requirements of uranium mining, as function of several variables, such as: total amount of rock handled per unit mass of ore, and transport distance from mine to mill. The values of these quantities vary widely from mine to mine. Mortimer made no distinction between rock types, probably his figures are based only on mines in the USA and Canada, with easily mineable ores.

Orita 1995 calculated the energy requirements per Mg U; his low values refer to rich ores with  $G = 2.5\%$ , the high values to lean ores with  $G = 0.01\%$ . The figures in Table 2 are corrected for energy requirements per Mg ore. neither the high values, nor the wide range are explained in his paper. Presumably they are based on actual mines and may include different rock types.

In this study the value from Rotty et al. 1975 is used, as is done in ERDA-76-1. It is derived from an average over many U.S. sources. This value is an average of 60% open-pit and 40% underground mining and includes indirect energy consumption, energy embodied in chemicals and equipment:

$$J_{\text{mining}} = J_e + J_{\text{th}} = 1.06 \text{ GJ/Mg ore with } R = J_{\text{th}}/J_e = 8.0.$$

These figures may be a low estimate, even for soft ores. The figures of Mortimer and Ortita seem to be more reliable as an average of world mining activities, because they are based on actual mines.

### Energy cost of milling

In the mill, the mined uranium ore is ground to a fine powder and treated with chemicals, often sulphuric acid, to extract the uranium compound from the rock. The uranium is shipped to the next process in the nuclear chain as a compound called 'yellow cake', a yellow solid with a variable

content of sodiumdiuranate  $\text{Na}_2\text{U}_2\text{O}_7$  and/or ammoniumdiuranate  $(\text{NH}_4)_2\text{U}_2\text{O}_7$ . The uranium content of the ore is expressed in mass percent  $\text{U}_3\text{O}_8$ .

### Specific energy requirements

Uranium occurs in many kinds of ore in the earth's crust. Roughly, the ores can be divided into two groups (see e.g. Orita 1995, [UIC-34 2005](#), [WNA-mining 2005](#))

- (1) soft ores, e.g. sandstones, shales and calcretes, with typical grades ranging from more than 10% down to about 0.01%  $\text{U}_3\text{O}_8$ , and
- (2) hard ores, e.g. quartz pebble conglomerates and granites, with grades varying typically from about 0.1% down to about 0.001%  $\text{U}_3\text{O}_8$ , or less. Some high-grade vein-type ores are also hard.

The specific energy expenditure of the milling of the two types of ore are quite different. This is caused by different properties of the rocks.

Values of specific energy requirements from the literature are compiled in Table 3.

Table 3. Energy requirements for milling uranium ore

reference	specific energy, $J_e + J_{th}$ GJ/Mg ore	thermal to electric energy ratio, $J_{th}/J_e$	remarks
Ross&Guglielmin 1968	0.061-0.126	–	preparation only, electric only
Franklin et al. 1971	1.40	9.7	
Rombough & Koen 1974	1.004	6.78	mining + milling direct
Rombough & Koen 1974	1.564.	4.61	mining + milling direct + indirect
Rotty et al.	1.06	7.0	
SRI 1975	0.96	8.9	
Kolb et al. 1975	0.504 - 2.50	6 - 16	
Chapman 1975	0.927	8.4	
Huwyler et al. 1975	41.1		electric dissociation of granite
ERDA-76-1	1.27	7.0	
Kistemaker 1976	4.432	0.093	
Mortimer 1977	1.76 - 6.90	3.3 - 33	
Orita 1995	6.2 – 95	1.54 - 2270	
WNA 2005	0.41	–	mining + milling, thermal only

Differences in the values are (partly) explainable by differences in ore type. The value given by Kistemaker 1976, for instance, is the empirical value for hard low grade ore in South African uranium mines, including the direct energy expenditure plus the indirect energy embodied in chemicals, but excluding energy embodied in equipment. Some authors give only the direct energy requirements.

Kistemaker (1976) stated that extraction losses at ore grades  $G < 0.02\%$  become so high (more than 50%) that rocks with such low grades can no longer be considered to be ores. His observation was based on the practice in South African uranium mines. This point will be treated more thoroughly when we discuss mining and milling yields, below. Most studies ignore the yield-grade relationship.

To be sure that the specific energy requirements are not overstated, the values from ERDA-76-1 are used in this study for soft ores:

$$J_{\text{milling}} = J_e + J_{\text{th}} = 1.27 \text{ GJ/Mg ore with } R = J_{\text{th}}/J_e = 7.0,$$

and the Kistemaker value for the hard ores:

$$J_{\text{milling}} = J_e + J_{\text{th}} = 4.43 \text{ GJ/Mg ore with } R = J_{\text{th}}/J_e = 0.093$$

Kistemaker did not include the energy input for equipment (capital goods). If the same estimate for this indirect energy input as in [Rotty at al. 1975] and in [ERDA-76-1 1976] is used, the milling energy requirements per metric tonne hard ore becomes:

$$J_e + J_{\text{th}} = 4.49 \text{ GJ/Mg ore with } R = J_{\text{th}}/J_e = 0.10.$$

Judging from the figures of Mortimer and Orita, above figures may be a low estimate.

### **Total energy requirements for mining and milling:**

From the above we find the total energy needed for the mining and milling of soft ores:

$$J_{\text{mm}} = J_e + J_{\text{th}} = 2.33 \text{ GJ/Mg ore with } R = J_{\text{th}}/J_e = 7.5,$$

and for hard ores:

$$J_{\text{mm}} = J_e + J_{\text{th}} = 5.55 \text{ GJ/Mg ore with } R = J_{\text{th}}/J_e = 1.6$$

### **The energy cost of 1 kg of uranium**

The energy cost of extracting 1 kg of uranium is found by dividing  $J_{\text{mm}}$  by the mining and milling yield,  $Y$ , and the mass fraction of uranium in the ore,  $\gamma$ . This must be corrected to account for the weight of uranium (0.848 kg) in 1 kg  $\text{U}_3\text{O}_8$  and the convention of giving ore grades in %-mass  $\text{U}_3\text{O}_8$ .

$$J = \frac{J_{\text{mm}}}{Y \cdot \gamma} = \frac{J_{\text{mm}}}{Y \cdot 0.00848 \cdot G} \text{ (GJ/MgU)}; \quad J = \frac{J_{\text{mm}}}{8.48 \cdot Y \cdot G} = \frac{c}{Y \cdot G} \text{ (GJ/kgU)} \quad (\text{Eq. 2.1})$$

$$Y = 0.980 - 0.0723 \cdot (\log G)^2 \quad (\text{Eq. 2.2})$$

with:

$$c = J_{\text{mm}} / 8.48 \text{ GJ/kg U};$$

$$G = \text{ore grade (mass-\% } \text{U}_3\text{O}_8\text{)};$$

$$Y = \text{yield of the mining and milling process (fraction extracted U)}.$$

The yield  $Y$  vs. ore grade is found from the data presented in Table 5 below. The sources of the data are also given there. The complete references are to be found in the *References* chapter. The two values of  $c$  are given in the table 4:

Table 4. The energy-cost parameter,  $c$ , to be used in Eq. 2.1 to determine the total energy cost of mining and milling

ore type	$c$ (GJ/kg U)	$c$ (PJ/Mg U)	$R, J_{\text{th}}/J_e$
soft	0.275	0.000275	7.5
hard	0.654	0.000654	1.6

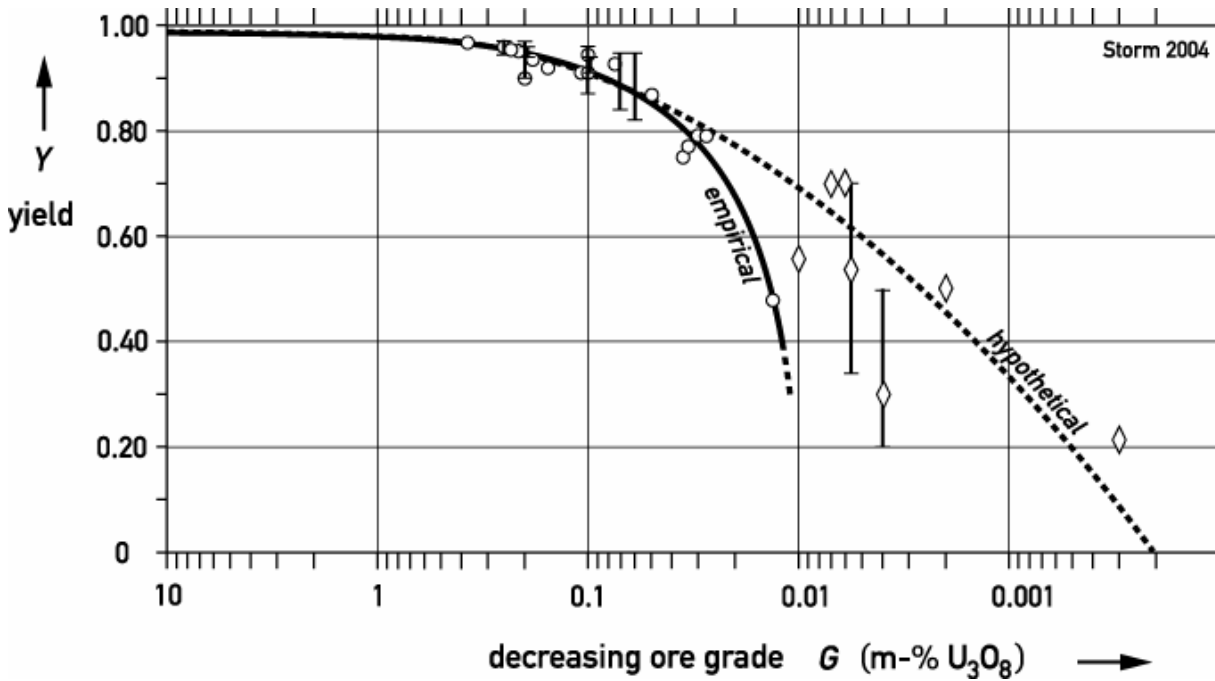


Figure 7 The extraction yield, or recovery, Y, of uranium oxide,  $U_3O_8$ , as a function of ore grade. The circles are empirical values from the literature, based on actual mining operations. The diamonds are taken from hypothetical mining and milling operations.

Table 5. Literature values of yield vs ore grade.

ore grade in mass-% $U_3O_8$	yield of mining and milling	References (see chapter "References" for details)
0.34	0.97	Franklin et al. 1971
0.26	0.958	James & Simonson 1978
0.25	0.962	Ross & Guglielmin 1968
	0.95-0.97	Simonson, Boydell & James 1980
0.24	0.95	James & Simonson 1978
0.22	0.949	James & Simonson 1978
0.208	0.95	Rotty, Perry & Reister 1975
0.20	0.959	Ross & Guglielmin 1968
	0.94-0.96	Simonson, Boydell & James 1980
	0.90-0.97	GJO-100 1980
	0.90	SRI 1975
	0.95	Rombough & Koen 1975
0.18	0.937	James & Simonson 1978
0.156	< 0.92	James & Simonson 1978
0.15	0.954	Ross & Guglielmin 1968
	0.935-0.95	Simonson, Boydell & James 1980

0.14	0.924	James & Simonson 1978
0.125	0.95	Ross & Guglielmin 1968
0.11	0.909	GJO-100 1980
0.10	0.944	Ross & Guglielmin 1968
	0.91-0.94	Simonson, Boydell & James 1980
	0.87-0.96	GJO-100 1980
	0.910	James & Simonson 1978
0.075	0.933	Ross & Guglielmin 1968
0.07	0.84-0.95	GJO-100 1980
0.06	0.82-0.95	GJO-100 1980
0.05	0.87	Simonson, Boydell & James 1980
0.035	0.75	James, Boydell & Simonson 1975
0.033	0.77	Simonson, Boydell & James 1980
0.030	0.79	Huwylar, Rybach & Taube 1975
0.0267	0.786	Kistemaker 1976
0.013	0.472 (see text)	Kistemaker 1976
0.010	0.55	Simonson, Boydell & James 1980
0.007	0.70	Rotty, Perry & Reister 1975
0.006	0.70	Burnham et al. 1974
0.0055	0.34-0.70	Mutschler, Hill & Williams 1976
0.004	0.30	Simonson, Boydell & James 1980
	0.20-0.50	James & Simonson 1978
0.0003	0.21	Huwylar, Rybach & Taube 1975

In the table we have presented all of the data (empirical and hypothetical) that we have found. Only those data where the ore grade is specifically stated are used in Figure 7.

The solid curve in this figure, beginning at the y-axis and changing to a dotted curve where the empirical points begin to deviate strongly, is given by Eq. 2.2 which is a fit to the empirical data down to about  $G=0.05\%$ . It is seen to be a reasonable fit to the hypothetical points also down to much lower ore grades. The solid curve is fitted to the empirical data down to the leanest ore for which there is data. This shows much lower yield than the extrapolation. It is quite possible that the real yield will turn out to be much less than the extrapolation of Eq. 2.2 to leaner ores.

Note that there are two points given by Kistemaker (1976). He cites the yield (0.786) at ore grade 0.0267%, but also mentions that this is not the true yield or ore grade. The mined ore is first crudely selected on the basis of the radioactivity of the rocks. If one takes the amount of debris that is thrown away into account, a totally different picture arises. The true ore grade then drops to 0.013% and the yield to 0.472. In our calculations we have not used this datum, but have assumed that Eq. 2.2 remains valid down to very lean ore grades. The reason that we do not use it is that it is the only one that deviates strongly from the value given by Eq.2.2. It is certainly correct for that particular case, but may be an exception to the general trend.

Although the strong dependence of the energy expenditure of uranium extraction on the ore grade (and type) determines what portion of the world's uranium reserves are usable for net electricity

production, this dependence is still ignored by the World Nuclear Association (formerly Uranium Institute London) [WNA-11 2001] or [UIC-57 2001].

In retrospect it is interesting to note that the ore utilized in South Africa in the seventies was so poor that, had the object been to produce energy for household and industrial consumption, the yield would have been negative.

It should be noted that the values of  $c$  as listed in Table 4, may greatly underestimate the specific energy requirements of mining + milling for ores with grades below 0.03% U<sub>3</sub>O<sub>8</sub>. There are several reasons for this.

- As far as can be deduced from the open literature, the energy requirements for mining alone probably are underestimated with a factor of 3 or more, as is pointed out above, see Table 3.
- The relatively high extraction yields at low ores grades are based on assumed, but yet unproven techniques. An example of such an unproven separation technique is described by Huwyler et al. 1975. It is doubtful whether such techniques will ever be developed.
- Higher extraction yields imply more stages in the extraction process and require higher specific energy and materials inputs.
- In this study the specific embodied energy in the materials and equipment for advanced extraction methods at low ore grades are assumed to be the same as that of the relatively simple milling processes which are operational at present to process rich ores.

### **In-situ leaching**

In some places in-situ leaching (ISL) is applied to extract uranium from ore still in the ground. Chemicals are pumped down via injection wells into the ore body and the uranium-bearing liquor is pumped up from production wells, after a residence time of 3-25 years. Large quantities of chemicals are needed: sulphuric acid, nitric acid, hydrofluoric acid, ammonia and other, together tens to hundreds of tonnes (Mg) chemicals per Mg uranium extracted (Mudd 2000). The extraction yield ( $Y = 50-80\%$ ) is lower than of conventional milling. From the literature it is not clear whether the stated yields relate to the extraction from the rock, the extraction of uranium from the resulting solution or the combination of both. Given the low uranium content of the parent rock and of the solution pumped from the production wells, the cited yield probably relates only to one of the stages. The overall yield, extraction from ore in the ground to yellow cake, may be in the range of, say, 20-40%.

A major problem of ISL is the large-scale contamination of aquifers, not only by the added chemicals, but also by radioactive and toxic elements, such as radium, heavy metals and arsenicum, which are chemically mobilized from the parent rock as well.

The ISL technique cannot be reconciled with sustainable development, because of the harmful and irreversible effects for the environment.

A rough impression of the energy requirements embodied in the chemicals for extraction can be made. Assuming 100 Mg sulfuric acid plus 3 Mg ammonia are needed to extract one Mg uranium from the ground – in some places two to three times as much is consumed – the embodied energy in these two chemicals alone is:

$$J_{isl} = 0.46 \text{ GJ/kg (U)} \qquad R = J_{th}/J_e = 2.8$$

These figures are based on the specific energy intensities according to Rotty et al. 1975 [Q95]:

sulfuric acid H<sub>2</sub>SO<sub>4</sub>  $J_{spec} = 2.87 \text{ GJ/Mg}$  with  $R = 100$ , and

ammonia NH<sub>3</sub>  $J_{spec} = 86.65 \text{ GJ/Mg}$  with  $R = 1.41$

The above figure is about the same as mining and milling rich soft ores in the conventional way. It should be emphasized that the figures represent only a fraction of the total specific energy requirements of ISL per kg extracted uranium. Not included are, for example, the energy

requirements for drilling injection and production wells, pumping and the extraction of uranium from the solution.

The energy requirements of in situ leaching will vary over a wide range, because geochemical conditions, depth of ore body, number of wells, operational life of each well and ore properties vary widely. In addition the energy requirements depend on the ore grade, as with conventional mining and milling. Data on actual mines are lacking in the open literature, so the average values are unknown.

In this study the specific energy requirements of ISL are assumed to be the same as of open pit mining, taken as the average of all mines and mills. This assumption will probably not lead to overestimation of the overall specific extraction energy of uranium from ore. Mortimer 1977 gives figures for ISL in the same range as soft ore mining and milling.

### Conversion of U<sub>3</sub>O<sub>8</sub> into UF<sub>6</sub>

As stated above, the uranium is transported from the mill in the form of yellow cake. This mixture must be refined and converted to very pure U<sub>3</sub>O<sub>8</sub> (triuraniumoctaoxide) in order to carry out the following step, i.e. the (chemical) conversion to very pure UF<sub>6</sub>. This is done in a number of chemical processes. This must be done because a gaseous compound is necessary for the enrichment in <sup>235</sup>U, and UF<sub>6</sub> is the only uranium compound that is gaseous at low temperatures. Table 6 gives the literature values that we have found.

Table 6. Literature values of the energy cost of conversion of U<sub>3</sub>O<sub>8</sub> to UF<sub>6</sub>.

reference	specific energy, $J_e + J_{th}$ GJ/Mg U	R, $J_{th}/J_e$
Franklin et al. 1971	171	3.3
Rombough & Koen 1974	288	4.3
Rotty et al. 1975	1478	27.1
SRI 1975	1350	35.5
Chapman 1975	252	3.4
ERDA-76-1	1478	27.1
Mortimer 1977	196-360	3.6 - 7.2
Orita 1995	25.2 - 176000	1000 - 1200

For the same reasons as given for mining and milling we have chosen the values of [ERDA-76-1, 1976] for this study. The specific energy expenditure is given by:

$$J_{conv} = J_e + J_{th} = 1.478 \text{ GJ/kg U, with } R = J_{th}/J_e = 27 \quad (\text{Eq.3})$$

Process loss is 0.5% according to [NRC 1996]

### Enrichment

The separative work S (number of separative work units SWU, with dimensions of kg) can be calculated from DOE/EIA 1997. The equations used for the calculation are:

$$\frac{F}{P} = \frac{x_p - x_t}{x_f - x_t} \quad (\text{Eq. 4.1})$$

$$S = P \cdot V(x_p) - F \cdot V(x_f) + (F - P) \cdot V(x_t) \quad (\text{Eq. 4.2})$$

$$V(x) = (2x - 1) \bullet \ln \left( \frac{x}{1-x} \right) \tag{Eq. 4.3}$$

- $F$  = feed mass uranium      kg U
- $P$  = product mass uranium    kg U
- $S$  = separative work            SWU = separative work units
- $x_f$  = feed assay = 0.0071      fraction  $^{235}\text{U}$
- $x_p$  = product assay            fraction  $^{235}\text{U}$
- $x_t$  = tails assay = 0.0020      fraction  $^{235}\text{U}$

The resulting values of  $S$  for several typical cases are given in Table 7:

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Table 7. Separative work for typical values of the product assay. Tails assay is assumed 0.0020.

$x_p$ = product assay, in % $^{235}\text{U}$	$S$ = separative work units (SWU)	$F/P$ ratio
2.6	3.45	4.71
3.3	4.98	6.08
4.2	7.01	7.84

##

**Energy costs of enrichment**

There are two principle techniques used in enrichment: ultracentrifuge and gas diffusion. In our original paper we only considered gas-diffusion enrichment. The share of the latter process is, however, declining in the world market. For this reason in this revision of our study we assume that only 30% of the enrichment in the future will be done by gas diffusion.

The literature values of the energy requirements are listed in Table 8.

Table 8. Energy requirements for enrichment

- n.a. = not available
- \* = only electric input given
- n.c = not comparable

Source	gas diffusion		ultra centrifuge	
	$J_e + J_{th}$ GJ/SWU	$R$ $J_{th}/J_e$	$J_e + J_{th}$ GJ/SWU	$R$ $J_{th}/J_e$
Franklin et al. 1971	n.a.-	n.a.	n.a.	n.a.
Rombough & Koen 1974	9.30	*-	-	-
Oregon 1974	9.78	0.052	-	-
Kolb et al. 1975	8.91	0.032	1.116	0.55
Kistemaker 1975	8.09	0.074	1.342	0.78
Rotty et al. 1975	11.00	0.083	-	-
SRI 1975	not per swu	-	-	-
Chapman 1975	9.06	0.040	-	-
ERDA-76-1 1976	11.00	0.083	-	-
Eaton et al. 1977	11.16		1.08-1.44	
Mortimer 1977	8.60 - 11.71	0.039 - 0.015	1.04 - 1.98	0.20-0.71
Crossley 1980	8.6	-	0.36	-
Wilkie 1980	8.76	-	-	-
INFCE 2 1980	8.28-9.00	-	0.378 - 1.44	-
Orita 1995	0.18 - 10.9	-	-	-
WNA 2003	8.64	-	0.23	*-

Specific energy expenditure should include construction, operation and maintenance of the enrichment plant. Enrichment by ultracentrifuge has a lower direct energy cost but costs of operation and maintenance are much higher because of the short technical life of the centrifuges (INFCE2, 1980, Becker et al, 1982 and Crossley, 1980). The net difference in specific energy use between the ultracentrifuge and the gas- diffusion technologies is only about a factor of two. We use the results of the ERDA-76-1 study for gas diffusion, as shown in Table 8.

$$J_{diff} = 11.00 \text{ GJ/SWU} \quad R = J_{th}/J_e = 0.083 \quad (\text{Eq. 5.1})$$

Process loss is 0.5%, according to (NRC, 1996)

For the ultracentrifuge process there are only three usable values, i.e. Kolb et al., Kistemaker, and Mortimer. The operational energy requirements are not included in any of these. We use the value of Kistemaker, 1975, which does include the energy costs of plant construction. Rotty, 1975 gives the operational costs of gas diffusion, and if we assume that these costs are twice as high for the ultracentrifuge process we get for the operating costs:

$$J_{operUC} = 1.76 \text{ GJ/SWU} \quad R = J_{th}/J_e = 21$$

Adding this value to the energy costs as given by Kistemaker, we find for the total specific energy cost of ultracentrifuge enrichment:

$$J_{UC} = 3.10 \text{ GJ/SWU} \quad R = J_{th}/J_e = 2.72 \quad (\text{Eq. 5.2})$$

Assuming, as explained above, a 30/70 world-wide ratio for the two processes, we find the total specific energy costs of enrichment to be

$$J_{enrich} = 5.47 \text{ GJ/SWU} \quad R = J_{th}/J_e = 0.51. \quad (\text{Eq. 5.3})$$

This value is, of course, an estimate, since the full energy costs for the ultracentrifuge process have not been worked out directly. But it is probably not too high because both Rotty et al. and Kistemaker assumed a 100% load factor for the plants.

Because the SWU is too small to be practical, we introduce a unit 1000 times as large, and call it the "Mg SWU". Using this enrichment unit and PJ for the energy unit, we have for the enrichment energy costs:

$$J_{enrich} = 0.0055 \text{ PJ/Mg SWU with } R \text{ as above.} \quad (\text{Eq. 5.4})$$

### **Fuel element fabrication:**

The enriched uraniumhexafluoride from the enrichment plant has to be converted to a ceramic solid, uraniumdioxide  $\text{UO}_2$ , before it can be used as fuel for the reactor. The uraniumdioxide tablets are packed in Zircalloy tubes (an alloy of zirconium with a few percents of other metals, e.g. tin). A number of this tubes, called fuel pins, are bundled to form fuel elements, which are placed in the reactor. The literature values of the energy requirements are listed in Table 9.

*# svl toegevoegd:*

Zircalloy is made by alloying extremely pure zirconium with a few percents of other metals, e.g. tin. The zirconium should be completely hafnium-free, because of the adverse effects of hafnium in a nuclear reactor, and technical grade zirconium always contains some hafnium. The purification of zirconium involves chlorination of the metal and its impurities, followed by distillation of the

chlorides formed in the chlorination process. Afterwards the purified zirconiumtetrachloride ( $ZrCl_4$ ) is converted back into the metal. The whole process may be very energy-intensive.

Table 9. Energy requirements for fuel element fabrication

reference	specific energy, $J_e$ + $J_{th}$ GJ/Mg U	R, $J_{th}/J_e$	remarks
Franklin et al. 1971	392	0.008	
USAEC 1972	310	0.62	probably reconversion only
Rombough & Koen 1974	6775	10.2	all processes included
Rotty et al. 1975	585	0.64	direct input only
Rotty et al. 1975	3791	2.50	direct +construct. + materiels
SRI 1975	572	0.64	direct only
Chapman 1975	288	0.66	
ERDA-76-1	3792	2.50	
Mortimer 1977	493 - 611	1.3 - 1.5	
Orita 1995	330	0.65	
WNA 2005	3792	2.50	

# *svl toegevoegd*:

The production of Zircalloy itself may be an underestimated factor in the nuclear fuel production process. For each Mg uranium in the fuel, 1-2 Mg Zircalloy is needed.

The figure of Rombough & Koen 1974 calculated their figures in a different way: they multiply the costs (in \$) by the energy intensity of the appropriate industrial sector. In this way all partial processes of the fuel fabrication process may be included.

The lower value from Rotty et al. 1975 may be explained either by neglecting the difference in energy intensity between Zircalloy and commercial zirconium, or by neglecting the high turnover rate of some equipment in the fuel fabrication plant. Rotty et al. assumed a 30 year plant life, whilst some parts have to be replaced after 5 and others after 15 years, according to Kistemaker 1975.

The figure of Rombough & Koen 1974 may be a more realistic figure than of Rotty et al. 1975, although the thermal/electric energy ratios (R) do not seem to be mutually compatible.

To be sure not to overestimate the specific energy requirements of the fuel fabrication process in this study, the value according to ERDA-76-1 is adopted:

$$J_{fabric} = J_e + J_{th} = 0.00379 \text{ PJ/Mg U, with } R = J_{th}/J_e = 2.50 \quad (\text{Eq. 6})$$

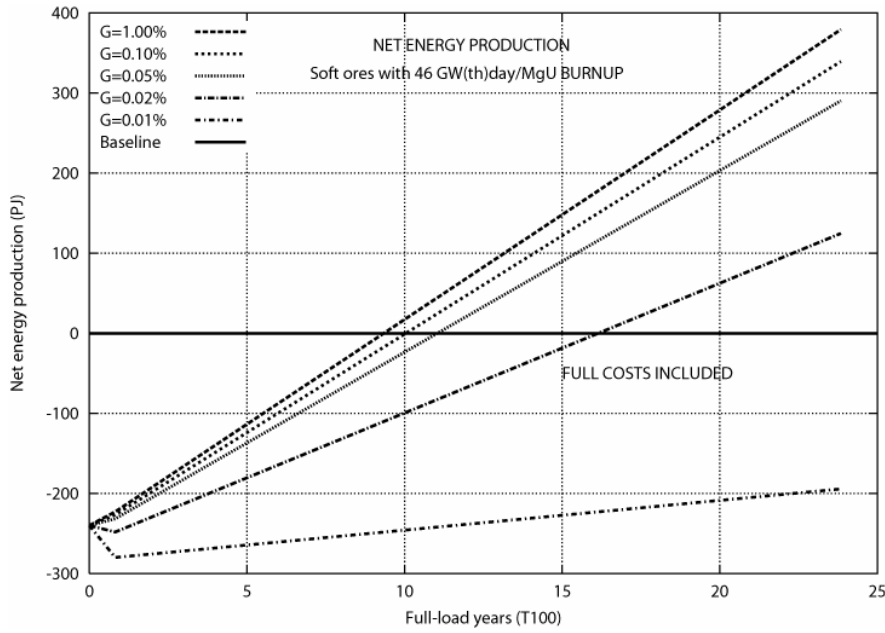
## Fuel availability

### The contribution of nuclear power to the world's energy supply

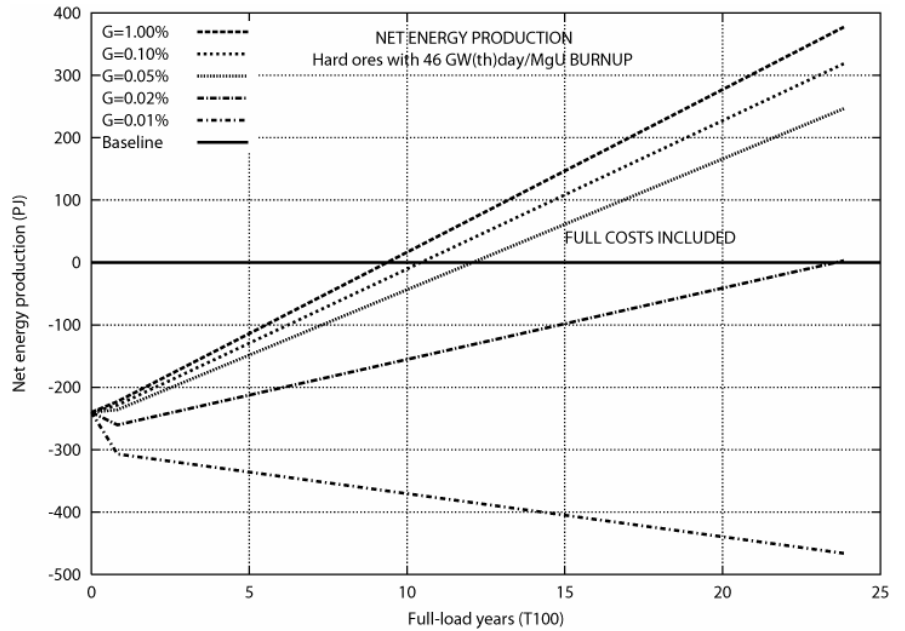
In 2000 the world energy use was about 400 EJ/a [BP 2001]. The electrical energy used was about 55 EJ (in 2004: 63 EJ). If one assumes that in the future this electrical energy will be produced by nuclear energy, it is important to ask how many years nuclear energy could supply this electricity. Naturally the demand will increase with time but it is relevant to ask how many years nuclear energy could satisfy the present demand. In order to answer this question we need to know how much uranium can be recovered (i.e. mined and milled), and how much (electrical) energy can be produced from it. The net electrical energy that a nuclear reactor can be expected to deliver in its

useful lifetime is shown in figures 8a, 8b 9a and 9b. The most favorable operating conditions, have been chosen (46 MW(th)day/MgU).

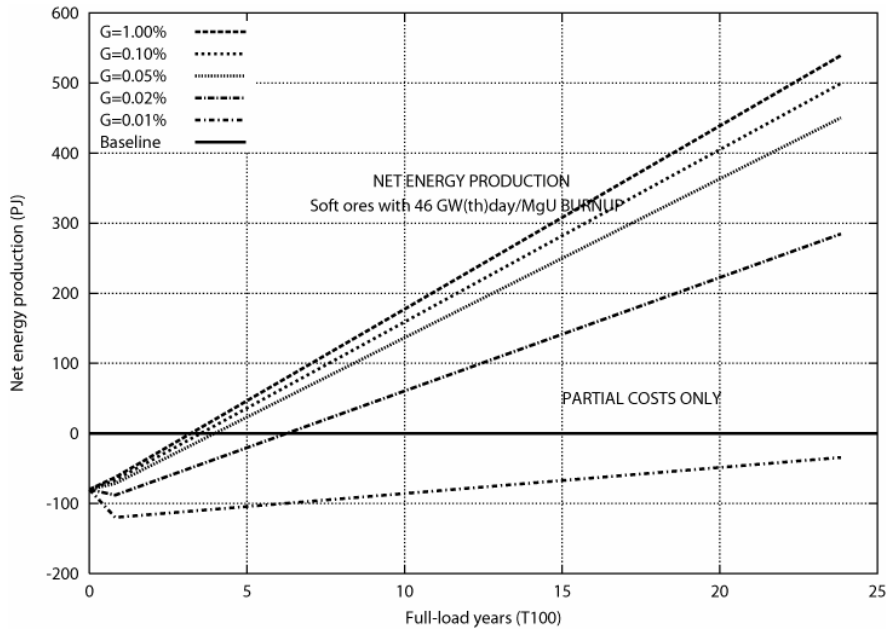
In the Figures 8a and 8b the costs are taken to be the total costs, i.e. including the full costs of dismantling the reactor, as well as all operating costs and debts. Figure 8a gives the energy delivered during the reactor's lifetime for different soft ore grades, and Figure 8b the same but for different hard ores grades. Figures 9a en 9b are the same but with neglect of the dismantling costs.



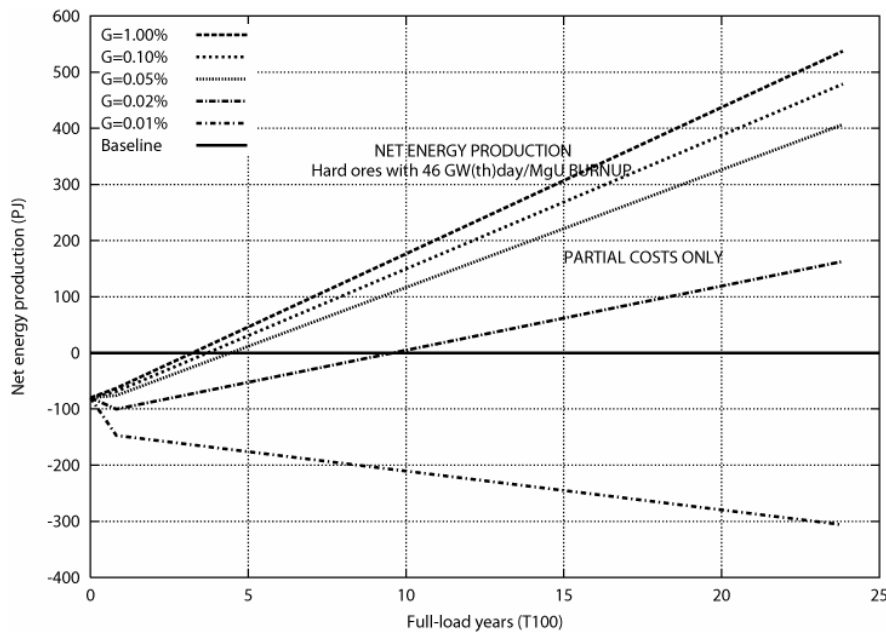
**Figure 8a.** The net energy production as a function of time of a reactor fed with uranium from soft ores, and operated with a burnup of 46GW(th)day/MgU. All of the costs are subtracted from the energy output, including the dismantling costs of the reactor.



**Figure 8b.** The net energy production of a reactor operated under the same conditions as in Figure 8a, except with hard ores as the source of the uranium.



**Figure 9a.** This figure, giving the net energy production from soft ores but with neglect of the dismantling costs.



**Figure 9b.** This figure is the same a Figure 9a, but for hard ores.

A more convenient picture is given in Figure 10, where the four curves shown are constructed by connecting the endpoints of the curves of figures 8 and 9, respectively in order of ore grade and dividing the resulting curve by the mass of uranium used in the life cycle of the reactor. As explained in Chapter 3 the useful lifetime is optimistically set at 24 full-load years. The curves in this figure all have the generic form of Eq. 2.3, i.e. a constant term and a term that follows the yield curve of mining and milling, (as in Eq. 2.2), since each corresponds to constant conditions with the ore grade the only variable.

$$B + \frac{A}{0.98 - 0.0723 * G * (\log G)^2} \tag{Eq. 2.3}$$

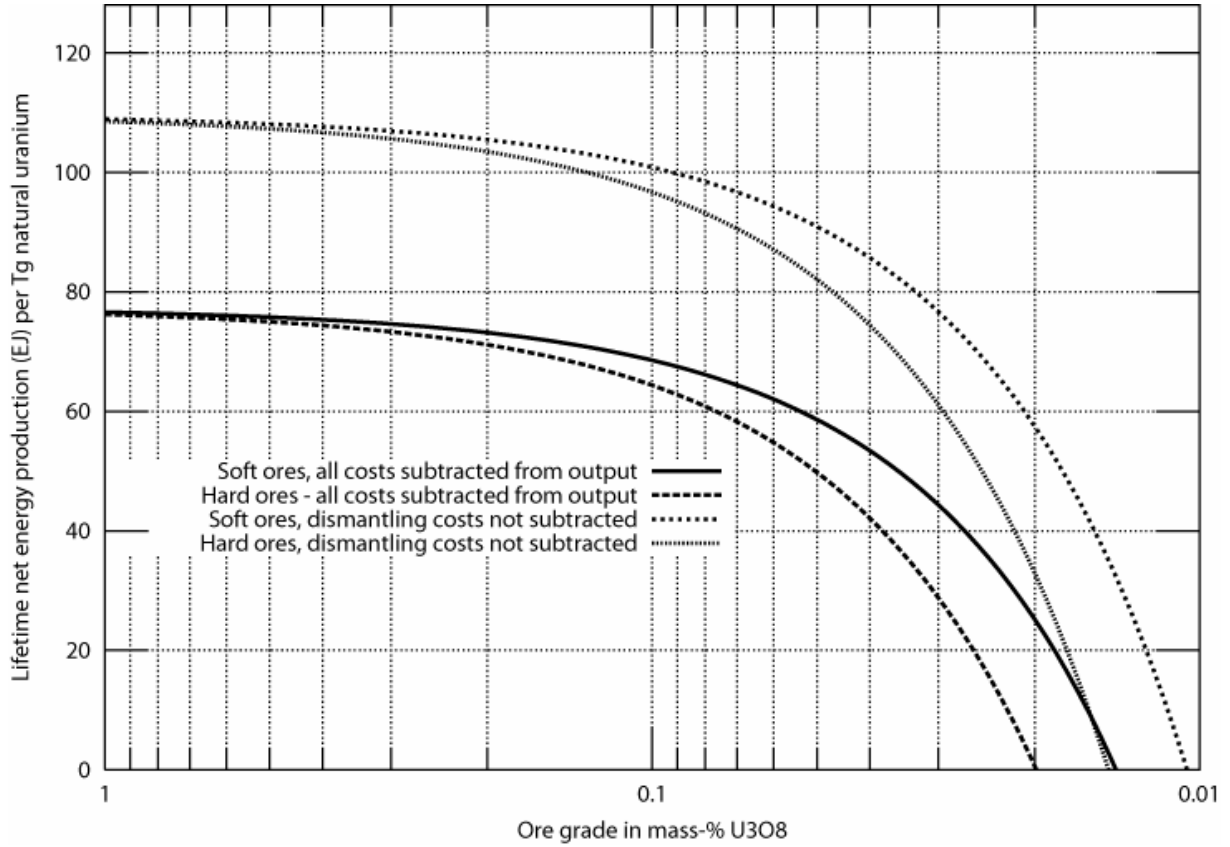
with the values of A and B in units of EJ per Tg uranium, as given in the table below .

	soft ores, all costs subtracted from output	hard ores, all costs subtracted from output	soft ores, costs of dismantling not subtracted	hard ores, costs of dismantling not subtracted
B	77.42	77.42	109.7	109.7
A	-0.8060	-1.185	-0.8060	-1.185

The two pairs of curves are shown in Figure 10. The first pair assumes that all (energy) costs are subtracted from the energy output, and the second assumes that all costs except for the dismantling of the reactor are subtracted from the total net energy production. These curves demonstrate that if society decides that at the end of the useful life reactors do not need to be completely dismantled and the debris carefully sequestered, the total amount of energy available from the existing uranium reserves would be greatly increased, and the time before the reserves are exhausted would be about 60% longer.

With Figure 10 one finds the total net energy that can be produced from the uranium of the given type (hard or soft) in an ore body by multiplying the value given by Eq. 2.3 (the ordinate of the appropriate curve), or reading from the graph for the ore grade, by the mass of uranium in the ore body.

The results are shown in Table 10, below, for all of the ores bodies reported in the literature. Most of the data for Table 10 comes from the World Nuclear Association (WNA 2003). Not all of the data from different sources quoted on the website are internally consistent. We have chosen, in case of doubt, the highest value quoted. The total available energy is shown for 46 GW(th)day/MgU (46GW) burnup for each ore body, region, or location, calculated as explained above. As already noted, the time to exhaustion is less than ten years, whether or not one takes the dismantling costs into account.



**Figure 10.** The total electrical energy that can be obtained from a given ore body is found by multiplying the ordinate of the desired curve (read off the graph or calculated from Eq. 2.3) at the applicable ore grade by the total mass of uranium in the ore body.

**Table 10**

Continent	Country, location or ore body	Ore grade in mass-% U <sub>3</sub> O <sub>8</sub> . ¶	Mass U in soft ores Tg	Mass U in hard ores Tg	Net electrical energy (EJ) delivered to the grid, with a burnup of 46 GW(th)day/MgU	
					Dismantling costs not subtracted	All costs subtracted from output
Australia	Olympic Dam <sup>1</sup>	0.05		0.490	40.2	24.4
	Olympic Dam <sup>1*</sup>	0.04		0.797	59.3	33.6
	Ranger <sup>1*</sup>	0.24	0.083		8.8	6.1
	Jabiluka <sup>2*</sup>	0.52	0.198		21.4	15.0
	Koongarra <sup>2</sup>	0.8	0.012		1.3	0.9
	Kintyre & Ben Lomond <sup>2</sup>	0.2-0.4	0.034		3.6	2.5
	Honeymoon and Yeelirrie <sup>2</sup>	0.15	0.050		5.2	3.6
	Westmoreland (Beverley) <sup>2</sup>	0.04-0.2	0.018		1.8	1.2
	Valhalla <sup>2*</sup>	0.144	0.035		3.6	2.5
rest of Australia <sup>2</sup>	0.12	0.046		4.7	3.2	
North and South America	Canada <sup>4,7</sup>	≥0.5	0.439		47.4	33.3
	U.S.A. <sup>5,7</sup>	0.05-0.4**	0.102		10.6	7.3
	Brazil <sup>5,7</sup>	0.05-0.4**	0.143		14.8	10.2
Africa	South Africa <sup>3,7</sup>	0.02		0.298	9.8	0.2
	Namibia <sup>6</sup>	0.035		0.213	14.6	7.8
Eurasia	Kazakhstan <sup>3,7</sup>	0.02-0.07	0.622		52.0	33.2
	Russian Federation <sup>3,7</sup>	0.05-0.4**	0.158		16.4	11.3
	Uzbekistan <sup>3,7</sup>	0.05-0.4**	0.093		9.6	6.6
Rest of the world***	total <sup>3</sup> (assumed soft)	0.05-0.4***	0.480		49.7	34.2
Total mass hard and soft reserves and available electrical energy			2.513 Tg	1.798 Tg	374.8 EJ	237.1 EJ
<b>Total mass of all reserves</b>			4.311 * Tg			
<b>Length of time that these reserves could provide the present global electrical energy demand of 55 EJ/yr</b>					6.8 yr	4.3 yr
<b>Total reserves according to Uranium 2003, plus time to exhaustion.</b>			3.537 Tg		5.8 yr	3.5 yr

¶ In the calculations, when a range of ores grade is given in the table, the geometric mean of the range is used for the total available energy.

\* includes inferred and indicated reserves. The world total without these would be about 3.5 Tg instead of the sum given in the table (4.311 Tg). According to OECD, NEA & IAEA, Uranium 2003, per 0101003, the world total is 3.537 Tg uranium

\*\* assumed sandstone, based on reference 5 (see below for reference).

\*\*\* figures not given in a specific reference; calculated by the authors from data of reference 3 (see below for reference), with the ore type and grade speculative (assumed to be average sandstone).

References numbered by superscripts in Table 10:

<sup>1</sup> Australia's uranium mines, UICemine 2003, [www.uic.com.au/emine.htm](http://www.uic.com.au/emine.htm).

<sup>2</sup> Australia's uranium deposits and prospective mines, UICpmine 2004, [www.uic.com.au/pmindex.htm](http://www.uic.com.au/pmindex.htm).

<sup>3</sup> Nuclear Assurance Corporation, 1982, personal communication.

<sup>4</sup> Canada's uranium production, WNA49, 2003, [www.world-nuclear.org/info/inf49.htm](http://www.world-nuclear.org/info/inf49.htm).

<sup>5</sup> Geology of uranium deposits, UIC 34, 2004, [www.uic.com.au/nip34.htm](http://www.uic.com.au/nip34.htm).

<sup>6</sup> Australia's uranium and who buys it, UIC34, 2004, [www.world-nuclear.org/info/inf48.htm](http://www.world-nuclear.org/info/inf48.htm).

<sup>7</sup> Supply of Uranium, [www.world-nuclear.org/info/inf75.htm](http://www.world-nuclear.org/info/inf75.htm), based on OECD, NEA & IAEA, Uranium 2003, per 0101003.

Even if the useful uranium resources were found to be much larger than now estimated, the maximal contribution to the electrical energy supply would only satisfy the global demand for several decades. The world would then be left with a heritage of astronomical quantities of

radioactive waste with no source of energy left to sequester it safely. On this ground alone, one can not speak of nuclear energy as a serious candidate for a future energy supply.

It should be noted that the present statistics of world energy supply, e.g. [BP Amoco, 1999] are based on the gross electricity production of nuclear power plants, without taking any account of the energy consumption of the rest of the nuclear fuel chain. We have corrected these overly optimistic figures in this study by specifically including these costs. The realistically calculated reserves, shown in Table 10, are remarkably lower than given by BP and others. At this point it is quite relevant to ask what the chances are that in the future, when the energy crunch is on, much more uranium than is now known and reported in Table 10 will be discovered.

### Discoveries of uranium deposits: what can we expect in the future?

Deposits of uranium (and other trace elements in the earth's crust) are formed only where several improbable circumstances occur together: there must be a source of the element, enough water to transport it, suitable subsurface conduits, suitable chemical conditions (oxidizing or reducing), complexing agents to carry the element in solution and other agents to finally precipitate the mineral. If any of these are missing, no uranium deposit can be formed. The probability that an ore

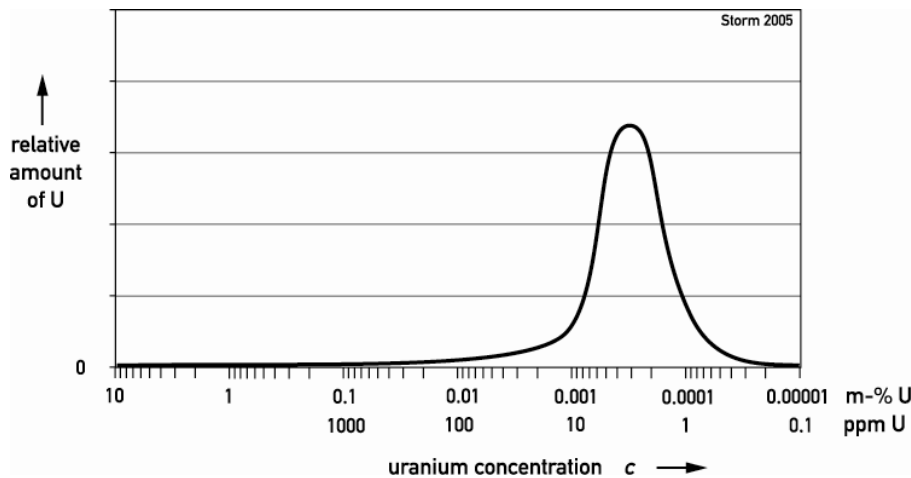


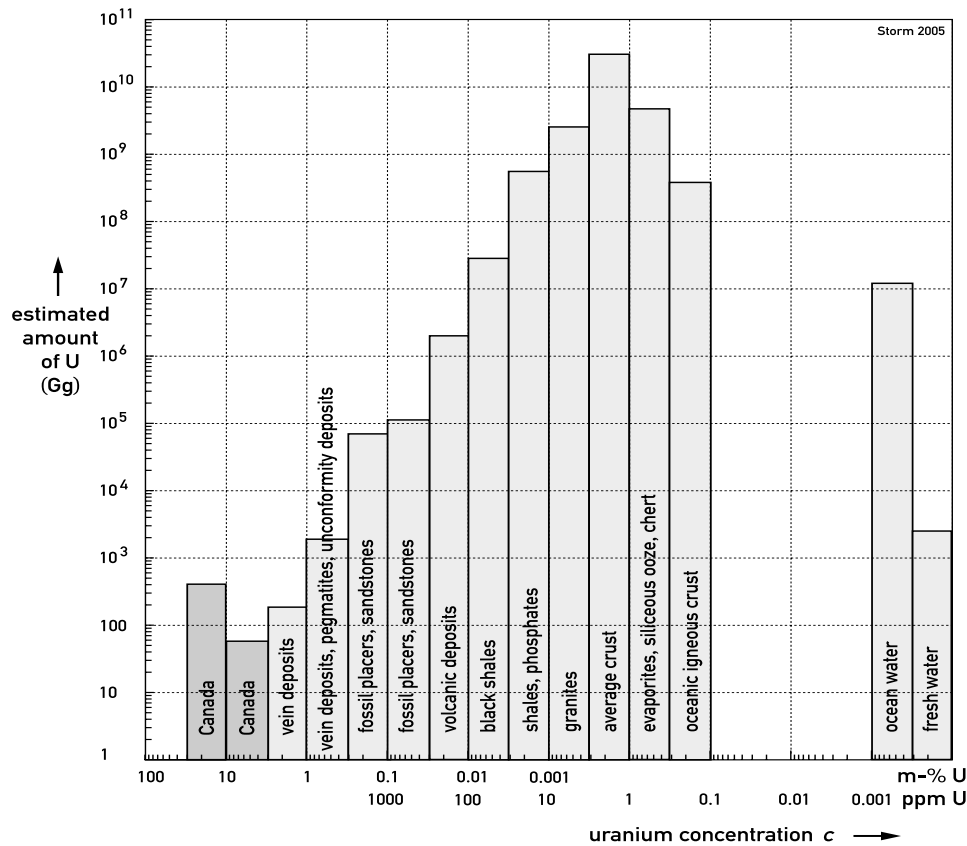
Figure 11. Hypothetical log-normal distribution, assuming that this distribution applies to uranium, Source: Deffeyes & MacGregor 2000.

deposit will be formed at a given site is determined by multiplying the probabilities of each essential ingredient being present. When probabilities add, the central-limit theorem of statistics holds that the final distribution approaches the bell-shaped normal distribution, as in Figure 11. Since the multiplication of probabilities corresponds to adding on a logarithmic scale, it is not surprising to see the

familiar bell-shaped curve appear when elemental abundances is plotted on a logarithmic scale.

Inventarising and aggregating all known uranium deposits in the world, Deffeyes and MacGregor, 1980 found a global abundance for uranium as shown in Figure 12. They considered the odds small that a major category had been systematically underestimated. According to these authors this diagram appears to support the hypothesis of a single log-normal distribution as illustrated in Figure 10, although they point out that no rigorous statistical basis exists for expecting a log-normal distribution of uranium in the earth's crust. Other possible distributions are bimodal, with two peaks, or a multi-modal one with several peaks in the distribution curve. It is not possible to determine the distribution directly. Uranium exhibits as complex range of geochemical behaviour and a wide variety of economically significant deposits.

The large resources at rich ore grades ( $\geq 10\%$   $U_3O_8$ ) in Figure 12 may suggest a bimodal distribution, but there is no other evidence pointing toward this hypothesis. The high value may represent a single, isolated, peak.



**Figure 12.** Distribution of uranium in the major geological reservoirs, according to Deffeyes & MacGregor 1980. The height of each bar shows the estimated total amount of uranium at the given concentration. It should be noted that both horizontal and vertical scales are logarithmic in this diagram, while the vertical scale in Figure 1 is linear. The logarithmic scale gives one a distorted picture of the relative abundances, by giving the impression that the content of different reservoirs are comparable, whereas the amounts differ by many orders of magnitude. Note that the rich deposits in Canada (more than 3% U) were missing in the original publication of Deffeyes & MacGregor, and have been added here (dark shaded bars).

The crustal distribution of uranium has profound consequences for the uranium supply in the future. Based on these data – which are considered representative by, among others, the authors of MIT 2003 – it may be concluded that the chances to discover new large rich ore deposits, like the deposits in Canada, are dim.

Some authors, however, e.g. WNA-75 2004, Wikdahl 2004, MacDonald 2001 and MacDonald 2003 expect new large and rich discoveries when intensive exploration is resumed. Their views are based on traditional economic considerations: more exploration will automatically yield more known resources. This reasoning is quite analogous to that one find in the oil industry. Although it has been clear for many years that the world's oil reserves are running rapidly running out, there are still those who insist that the oil reserves are inexhaustible and immense new oil discoveries are just around the corner. This reasoning flies in the face of the simple fact that reserves of anything have to be there in the first place before they can be discovered, i.e. this is a geological, not an economic, question.

Exploration for uranium deposits was very intensive in large parts of the world during the 1960's and 1970's. Almost all presently mined ore bodies were already known or were discovered in that period. From the past 30 years, no reports are known to us of rich and large discoveries have

appeared in the open literature. On these grounds discovery of new rich and large ore bodies is unlikely, but, of course, can't be excluded.

Relatively few data are available on uranium resources in China. It may be that few deposits have been found. or it may be that the exploration for uranium in the past was not very intensive, or, of course, that data are being withheld from the open literature.

## Conclusion

Large-scale uranium mining in the future, when the deposits in the Athabasca basin are beginning to run out, will most likely have to shift to leaner ores than mined presently. Consequently specific energy consumption of mining and milling (per kilogram uranium leaving the mill) will rise significantly. At ore grades of 0.02% and lower, the specific energy consumption rapidly becomes prohibitive.

### Other possible sources of nuclear fuels and/or energy

#### *Nuclear energy from the uranium in the earth's crust*

The uranium content of granitic rocks varies, but is of the order of 0.0003%. One of the standard assumptions concerning nuclear energy is that if it came down to it, this uranium could be used in nuclear power generation. It is easy to see that this is completely impossible. Taking the yield of the last diamond in Figure 6 ( $G=0.0003\%$  and  $Y=0.2$ ), a very optimistic assumption (Huwyler, Rybach and Taube, 1975) and using the value of  $c$  for hard ores in Table 4 and the total mass of the natural uranium needed (see page 2 of the appendix) one finds that  $4.952 \times 10^3 \times 0.654 / (0.0003 \times 0.2) = 54$  EJ would be needed to mine and extract the uranium needed for 24 years full-load operation of a 1 GWe plant operating with a uranium burnup of 46GW(th)day/MgU. The total electrical energy delivered to the net by a 1GWe nuclear power plant in this period is 0.751 EJ, or **seventy times less than the energy required for mining and milling alone**. As pointed out above, the lessons of practice in the chemical industry indicate that the actual energy needed for extraction would be much more than was assumed by Huwyler et al. for such a lean ore, so that, if it were to be attempted, the recovery of uranium from granitic rocks would be much higher than the theoretical calculations predict. There is thus not the slightest possibility that the uranium in the earth's crust could deliver energy in nuclear reactors.

#### *Nuclear energy from the uranium in the oceans.*

The same conclusion can be reached in this case, although the argumentation is more complicated, and the certainty less. Seawater contains uranium, mainly as uranyltricarboxylate ions  $[\text{UO}_2(\text{CO}_3)_3]^{4-(\text{aq})}$ , with an average concentration of 3.34 mg uranium per cubic metre seawater. As the total volume of seawater is about 1.37 million  $\text{km}^3 (=1.37 \times 10^{15} \text{ m}^3)$ , the total amount of uranium in the oceans is about 4.5 Tg. This huge resource is assured. The studies discussed below, however, show that it is not an usable *energy* source?

Technically it is possible to extract uranium from seawater. Probably the most promising method is based on adsorption of the dissolved uranium-complex ions on hydrous titaniumhydroxide gel, followed by elution with a 1.5 M ammoniumcarbonate solution. In subsequent chemical processes the eluant is treated to regain the ammoniumcarbonate. The uranium solution is further concentrated and converted into yellow cake.

In the past numerous studies have been published on this subject, some of which are detailed, e.g. ORNL 1974, Burnham et al. 1974, and Mortimer 1977, while many others are confined to only a part of the process, e.g. Nobukawa et al. 1994, Burk 1989, Saito 1980 and Koske 1979. Brin 1975 gives a concise bibliography.

Because of the low concentration of uranium, very large volumes of seawater would have to be processed in order to extract useful quantities of uranium. With an expected extraction efficiency of about 0.3, 1 gram of uranium could be extracted from 1000 m<sup>3</sup> seawater. There are several unsolved problems in the extraction process, such as large losses of titaniumhydroxide (about 15 kg titanium per kg uranium extracted, ORNL 1974), and pollution of the adsorber beds by organic materials of the sea, however. In any case the process needs enormous pumps that would consume large amounts of electricity. Regeneration of the eluant by steam stripping is also an energy-intensive process.

In the table below the estimated extraction costs of uranium are summarized. In the second column of the table the energy expenditure per Mg uranium is given. In the third and fourth columns the dollar cost (per Mg extracted uranium) is given, normalized to the dollar value in the year of the publication of the given study, and for the year 2000, respectively. The costs given in these two columns is derived from the dollar cost using the dollar/GNP ratio,  $e$ , for industrial activity.

*Estimates of costs and energy consumption of extracting uranium from seawater.*

- \* only pumping energy, deduced from data in publication
- \*\* pumping costs only for a pump-fed plant with a pump head of 20 m.
- \*\*\* pumps in tidal plant
- \*\*\*\* deduced from data in publication; only pumps in first stage, pump head 0.5 m

reference	energy costs TJ/Mg(U)	dollar cost \$(yr)/kg U	dollar cost (\$2000)/kg U
Burnham et al, 1974	26 *		
ORNL, 1974	90 **	>> 796 \$(1974)	>> 2766
	1.2 ***	>> 796 \$(1974)	
Brin, 1975	390 *	91-2,600 \$(1975)	289-8,280
Mortimer, 1977	20-600	260-780 \$(1977)	728-2,210
		from ore 21-78, \$(1977)	from ore 60-221
Koske, 1980	5-10 ****		
INFCE 1,1980	-	7,000-9,000 \$(1978)	18,400-23,700

Because extraction plants would need to be very large, with dimensions in kilometers, there are only a few estuaries in the world that could accomodate a tidal plant. For this type of plant, sufficient refreshing of the seawater is one of the problems to be solved. The relatively low pumping costs in the case of an estuary would be offset by the construction and maintenance costs of huge civil works (dams, etc.).

The most extensive study reported, ORNL 1974, assumes the use of pumps, since the authors concluded that an estuary solution is not suitable. The energy costs taken into account include plant construction, chemicals, and pumping and maintenance costs. A rough estimate of the costs is about 100 TJ/Mg. This is a low estimate: the authors emphasize that the values they found are based on very optimistic assumptions, so the real values will probably be several times higher.

The dollar costs shown in the bottom row of columns three and four of the table were derived in another large effort, INFCE-1 1980, of the US Department of Energy in which the dollar costs of extraction were estimated, including plant construction, chemicals and operation and maintenance. This study arrived at dollar costs enormously higher than the price of mined uranium at that time. Using their cost estimates, of 18,400-23,700 \$(2000)/kg, converted to energy units, one finds a somewhat higher energy cost of uranium extracted from the sea than found in the ORNL 1974 study: 195-250 TJ/Mg. But the since the latter authors, as remarked above, emphasize that their

estimate is certainly too low, there is no real disagreement between the different studies, although theirs is certainly an underestimate, because the chemical industry is more energy-intensive than the average economic activity. There is general agreement between these studies and that of Mortimer 1977.

The lowest found value of the specific energy consumption  $J_{sea} = 100 \text{ TJ/Mg}$  is about the same as that of obtaining uranium from low grade hard ores ( $G < 0.01\% \text{ U}_3\text{O}_8$ ). As is clear from the energy costs calculations in chapter 5, the energy produced by a nuclear energy plant with fueled with uranium from such lean ores is always less than the total energy costs of operation of the plant. The upshot of existing studies is that, energetically, uranium from seawater can't be considered an option for the global energy supply.

This is confirmed by the conclusion of INFCE1 1980.

“Therefore it would be unrealistic to expect uranium from seawater to contribute significant amounts of the world's uranium demand for thermal reactors on an acceptable time scale”, and it is also not in disagreement with the more general conclusion of Mortimer 1977 that the energy consumption of most of the extraction processes that the author had examined approximately equals the energy content of the uranium.

We have reviewed up to this point in this chapter of providing electrical energy over the long term by the use of thermal neutron reactors in a once-through fuel cycle. We started with an evaluation of the energy content of the presently known reserves and proceeded toward less and less probable future discoveries and exploitation of at present unknown reserves. The overall picture is that it is feasible to provide (electrical) energy with nuclear reactors for at most several decades, although the possibility cannot be excluded that unexpected large discoveries in the future will radically change this picture. There are several other possible sources of nuclear energy. We review briefly below two of these other sources of energy which in principle could be utilized by releasing for practical use the immense amount of energy stored in the nuclei of atoms.

#### *The fast-neutron breeder*

The first, quite popular up to around 1970, was the *fast-neutron breeder reactor*. Although in the four countries mentioned there (U.K., U.S.A., France, and Germany), the breeder programs turned into failures. We are not certain of the status of the Japanese breeder-reactor effort. There is an operating fast-neutron reactor in Russia that has been in service for more than 21 years, but as of June 2001 it had only achieved a full-load service time of 14.2 years, replete with accidents. The record of accidents and down-time is such that the program would probably have been discontinued in the other countries mentioned. More importantly, there is no mention in the relevant literature (Saraev 1998) of the reactor ever having bred fuel.

One does still hear, however, occasional references, eg. Charpak and Garwin, 1997, to the "unlimited" energy that this type of reactor could provide. It is worthwhile noting that in the 1950's the breeder was presented, with a great deal of publicity, as the perfect solution to the "energy problem", but that by the 1970's such sounds were more muted. The breeder, as originally conceived, would be able to breed enough fuel in seven years to fuel a new reactor. This re-fuelling period "grew" longer and longer in the following decades. At the Pugwash conference in Munich in 1977, Prof. Francis Perrin, who headed the French Atomic Authority from 1951 to 1971 stated that the question at that time was if a breeder could *ever* refuel itself [Perrin 1977].

It is, for the viability of nuclear energy, crucial to prove that a fast-neutron reactor system can breed and prepare for use, more fuel than it uses. We have shown above that the amount of  $^{235}\text{U}$  that can be mined and milled profitably (in energy terms) is simply too small to make nuclear energy into a long-term solution of energy production, unless means are found, such as the breeder reactor, to make the much larger energy content of the  $^{238}\text{U}$  available for energy production. We refrain from making any prediction as to the viability of the fast-neutron breeder in the long term.

#### *Energy from nuclear fusion*

This possible source of energy from the nucleus has been the object of considerable, generally fanciful, interest, and has received immense subsidies. The attractiveness of this option is that ordinary water contains deuterium (in the form of "heavy" water). It constitutes only 0.015% of water, but in the totality of the oceans there is an unimaginably large amount. In recent years it has become doubtful, however, whether a self-sustaining plasma is realizable. But what really seals the fate of fusion energy is that the D-D reaction, for fundamental physical reasons, cannot provide a net production of energy (above that needed to maintain a stable plasma, plus all of the other energy cost factors) [Pease, 1997] in a plasma with presently conceivable densities and temperatures. There are losses that cannot, even theoretically, be eliminated. The only reaction that could theoretically achieve net energy production is the deuterium-tritium (D-T) reaction. In the enthusiastic promotion publicity concerning fusion energy this reaction was quietly substituted for the deuterium-deuteron reaction, without drawing attention to the fact that tritium, the heaviest isotope of hydrogen, only exists in nature in the minute amounts formed by cosmic rays. Gone was the original euphoria about an unlimited fuel supply (although there are those who speak of "importing" tritium from the moon). The silent assumption is made that the tritium would be produced by bombarding a blanket of lithium with the large excess of high energy neutrons emanating from the tritium-deuteron reaction. A viable breeder process would have to be developed, of course. Experience with breeder systems is not encouraging. We need not discuss this possibility further, since the expected arrival date of a net **electrical** energy producing fusion system has long since disappeared over the time horizon of even the most technologically optimistic.

(iv) *Exotic nuclear reactors.* A half a century of attempts to design and then build a functioning fast-neutron breeder reactor or to show the feasibility of producing energy from nuclear fusion, both at gigantic cost, have led to disappointing results. Recently a new type of nuclear reactor has been conceived that differs fundamentally from all previous designs in that it is not "critical", i.e. it only amplifies the energy that is applied to it. This has the obvious charm that it cannot explode - cutting off the flow of input energy stops its working, immediately. The design differs in almost every detail from the present day reactor. It uses thorium, not uranium, of which there is more in the earth's crust. But its input energy must be supplied by a particle accelerator. If this can be done reliably on the scale required is an unanswered question. Since private financing is out of the question, it is also uncertain if any government would be willing to take the risks involved in financing such a gigantic project.