

Nuclear power - the energy balance

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Note

In this document the references are coded by Q-numbers (e.g. Q6). Each reference has a unique number in this coding system, which is consistently used throughout all publications by the author. In the list at the back of the document the references are sorted by Q-number. The resulting sequence is not necessarily the same order in which the references appear in the text.

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Part D

Uranium

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D1 World known recoverable uranium resources

The world known recoverable uranium resources, as given by the Red Book 2006 [Q90], are listed in Table D.1. The total amount of uranium resources jumped from about 3.6 Tg in 2005 to 4.7 Tg in 2006 (1 Tg = 1 teragram = 10^{12} gram = one million tonnes). The increase from 3.6 to 4.7 Tg is due almost completely to the inclusion of the higher price category of 80-130 US\$/kg U. This category comprises uranium ores of lower quality: lower ore grades, greater depth, longer transport distances and/or harder to mine.

Table D.1

World known recoverable resources of uranium. Reasonably Assured Resources (RAR) plus Estimated Additional Resources (EAR)-category 1, to 80 US\$/kgU for the years 2003-2005; Reasonably Assured Resources plus Inferred Resources (IR) to 130 US\$/kgU for the year 2006.

	2003		2005 *		2005 **		2006 ***	
	< 80 US\$/kg U		< 80 US\$/kg U		< 80 US\$/kg U		< 130 US\$/kg U	
	Gg U	%	Gg U	%	Gg U	%	Gg U	%
Australia	863	28	989	28	1074	30	1143	24
Kazakhstan	472	15	622	18	622	17	816	17
Canada	437	14	439	12	439	12	444	9
South Africa	298	10	298	8	298	8	341	7
Namibia	235	8	213	6	213	6	282	6
Brazil	197	6	143	4	143	4	279	6
Niger							225	5
Russian Federation	131	4	158	4	158	4	172	4
USA	104	3	102	3	102	3	342	7
Uzbekistan	103	3	94	3	93	3	116	2
Ukraine							90	2
Jordan							79	2
India							67	1
China							60	1
Other	267	9	480	14	480	13	287	6
Total world	3107	100	3537	100	3622	100	4743	100

1 Gg = 1 gigagram = 10^9 gram = 1000 metric tonnes.

Sources:

* WNA-75 2003 and WNA-48 2003

** WNA-75 2005 [Q85] and WNA-48 2005 [Q210], from update February 2006.

*** OECD/NEA, Red Book 2006 [Q90].

Virtually no new discoveries of uranium deposits have been added to the listed resources since 2005. Very few new resources, if any, are discovered during the last decades anyway. The increase of uranium resources during the past years is almost completely due to higher uranium prices and consequently reclassification of previously known economically unrecoverable resources into 'recoverable' resources.

The economic classification of uranium resources has no direct relationship with the thermodynamic quality of the uranium resources. It is not possible to assess the thermodynamic quality of the uranium resources from the information in the Red Book. The nuclear industry, through IAEA, OECD/NEA, WNA and other institutions, seems to avoid any discussion on ore quality and ore grade. In Part D2 the ore grade distribution of the known uranium resources is addressed.

Table D.1 has been compiled on economic criteria, not primarily on physical criteria. In this system several categories of uranium resources are distinguished, according to the production costs and assurance of the resources. For a full description of that classification we refer to the Red Book 2006 [Q90].

Table D.1 includes the following categories: *Reasonably Assured Resources* (RAR) plus *Estimated Additional Resources* (EAR)-category 1, to 80 US\$/kg U for the years 2003-2005; for the year 2006 *Reasonably Assured Resources* plus *Inferred Resources* (IR) to 130 US\$/kg U. In 2006 a modified classification system has been introduced. Together the categories RAR + IR are called the *Identified Resources*.

Other uranium resources

The Red Book mentions also *Undiscovered Resources* (*Prognosticated* and *Speculative Resources*) with a total mass of 10 Tg.

As the name implies, no assurance exists with regard to the Undiscovered Resources. Unclear is how these estimates came about and no clues on the ore quality have been published.

The sum of the Identified and Undiscovered Resources is 14.7 Tg. This number seems to be used without reservations in publications of the nuclear industry regarding the nuclear future, as if these resources are assured and as if the only uncertainty is the price of uranium on the global market (see for instance Omoto 2007 [Q359]). The IAEA states that these so-called conventional uranium resources will last for about 270 years at the current consumption rate of 68 Gg/a.

In addition to the Undiscovered Resources, the IAEA states that beyond the conventional uranium resources abundant uranium (22 Tg) will become available in phosphates and ultimately in seawater: 4000 Tg.

The conventional resources plus the phosphates would last for 675 years at the current consumption rate. With fast reactors these resources would last for 8000 - 16000 years. How valid are these assertions?

In Part D9 we will return to the recovery of uranium from phosphates and in Part D10 to uranium from seawater.

D2 World known recoverable uranium resources, by grade

Figure D.1 presents the world known uranium reserves of 2006, as listed in the last column of Table D.1, as a function of the ore grade. The ores are roughly classified into two groups: soft ores and hard ores, the former being relatively easy to process, the latter hard, consuming more energy and materials than the soft ores. Lower grade ores tend to be harder. Figure D.1 shows the largest resources to be present in rocks with low ore grades, a common geologic feature of metal ores. The low peak at the highest grades represent a number of deposits in Canada.

The distribution of the world uranium resources and the classification into hard and soft ores are based on a number of sources, listed in the caption of Figure D.1 and in Table D.2. Figure D.1 is based on Tables D.2 and D.3 below.

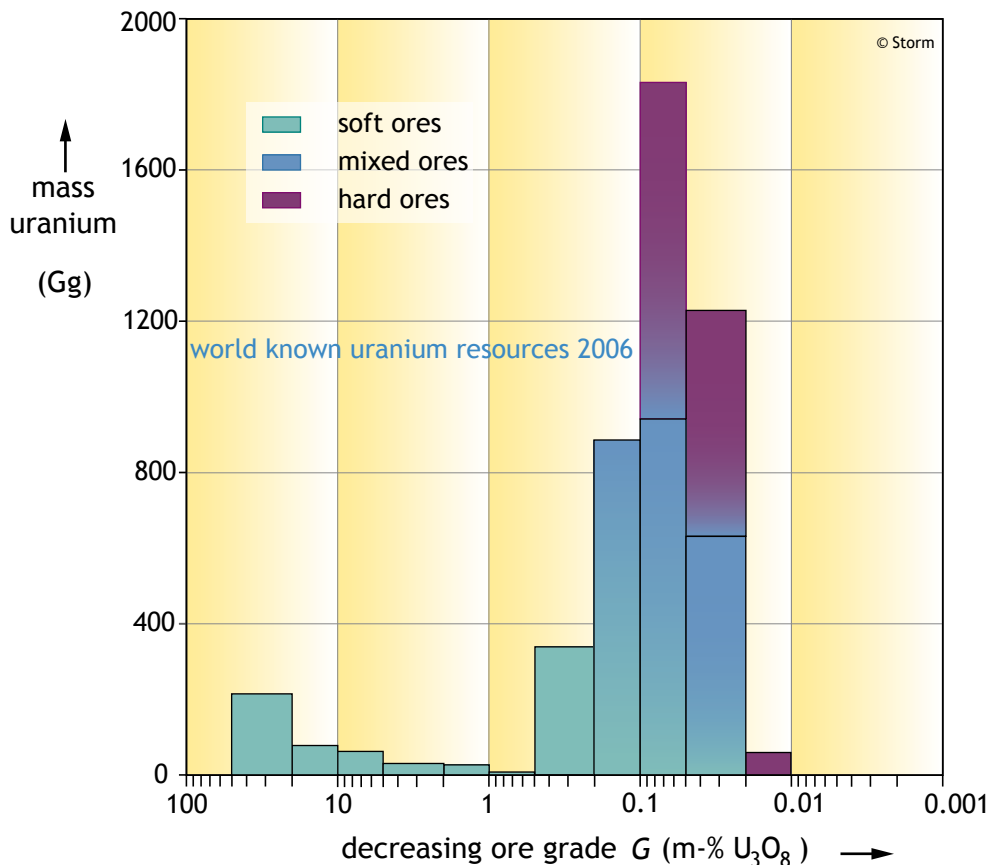


Figure D.1

The distribution of the known uranium resources as function of the ore grade. The horizontal axis represents the uranium ore grade on a logarithmic scale. On the vertical axis the size of of a given resource is given, on a linear scale. The amount of uranium present in deposits of a given grade interval is represented by the height of each bar, not by its area. The different colors indicate different qualities of the ores. Soft ores (greenish) are the easiest to process, hard ores (purple) the hardest to process.

Sources: [Q53], [Q85], [Q86], [Q87], [Q90], [Q210], [Q211], [Q212], [Q213], [Q314] and [Q324].

Table D.2

Survey of the world known uranium resources by ore grade. This table includes resources classified as RAR + EAR1 (= IR in 2006) to 80-130 USD/kg U.

u = underground mining

isl = in situ leaching

h = hard ore (the other are classified as soft ores)

• = remotely controlled

mine and country	U ₃ O ₈	G	G geom	m(U)				ore type
	Mg	%	%	Mg		ref		
• Australia								
Olympic Dam proven reserves	71000	0.06		60208	u •	Q211	h	granite-breccia
OD probable reserves	321000	0.05		272208	u •	Q211	h	
OD measured resources	325000	0.05		275600	u •	Q211	h	
OD indicated resources	568000	0.04		481664	u •	Q211	h	
OD inferred resources	522000	0.03		442656	u •	Q211	h	
sum Olympic Dam	1807000	0.04		1532336				
Ranger stockpile (= A)	14716	0.15		12479		Q211		unconformity
proved & probable reserves (=B)	29741	0.23		25220		Q211		
total reserves = A + B	44457	0.20		37700		Q211		
measured+indicated resourc.(=C)	20133	0.14		17073		Q211		
inferred resources (=D)	22454	0.14		19041		Q211		
total resources = C+D	42587	0.14		36114		Q211		
sum Ranger	87044			73813				
sum Olympic Dam + Ranger	1894044			1606149				
Jabiluka reserves	67000	0.52		56816	u	Q213		unconformity
Jabiluka meas+indic resources	21000	0.39		17808	u	Q213		
Jabiluka inferred resources	75000	0.48		63600	u	Q213		
sum Jabiluka	163000			138244				
Kintyre, reserves+resources	36000	0.15-0.4	0.28	30528		Q213		unconformity
Honeymoon, indic resources	2900	0.24		2459	isl	Q213		sandstone
Billaroo West (Gould Dam) indicated reserves	2500	0.045		2120		Q213		sandstone
Koongarra reserves	14540	0.8		12330		Q213		unconformity
Yeelirrie indicated reserves	52500	0.15		44520		Q213		surficial: calcrete
Beverley	21000	0.18		17808	isl	Q213		
Ben Lomond, resources	4760	0.25		4036	o/u	Q213	h	volcanic
Maureen, resources	2940	0.123		2493		Q213	h	volcanic
		0.09-						
Mayingee, resources	9500	0.12		8056		Q213		sandstone
Oobagooma, inferred	10700	0.1-0.15		9074		Q213		sandstone
Valhalla, indicated	16900	0.077		14331		Q213	h ?	tuff & shale
Valhalla, inferred resources	9000	0.077		7632		Q213	h ?	tuff & shale
Angela, resources	11250	0.13		9540		Q213		sandstone

Lake Way resources	4600	0.054		3901	Q213	sediment calcrete
Centipede, resources	4400	0.063		3731	Q213	calcrete
Curnamona					Q213	(no values given)
Prominent Hill, inferred re- sources	9900	0.0120	0.0103-	8395	Q213	h as Olympic Dam?
Mulga Rock, resources	15000	0.13		12720	Q213	lignite/sediment
Mt Fitch, resources	4050	0.046		3434	Q213	
Bigly, resources	5200	0.14		4410	Q213	sandstone
Nolans Bore, resources	3977	0.02		3372	Q213	
Lake Maitland, resources	4759	0.05		4036	Q213	calcrete
Lake Maitland, resources	5941	0.03		5038	Q213	calcrete
Thatchers Soak, resources	4100	0.03		3477	Q213	
Mt Gee, inferred resources	33000	0.073		27984	Q213	
Crocker Well, resources	8576	0.048		7272	Q213	h igneous
Skal, resorces	5000	0.119		4240	Q213	
Andersons Lode, inferred resourc	6500	0.143		5512	Q213	
		0.04-				
Westmoreland, inferredresources	22500	0.15	0.0774	19080	Q213	
sum other deposits Q213	657993			557978		
total Australia Q211+Q213	2552037			2164127		
Australia Identified Resources <130 USD/kgU not included in Red Book				1143000	Q90	
sum Ol.Dam 1+2+3+4 + Ranger +Beverley				1021127		
sum Ol.Dam 1+2+3+4 + Ranger +Beverley +Jabiluka				1181301		
				1143000		
Australia 1		0.06		60208	u	
Australia 2		0.05		272208	u	
Australia 3		0.05		275600	u	
Australia 4		0.04		305139	u	
Australia 5		0.23		25220		
Australia 6		0.14		48593		
Australia 7		0.18		17808	isl	
Australia 8		0.48		138224	u	
sum Australia				1143000		
• Canada						
Key Lake proven reserves	300	0.53		254	Q212	unconformity
Rabbit Lake proven reserves	4800	1.20		4070	u Q212	
Rabbit Lake indicated resources	2800	1.62		2374	u Q212	
Cluff Lake reserves	2130	2.5		1806	Q212	
McClellan Lake Sue, reserves	16650	1.8		14119	u Q212	
McClellan Lake McClellan	5850	2.1		4961	u Q212	
McArthur River proven reserves	152000	26.56		128896	u • Q212	
probable reserves	39000	19.06		33072	u • Q212	
measured & indicated resources	7900	9.42		6699	u • Q212	
inferred resources	48000	9.51		40704	Q212	
sum McArthur River	246900			209371		

Cigar Lake proven reserves	102860	20,67	87225	u •	Q212	
probable reserves	2400	4,41	2035	u •	Q212	
inferred resources	53600	16.92	45453	u •	Q212	
sum Cigar Lake	158860		134713			
Midwest proven+probable						
reserves	18900	5.47	16027	u	Q212	
Dawn Lake indicated resources	5800	1.69	4918		Q212	
Elliot Lake		0.087	29000	Q53	h	granite-breccia
<u>total Canada Q212</u>	<u>462990</u>		<u>421616</u>			

Canada, Identified Resources <130 USD/kgU			444000	Q90	
additional high-grade			51384		
additional high-grade			22384		

Canada 1		>20	216121	
		19.99-		
Canada 2		10	78525	
Canada 3		9.99-5	63430	
Canada 4		4.99-2	8802	
Canada 5		1.99-1	25482	
Canada 6		0.99-0.5	254	
Canada 7		0.087	29000	
Canada 8	asumed	2	22384	
sum Canada 1-8			444000	

• **Kazakhstan**

Akdal, ind.rec		0.057	11756	Q324
inf		0.063	6214	Q324
sum				isl Q324
Inkay, prov.res		0.059	13615	Q324
prob		0.051	30385	Q324
ind		0.051	5462	Q324
inf		0.042	103077	Q324
				isl Q324
South Inkay, inf resc		0.043	14068	isl Q324
North Kharasan, ind.resc		0.2	5301	Q324
inf		0.095	29051	Q324
				Q324
Mundukuk		0.035	49000	isl Q324
MUyunkum, prov res		0.06	3140	Q324
prob		0.06	11319	Q324
inf resc		0.06	1210	Q324
				isl Q324
Tortkuduk, prob res		0.06	14202	Q324
inf resc		0.06	10937	Q324
				isl Q324
		0.10-		
Zarechnoye, prov res		0.11	14500	Q324
		0.10-		
prob		0.11	4500	Q324
				isl Q324

Korsan		?	60000	isl	Q324
total Kazakhstan			387737		
Kazakhstan <80			607000		Q90
Kazakhstan additional <130			209000		Q90
		0.042-			
Kazakhstan 1		0.086	0.060	387737	
		0.035-			
Kazakhstan 2		0.06	0.046	219263	
		0.02-			
Kazakhstan 3		0.05	0.032	209000	
	sum			816000	
• Namibia					
Rössing, stockpiled reserves		0.018		1033	Q324
proven		0.032		3222	Q324
probable		0.028		34700	Q324
measured resources		0.042		4410	Q324
indicated		0.025		66678	Q324
inferred		0.025		47471	Q324
	sum Rössing			157514	Q324
Trekkopje measured resources		0.011		214	Q324
indicated		0.011		1688	Q324
inferred		0.011		5025	Q324
Klein Trekkopje measured resources		0.012		511	Q324
indicated		0.012		4671	Q324
inferred		0.011		48508	Q324
	sum Trekkopje + Kl Tr			60617	Q324
Langer Heinrich, proven reserves		0.061		9261	Q324
probable		0.057		4130	Q324
measured resources		0.063		9504	Q324
indicated		0.056		5003	Q324
inferred		0.060		13316	Q324
	sum Langer Heinrich			41214	Q324
Valencia		0.019		5970	Q324
Klein Spitzkoppe		0.024		1217	Q324
Engo Valley		0.029		1638	Q324
total Namibia				268170	
Namibia < 80				238000	Q90 h
additional <130				44000	Q90 h
Namibia 1		0.028		157514	h
Namibia 2		0.07		41214	h
Namibia 3		0.011		60617	h
		0.019-			
Namibia 4		0.029	0.023	8825	
Namibia 5	assumed	< 0.03		13830	
sum Namibia 1-5				282000	
• South Africa					

Hartebeest/Buffelsfontein, meas.resources	0.018		3178	Q324
indicated	0.016		1901	Q324
tailings dam, measured resources	0.0070		7504	Q324
indicated	0.0052		8938	Q324
sum Buffelsfontein			21521	Q324
Randfontein, measured resources	0.061		1500	Q324
indicated	0.081		1100	Q324
inferred	0.064		87846	Q324
sum Randfontein			90446	Q324
Beaufort West	0.08		23000	Q324
Springbok Flats	0.06		18000	Q324
Vaal River, probable reserves	0.039		13920	Q324
measured resources	0.055		1081	Q324
indicated	0.109		37713	Q324
inferred	0.126		8347	Q324
sum Vaal River			61061	Q324
Bonanza South (Klerksdorp)	0.0076		270	Q324
Dominion, indicated resources	0.071		11568	Q324
inferred	0.033		28417	Q324
sum Dominion			39985	Q324
Rietkuil, indicated resources	0.069		6698	Q324
inferred	0.054		48196	Q324
sum Rietkuil			54894	Q324
Ryst Kuil	?		30385	Q324
Denny Dalton	0.03		9350	Q324
<u>total South Africa</u>			<u>307642</u>	
South Africa			249000	Q90 h
South Africa, additional <130			92000	Q90 h
	0.035-			
SA 1	0.08	0.053	249000	
	0.016-			
SA 2	0.035	0.024	92000	
• Brazil				
Poços de Caldas				
Cachoeira	0.3		12700	Q90 h
Santa Quitéria	0.08		76100	Q90
Lagoa Real	0.25		79712	Q324
Itataia	0.085		67246	Q324
<u>total Brazil</u>			<u>235758</u>	
Brazil			231000	Q90
Brazil, additional <130			48000	Q90
Brazil 1	0.25-0.3	0.27	92412	
	0.08-			
Brazil 2	0.085	0.082	143346	
Brazil 3	assumed 0.08-0.2	0.126	43242	

• Niger

Arlit, proven reserves	0.3		16716	Q324
measured resources	0.16		6397	Q324
inferred	0.3		10115	Q324
sum Arlit			33228	Q324
Akouta, proven reserves	0.46		7909	Q324
probable	0.43		15737	Q324
measured resources	0.33		2544	Q324
indicated	0.31		5725	Q324
inferred	0.25		18718	Q324
sum Akouta			50633	Q324
Imouraren, measured resources	0.11		118100	Q324
indicated	0.1		25500	Q324
sum Imouraren			143600	Q324
Teguidda	?		12764	Q324
Madaouéla	?		6191	Q324
total Niger			246416	
Niger			225000	Q90
Niger, additional <130			0	Q90

Niger 1	0.16-0.3	0.22	33228
	0.25-		
Niger 2	0.46	0.34	50633
Niger 3	0.1		141139

• Russian Federation

Karkhu (Karelia)	0.1		7000	Q324
Aldansky district	0.1-0.15	0.122	340000	Q324
Dalmatovkoye	0.04		10200	Q324
Malinovskoye	?		10000	Q324
Vitimsky	0.05		11000	isl Q324
Streltskovsk	0.2		128200	Q324
Zherlovskoyo	?		3000	Q324
Pyatiletneye	?		1300	Q324
Argunskoyo	?		4800	Q324
total Russia			515500	
Russian Federation <80 = <130			172000	Q90

Russia 1	0.2		128200
	0.04-		
Russia 2	0.05	0.045	21200
Russia 3	assumed	0.1	22600

• USA

Anderson	0.068		3885	Q324
Arizona 1	0.55		385	Q324
Canyon	0.76		770	Q324
Pinenut	0.35		347	Q324
Wate	0.68		431	Q324
Workman Creek	0.093		3773	Q324
Schwartzwalder	?		6150	Q324

Sunday	0.2	412		Q324
Hansen	0.086	10654		Q324
Whirlwind Claim	0.21	385		Q324
Coyote Basin	0.17	13615		Q324
Crow Butte, proven reserves	0.33	2615		Q324
probable	0.20	386		Q324
inferred resources	0.14	3885		Q324
sum Crow Butte				Q324
Big Red	?	2348	isl	Q324
Kings Valley	0.085	6577		Q324
Churchrock, measured resources	0.085	1244	isl	Q324
indicated	0.076	3313		Q324
inferred	0.076	1356		Q324
sum Churchrock (Strathmore)				Q324
Churchrock (HRI)	?	7350	isl	Q324
Crownpoint	?	15000	isl	Q324
Crownpoint section 19/29, indicated	0.075	5258		Q324
Crownpoint section 24, indicated	0.086	3833	isl	Q324
Hosta Butte, indicated	0.095	5700	isl	Q324
La Jara Mesa, inferred resources	0.17	1220		Q324
Mt. Taylor	?	38500		Q324
Ram Claims	?	2300		Q324
Roca Honda, measured resources	0.20	2321		Q324
indicated	0.20	4414		Q324
inferred	0.14	6089		Q324
sum Roca Honda				Q324
Nose Rock	0.12	5624		Q324
Treeline	0.11	577		Q324
Aurora Property	0.044	7038		Q324
Dewey/Burdock Property	?	2387		Q324
	0.042-			
Goliad	0.064	0.051	2000	Q324
Palangana	0.13	2195	isl	Q324
Kingsville dome	?	122	isl	Q324
Rosita	?	134	isl	Q324
Vasquez	?	1192	isl	Q324
Bullfrog, inferred resources	0.28	4971		Q324
Rim	0.15	140		Q324
San Rafael	0.14	800		Q324
Cedar mountain	0.044	770		Q324
Green River North	0.2	250		Q324
Green River South	0.2	192		Q324
Tony M	0.13	4192		Q324
Velvet	0.36	764		Q324
Frank M	0.1	1350		Q324
Coles Hill	0.17	11538		Q324
Allemand-Ross, ind.+ inf.resources	?	2149		Q324
Antelope	0.064	926		Q324
Charlie	?	1610		Q324
Cyclone	0.11	770		Q324
East Shirley Basin, measured resources	0.13	115		Q324

indicated	0.093	1577	Q324
inferred	0.085	423	Q324
sum East Shirley			Q324
Gas Hills, probable reserves	0.12	7577	Q324
measured resources	0.076	1269	Q324
indicated	0.068	885	Q324
inferred	0.06	308	Q324
sum Gas Hills			isl Q324
Highland, proven reserves	0.10	692	Q324
probable	0.10	1038	Q324
measured resources	0.085	654	Q324
indicated	0.076	38	Q324
inferred	0.13	769	Q324
sum Highland			isl Q324
JAB property	0.06	1154	Q324
Jackpot	0.165	22192	Q324
Nine Mile Lake	0.047	3465	Q324
North Butte, probable reserves	0.085	3269	Q324
measured resources	0.068	731	Q324
indicated	0.059	2423	Q324
inferred	0.059	385	Q324
sum North Butte			isl Q324
Ruth, probable reserves	0.085	654	Q324
measured resources	0.085	77	Q324
indicated	0.059	77	Q324
inferred	0.059	154	Q324
sum Ruth			Q324
Peterson, measured resources	0.075	606	Q324
indicated	0.10	100	Q324
sum Peterson			Q324
	0.056-		
Reno Creek	0.069	0.0621 2271	Q324
Lye property	?	212	Q324
Reynolds Ranch, measured resources	0.059	1731	Q324
indicated	0.051	2692	Q324
inferred	0.034	1885	Q324
sum Reynolds Ranch			Q324
Sheep Mountain, inferred	0.14	6000	Q324
Smith Ranch, proven reserves	0.076	1077	Q324
probable	0.10	3192	Q324
measured resources	0.17	38	Q324
indicated	0.076	1923	Q324
inferred	0.059	346	Q324
sum Smith Ranch			isl Q324
Northwest Unit, indicated resources	0.034	885	Q324
inferred	0.034	192	Q324
sum Northwest Unit			isl
Bear Creek	?	1848	isl Q324
Ruby Ranch, probable reserves	0.076	2115	Q324
measured resources	0.15	231	Q324
indicated	0.051	38	Q324

inferred	0.013		77	Q324
sum Ruby Ranch				isl Q324
Taylor Ranch	?		3850	isl Q324
Moore Ranch, measured resources	0.085		2261	Q324
inferred	0.086		34	Q324
sum Moore Ranch				Q324
Lost Soldier (Sweetwater), measured re- sources	0.054		1923	Q324
indicated	0.055		2769	Q324
inferred	0.047		692	Q324
sum Lost Soldier				Q324
Lost Creek, indicated resources	0.049		1769	Q324
inferred	0.064		423	Q324
sum Lost Creek				Q324
Red Rim, indicated resources	0.14		439	Q324
inferred	0.14		592	Q324
sum Red Rim				
<hr/>				
total USA			298344	
USA, < 80 USD/kgU			102000	Q90
USA, < 130 USD/kgU			240000	Q90
USA 1	0.1-0.3	0.173	102000	
USA 2	0.03-0.1	0.055	240000	
• Uzbekistan			91000	Q90
Uzbekistan additional < 130 USD/kgU			25000	Q90
	0.06-			
Uzbekistan 1	0.18	0.104	91000	
	0.026-			
Uzbekistan 2	0.06	0.039	25000	
• Ukraine				
Ingul'skii	0.1		27000	Q324
Vatutinskii	?		25500	Q324
Severinskoye	0.1		50000	Q324
<hr/>				
total Ukraine			102500	
Ukraine			76000	Q90
Ukraine, additional < 130 USD/kgU			14000	Q90
Ukraine 1	0.1		76000	
	0.01-			
Ukraine 2	0.06	0.024	14000	Q90
Jordan	*	0.05	79000	Q90
India		0.05	67000	Q90
China	*	0.1	60000	Q90

other	*	0.1-0.2	0.14	156000	Q90
other, additional < 130 USD/kgU	*	0.05-0.1	0.071	131000	Q90

* deduced by the author

world total Red Book 4743000 Q90

NB Data from references Q210, Q211, Q212, Q213, Q85 are not consistent

Table D.3

The ore grade distribution of the known uranium resources. This table is based on Table D.2 above.

diagram			G or geom	actual				
G-range	fraction	resource	mean G	G-range	sub m(U)	total m(U)		
%U ₃ O ₈	%		%U ₃ O ₈	%U ₃ O ₈	Mg	Mg		
50 - 20	4,6	Canada 1				216121	u *	
		Mac Arthur proven reserves	26,56		128896			
		Cigar Lake proven reserves	20,67		87225			
		sum			216121	216121		
20 - 10		Canada 2				78525	u *	
		Mac Arthur probable reserves	19,06		33072			
		Cigar Lake inferred resources	16,92		45453			
	1,7	sum			78525	78525		
<10 - 5		Canada 3				63430	u *	
		Mac Arthur indicated re-sources	9,42		6699		u*	
		Mac Arthur inferred resources	9,51		40704			
		Midwest prov.res	5,47		16027			
	1,3	sum			63430	63430		
<5 - 2		Canada 4	3,04	2,1 - 4,41		8802		
		Canada 8	2			22384	u	
	0,7	sum				31186		
<2 - 1,0		Canada 5	1,42	1,2 - 1,69		25482	u	
	0,5	sum				25482		
<1 - 0,5		Canada 6	0,53			254		
	0,01	sum				254		
<0,5 - 0,2		Australia 5	0,23			25220		
		Australia 8	0,48			138224	u	
		Niger 1	0,22	0,16 - 0,3		33228		
		Niger 2	0,34	0,25 - 0,46		50633		
		Brazil 1	0,27	0,25 - 0,3		92412	h	
	7,2	sum				339717		

<0,2 - 0,1	Australia 6	0,14			48593		
	Australia 7	0,18			17808	isl	
	Brazil 3	0,14	0,1 - 0,2		43242		
	Niger 3	0,1			141139		
	Russia 1	0,2			128200		
	Russia 3	0,1			22600		
	USA 1	0,173	0,1 - 0,3		102000		
	Uzbekistan 1	0,104	0,06 - 0,18		91000		
	Ukraine 1	0,1			76000		
	China	0,1			60000		
	other (<80\$)	0,14	0,1 - 0,2		156000		
18,7	sum				886582		
<0,1 - 0,05	Australia 1	0,06			60208	u	h
	Australia 2	0,05			272208	u	h
	Australia 3	0,05			275600	u	h
	Canada 7	0,087			29000		h
			0,042				
	Kazakhstan 1	0,060	- 0,086		387737		
	Namibia 2	0,07			41214		
	SA 1	0,053	0,035 - 0,08		249000		h
	Brazil 2	0,082	0,08 - 0,085		143346		
	USA 2	0,055	0,03 - 0,1		240000		
	other (<130\$)	0,071	0,05 - 0,1		131000		
38,6	sum				1829313		
	of which hard ores				886016		
	soft or mixed ores		sum	1829313	943297		
<0,05 - 0,02	Australia 4	0,04			305139	u	h
	Kazakhstan 2	0,046	0,035 - 0,06		219263		s+h
	Kazakhstan 3	0,032	0,02 - 0,05		209000		
	Uzbekistan 2	0,039	0,026 - 0,06		25000		
	Russia 2	0,045	0,04 - 0,05		21200		
	Namibia 1	0,028			157514		h
	Namibia 4	0,023	0,019 - 0,029		8825		h
	Namibia 5	0,3			13830		h
	SA 2	0,024	0,016 - 0,035		92000		h
	Ukraine 2	0,024	0,01 - 0,06		14000		
	Jordan	0,05			79000		
	India	0,05			67000		
25.5	sum				1211771		
	of which hard ores				577308		
	soft or mixed ores		sum	1211771	634463		
<0,02 - 0,01	Namibia 3	0,011			60.617		h
1.3	sum				60617		
100	total world				4742998		

D3 Recovery of uranium from the earth's crust

Ore types

Uranium occurs in many kinds of minerals in the earth's crust. The nuclear industry distinguishes sometimes two categories of uranium resources, based on economic considerations: conventional and unconventional resources.

Conventional resources are deposits of the kind now being mined. Unconventional resources are shales, phosphates, granites or even seawater. To extract uranium from unconventional resources, new techniques have to be developed and the market price should rise substantially to make the extraction economic.

However, unconventional resources all have very low grades. We return to this topic later.

In this study the conventional ores are divided into two groups, based on information from (among other) Orita 1995 [Q23-14], UIC-34 2005 [Q86], WNA-mining 2007 [Q87]):

- soft ores, easily mineable and millable, e.g. sandstones and calcretes, with typical grades ranging from more than 10% down to about 0.02% U_3O_8 ,
- hard ores, hard to mine and mill, e.g. quartz pebble conglomerates, with grades varying typically from about 0.1% down to the mineralization limit (presumably at about 0.01% U_3O_8 , see next section). Some high-grade vein-type ores are also hard to mill.

These ore two classes are not used by the nuclear industry, but become roughly evident when assessing the specific energy requirements for mining and milling of uranium ores.

Table D.4

Mining methods in 2006. Percentage of world uranium production

	Percentage world uranium production 2006	
	WNA [Q89]	This study
underground	26	35 **
open pit	43	43
in situ leaching (ISL)	22	22
by-product	9 *	< 1 **

Source: WNA-23 2007 [Q89].

* Includes Olympic Dam (Australia, by-product of copper), with 7.2% of world production, and AnloGold (South Africa, by-product gold) with 1.6%, both underground mines.

** In this study Olympic Dam and AngloGold are classified as underground uranium mines.

In 2006 the world annual uranium consumption was some 67000 Mg, the actual uranium production from the uranium mines was some 40000 Mg. The balance, 37000 Mg, was

supplied by mixing up highly enriched uranium (HEU) from military stocks with depleted uranium (enrichment tails).

Mineralogical barrier

The earth's crust consists 99.5% by weight of silicate and oxide minerals in which aluminium, iron calcium, magnesium, sodium, potassium and titanium are the cations (ions with a positive charge). The remaining 79 elements total only about 0.5% of the crust and are therefore present in essentially trace amounts. The most important of the minor mineral families are the carbonates, sulphides and sulphates.

Because most chemical elements occur in such tiny amounts in the crust, their ions can readily be accommodated in common oxide and silicate minerals by substitution of more abundant ions. The only way minerals of the scarce elements can form is for some unusual, and therefore rare, geological process to cause a local enrichment of element, in case uranium. At a certain enrichment level, a substitution limit is reached, set by the geochemical properties of uranium. Above the substitution limit a separate uranium mineral forms, in which uranium becomes the major cation. The uranium mineral, for example pitchblende UO_2 , is found as small grains between the common silicate crystals in the host rock.

A typical uranium ore consists of a large mass of valueless silicate minerals mixed with a small quantity of a uranium mineral. The extraction of uranium from the mix implies dissolving the small grains of the uranium mineral in an appropriate solvent (e.g. dilute sulphuric acid), without attacking the silicate minerals, which compose the bulk of the ore mass.

In rocks with a uranium content below the substitution limit, no uranium mineral grains are present. The uranium is dispersed, instead, in the lattices of the bulk minerals (usually silicates) of the host rock, replacing other cations. This condition is called a solid solution of uranium. Examples of a solid solution of uranium are phosphate rocks and granites. Solid solutions are not amenable to selective physical and chemical extraction. To recover the uranium from such a rock type, the complete mass of minerals of the host rock has to be brought into solution.

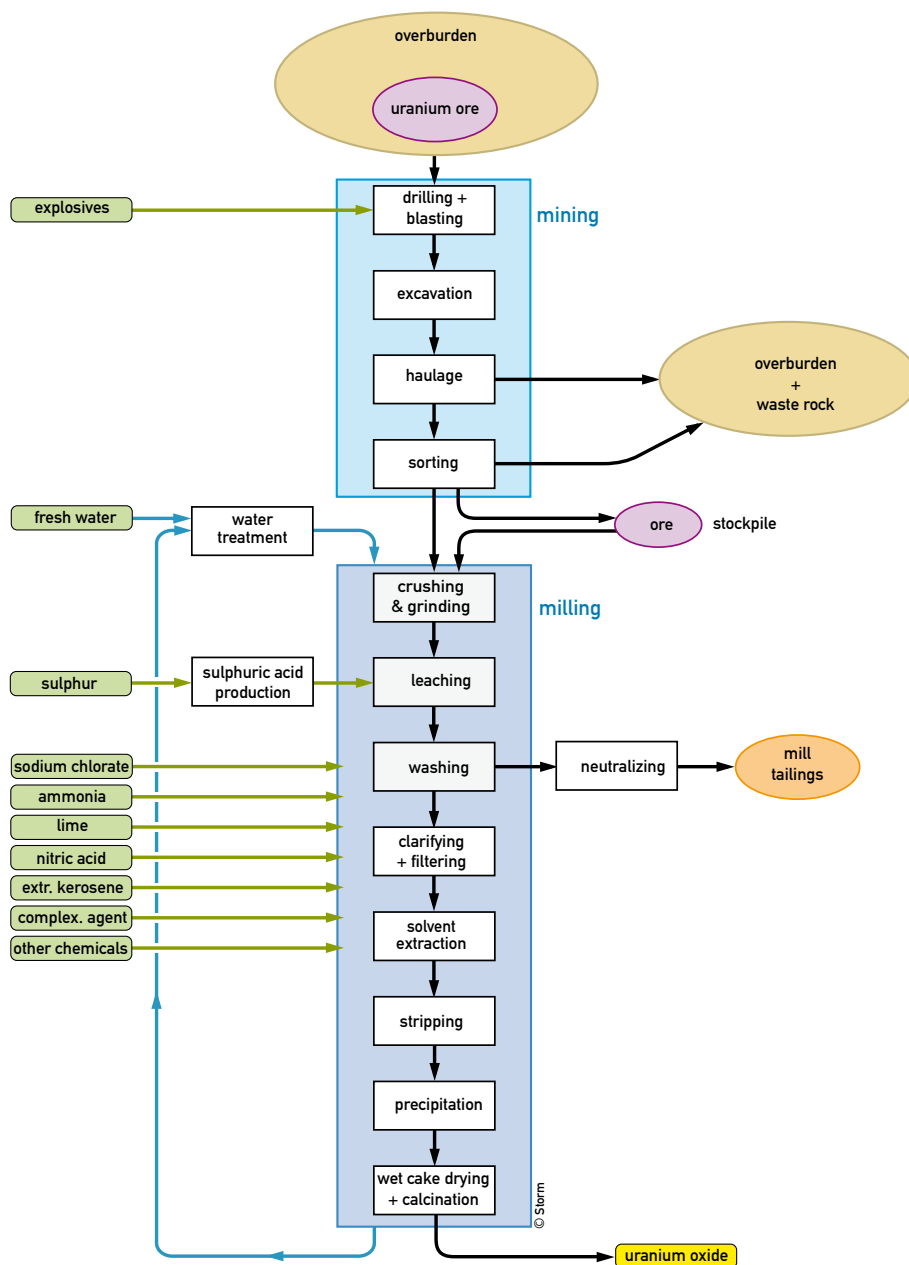
The grade boundary between rocks with and rocks without separate uranium minerals is called the mineral barrier by Skinner 1979 [Q322]. Any scarce metal has its own mineral barrier. Copper for instance has its barrier at a grade of about 0.1% by weight of copper. The uranium mineralization limit has a lower value than copper, due to the special geochemical properties of the element uranium.

For mining operations the mineralogical barrier has profound consequences, for the inputs of energy and materials required for the extraction jump by a factor 10 or more beyond the barrier.

The mineralogical barrier is not mentioned in the open nuclear literature. The gold industry reckons with this barrier indeed (DRDGold 2005 [Q334]).

Main variables: ore grade and recovery yield

The recovery of uranium from its natural occurrence in the earth's crust requires a sequence of mechanical and chemical processes. Each of these processes consumes energy and materials and consequently emits carbon dioxide and other gases. The uranium-bearing rock has to be drilled and blasted, excavated from the earth, separated from waste rock and transported to the mill. In the mill the rock is crushed, ground to powder and put through a number of physical and chemical separation processes (see Figure D.2).



Flowsheet Ranger uranium mine

Figure D.2

Flowsheet of Ranger, a large open pit uranium mine in Australia. This flowsheet is based on ERA 2006 [Q320] and ERA-AR 2005 [Q321] and can serve as model for most uranium mines.

The energy consumption of the uranium recovery comprises not only direct energy inputs - e.g. diesel fuel for the excavators and trucks and electricity for the mills and other equipment - but also indirect energy inputs, embodied in chemicals, capital goods, equipment, human labour and services.

There is a strong relationship between the energy input of the extraction of one kilogram uranium from the earth's crust and the grade of the rock from which the uranium is extracted.

At low ore grades, the specific energy inputs rapidly rise as the grade decreases.

Three grade-related mechanisms contribute to this phenomenon:

- dilution factor, hyperbolically increases as ore grade decreases
- milling efficiency, decreases as ore grade decreases
- extraction efficiency, decreases as ore grade decreases.

Dilution factor

The amount of rock to be mined and milled to obtain 1 kg uranium depends on the ore grade. At a grade of 0.1% uranium, one megagram (1 Mg = 1 metric tonne) of rock has to be mined and transported to obtain 1 kg uranium in the mill. This is ten times as much as from rock at a grade of 1%, containing 10 kg uranium per Mg rock. Consequently, the mining energy input per kilogram uranium is at least ten times as large. At a grade of 0.01% the energy input is 100 times as large.

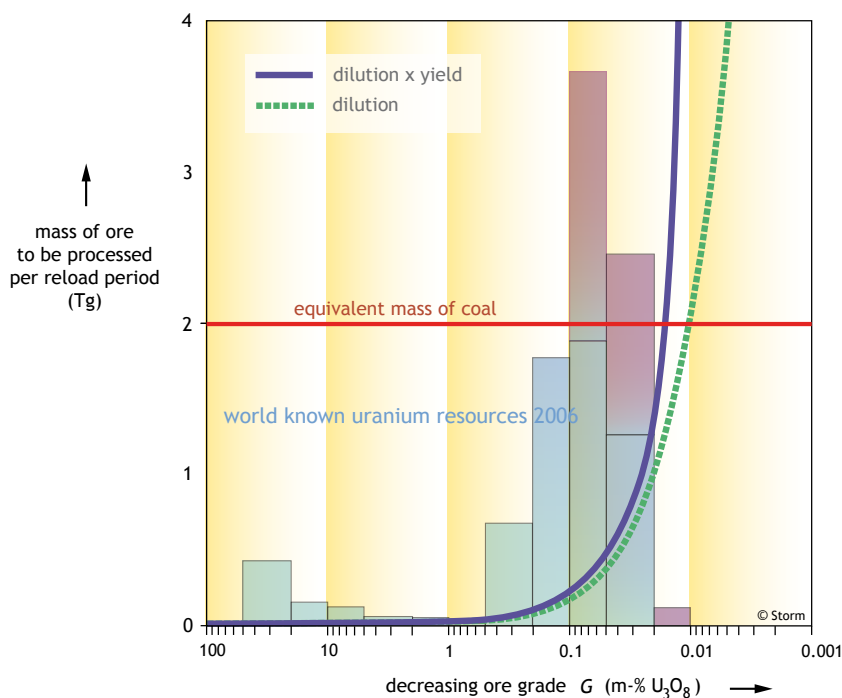


Figure D.3

The dilution factor and the coal ceiling. The mass of rock to be processed to fuel one reactor for one year with uranium rises hyperbolically with falling ore grade. At a grade below 0.02% U_3O_8 the mass of ore equals the mass of coal consumed by a coal-fired station to generate the same amount of electricity: this is the *coal ceiling*. See also the explanation in the text.

The dilution factor is a simple mathematical relationship between ore grade and mass of rock to be processed per mass unit of uranium, which does not depend on technology or ore type. The dotted green curve in Figure D.3 illustrates the mathematical relationship between the mass of ore to be processed each year to fuel one reactor and the ore grade.

Coal ceiling

Actually, more ore has to be processed than the mathematical minimum, due to the unavoidable imperfections of the extraction process (see sections below). The blue drawn curve in Figure D.3 illustrates the practical dilution relationship. The horizontal red line represents the mass of coal (2.0 Tg) consumed by a coal-fired power plant to generate the same amount of electricity as the reference reactor during one year (25.86 PJ). Below an ore grade of 0.02% U_3O_8 the annual mass of uranium ore to be processed to fuel one nuclear power plant equals the mass of coal: the *coal ceiling*.

Figure D.3 shows that below the critical ore grade, at which the nuclear system will pass through the coal ceiling, virtually no recoverable uranium resources have been reported. This is a striking coincidence in one respect, because no relationship exists between uranium ore grade and the combustion heat of coal. However, some explanation may be possible based on the Second Law of thermodynamics.

Decreasing efficiency mechanisms

Physical and chemical effects exacerbate the dilution effect, as mentioned above.

Mining losses

At lower grades the radiometrical distinction between ore and waste rock, with a uranium content below the cut-off grade, becomes increasingly difficult. This will lead to higher uranium losses and higher energy inputs of the following processes.

Milling efficiency

Lower grade usually means smaller grains of the uranium mineral in the ore rock. The ore has to be ground to a finer mesh, in order to liberate the smaller uranium grains from the rock matrix. Consequently the energy input of grinding per Mg ore increases. A second effect of the smaller grains is that a physical separation process to separate the uranium mineral grains from the bulk material (e.g. washing and flotation), becomes less efficient. The percentage of uranium recovered in milling decreases with decreasing grade.

Extraction efficiency

The powder from the mill is treated with chemicals, usually sulphuric acid with an added oxidizer, to dissolve the uranium compound from the host rock. The resulting raw solution, containing uranium ions and many other kinds of ions, goes through a number of chemical processes to extract the uranium ions selectively from the solution. After the separation from the other dissolved species in the mother liquid, the uranium solution is concentrated and refined to the product (yellow cake, ammoniumdiuranate).

The separation of the uranium ions from the raw solution involves several chemical equilibria. Separation equilibria never go to completion. The effectiveness of the extraction process decreases with decreasing content of uranium ions in the mother liquid. The inevitable presence of many other (unwanted) ions in the raw solution, with competing equilibria, exaggerates that problem. A lower grade leads to:

- larger volumes of solutions to be treated,
- lower extraction efficiency and
- need for more selective separation processes.

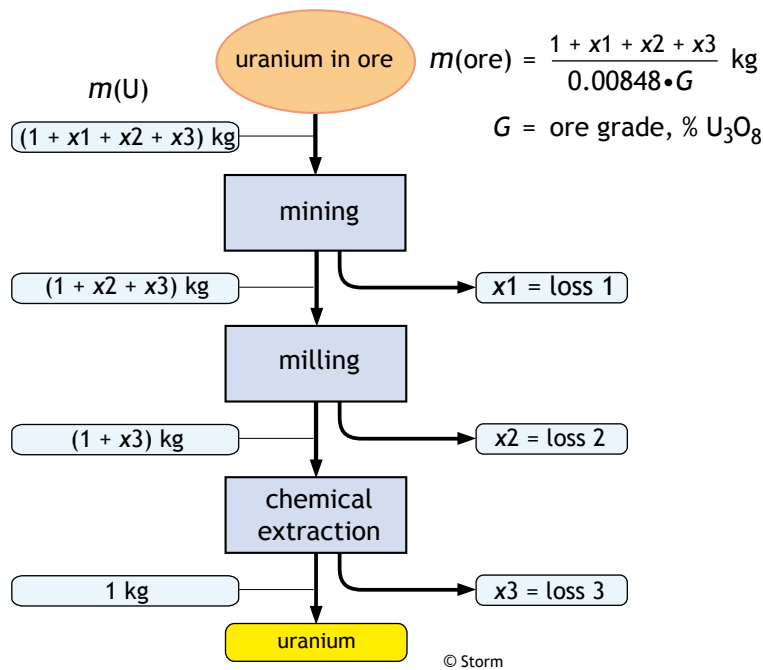


Figure D.4

The mass balance of the extraction processes of uranium from ore. The quantities x_1 , x_2 and x_3 are a function of the ore grade, becoming larger as the ore grade G decreases. The mass of ore to be mined is also a function of the grade due to the dilution factor $1/G$.

Extraction yield

The effects of the three mechanisms discussed above (mining losses, milling efficiency and extraction efficiency) become visible in the overall extraction yield of the uranium recovery.

The *extraction yield*, also called the *recovery yield* Y , is the ratio of the mass of uranium actually extracted and the mass of the uranium in the treated amount of rock. For instance, if a given amount of rock contains 800 grams of uranium and 768 grams of uranium is actually recovered from it, the yield $Y = 768/800 = 0.96$ or 96%.

The extraction yield never can be unit (or 100%), due to the inherent imperfections of extraction techniques, as is discussed above. The relationship between ore grade and empirical extraction yield of the uranium extraction is shown by Figure D.5.

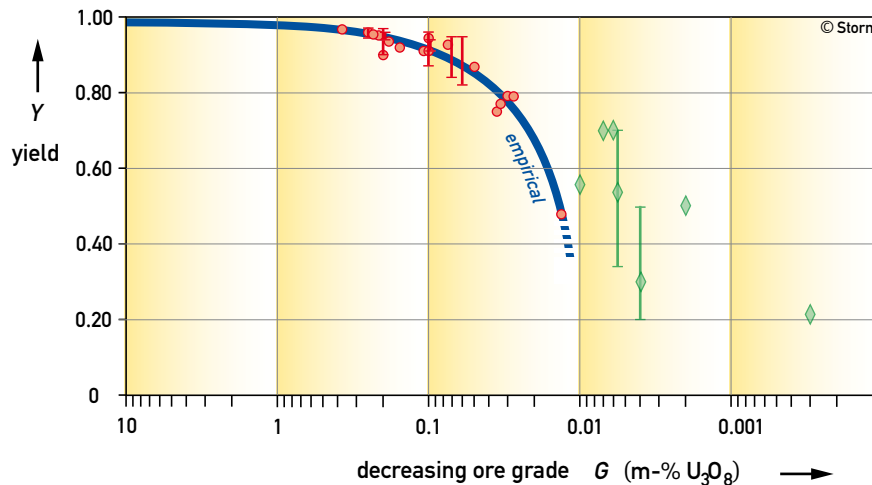


Figure D.5

The extraction yield of uranium from ore as function of the ore grade. The red dots are empirical values from the literature, based on actual mining operations. The green diamonds represent values taken from hypothetical mining and milling studies, which are not backed by practical experience. The blue curve (empirical) shows the empirical relationship between yield and grade.

The data used in this diagram have been taken from thirteen references:

Burnham et al. 1974 [Q136], Franklin et al. 1971 [Q111], GJO-100 1980 [Q141], Huwyler et al. 1975 [Q134], James & Simonson 1978 [Q138], James, Boydell & Simonson 1978 [Q137], Kistemaker 1976 [Q116], Mutschler, Hill & Williams 1976 [Q135], Rombough & Koen 1975 [Q120], Ross & Guglielmin 1968 [Q140], Rotty et al. 1975 [Q95], Simonson, Boydell & James 1980 [Q139], SRI 1975 Q110].

Recently reported yield data are not always unambiguous. In most cases it is not clear on which quantity of uranium the reported yields are based (see also Figure D.4):

- the in situ uranium (as present in the undisturbed ore body),
- in the actually mined ore
- in the ore entering the mill (the mined ore minus the waste)
- in the ore entering the chemical separation processes.

Some mining companies publish data which would result to yields of 100% or higher.

In this study the extraction yield is related to the reported uranium resources, assuming these being in situ resources. In some cases this assumption may lead to an underestimation of the true energy input of the uranium extraction.

In Figure D.5 the red dots represents the empirical extraction yields of uranium mines, as reported in the literature. Apparently, with the currently used extraction methods the yield drops to very low values at grades below 0.02% uranium oxide U_3O_8 (blue curve). The steep drop of the empirical curve may point to the mineralogical barrier of uranium being at a grade of about 0.01% U_3O_8 .

In a small number of studies on mining and milling of unconventional ores some values of extraction yields are mentioned, either based on small-scale laboratory experiments or on undisclosed assumptions.

The hypothetical yield values (the green diamonds in Figure D.5) refer to rocks beyond the mineralogical barrier. The specific energy input of the recovery of uranium from

rocks without uranium mineralization are much higher (10-100 times) than from rock with uranium mineralization, both at the same average grade. Therefore the unproved hypothetical yield values are not comparable with the empirical values.

The calculations in this paper are based on the empirical yield curve, down to the empirical data with the lowest reported grade of about 0.013% U_3O_8 .

The original calculations in the Storm&Smith [Q6] study are based on a mathematical relationship including the hypothetical yield values at grades below 0.01% U_3O_8 . Below a grade of 0.013% U_3O_8 no scientifically verifiable data are available and so the results of the analysis using data at grades below that value have essentially no scientific meaning.

Summarized:

The specific energy requirements of the extraction of uranium rise as the ore grade decreases. The rise becomes very steep at low grades, due to the exponentially rising dilution effect and the exponentially falling recovery yield.

Beyond the mineralization limit

Technically it is possible to extract uranium from rock with very low uranium content, for instance granites and shales, but at the expense of high consumption rates of materials and energy.

In rocks with grades beyond the mineralization limit uranium is present in solid solution (see section above on the mineralogical barrier). Consequently the whole rock mass has to be dissolved to liberate the uranium ions. Due to the very low uranium concentration and high concentrations of many other species in the solution, exceedingly selective extraction techniques are necessary. The high selectivity and the large volumes of solutions to be treated imply a high specific energy consumption.

Entropy

The mixing entropy of uranium increases with decreasing concentration of the uranium ions in the solution increases as more kinds of other ions are present and increases as the concentrations of the other species are higher. If the mixing entropy of uranium in a solution is higher, the required entropy reduction to the value of pure uranium is larger.

The entropy reduction of a certain quantity of matter is possible only with the input of work: usable energy, called Gibbs Energy in thermodynamics. The larger the entropy difference between uranium in its host rock and uranium in pure form, the more work (e.g. electricity) and ordered matter (equipment and chemicals) are needed. The lower limit of the required work is governed by the basic laws of thermodynamics, not by technology.

D4 Energy requirements of uranium recovery, from conventional ores

Mining

Other studies

The energy inputs of mining uranium ore have been estimated in a number of studies, most of which have been published in the 1970s. In our original report [Q6] most known energy analyses of the mining process are reviewed. Not all studies happen to be original energy analyses. The values of the energy requirements given in these studies vary widely, from a low of 0.08 GJ/Mg ore (Franklin et al. 1971 [Q111]) to a high of 77 GJ/Mg ore (Orita 1995 [Q23-14]). The large spread may be partly explained by individual differences between mines and partly by methodological differences.

The overburden ratios of open pit mines vary roughly from 3-50. The overburden ratio in the USA averaged 50, according to Blanchard et al. 1982 [Q252]. A overburden ratio (or stripping ratio) of 3 means that for every Mg ore 3 Mg of waste rock has to be removed. Stripping ratio variations may introduce a spread in the energy requirements with a factor of 5 (Chapman 1976-2 [Q106]), with the same ore. In addition different rock types: varying from earth to granite, have to be taken into account.

The methodological differences between the various studies are wide. Some studies take only the direct electrical input, other only the thermal, a third includes embodied energy in chemicals, and so on. In a number of studies it is not clear how the figures have been found.

This study

This study uses the figures from Rotty et al. 1975 [Q95]. See equation D.1. We assumed energy requirements of mining to be the same for soft and hard ores.

$$J_{\text{mining}} = 1.06 \text{ GJ/Mg ore} \quad R = J_{\text{th}}/J_{\text{e}} = 8.0 \quad \text{eq D.1}$$

As explained in Part C, the electrical and thermal (fossil fuel) energy inputs are kept separated in this study.

The figures from Rotty et al. are based on an unpublished survey of energy consumption in the USA mining and milling operations, conducted by the US Bureau of Mines in 1973. At that moment virtually all uranium in the USA was recovered from high-grade sandstone deposits. The figures represent the average of 60% open-pit and 40% underground mining and include, according to the authors, indirect energy consumption: the energy embodied in chemicals and equipment. The figures from Rotty et al. are used in ERDA-76-1 [Q109], a study quoted by WNA-11 2007 [Q155].

Milling

In this study the value from ERDA-76-1 [Q109] is used for soft ores:

$$J_{\text{milling}} = 1.27 \text{ GJ/Mg ore} \quad R = J_{\text{th}}/J_e = 7.0 \quad \text{eq D.2}$$

and the value from Kistemaker 1976 [Q116] for the hard ores:

$$J_{\text{milling}} = 4.49 \text{ GJ/Mg ore} \quad R = J_{\text{th}}/J_e = 0.10 \quad \text{eq D.3}$$

Mining + milling

The total specific energy requirements for mining + milling per Mg ore, as adopted in this study, are the sum of mining and milling energy inputs.

Soft ores:

$$J_{\text{m+m}} = 2.33 \text{ GJ/Mg ore} \quad R = J_{\text{th}}/J_e = 3.0 \quad \text{eq D.4}$$

and for hard ores:

$$J_{\text{m+m}} = 5.55 \text{ GJ/Mg ore} \quad R = J_{\text{th}}/J_e = 0.69 \quad \text{eq D.5}$$

The energy requirements of the recovery of one mass unit of uranium leaving the mill, $J_{\text{m+m}}$, depend on the ore grade G and the recovery yield Y , and can be calculated with the following equation D.6:

$$J_{\text{m+m}}(\text{U}) = \frac{c}{Y \cdot G}$$

$J_{\text{m+m}}(\text{U})$ = specific energy expenditure per mass unit uranium GJ/kg

c = specific energy constant = 0.275 GJ/kg(U) soft ores
= 0.654 GJ/kg(U) hard ores

Y = yield = fraction extracted U

G = ore grade mass-% U_3O_8

eq D.6

In our study Q6 we calculated the values of c , being:

$$\begin{array}{ll} \text{soft ores} & c = 0.275 \text{ TJ/Mg (U)} \quad R = J_{\text{th}}/J_e = 7.5 \\ \text{hard ores} & c = 0.654 \text{ TJ/Mg (U)} \quad R = J_{\text{th}}/J_e = 1.6 \end{array} \quad \text{eq D.6a}$$

To verify equation D.6, we compared the results of the process analysis of the Ranger mine in Australia (see Part D7) with the outcome of the equation under the conditions of Ranger. The actual figures of Ranger turn out to be slightly higher than the outcome of the equation D.6: see Table D.5. As Ranger is a large high-quality uranium mine, with production costs in the lowest region of the cost spectrum (see Figure D.16 in Part D12), we may conclude that Equation D.6 will not lead to overestimation of the energy input of the uranium recovery.

Some basic data of the uranium recovery (mining + milling) from soft ores are compiled in Table D.6 and from hard ores in Table D.7. The energy requirements and CO_2 emis-

sion are given per Mg uranium leaving the mill and the specific CO₂ emission also in g/kWh electricity put into the grid by the reference reactor.

Table D.5

Specific recovery energy requirements and carbon intensity of the Ranger uranium mine in Australia and of a reference mine with the same ore grade in this study. The Ranger ore is assumed to be soft. This process analysis is based on data from ERA 2005 [Q321] and ERA 2006 [Q320].

	Ore grade % (w/w) U ₃ O ₈	Specific energy requirements TJ/Mg Unat	Carbon intensity gCO ₂ /Kwh
Ranger 2005	0.288	1.08	1.78
Reference mine	0.288	0.97	1.46

Table D.6

Recovery data of uranium per Mg uranium leaving the mill (= m3, see Part E) from soft ores. Based on the empirical recovery yield curve from Figure D.5.

Grade, G % U ₃ O ₈	Yield, Y	$E_{th} + E_e$ TJ/Mg U	E_{th} TJ/Mg U	E_e TJ/Mg U	$m(\text{CO}_2)$ Mg/Mg U	CO ₂ emission g/kWh *
10	0.990	0.028	0.025	0.003	1.84	0.04
1	0.980	0.281	0.248	0.033	18.6	0.42
0.5	0.973	0.565	0.499	0.066	37.4	0.84
0.15	0.931	1.97	1.74	0.23	130	2.95
0.10	0.908	3.03	2.67	0.36	200	4.53
0.06	0.872	5.26	4.64	0.62	348	7.86
0.05	0.850	6.47	5.71	0.76	428	9.68
0.04	0.825	8.33	7.35	0.98	551	12.5
0.03	0.775	11.8	10.4	1.4	783	17.7
0.02	0.700	19.6	17.3	2.3	1300	29.4
0.013	0.472	44.8	39.5	5.3	2966	67.0

* Taken on the gross electricity production of the reference reactor, per reload charge in the steady state. This figure depends on the characteristics of the reactor in which the uranium is used, e.g. enrichment assay and burnup. Only the thermal (= fossil) inputs are accounted for in the CO₂ emission. The CO₂ emission of the energy debt is not taken into account.

Thermodynamic quality of a uranium resource

The thermodynamic quality of a uranium resource is the determinant of being an energy resource or not. Here we define the thermodynamic quality of a uranium resource as the quantity of useful energy (work) required to extract 1 kg pure uranium from that resource. If the extraction of 1 kg uranium requires as much work as can be generated from that uranium, it is no energy source. As pointed out above, the minimum amount of extraction work is governed by basic physical laws, not by technology.

Shift effect

Apart from the ore grade, the thermodynamic quality of uranium ore is also set by other factors, such as: depth of the ore body, accessibility of the ore (physically and chemically) and hauling distance. As the easiest accessible and richest ores are exploited first, the mining activities gradually have to shift to lower quality ores. From an energy point of view the equivalent ore grade then shifts to lower values. This is here called the 'shift effect'.

Table D.7

Recovery data of uranium per Mg uranium leaving the mill (= m³, see Part E) from hard ores. Based on the empirical recovery yield curve from Figure D.5.

Grade, G % U ₃ O ₈	Yield, Y	$E_{th} + E_e$ TJ/Mg U	E_{th} TJ/Mg U	E_e TJ/Mg U	$m(\text{CO}_2)$ Mg/Mg U	CO ₂ emission g/kWh *
10	0.990	0.066	0.041	0.025	3.08	0.07
1	0.980	0.667	0.411	0.257	30.8	0.70
0.5	0.973	1.34	0.827	0.517	62.0	1.40
0.15	0.931	4.68	2.88	1.80	216	4.89
0.10	0.908	7.21	4.43	2.77	333	7.52
0.06	0.872	12.5	7.69	4.81	577	13.0
0.05	0.850	15.4	9.47	5.92	710	16.1
0.04	0.825	19.8	12.2	7.62	915	20.7
0.03	0.775	28.1	17.3	10.8	1298	29.3
0.02	0.700	46.7	28.7	18.0	2156	48.7
0.013	0.472	107	65.6	41.0	4919	111.2

* Taken on the gross electricity production of the reference reactor, per reload charge in the steady state. This figure depends on the characteristics of the reactor in which the uranium is used, e.g. enrichment assay and burnup. Only the thermal (= fossil) inputs are accounted for in the CO₂ emission. The CO₂ emission of the energy debt is not taken into account.

D5 In-situ leach (ISL) uranium mining

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.

ISL has been applied to common low-grade ores containing 0.03-0.05% uranium. Common practice has mainly used sulfuric acid leaching at a concentration of 2-5 g/l (0.02-0.05 M H_2SO_4). However, an initial concentration of 15-25 g/l (about 0.15-0.25 M H_2SO_4) is generally used to reduce the ore preparation period. Often an oxidant is required to dissolve the uranium mineral. Oxidants in use include hydrogen peroxide, nitrate ions (nitric acid) and sodium chlorate. Acid consumption is typically 5-6 kg per Mg rock, but up to 10-15 kg/Mg rock. Overall recovery is typically 65-80% of the in-the ground resource CSIRO 2004 [Q363].

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 (Mudd 2000 [Q291]).

The reported recovery yield of $Y = 50-80\%$ refers only to the extraction from the rock. Given the low uranium content of the parent rock and of the pregnant solution pumped from the production wells, the extraction yield of uranium from the pregnant solution may be low. The overall yield, extraction from ore in the ground to yellow cake, may be in the range of 20-40%.

A major problem of ISL is the large-scale and irreversible 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 (see also Mudd 1998 [Q364]).

In our view the ISL technique cannot be reconciled with any sustainable development, for reason of its harmful and irreversible effects in the environment.

A rough impression of the energy requirements embodied in the chemicals for extraction can be figured out. 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:

$$J_{isl} = 0.547 \text{ TJ/Mg (U)} \quad R = J_{th}/J_e = 2.8 \quad \text{eq D.7}$$

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

$$\begin{array}{l} \text{sulfuric acid } H_2SO_4 \\ J_{spec} = 2.87 \text{ GJ/Mg} \end{array} \quad R = J_{th}/J_e = 100 \quad \text{eq D.8}$$

$$\begin{array}{l} \text{ammonia } NH_3 \\ J_{spec} = 86.65 \text{ GJ/Mg} \end{array} \quad R = J_{th}/J_e = 1.41 \quad \text{eq D.9}$$

It should be emphasized that equation D.7 represents only a fraction of the total

specific energy requirements of ISL per Mg extracted uranium. Not included are, for example, the energy requirements of:

- drilling the injection and production wells,
- pumping the fluids into the ground, via injection wells, and from the ground via production wells
- extraction of the uranium from the solution.

The energy requirements of in situ leaching will vary over a wide range, due to widely different geochemical conditions, depth of ore body, number of wells, operational life of each well (clogging is a major problem) and ore properties. In addition the energy requirements depend on the ore grade, as with conventional mining and milling. Data on actual ISL mines are missing from 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. This assumption may not lead to overestimation of the specific extraction energy of uranium from ore, the average of all mines and mills.

Mortimer 1977 [Q98], one of the few studies which include ISL, gives figures in the same range as soft ore mining and milling (see Table D.8).

The direct energy consumption of the Beverley ISL mine happens to be as high as that of the Ranger open pit mine, both mines in Australia (Mudd & Diesendorf 2007 [Q338]).

Table D.8

Energy input of mining + milling according to Mortimer 1977 [Q98].

	$J_{th} + J_e$ GJ/Mg ore	$R = J_{th}/J_e$
Soft ores	0.324 - 11.5	5.0 - 1.5
In situ leaching (ISL)	0.150 - 10.9	0.9 - 17
Phosphate clay	1.12 - 2.47	7.7 - 13.9

If ISL were to be cheaper than other methods, its specific energy consumption might be as high as of conventional methods or even higher. Moreover, the mineralogical barrier cannot be circumvented by any technology.

D6 Mine reclamation

Mining and milling operations disturb the geological formations that retain naturally occurring radioisotopes and allow public exposure. In no way uranium ore itself can be considered to be a harmless substance. In other studies the problems mill tailings pose are ignored.

Apart from the remaining uranium (U-238, U-235, U-234) in the mill tailings - the extraction yield or recovery always is less than 100% - the mill tailings contain the uranium decay daughters: Th-230, Ra-226, Rn-222, Pb-210, Po-210 and Pa-231. Some uranium ores contain a significant fraction of Th-232 and its decay products. As a consequence in some mill tailings the activity is comparable to that of U-238.

Blanchard et al. 1982 [Q252] summarize the results of an EPA study that addresses radioactive effluents (gaseous, wind blown dust, rain water seeping into the ground water table) associated with active uranium mines in the USA. Principal exposure routes considered are inhalation of contaminants released into the air, external exposure from air submersion and radionuclides deposited on the ground, use of contaminated water and ingestion of food contaminated via either air or water.

All radionuclides present in the tailings pose a serious long term environmental risk (NRC 1996 [Q16], Andriessse 1994 [Q77], Lipschutz 1980 [Q54], INFCE-7 1980 [Q277]) all the more so because the elements are chemically mobile after the milling process. The publications by Blanchard et al. 1982 [Q252], WISE-U 2006 [Q324] and Diehl 2006 [Q343] deepen that worries.

Until the present day, mill tailings and other waste of the uranium mining industry mostly are being discharged into the environment. In a sustainable nuclear cycle the mill tailings should to be shielded from the biosphere.

In this study the uranium mining area is restored to green field conditions after depletion of the ore deposit. and isolating the mill tailings - a sandy material saturated with several chemicals (e.g. sulfuric acid) - from the biosphere comprises several activities:

- neutralizing the acids in the tailings, in this study with limestone
- immobilization of the tailings
- transport of the immobilized tailings into the mine
- replacing the overburden
- replanting the area with indigenous vegetation.

The uranium mass balance of the mining and milling process is given in Figure D.6. For the complete mass balance of the front end see Figure E.2.

$$m_3 = m_4 - \text{loss } 4 \quad \text{eq D.10}$$

The total mass of the tailings is:

$$m_{\text{tailings}} = m_{\text{ore}} + m_{\text{chem}} - m_3 \quad \text{eq D.11}$$

The amount of process chemicals, needed to leach out the uranium compounds from the ore, may be a significant part of the total tailings mass. As a consequence the mass and volume of the tailings are larger than those of the processed ore, the study INFCE-7 1980 [Q277] estimates an additional mass of 8.6%.

Besides, a very large mass and volume of overburden of rock and earth has to be removed and stored. A significant part of the overburden mass also contains radionuclides and has to be isolated from the biosphere. Blanchard et al. 1982 [Q252] cite an average overburden ratio of 50 of all US surface mines. This implies that for each Mg of ore, 50 Mg of overburden has to be removed. Apart from the ore about an equal mass of sub-ore is mined, which is not processed. The average ratio sub-ore/ore for underground mines is slightly more than 0.1.

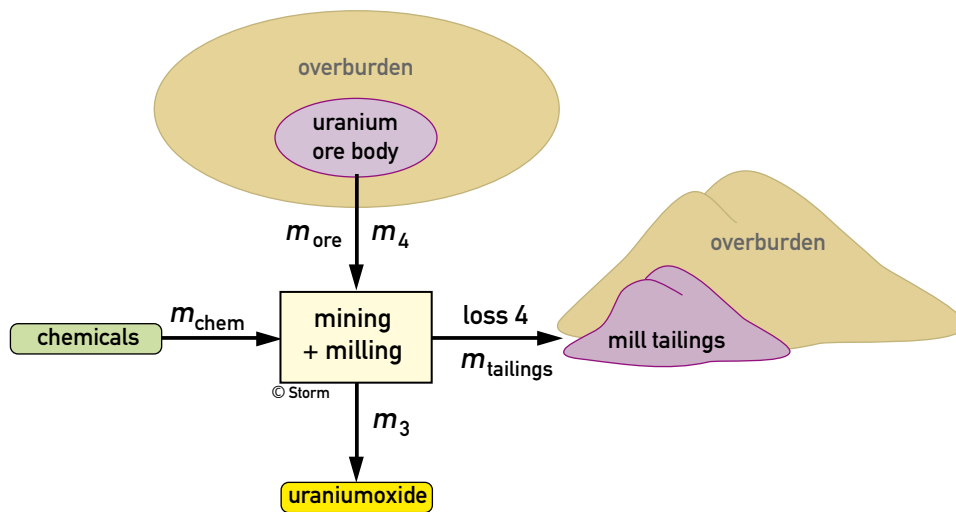


Figure D.6
Mass balance of the mining and milling process

The mass of mill tailings originating from the processed ore can be calculated by equation D.12 below. Here the mass of the process chemicals is ignored.

$$m_{\text{tail}} = 1.18 \cdot m_U \cdot \left(\frac{100}{Y \cdot G} - 1 \right)$$

m_{tail}	= mass of tailings	Mg
m_U	= mass of produced uranium	Mg
Y	= yield	fraction extracted U
G	= ore grade	mass-% U_3O_8 in ore
1.18 kg U_3O_8 contains 1 kg U		

Eq D.12

Estimation of the energy input of the mine reclamation is difficult, because no cases are known in the open literature. We assume a two-step process for neutralization and immobilization of the chemicals in the mill tailings:

- the tailings are mixed with powdered limestone to neutralize the acids and a phosphate (e.g. sodiumphosphate) to render the radioactive nuclides and other heavy metal ions insoluble in water.
- the resulting mass is poured between thick layers of bentonite, in order to isolate the tailings from the ground water.

The overburden and waste rock have to be replaced and replanted. Underground mines are filled up with the neutralized and immobilized tailings, in this concept.

It should be noted that this procedure is not an operational process, but a model aimed at a rough estimate of the energy expenditure of the reclamation process.

The energy expenditure of above process is estimated to be four times that of mining, $J_{\text{mining}} = 1.06 \text{ GJ/Mg (ore)}$. The mass of the tailings, including the limestone and bentonite, is assumed to be about twice the mined ore mass. The limestone and bentonite have to be mined as well, the sodiumphosphate has to be produced from phosphate rock.

So, the reclamation of the mine area may take approximately:

$$J_{\text{tailings}} = 4.2 \text{ GJ/Mg (tailings)} \quad R = J_{\text{th}}/J_e = 8.0 \quad \text{eq D.13}$$

We are aware of the somewhat speculative character of this estimate, as no empirical data exist today.

In our view above approximation yields a reasonable indication of the order of magnitude of the energy input. We think it far better to include a rough estimate of the energy input of this phase of the nuclear chain, rather than deleting it at all, as other studies do.

Table D.9

Reclamation of the mine area per Mg uranium leaving the mill. Based on the empirical recovery yield curve.

Grade, G % U ₃ O ₈	Tailings Gg/ Mg U	$E_{\text{th}} + E_e$ TJ/Mg U	E_{th} TJ/Mg U	E_e TJ/Mg U	$m(\text{CO}_2)$ Mg/Mg U	CO ₂ emission g/kWh/Mg U
10	0.011	0.0451	0.0401	0.0050	3.007	0.07
1	0.119	0.501	0.445	0.0556	33.38	0.75
0.5	0.241	1.01	0.90	0.113	67.6	1.53
0.15	0.844	3.54	3.15	0.394	236	5.34
0.10	1.30	5.46	4.85	0.606	364	8.22
0.06	2.25	9.47	8.42	1.05	631	14.3
0.05	2.78	11.7	10.4	1.30	777	17.6
0.04	3.58	15.0	13.3	1.67	1001	22.6
0.03	5.07	21.3	18.9	2.37	1421	32.1
0.02	8.43	35.4	31.5	3.93	2360	53.3
0.013	19.2	80.8	71.8	8.97	5384	121.7

D7 Process analysis of the Ranger mine

The Ranger mine in Australia is one of the best-performing uranium mines today. So it appears appropriate to use a process analysis of this mine to validate the equations and specific energy requirements used in this study. Fortunately, Energy Resources of Australia (ERA), the owner of Ranger, published enough data of the mine to make possible a rough process analysis of the mine. Apart from the ERA data, several other sources have been used.

Table D.10

Uranium mining and milling at the Ranger mine in 2005. This table is a copy of an Excel spreadsheet (filename: Ranger4.xls), so its lay-out may not look optimal.

Ranger uranium mine basic data 2005, as quoted in source			source
mass of ore milled	2,293	million Mg	Q321
processing recovery	89,2	%	Q321
mass of U3O8 (drummed) produced	5910	Mg	Q321
product grade of drummed U3O8	98,7	% U3O8	Q321
processing head grade	0,288	% U3O8	Q321
cutoff grade	0,12	% U3O8	Q321
number of dump trucks (Caterpillar)	6		Q320
excavator (Hitachi EX-2500)	1		Q320
explosives	0,25	kg/Mg rock	Q320
sulphuric acid plant production capacity	250	Mg/day	Q320
diesel fuel consumption electric station	50-60	Mg/day	Q320
stripping ratio (=overburden/ore ratio) Pit#1	3		Q320
average ore grade Pit#3	0,27	% U3O8	Q320
water treatment	1,5	million m3/a	Q321
capacity haul trucks	90-135	Mg	Q320
diesel fuel			
density	0,839	Mg/m3	BP 2006
energy content	38,66	GJ/m3	BP 2006
specific CO2 emission	75	g/MJ(th)	Q27
			R
embodied energy	Je + Jth	Jth/Je	
ammonia NH3	86,65	GJ/Mg	1,41
production of sulphur (average)	41,5	GJ/Mg	26,7
sodiumchlorate NaClO3	45,0	GJ/Mg	1,12
hydrofluoric acid HF	52,3	GJ/Mg	11,2
nitric acid HNO3	35,5	GJ/Mg	2,62
explosives (average)	71	GJ/Mg	70
ammoniumnitrate NH4NO3	46,37	GJ/Mg	2,02
lime CaO	8,47	GJ/Mg	120
extraction grade kerosene	?		

	?		R	
	Je + Jth		Jth/Je	
complexing agent				
indirect energy requirements				
drilling + blasting, excluding explosives	12 MJ(th)/Mg rock		-	Q98
excavation + haulage	38 MJ(th)/m3 earth		-	Q98
ore processing (mill), excluding chemicals	50 MJ/Mg ore		7,33	Q98
direct energy requirements				
drilling + blasting (average)	5 MJ(e)/Mg rock		-	Q98
excavation	14,6 MJ(th)/m3 earth		-	Q98
haulage (average)	1 MJ(th)/Mg.km		-	Q98
crushing & grinding	74 MJ(e)/Mg ore		-	Q98
leaching	0,27 GJ/Mg ore		5,23	Q98
solvent extraction	2 GJ(e)/Mg U3O8		-	Q98
drying wet cake	190 GJ(th)/Mg U3O8		-	Q98

1 kg U3O8 contains 0,848 kg U

Uranium mining + milling, Ranger, 2005

Process analysis

category	material	quantity		energy requirem.		CO2
		Gg	103 m3	electric TJ	thermal TJ	Gg
direct energy inputs						
drilling + blasting (excl explosives)	rock	9200		46		
excavation	rock	9200			56	4,2
haulage	rock	9200			44	3,3
crushing and grinding	ore	2293		170		
leaching	ore	2293		97	528	39,6
solvent extraction	U3O8	5,910		12		
drying wet cake and calcination	U3O8	5,910			1123	84,2
water treatment	water	1500	1500			
transport to Ranger	various	119			24	1,8
transport from Ranger	various	p.m.				
	sum			<u>325</u>	<u>1775</u>	<u>133,1</u>
electricity production (mean)	diesel fuel	20,1	23,9	-315	925	69,4
sulphuric acid production	H2SO4	91,25			278	0
	sum direct energy inputs			<u>10</u>	<u>2978</u>	<u>202,5</u>

indirect energy inputs, equipment, maintenance

drilling and blasting				112	8,4
excavation and haulage				14	1,1
ore processing (mill)				<u>54</u>	<u>404</u>
	sum			<u>54</u>	<u>530</u>

energy embodied in chemicals

explosives	?	2,300	2	160	12,0
sulphur	S	29,8		1200	90,0
sodiumchlorate *	NaClO3	2,75	58	66	5,0
ammonia	NH3	1,08	39	54	4,1
extraction grade kerosene	? ?				
complexing agent	? ?				
lime	CaO	26,04	1,8	219	16,4
nitric acid	HNO3	?			
water	H2O	?			
sum					
			101	1699	127,4
total energy requirements			155	5207	369,7
CO2 production CaCO3 -> CaO+CO2					20,5
total CO2 production, Tg					390,2

Summary

		unit	source
drummed U3O8	5910	Mg	Q321
grade (fraction U3O8)	0,987		Q321
pure U3O8	5833	Mg	
extracted uranium	4947	Mg	•0,848
natural uranium for 1 reload charge	162,48	Mg	Q6
number of reloads	30		
gross electricity production per reload	25,86	PJ	
	7,18	10E9 kWh	
gross electricity production total	219	10E9 kWh	30•7,18
	787,2797773	PJ	
gross electricity production per Mg U	0,1592	PJ/Mg U	
	44,2	10E6 kWh/Mg U	
fraction energy input:gross electr.production	0,006810539		
specific energy requirements mining+milling			
thermal (fossil fuel)	1053	GJ/Mg U	
electric	31	GJ/Mg U	
total (th + e)	1084	GJ/Mg U	
R = Jth/Je	33,6		
specific CO2 emission			
per Mg uranium	78,88	MgCO2/Mg U	
gross electricity production	1,78	gCO2/kWh	

Calculated by equation 2.1 from Q6

$J(m+m) = c/Y \cdot G = 275/0,98 \cdot 0,288$	974	GJ/Mg U	R=7,5
thermal	860	GJ/Mg U	
electric	115	GJ/Mg U	
CO2 emission = J(th)*75	64,48	MgCO2/MgU	
gross electricity production	1,46	gCO2/kWh	

Summary results

specific energy requirements mining+milling			
total (th + e), Ranger	1084	GJ/Mg U	R=34, special case
total (th + e), Q6	974	GJ/Mg U	R=7,5
specific CO2 emission			
gross, Ranger	1,78	gCO2/kWh	

gross, Q6

1,46 gCO2/kWh

Note

This spreadsheet uses commas instead of dots in the numbers

relationship between mining energy requirements and

- strip ratio (overburden/ore)
- haulage distance

$$m0 = m(\text{ore}) = 2300 \text{ Gg} = 2,3 \text{ Tg}$$

$$m1 = m(\text{rock}) = 2,3 + r \cdot 2,3 \text{ Tg}$$

extracted uranium 4947 Mg

mining

strip ratio	mining only									mining +milling				
	m1	dr+blst	excav	haul	expl	expl	expl	equipm	sum	sum	sum	sum	per Mg U	
	Tg	TJ(e)	TJ(th)	5 km	Gg	J(e)	J(th)	J(th)	TJ	TJ	TJ	TJ	GJ/Mg	GJ/Mg
0	2,3	11,5	14	11	0,575	0,5	40	31,4	12	96	444	3992	90	807
3	9,2	46	56	44	2,3	2	160	126	48	386	480	4282	97	865
5	13,8	69	84	66	3,45	3	240	188	72	578	504	4474	102	904
10	25,3	126,5	154	121	6,33	5,5	440	345	132	1060	564	4956	114	1002
20	48,3	241,5	294	231	12,08	10,5	840	659	252	2024	684	5920	138	1197
30	71,3	356,5	434	341	17,83	15,5	1240	973	372	2988	804	6884	163	1392
40	94,3	471,5	574	451	23,58	20,5	1640	1287	492	3952	924	7848	187	1586
50	117,3	586,5	714	561	29,33	25,5	2040	1601	612	4916	1044	8812	211	1781

haulage of ore, 2,3 Tg, distance in addition to 5 km

additional energy requirements

assume overburden ratio (=strip ratio) = 3

sum J(th) = 386 TJ

a	direct	+sum
km	J(th)	J(th)
	TJ	r = 3

0	0	386
5	11	397
10	22	408
20	44	430
50	110	496
100	220	606
200	440	826
300	660	1046
400	880	1266

10 km additional haul distance ore to mill adds 408 TJ (th)
per Mg U: 408/4947 = 82,5 GJ/Mg U

milling: fixed energy requirements, m0 = 2.3 Tg

	J(e)	J(th)
	TJ	TJ
crushing and grinding	170	
leaching	97	528
solvent extraction	12	
drying wet cake and calcination		1123
water treatment		
indirect, ore processing (mill)	54	404
sulphur		1200
sulphuric acid production		278
sodiumchlorate *	58	66
ammonia	39	54
extraction grade kerosene		
complexing agent		
lime	1,8	219
sum	432	3872
transport chemicals 200 km		24
total		3896

(Ranger)

Mining energy requirements

Based on the process analysis of the Ranger mine (file: Ranger4.xls) the generic energy requirements of uranium mining activities can be approximated.

Table D.11
Energy input of uranium mining at the Ranger mine

	Item	Energy electric	Energy thermal	unit
	Direct energy input, excluding explosives			
1	Drilling and blasting	5		MJ/Mg
2	Excavation		14.6 5.84	MJ/m ³ MJ/Mg (1)
3	Haulage		1	MJ/Mg/km
	Indirect energy input (1)			
4	Drilling and blasting, excluding explosives		12	MJ/Mg
5	Excavation + haulage		38 15.2	MJ/m ³ MJ/Mg (2)
6	Explosives		18	MJ/Mg (3)

(1) Energy consumption of maintenance, chemicals and auxiliary materials. Energy requirements of construction and transport of the equipment (trucks, excavators, etcetera) are not included.

(2) Assumed mean density of rock $d = 2.5 \text{ Mg/m}^3$

(3) Based on Ranger usage: 0.25 kg explosives per Mg rock. Specific (thermal) energy input of explosives $J = 71 \text{ MJ/kg}$.

Assuming a minimum haulage distance of $s = 10 \text{ km}$, the minimum energy input of mining one Mg rock (ore or overburden) amounts to:

$$J_{\text{mining}} = 66.0 \text{ MJ/Mg} \quad R = J_{\text{th}}/J_{\text{e}} = 12 \quad \text{eq D.14}$$

Or as function of the haulage distance:

$$J_{\text{mining}} = (56.0 + s) \text{ MJ/Mg} \quad R = J_{\text{th}}/J_{\text{e}} = 12 \quad \text{eq D.15}$$

with $s =$ haulage distance in km.

Table D.12

Comparison of the specific recovery energy requirements and carbon intensity of the Ranger uranium mine in Australia and of a reference mine with the same ore grade in this study. The Ranger ore is assumed to be soft.

	Ore grade % (w/w) U ₃ O ₈	Specific energy requirements TJ/Mg Unat	Carbon intensity gCO ₂ /Kwh
Ranger 2005	0.288	1.08	1.78
Reference mine	0.288	0.97	1.46

D8 Olympic Dam

Olympic Dam, in the south of Australia, is the largest uranium deposit known in the world. Today it is an underground mine with copper as main product and uranium as by-product. In addition relatively small amounts of gold and silver are extracted from the large mineralisation. The owner of Olympic Dam, BHPBilliton, is considering to convert the underground mine into a huge open pit mine. In this Part D8 the so-called Olympic Dam Expansion, or ODX, is addressed.

Uranium reserves and resources of Olympic Dam

The uranium resources and reserves of Olympic Dam as published by the IAEA, OECD/NEA [Q90] and UIC [Q211] are listed in Table D.13. The corresponding uranium content, mass of ore and volume of the ore are added by the author. To calculate the volume of the corresponding ore bodies an average density of $d = 2.7 \text{ Mg/m}^3$ is assumed (the average density of granite).

In its Annual Report 2007 [Q361] BHPBilliton published new, greatly upgraded figures. We return to the new data in a later section. The next sections are to show the close relationship between the reported resource figures and some basic arithmetic based on other data from BHPBilliton.

Table D.13

Uranium reserves and resources of Olympic Dam. The data of the first three columns are from Red Book 2006 [Q90] and UIC-emine 2005 [Q211].

The volume of the ore is based on an assumed density of $d = 2.7 \text{ Mg/m}^3$.

Olympic Dam	$m(\text{U}_3\text{O}_8)$ Mg	G % U_3O_8	$m(\text{U})$ Gg	$m(\text{ore})$ Tg	$V(\text{ore})$ 10^6 m^3
Proven reserves	71000	0.06	60	118	44
Probable reserves	321000	0.05	272	642	238
Measured resources	325000	0.05	276	650	241
Indicated resources	568000	0.04	482	1420	526
Inferred resources	522000	0.03	443	1740	644
Sum Olympic Dam	1807000		1532	4570	1693

Table D.14

Various combinations of resources and reserves of Olympic Dam

Olympic Dam	$m(\text{U})$ Gg	$m(\text{ore})$ Tg	$V(\text{ore})$ 10^6 m^3
Inferred + indicated resources	924	3160	1170
Inferred + indicated + measured resources	1200	3810	1411
Inferred + indicated + measured resources + probable reserves	1472	4452	1649

Data from BHP Billiton

Quoted from source: ODX 2007 [Q354]:

Excavating the 'Starter Pit', with a depth of 350 m and a diameter at the surface of 1700 m, will take four years at a movement of estimated 350 million tonnes overburden a year.

Once completed, the expanded mine will produce around 40 million tonnes of ore to the new processing plant each year.

The pit is in flat lying sedimentary cover including limestone and sandstone. At a depth of 350 m the ore (Breccia Complex) will be exposed.

The final depth of the Starter Pit will be 450 m.

15-18 m benches

65 m wide ramps (2 lanes up for traffic management).

Estimated 100 000 tonnes per day of ore will be transported to the processing plant once commissioned.

Existing underground and the proposed open-cut mining operations will run concurrently for a number of years.

The ultimate open pit will be about 1200 m deep and with dimensions at the surface of about 3.5 by 4.5 km.

Processing capacity. Current capacity close to 10 million tonnes ore a year.

The ODX will result in an expected working life extension up to 70 years or more.

Power. OD is connected to the state electricity supply grid with an average total load of 120 MW. The proposed development may require 300-400 MW of additional power.

Water. The current operation uses about 32 ML (million liters) a day (= 32 000 m³/day). The proposed development requires about 120 ML/day (= 120 000 m³/day) of additional water. Feasibility investigations are underway of a coastal desalination plant ((Upper Spencer Gulf) to meet long-term water demand.

Transport. About a million tonnes of material is currently transported to and from Olympic Dam each year by road. The proposed expansion is likely to increase this to about 2.2 million tonnes of material per year.

Geology

Following initial underground development in 1985-1987, it became apparent that none of the mineralisation was stratabound and that it was hosted in a highly variable magmatic-hydrothermal breccia complex that displays multiple brecciation, diatreme intrusions, mafic and ultramafic dyke intrusions along with interpreted gravity collapse of high level volcanic edifice material. The deposit is considered to be a member of the Iron Oxide Copper Gold (IOCG) family of deposits and has a close temporal and spatial association with the Hiltaba Suite of granites, dated at 1590Ma and more specifically the more fractionated and oxidised Roxby Downs subsuite (1588Ma).

The main lithologies at Olympic Dam comprise a continuum of breccias starting at the

periphery with granite clasts set in little haematite matrix progressing into the centre where the clasts are wholly haematite in a haematite matrix. The principal gangue minerals are haematite, sericite and quartz. Minor gangue minerals include; siderite, chlorite, fluorite and barite.

The dominant sulphide minerals are chalcopyrite, bornite, chalcocite and pyrite. Minor sulphides include; carrollite, cobaltite, galena, sphalerite and molybdenite. Other minerals of interest are; metallic copper, electrum, Ag-, Hg-, Pb-, Bi-selenides and tellurides. The uranium at Olympic Dam occurs mostly as uraninite, coffinite or brannerite with trace amounts in zircon, monazite, florencite and bastinite.

Assumptions in this study

The text published by BHP Billiton [Q354] gives only a few details, so a number of assumptions must be made to estimate the amounts of materials:

- The overburden consists of sedimentary rock, including limestone ($d = 2.68\text{-}2.76 \text{ Mg/m}^3$) and sandstone ($d = 2.14\text{-}2.36 \text{ Mg/m}^3$, according to Handbook of Chemistry and Physics 75th ed). The density of the waste rock is assumed to be $d = 2.5 \text{ Mg/m}^3$.
- The mean density of the ore is assumed to be $d = 2.7 \text{ Mg/m}^3$ (the average density of granite).
- The average slope angle of the pit is assumed to be 45° (see Figure D.7). Much steeper or much shallower slopes seem not likely, judging from photographs of other open-pit mines, including BHP Billiton's Escondida mine in Chile.

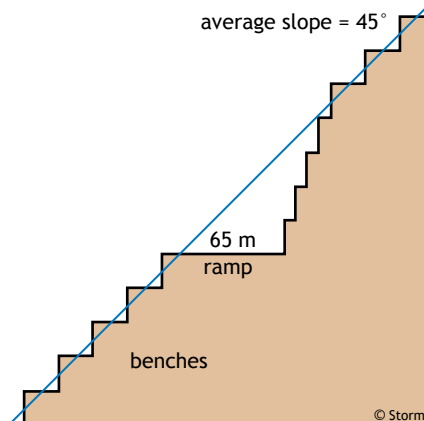


Figure D.7

A possible configuration of the ODX open pit mine, with benches of 18 m high and 18 m wide. The average slope angle of the pit is 45° .

- The haul distance is difficult to estimate. If we assume an average slope of the ramp to the surface of 10%, the minimum haul distance from the bottom to the surface of the starter pit would be some 4.5 km and of the final pit some 12 km.

The total haul distance from the pit bottom to the waste rock dump area is assumed to

be at least 20 km for the starter pit and 30 km for the final pit. The empty return trip of the dump trucks is considered to be equivalent to some 5 km loaded and is included in above figures.

- The final pit is assumed to encompass the complete mineable ore body.
- To approximate the maximum amount of ore in situ, the mineable ore body is assumed to be a homogeneous ellipsoid with semi-axes a , b and c . The values of the semi-axes are determined by the dimensions of the final pit, and are graphically derived (see Figure D.9).



Figure D.8

BHP Billiton's Escondida copper mine in Chili (source: [Q354]). The future appearance of Olympic Dam?

Possible geometry of ODX

Based on above assumptions a possible geometry of the ODX can be conceived, as illustrated by Figure D.9. The pits are considered to be inverted truncated cones: the starter pit circular and the final pit elliptical. The volume of a frustum can be calculated by the equation D.16 in Figure D.10. The volumes of the three open pits, as illustrated by Figure D.9, and the volumes of the rock to be excavated, are compiled in Table D.15.

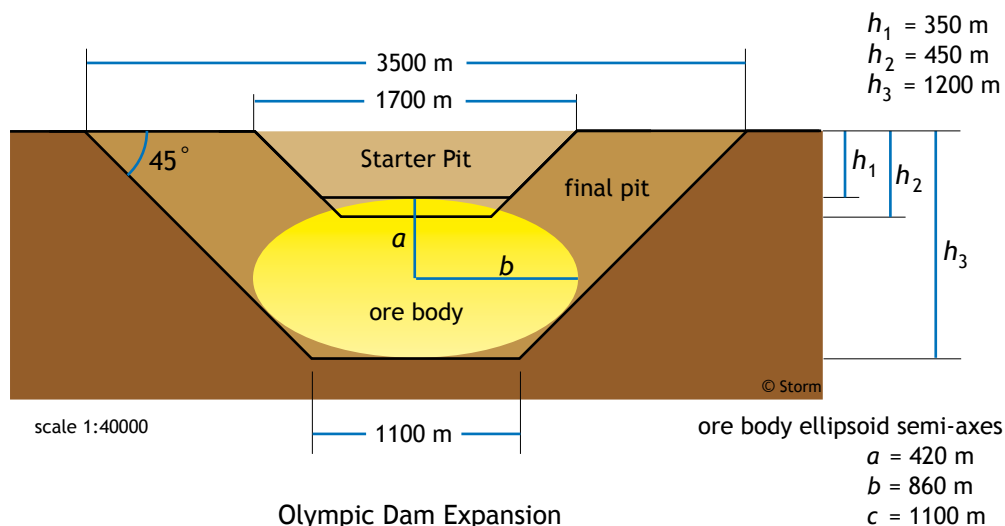
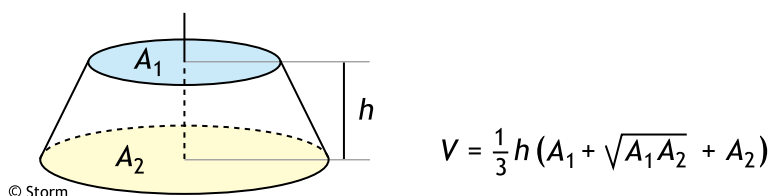


Figure D.9

Estimated geometry of the starter pit, final pit and the mineralization body of the Olympic Dam mine expansion.



eq D.16

Figure D.10

Truncated cone (frustum) and its volume. The top and bottom areas can be either circular, elliptical (provided both ellipses have the same a/b ratio) or polygonal (truncated pyramid). A_1 and A_2 are the areas of the top and bottom of the frustum respectively.

Table D.15

Dimensions and volumes of the proposed open pits of ODX. The figures in the columns h , a_1 and b_1 are from BHP Billiton, the last 5 columns are based on above assumptions.

pit	h m	a_1 m	b_1 m	a_2 m	b_2 m	A_1 10^6 m^2	A_2 10^6 m^2	V 10^6 m^3
Starter pit	350	850	850	500	500	2.270	0.785	512
Starter pit	450	850	850	400	400	2.270	0.503	576
Final pit	1200	1750	2250	550	1050	12.37	1.814	7569

To approximate the maximum volume of an ore body fitting within the boundaries of the final pit we assumed it to be an ellipsoid. The volume of an ellipsoid with semi-axes a , b and c can be calculated by equation D.17:

$$V_{\text{ellipsoid}} = \frac{4}{3} \pi a b c \quad \text{eq D.17}$$

From Figure D.9 the semi-axes of the assumed ellipsoid are deduced, being: $a = 420$ m, $b = 860$ m and $c = 1100$ m.

The volume of the thus estimated ore body is $V_{\text{ore}} = 1.66 \cdot 10^9 \text{ m}^3$, or 1.66 km^3 .

At a depth of 350 m the ore body will be exposed (BHP Billiton). We assume the volume difference ($\Delta V = V_2 - V_1$) of pits 1 and 2 containing ore (see Table D.16). Figure D.9 shows this is a rough approximation of the segment of the ore ellipsoid falling within that part of pit 2. As the ore body is a highly variable complex no better estimate seems possible.

From Table D.16 the overburden ratio (or stripping ratio) can be found: see Table D.17.

Table D.16

The four main parts of the ODX rock bodies, as estimated in Figure D.9.

part	Depth m	designation	V 10^9 m^3	$m(\text{content})$ 10^9 Mg	d Mg/m^3
Pit 1	350	V1	0.512	1.28	2.5
Pit 2	450	V2	0.576	1.44	2.5 *
Pit 3	1200	V3	7.57	18.9	2.5 *
Ore ellipsoid		V4	1.66	4.49	2.7
First ore		V2 - V1	0.064	0.173	2.7
Overburden final mine		V3 - V4	5.90	14.8	2.5

* Supposed this volume would consist of waste rock only. A part of this volume comprises ore, so the real mass will be higher. The content of pit 2 would have a mass of $m_2 = 1.28 + 0.17 = 1.45 \cdot 10^9 \text{ Mg}$ and of pit 3: $m_3 = 14.8 + 4.5 = 19.3 \cdot 10^9 \text{ Mg}$.

Table D.17

Overburden ratio (stripping ratio)

• First ore	volume	$r_V = 0.512/0.064 = 8.0$
	mass	$r_m = 1.28/0.160 = 8.0$
• Final ore	volume	$r_V = 5.90/1.66 = 3.6$
	mass	$r_m = 14.8/4.49 = 3.3$

The volume and mass of the estimated ore ellipsoid come strikingly close to the official figures from Table D.14, see Table D.18. The results indicate that the conceived geometry of ODX in Figure D.9 may be not far from the truth.

Table D.18

Comparison of the geometrically estimated ore body mass and volume with the figures deduced from Q90 and Q211.

Olympic Dam Expansion	$m(\text{ore})$ Tg	$V(\text{ore})$ 10^6 m^3	source
Estimated ore body ellipsoid	4494 *	1664	this study
Inferred + indicated + measured resources + probable reserves (see Table D.14)	4452	1649 *	Q90 + Q211

* Assumed average density of the ore $d = 2.7 \text{ Mg/m}^3$.

Revised resource data of Olympic Dam (September 2007)

In its Annual Report 2007 [Q361] BHPBilliton published greatly upgraded figures of its resources, see Table D.19.

Table D.19

Uranium reserves and resources of Olympic Dam, according to BHPBilliton 2007 [Q361]. The volume of the ore is based on an assumed density of $d = 2.7 \text{ Mg/m}^3$.

Olympic Dam	$m(\text{ore})$ Tg	G % U_3O_8	$m(\text{U}_3\text{O}_8)$ Gg	$m(\text{U})$ Gg	$V(\text{ore})$ 10^6 m^3
Proved ore reserve	61	0.063	38.4	32.6	23
Probable ore reserve	339	0.057	193.2	163.9	126
total reserve	399		232	196	148
Measured resource	1311	0.036	472	400	486
Indicated resource	3129	0.029	907	770	1159
Inferred resource	3298	0.026	857	727	1221
total resource	7738		2237	1897	2866
Sum Olympic Dam	8538		2469	2093	3014

Table D.20

Uranium ore reserve and resource of OD, about fitting within the dimensions of Figure D.9 and Table D.18.

Olympic Dam	$m(\text{U})$ Gg	$m(\text{ore})$ Tg	$V(\text{ore})$ 10^6 m^3
Total reserve	196	339	148
Measured resource	400	1311	486
indicated resource	770	3129	1159
sum	1366	4779	1793

Very likely the figures published on the dimensions of ODX in the first half of 2007 were based on the most recent figures of reserves and resources. As Table D.20 shows, the sum of the reserve, measured resource and indicated resource comes close to the figures in Table D.18. If so, this would imply that the plans for ODX exclude the Inferred Resource, the least assured and leanest ore resource. This part of the ore body may lay deeper or may extend horizontally. In both cases exploitation of that part might require a far larger open pit and so a larger overburden ratio. The energy consumption of mining and milling per kg uranium of that part may rise substantially.

Energy consumption and CO₂ emission of the uranium extraction from ODX

The main product of Olympic Dam is copper, uranium is by-product as are gold and silver. That may imply that the separation processes following the mill are optimized to extract copper. The consumption of energy and auxiliary materials by OD to run the separation processes surely will be higher than if uranium were the only product. Without detailed data on the separation processes it is difficult to estimate the fraction of the energy and material consumption which should be attributed to the uranium extraction. Mudd & Diesendorf 2007 [Q338] adopt a fraction of 20%, based on the average proportion of revenue from uranium at OD. From a physical/chemical point of view this criterion may not be very solid, but without sufficient process data there seems to be no other choice.

To get an impression of the energy quality of the ore of OD, we assume it to be exclusively a uranium ore. The calculation of the energy consumption and CO₂ emission are based on the specific energy requirements listed in Table D.21, which are used in this study. The process analysis of the Ranger mine (see Part D7) shows that the used values may lead to a slight underestimation, rather than overestimation.

Table D.21

Specific energy requirements of mining and milling hard uranium ores, considered to be world average values.

mining	$J_{th} + J_e = 1.06 \text{ GJ/Mg ore}$	$R = J_{th}/J_e = 8.0$
milling hard ores	$J_{th} + J_e = 4.49 \text{ GJ/Mg ore}$	$R = J_{th}/J_e = 0.1$
reclamation mine	$J_{th} + J_e = 4.2 \text{ GJ/Mg ore}$	$R = J_{th}/J_e = 8.0$

The thermal inputs of the processes are assumed to be supplied as diesel fuel. The specific CO₂ emission of burning diesel is taken at 75 g/MJ (heat), or 0.075 Tg/PJ.

In Table D.22 the energy requirements, thermal and electrical separately, and CO₂ production of the exploitation of the complete ore body of Olympic Dam (8538 Tg of ore) are listed. Evidence from the past shows that the nuclear industry tends to easily adopt favourable figures, how insecure they may be, as facts to base on their prognoses.

In the last three columns (E, F and G) of Table D.22 the CO₂ production are calculated, assumed that the electrical component of the energy consumption is generated by diesel-fuelled power plants, with an electrical/thermal efficiency of 40% (which may be a high estimate).

Table D.22

Energy requirements and CO₂ production of the mining and milling of the full mass of ore of Olympic Dam, amounting to 8538 Tg, plus mine reclamation. The columns E, F and G refer to the case the electrical component is supplied by diesel-fuelled power generation.

8538 Tg ore process					Assumed diesel-generated electricity, efficiency = 0.40		
	A	B	C	D	E	F	G
	$E_{th}+E_e$ PJ	E_{th} PJ	E_e PJ	$m(\text{CO}_2)$ $B \cdot 0.075$ Tg	$E_e \rightarrow E_{th}$ $C/0.40$ PJ	$m(\text{CO}_2)$ $E \cdot 0.075$ Tg	$m(\text{CO}_2)$ $D + F$ Tg
Mining	9050	8045	1006	603	2514	189	792
Milling	38336	3485	34851	261	87126	6535	6796
Sum mine+milling	47386	11530	35856	865	89640	6723	7588
Reclamation mine	35860	31875	3984	2391	9961	747	3238
Sum m+m+reclam	83246	43405	39841	3255	99601	7470	10725

To give an impression of the electric power consumption of the mining and milling, excluding mine reclamation, the following example. Assume an operational life of 100 years of ODX, then the average electricity consumption would be 359 PJ a year, corresponding with a continuous power consumption of 11.4 GW.

As pointed out above, the total energy consumption of ODX will be significantly higher than the figures from Table D.22, because the extraction and processing of copper, gold and silver are not included. The energy consumption to be attributed to the uranium extraction alone might be considerably lower than the figures of Table 22.

Energy potential from ODX

How much electricity could be generated from the uranium ore reserves and resources? The answer is not as simple as it may seem, for the amount of uranium extractable from the 8538 Tg of ore depends on the extraction efficiency, which in turn depends on several factors, such as ore grade and mineralogy of the uranium in the ore.

The mineralogy of the uranium in the ore body of ODX tends to become increasingly refractory with depth, due to an increasing proportion of the refractory uranium minerals brannerite and coffinite, and a decreasing proportion of the easily processible mineral uraninite.

uraninite = UO₂

coffinite = U(SiO₄)_{1-x}(OH)_{4x}

brannerite = (U,Ca,Ce)(Ti,Fe)₂O₆

A higher fraction of refractory minerals implies higher consumption of energy and materials per kg recovered uranium and lower extraction yield due to greater losses. In addition to a decreasing ore grade with depth, the ore gets harder to process. Both effects cause a declining energy quality of the ore with increasing depth.

In 2006 10 million tonnes of ore have been processed at a uranium grade of 0.06% U₃O₈ (proven reserves), with an uranium production of 3916 Mg U₃O₈ (BHP Billiton

Annual Report 2006). This figures point to a net extraction efficiency of 0.66 (66%). The historical extraction efficiency for uranium of OD is 0.653 and is expected to decline to 0.50-0.40 for ODX, according to Mudd 2007 [Q362].

Likely, the historical value of $Y = 0.653$ refers to the proven reserve of OD, with an ore grade of $G = 0.063\% \text{ U}_3\text{O}_8$, for these ore are mined first. In Figure D.11 two curves, A and B, are added to the Y -log G curve which is used in this study to assess the world uranium resources. Both curves start from the emprirical point at $G = 0.063\% \text{ U}_3\text{O}_8$. The extraction yields at OD will go down when the average ore grade decreases.

Curve A represents the same relationship between yield and grade as the blue main curve in Figure D.11, the only difference being the lower starting point.

Curve B is based on the assumption that the yield Y declines to 0.40 at $G = 0.026\% \text{ U}_3\text{O}_8$. There are reasons to suppose a more steeply declining yield at lower grades than curve A, in view of the increasingly refractory character of the ore when going to deeper and poorer deposits.

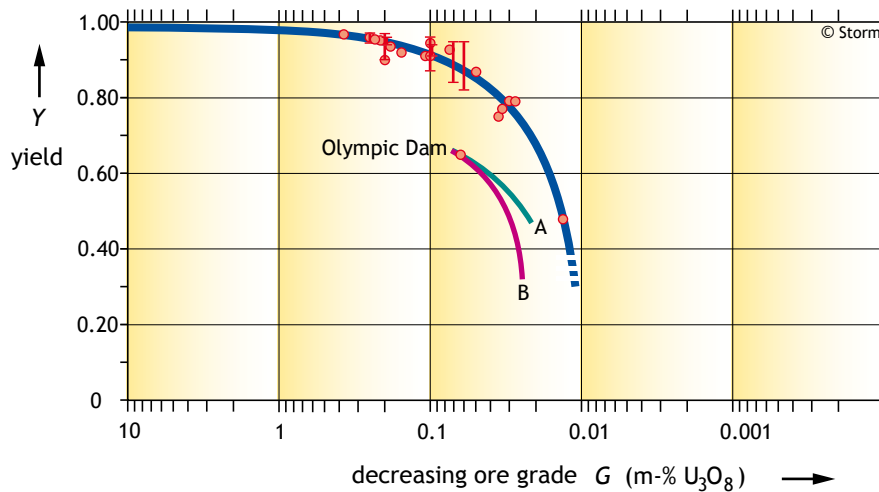


Figure D.11

Extraction yields of Olympic Dam. The value at an ore grade of $G = 0.0653\% \text{ U}_3\text{O}_8$ (red dot) is the historic mean yield. Curves A and B represent two possible relationships between yield and lower ore grades and are explained in the text. The upper (blue) curve represents the Y -log G relationship as used in this study (see Part D3).

In Table D.22 the recoverable amounts of uranium from the total reserve en resource of OD are listed. Column B gives the in situ amounts of uranium, column D gives the recoverable amounts assumed the yield declines according curve A and column F is based on curve B in Figure D.11.

The values in columns D and F should be considered speculative, as no empirical data are available.

The reference reactor in this study (see Part B), which comes close to the Generation III reactors, consumes 162.35 Mg natural uranium per reload period. During one reload period D (corresponding with one year at a mean load factor of 0.82) it produces a fixed gross amount of electricity of:

$$E_{\text{gross}} = 25.86 \text{ PJ/D} = 7.183 \cdot 10^9 \text{ kWh/D.}$$

Obviously, Olympic Dam will ultimately deliver less than 2093 Gg uranium, and likely

less than 1156 Gg. An amount of close to 996 Gg seems more realistic. For comparison: the annual uranium consumption rate of today's world nuclear fleet is about 68 Gg/a.

Table D.22

Masses of recoverable uranium from ODX, calculated at different extraction efficiencies Y . Column C is based on the yield curve A in Figure D.11 and column E on curve B.

Olympic Dam	A	B	C	D	E	F
	G % U_3O_8	<i>in situ</i> $m(U)$ Gg	Y curve A	D=B•C recovered $m(U)$ Gg	Y curve B	F=B•E recovered $m(U)$ Gg
Proven reserves	0.063	32.6	0.653	21.3	0.653	21.3
Probable reserves	0.057	163.9	0.640	104.9	0.635	104.1
total reserve		196.4		126.2		125.3
Measured resources	0.036	400.2	0.580	232.1	0.545	218.1
Indicated resources	0.029	769.5	0.545	419.4	0.470	361.7
Inferred resources	0.026	727.1	0.520	378.1	0.400	290.9
total resource		1896.8		1029.6		870.6
Sum Olympic Dam		2093.3		1155.8		996.0

Table D.23 compiles the gross electricity production possible from the amounts of recovered uranium (see columns D and F of Table D.22), and the CO_2 emissions per kilowatt-hour. Columns C and D of Table D.23 give the specific CO_2 emission if only the fossil fuel input is taken into account, as is done throughout this study (see Part C). In columns E and F of Table D.23 the CO_2 emission is given when all electric energy inputs were to be generated from diesel fuel, with a thermal to electric efficiency of 40% and a specific CO_2 emission of 75 $gCO_2/MJ(th)$. The last column gives the sum of both components.

Table D.23

Specific CO_2 emissions of the recovery of uranium of Olympic Dam at two different recovery yield curves. The energy consumption and CO_2 emission is fully attributed to the recovery of uranium. For explanation: see text.

$m(U)$ Gg	A	B	C	D	E	F	G
	n number of reloads D	Gross E product. 10^9 kWh	th $m + m$ CO_2 g/kWh	th $m+m +$ reclam CO_2 g/kWh	e > th $m+m$ CO_2 g/kWh	e > th $m+m +$ reclam CO_2 g/kWh	th+e>th $m+m +$ reclam CO_2 g/kWh
1156	7119	51135	17	64	131	146	210
996	6135	44065	20	74	153	170	243

In Table D.24 the CO₂ emission of the full nuclear chain is calculated, based on uranium from OD. In this table the energy consumption of mining and milling and mine reclamation is fully attributed to the uranium extraction, ignoring the production of copper, silver and gold. The total uranium production is assumed to be 996 Gg.

In Case A only the CO₂ emissions due to the thermal energy inputs of the nuclear chain are listed. In Case B the electrical inputs of OD are supposed to be generated by diesel-fuelled power stations. The electrical inputs of the other components of the chain are not converted and assumed to be supplied by nuclear power in a steady state (see also Part C).

Only the term of the 'repayment' of the energy debt - construction and decommissioning - depends on the operational lifetime of the nuclear power plant. In this part an operational lifetime of 30 years at an average load factor of 0.82 is assumed. The other energy inputs with their CO₂ emissions are running inputs and are constant per reload period.

Table D.24

The CO₂ emission of the full nuclear chain based on uranium from ODX, if all energy consumption is attributed to the uranium extraction. Assumed uranium production of 996 Gg. The term of 'repayment' of the energy debt depends on the operational lifetime of the nuclear reactor, here assumed to be 30 years at a mean load factor of 0.82.

Part of the nuclear chain	CO ₂ emission (g/kWh)	
	Case A thermal input only	Case B thermal + electrical input *
mining + milling ODX	19.6	172.2 *
front end, excluding mining + milling	5.8	5.8
operation + maintenance + refurbishments	24.4	24.4
back end, excluding reclamation mine	11.3	11.3
reclamation mine ODX	54.3	71.2 *
'repayment' energy debt (average)	58.0	58.0
sum (rounded)	173	343

* Assumed the electrical inputs of OD (mining + milling and mine reclamation) are generated by diesel-fuelled power plants.

D9 Uranium from unconventional resources

Uranium from phosphates

Phosphate deposits may be classified according to origin into the two main categories of sedimentary and igneous phosphate rocks, which, quite apart from their distinct characteristics as source of phosphate, also differ in terms of uranium content. Sedimentary deposits of marine origin have a uranium content in the range 40-300 ppm, and the igneous phosphate ores contain typically 10-100 ppm. The largest phosphate deposits of the world are in Morocco (55% of the world resources) and have an uranium content varying from 70-230 ppm (Bergeret 1979 [Q47]). Nearly all other known deposits contain less than 180 ppm uranium, most of them around 100 ppm or less (0.01% or less). Deffeyes & MacGregor 1980 [Q281] place the bulk of the phosphates in the grade range of 10-30 ppm.

To produce each year 68 Gg uranium, the current world uranium consumption rate, a theoretical minimum mass of 680 Tg/a phosphate rock must be processed at a uranium content of 100 ppm. The actual mass certainly will be much higher, due to an extraction efficiency considerably lower than 100%. At an extraction yield of some 30% (see below), about 2000 Tg/a phosphate rock must be processed.

The world production of phosphate rock in 2006 was about 145 Tg. The world phosphate reserves are 18000 Tg, enough for 9-10 years of uranium supply at the current consumption rate. The world reserve base of phosphate is some 50 000 Tg, enough for another 25 years of uranium supply.

Even if the average uranium content would be twice as high as in this example, or if the extraction yield would be twice as high, the phosphate reserves would last roughly 20-50 years at the current consumption rate.

The amount of phosphate processing needed to meet the world uranium demand is about 14 times the rate needed to supply the phosphorus demand for agriculture. So the energy consumption of the mining, milling and processing of phosphate rock should be attributed fully to uranium. At a grade of 100 ppm (0.01% U) phosphate rock is beyond the energy cliff (see Part G), so phosphates cannot be regarded a net energy source.

Another issue is the application of phosphate rock as source of agricultural fertilizer. There are no substitutes for phosphorus in agriculture. So the exploitation of phosphate rock should be geared to the demand for food production.

Extraction

Extraction of uranium from phosphate ores is not simple. Because of the low uranium content, the extraction of uranium has to be combined with the production of phosphoric acid, an essential chemical feedstock, e.g. for the production of fertilizers. The first stage of the process, the digestion of the ore, has to be carefully controlled to ensure a sufficiently high rate of extraction of the uranium from the phosphate rock into

the phosphoric acid. Uranium is present in phosphate rock as a solid solution, without separate uranium mineral grains. The whole mass of phosphate rock has to be brought into solution to make uranium extraction possible.

After the acidulation process the raw phosphoric acid has to be purified and concentrated in uranium. From that concentrated solution, the uranium is extracted with solvents, highly toxic organophosphorus compounds. The process has high losses, partly due to reactions with hydrofluoric acid HF, needed in the extraction process. Reactions between HF and the organophosphorus compounds may yield very toxic organofluorophosphorus compounds. In addition the formation of potent greenhouse gases is conceivable. No data are available on these aspects

To achieve a high efficiency of the extraction of uranium from the concentrated solution, a maximum of 70% is believed to be possible, the process conditions should be controlled within very narrow limits.

The overall yield of the extraction of uranium from the phosphate ore is difficult to estimate, because the yield of the each stage in the process is very susceptible to various process conditions. If a yield of some 70% is assumed in each of the three stages of the uranium extraction process, the overall yield would be around 30%.

A quantitative example, based on above yield values.

Processing a phosphate rock at a uranium content of 100 ppm, with sulfuric acid results in a 30% solution of raw phosphoric acid with an uranium content of some 21 ppm. If the recovery (extraction yield) in the following two process stages is about 50%, around 10 gram uranium is recovered from each Mg solution. To produce one kilogram uranium, some 100 Mg solution (more than 80 m³) have to be processed with extraction solvents.

The high production costs, 50-200 \$/lb U₃O₈ (300-1200 \$(2000)/kg U), according to Bergeret 1979 [Q47], indicate a high specific energy and materials consumption even in coproduction with phosphoric acid.

Using these cost estimates and the energy-intensity of the average economic activity, $e = 10.6 \text{ MJ}/\$(2000)$, we find:

$$J_{\text{phos}} = 3.2-12.8 \text{ GJ/kg (U)} \quad \text{eq D.18}$$

Above figures are comparable with the energy requirements of conventional mining and milling of soft ores, but may be underestimated, because:

- the cost estimate refer to uranium as by-product
- the chemical industry is more energy-intensive than the average economic activity
- the production costs cited by Bergeret likely will not be overstated.

Uranium from shales

Organic-rich shales often contain uranium as well. Well-known examples are the Chattanooga shales in the USA and the Ranstadt black shales in Sweden. The Chattanooga shales consist of two principal layers, each with a variable thickness of 1-2 meters [Q135]. The richest layer has an average grade of 60 ppm uranium and contains some 3 million Mg uranium. The other layer has grades in the range 20-50 ppm and contains some 4.5 million Mg uranium. The Ranstadt shales have grades up to 300 ppm and contain some 300 000 Mg uranium, according to Rotty et al.1975 [Q95].

Extraction of uranium from shales would be done in conventional mining and milling processes. Grades in uranium-rich Chattanooga shale in the USA vary from 20-50 ppm in one principal layer to an average of 60 ppm in the richest layer. The Ranstadt shales in Sweden have grades up to 300 ppm. According to Deffeyes & MacGregor 1980 [Q281] black shales generally have a uranium content of 10-100 ppm.

A study by the US Brureau of Mines (BureauMines 1976 [Q135]) recommended further work that should concentrate on:

- investigating into the net change in energy balance that will occur if and when lower grade ores (60 ppm rather than 2000 ppm uranium) are used as a source of nuclear fuel
- developing more geologic data on the Chattanooga formation
- determining the sociological and environmental impact that will occur if the shale is to be mined and procesed for its uranium.

As the authors of the study put it:

“Currently, steady-state Light Water Reactors (LWR) have an energy input-output ratio of approximately 1 to 15 when using uranium derived from high-grade deposits. If the Gassaway [. . Member of the Chattanooga Shale . .] were utilized (60 ppm uranium) in the absence of more economical uranium deposits, this ratio would change. A preliminary study is needed to determine the increased ammount of energy input needed to mine, mill, and process the Chattanooga shale versus higher grade deposits and analyze how this would affect the energy ratio of LWR's.”

From processing point of view shales can be taken as soft ores.

Mining and milling of shales would be conventional processes. The Chattooga shales would be mined underground. No pilot plant operations have been undertaken. In laboratory tests a recovery of uranium from the ore of 34-80% are achieved, depending on method, amount of reagent and number of extraction steps [Q135]. No data are given on the recovery from the resulting solution. So, the overall yield is unknown, but may be comparable to that from phosphates.

The extraction of 162.35 Mg of uranium - needed to feed the reference reactor for one year - from shale at a grade of 60 ppm and an assumed recovery yield of 30% , would imply mining, milling and processing of about 9 Tg/a of shale. A coal fired plant consumes about 2 Tg/a of coal to generate the same amount of electricity.

Uranium from granite

The uranium content of granitic rocks typically varies from 3-20 ppm U (0.0003-0.002% U or 3-20 gram U per Mg rock), with an average value of 4 ppm (0.0004%). Roughly, the average composition of granite is 80% SiO₂ and 20% Al₂O₃. For the mining and milling process granite is a hard 'ore'.

To fuel the reference reactor, each year about 162 Mg natural uranium has to be extracted from the earth's crust. The mass of 162 Mg uranium is present in 40 Tg of granite, with an average uranium content of 4 gram U per Mg rock. The rock has to be dug up, ground to fine powder and chemically treated with sulfuric acid and other chemicals to extract the uranium compound from the mass. Assumed an extraction yield of $Y = 0.50$, an optimistic assumption, 80 Tg granite have to be treated. This is a block 100 meters wide, 100 meters high and three kilometers long. Each year, for one reactor.

For comparison: a coal-fired power station of 1 GW(e) consumes about 2 Tg of coal each year.

Substituting $c = 0.654$ GJ/kg(U) for hard ores, we find a total energy (thermal + electric) requirement of:

$$J_{th} + J_e = 3.27 \text{ TJ/kg(U)}$$

As the ratio thermal/electric energy is $R = 1.6$ for hard ores, the electric requirements alone are:

$$J_e = 1.26 \text{ TJ/kg(U)}$$

To extract 162 Mg uranium from granite about 204 PJ electricity would be consumed, plus some 326 PJ thermal energy (fossil fuels). The gross electricity production possible from this amount of uranium is 25.86 PJ, ignoring the energy requirements of the other processes of the nuclear chain.

D10 Uranium from seawater

Introduction

Seawater contains dissolved uranium, mainly as uranyltricarboxylate ions $[\text{UO}_2(\text{CO}_3)_3]^{4-}$ (aq), at an average concentration of 3.34 mg uranium per cubic meter seawater. Since the total volume of seawater of the world is about 1.37 billion km^3 , the total amount of uranium in the oceans is about 4.5 billion Mg (1 Mg = 1 megagram = 1 metric tonne). This huge uranium resource is assured, but is it also an energy resource?

In the past a limited number of studies have been published on the extraction of uranium from seawater. Some of the studies go into detail, e.g. ORNL 1974 [Q133], Burnham et al. 1974 [Q136], Mortimer 1977 [Q98], Sugo et al. 2001 [Q298] and Sugo 2005 [Q303]. Other studies are confined to only a part of the process, e.g. Koske 1979 [Q223], Saito 1980 [Q224], Burk 1989 [Q73], Nobukawa et al. 1994 [Q72] and Seko et al. 2003 [Q312]. Brin 1975 [Q225] gives a concise bibliography. The comprehensive and leading international study INFCE-1 1980 [Q226] cites results of an unspecified study by the US Department of Energy in 1978.

The INFCE study concluded that uranium from seawater is not a viable option.

Though uranium from seawater is still cited as an energy resource option for the future, e.g. MIT 2003 [Q280], few feasibility studies have been published after 1980, which is not surprising after the conclusions of INFCE in 1980.

Only in Japan some interest in extraction of uranium from seawater appears to exist today, judging by the publications of Nobukawa et al. 1994 [Q72], Sugo et al. 2001 [Q298], Seko et al. 2003 [Q312], Sugo 2005 [Q303] and JAERI 2005 [Q304].

Extraction from seawater, general outline

Technically it is possible to extract uranium from seawater. The first stage of the extraction process is the adsorption of the dissolved complex uranium ions from the seawater on solid adsorption beds. Because of the extremely low concentration of uranium and the relatively high concentrations of many other kinds of ions, very selective adsorbents are required.

After the first stage, the adsorption, a series of additional processes is needed to recover the uranium (see Figure D.12):

- cleansing of the adsorption beds to remove organic materials and organisms.
- desorption: elution of the adsorbed uranium ions from the adsorption beds with a suitable solution; depending on the type of adsorbent, this process may be a two-stage process in itself,
- purification of the eluent: removal of other desorbed compounds, which are present in far higher concentrations than uranium ions,
- concentrating the solution,
- solvent extraction of uranium from the solution with a mixture of an organic solvent and a specific complexing agent (an organophosphorus compound),
- concentrating and purification of the extracted uranium compound and conversion

into yellowcake or uraniumoxide U_3O_8 .

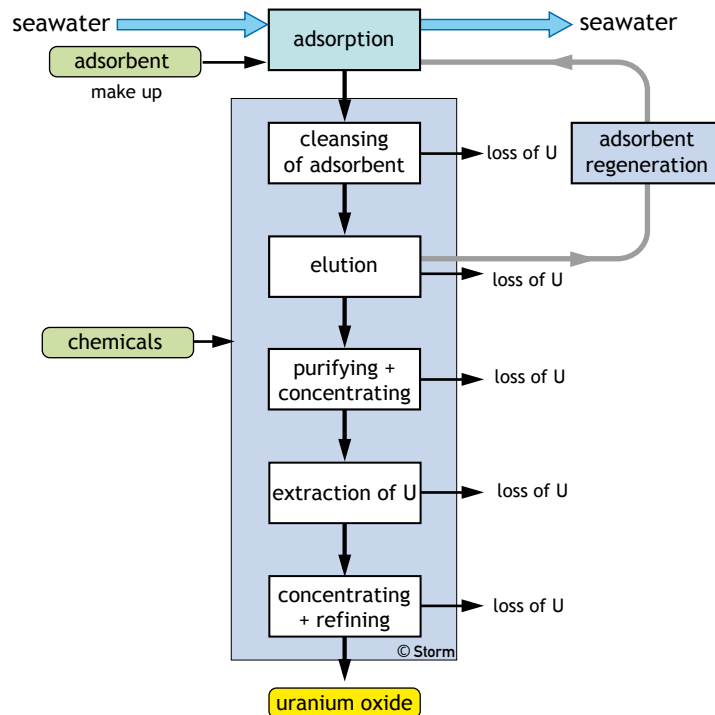


Figure D.12

General outline of the extraction process of uranium from seawater

After the elution (desorption) phase, the adsorbent has to be regenerated. A significant part of the adsorbent will be lost in the first stages of the process chain, so that it has to be replenished by fresh material in the restoration of the adsorption beds for the next cycle.

All processes have their unavoidable losses, this is an inherent feature of chemical equilibria involved in the extraction process. A five-stage process with an assumed yield of 80% of each stage, would have an overall yield of 33%. If each stage has an individual yield of 70%, the overall yield would be 17%. A rough estimate of the overall yield of a five-stage extraction process, excluding the first stage (adsorption from seawater), may be in the range of 20-30%. If more stages are needed, the total yield may be lower.

Volumes to be processed

The volumes of seawater to be processed in obtaining useful quantities of uranium are huge. The reference reactor of this study (see Part B), corresponding with the best reactors currently operating, needs 162.35 Mg natural uranium per year per GW. With an assumed overall system extraction yield of $Y = 17\%$, 285 km³ seawater per year per GW has to be processed, or 9040 m³ per second per GW. This is about 3-4 times the outflow of the river Rhine into the North Sea.

To fuel the current world nuclear power plant fleet (370 GW) a volume of about 105000 km³ seawater per year should be processed, or 3.3 million m³ per second.

Adsorption from seawater

Due to the extremely low concentration of uranium, very large volumes of seawater have to pass through the adsorption beds. With an assumed adsorption efficiency of about 0.3, 1 gram of uranium would be adsorbed from 1000 m³ seawater. The contact time of the adsorption bed with seawater, the time needed to attain a reasonable load of uranium on the adsorption beds, may vary from several days to several months. The adsorption stage therefore requires very large facilities, with dimensions measured in kilometers.

The adsorption rate strongly depends on the water temperature. Only locations in warm sea currents with water temperatures of 20 °C or more (e.g. the Gulf Stream in the Atlantic and the Black Current along Japan) are suitable to adsorption facilities, if the contact time of the adsorbent beds in seawater is to be kept within reasonable limits. This is because the establishment of the chemical equilibrium between the complex uranium ions in seawater and the bound uranium ions on the solid adsorbent phase, which is slow, is speeded up by higher temperatures. If the adsorption is an exothermic process, and probably it is, the ratio of adsorbed uranium/dissolved uranium becomes less favourable the higher the temperature, because the equilibrium shifts to the endothermic reaction (desorption) at higher temperatures. So an optimum between adsorption rate and shifting equilibrium has to be found.

Two adsorption methods are described in this paper in some detail: the titaniumhydroxide adsorption method and the polymer adsorption method. Neither of these have actually been tested other than in small-scale experiments.

Titaniumhydroxide adsorption method

This method, introduced by British studies in the 1960s and reviewed by ORNL in 1974 [Q133], is based on adsorption of the dissolved uranium-complex ions using hydrous titaniumhydroxide gel, Ti(OH)₄, on small titania spheres. The adsorbent is packed in large horizontal beds. The seawater is refreshed either by pumps in a facility on the shore, or by tidal currents in a facility on a dam in a large estuary. The pump-fed facility seemed more economical than a tidal facility.

According to ORNL, very few parameters of the British studies were backed by solid experimental data. The loading of uranium, for instance, was assumed to reach 240 mg U/kg adsorbent after 4 days immersion time, but laboratory experiments showed values no higher than about 130-150 mg U/kg adsorbent after 9 days contact time. Dependent on the production process of the adsorbent, the load values may be much lower. Considerable amounts of adsorbent are lost in the processing of the adsorbent: at least 15-65 kg titanium per kg uranium.

If an adsorption yield of 30% can be achieved, about 1 km³ seawater has to pass the adsorbent beds to get 1 Mg uranium adsorbed. Because of the low yield of the subsequent processes to recover the uranium from the adsorbent, the required amount of seawater has to be a multiple of 1 km³, to recover 1 Mg uranium.

There are many unsolved problems, such as clogging of the adsorption beds by organic materials from the sea. By the large uncertainties in a number of parameters, such as the recovery yield of the system, the feasibility of the concept will be speculative.

Polymer adsorption method

A Japanese concept (Sugo et al. 2001 [Q298], Seko et al. 2003 [Q312] and Sugo 2005 [Q303]) is based on adsorption on a fibrous polymer. This material is called a graft polymer because the active agent, consisting of amidoxime groups, is chemically bound to it via a special chemical process involving electron beams. The adsorbent sheets are packed in stainless steel cages which are moored in a warm sea current, hanging from anchored buoys. The plant to recover the uranium from the adsorbent is situated on the shore.

Seko et al. 2003 [Q312] describe adsorption experiments during the years 1999-2001 with three stainless steel cages containing 350 kg adsorbent. The calculated mass of uranium adsorbed during 12 immersion experiments was 1048 gram, involving 1800 adsorption stacks with a total dry mass of 1458 kg. This would correspond with an ideal (dry) 'uranium ore from sea' with a grade of 0.07% U. The actual 'ore grade', worked out on basis of the wet adsorbent, as it is processed by the extraction plant on the shore, would be much lower.

The authors did not mention the mass of the cages to be transported to the shore and back to the sea again in each cycle, nor the extraction yield of the uranium from the adsorbent, nor the way they calculated the mass of the adsorbed uranium.

The publication of Sugo et al. 2001 [Q298] is the most detailed one on this subject and gives hold for some practical insight. The following analysis is based on this publication.

Mooring the adsorbent at sea

In two mooring methods the cages with the adsorption beds are connected in units of 100 beds, spaced at 0.5 meter intervals, and hung under buoys (one unit per buoy) or large platforms (540 units per platform). Special crane ships hoist the beds aboard after a residence time of 60 days at sea, and bring them to the shore for processing.

In a third approach, the adsorption beds (cages) are connected in units of 10 beds, and are hung from a long horizontal line connecting anchored buoys. The line is 23 kilometers long and forms a loop. Both ends are connected with a processing plant at the coast. Recovery of the beds is done in the same manner as a ski lift: winching up one end of the rope and easing the other. At the plant the exposed beds are removed and fresh ones are attached on the outgoing line.

Table D.25

Basic parameters of one adsorption bed, from Sugo et al. [Q298]

mass adsorbent	125 kg
mass spacer	104 kg
mass stainless steel mesh container	685 kg
total mass (cage + adsorbent)	1 Mg (average dry and wet)
adsorbed uranium	2 g uranium per kg adsorbent
immersion time (= 1 cycle)	60 days
adsorbent loss replenishment	1/4 of total mass of adsorbent per annum
diameter	4 m
height	0.4 m

Hypothetical uranium extraction plant

Based on the data from Sugo et al. [Q298] some parameters of a hypothetical uranium extraction plant can be approximated, see Tables D.26, D.27 and D.28. The production capacity of this example is 10 Gg/a. The current world uranium consumption is some 68 Gg/a.

Table D.26

Parameters of a hypothetical extraction plant. Theoretical base case: without any process losses. Data from Sugo et al. [Q298]

annual uranium production	10000 Mg/a
annual operating time	300 days
number of cycles per annum	5
uranium production per cycle	2000 Mg
number of adsorption beds	8 million
total mass of adsorbent	1 million Mg
replenishment of adsorbent	250 000 Mg/a
daily number of beds to be processed	133333 each day
mass of beds to be processed	133333 Mg/day
loss of adsorption beds at sea	unknown, not mentioned in [Q298]
recovery yield uranium from adsorbent	unknown, not mentioned in [Q298]

Table D.27

Parameters of three mooring methods (theoretical), based on data from Sugo et al. [Q298]

parameter	buoys	floating bodies	chain loops
number	80000	148	350
area occupied at sea (km ²)	3200	1926	4025
number of ships unloaded per day	133	27	-
occupied coast line (straight, km)	-	-	350

Table D.28

Parameters per Mg uranium (theoretical)

effective uranium grade per bed	0.025%
number of beds to be processed	4000
mass of adsorption beds to be processed	4000 Mg
number of ships to be unloaded	4 (method 1) 1 (method 2)
adsorbent loss	25 Mg adsorbent/Mg U

Each adsorption bed (cage) has to go through the following processes during each cycle:

- hoisting mooring unit from the sea into the ship
- transport from sea to shore (tens to hundreds of kilometers), as a unit
- disassembling sea-going mooring unit of the adsorption beds
- disassembling the adsorption bed (cage)
- processing the adsorbent, a multi-stage process (see Figure D.12)
- replenishing the adsorbent
- reassembling and repair of the cages
- reconnecting the cages to the mooring unit
- transport from shore to sea (tens to hundreds of kilometers)
- remooring the units in the sea.

Uncertainties

The figures of the hypothetical plant above are calculated ignoring the losses in the processes of the recovery of uranium from the adsorbent. As pointed out above the recovery yield will certainly be much less than 100%. If a yield of 50% is assumed (probably a high estimate) all dimensions in above tables have to be multiplied by 2. If the recovery is lower, e.g. 33% or 20%, the multiplier will be 3 to 5.

It is unknown which fraction of the adsorption beds will be lost and damaged during each cycle by heavy waves at sea and by other causes. The minimum number of adsorption beds to be handled during each cycle theoretically would be 8 million, and practically a lot more, say 16-40 million, depending on the multiplier mentioned above. As long as the multiplier is unknown, the project is characterized by very large uncertainties, apart from the uncertainties posed by incorporating untested technology.

Costs

Some figures from the literature are listed in Table D.29. All figures except the last two (Nobukawa et al. 1994 [Q72] and Sugo et al. 2001 [Q298]) refer to the titaniumhydroxide adsorption method.

Estimates of the cost of deriving uranium from seawater range between approximately \$1000 and \$25000 per kg uranium. To put this into context, note that a nuclear reactor requires about 180 tonnes of uranium per full-power year, and the present (2006) average cost of uranium from conventional ores is around 40-80 \$/kg.

The authors of ORNL 1974 emphasize that the cost figures they found are based on very optimistic assumptions, so the real values will probably be several times higher.

The figures of Sugo et al. are based on apparently unrealistic assumptions:

- The processes following the adsorption stage, required to recover the uranium from the adsorbent (see Figure D.6), and their inherent losses are ignored.
- Sugo et al. assumed an uranium load of 6 grams uranium per kilogram adsorbent under operational conditions, expecting that this could rise to 10 g U per kg adsorbent. Their own experiments showed that a load of about 2 grams uranium per kilogram adsorbent could be reached after a residence time of 60 days and about 3 g U per kg adsorbent after 240 days immersion in seawater. Apparently the adsorbed uranium load then approaches an equilibrium value. The calculations of Sugo et al. are based on a contact time of about 60 days.

Taking above two factors into account, the cost estimates by Sugo et al. may be low by a factor of at least 10.

Table D.29

Estimates of costs and energy consumption of uranium from seawater

reference	energy consumption TJ/Mg (U)		costs \$(2000)/kgU
Burnham et al. 1974 [Q136]	26	(1)	-
ORNL 1974 [Q133]	90	(2)	>> 2766
	1,2	(3)	>> 2766
Brin 1975 [Q225]	~390	(1)	289-8280
Mortimer 1977 [Q98]	20-600		728-2210
Koske 1980 [Q223]	5-10	(4)	-
INFCE 1 1980 [Q226]	-		18400-23700
Burk 1989 [Q73]	-		-
Nobukawa et al. 1994 [Q72]	-		370
Sugo et al. 2001 [Q298]	-		280-560
Seko et al. 2003 [Q312]	-		-

- (1) only pumping energy, deduced from data in publication
 (2) only pumps, pump-fed plant, pump head 20 m
 (3) pumps in tidal plant
 (4) deduced from data in publication; only pumps in first stage, pump head 0.5 m

Upscaling

The cost figures in Table D.29 should be regarded as highly speculative, as no practical experience of uranium extraction from seawater exists with an integrated system, not even on a pilot plant scale. The sole empirical data known are the results of a few adsorption experiments at gram scale.

As every chemical engineer knows, upscaling complex chemical processes is not a simple task. For a meaningful contribution to the world uranium supply, an extraction plant with an annual production capacity of at least 10000 Mg uranium would be needed (current uranium consumption is 68000 Mg per annum). That would mean an upscaling of the adsorption process, coupled with the subsequent chemical processes, with a factor of *ten billion*.

Large cost escalations are intrinsic to new technology projects, as RAND 1981 [Q126] reports:

- Severe underestimation of capital costs is the norm for all advanced technologies; the underestimation for energy process technologies mirrored that seen in major weapon systems acquisition, very large advanced construction projects, and major public work activities. A number of advanced technologies brought to project completion had problems with reliability and performance.
- Capital costs are repeatedly underestimated for advanced chemical process facilities, just

as they are for advanced energy process plants. Furthermore, the performance of advanced energy process plants constantly falls short of what was predicted by designers and assumed in financial analyses.

- Greater than expected capital costs and performance shortfalls not anticipated by conventional estimating techniques can be explained in terms of the characteristics of the particular technology and the amount of information incorporated into estimates at various points in project develop.

According to RAND 1979 [Q127], escalations in cost estimates of energy process plants by factors of 2-5 are not uncommon. The nuclear industry itself provides many examples of these observations and those of RAND 1981 [Q126].

There are no indications that the mechanisms described in the RAND studies would not apply to new large technology projects in 2006, including the extraction of uranium from seawater. On the contrary, a number of factors leading to underestimation of the costs of uranium from seawater are clearly demonstrated in the publication of Sugo et al.:

- the assumed adsorption rate of uranium from seawater is far higher than experimentally demonstrated
- the unavoidable losses of materials and adsorption beds at sea by heavy waves and by corrosion and other causes are ignored
- the unavoidable extraction losses in the chemical processes needed to recover the uranium from the adsorbent are ignored
- maintenance and refurbishment are not accounted for
- starting from an ideal situation and disregarding technical imperfections
- estimating costs on theoretical grounds, without practical experiences (e.g. in a pilot plant)
- the components of the system are considered separately, not as an integrated system
- potential problems posed by the sheer size of the facilities at sea and on the coast are not evaluated.

In addition a number of problems are unsolved, including:

- the dynamic behaviour of the mooring equipment at sea under stress of high waves and strong winds. Violent movements and collisions may cause losses at sea of adsorption beds or even whole units of beds.
- recovery of the adsorption beds under adverse conditions.
- problems for shipping posed by the buoys and floating platforms.
- pollution of the adsorber beds by organic materials from the sea.

The evaluation by ORNL [Q133] is very critical of the cost estimates as done by the reviewed studies, for similar reasons.

Energy requirements

Titaniumhydroxide adsorption method

Mortimer 1977 [Q98] concludes that most of the techniques he reviewed have an energy consumption equalling the energy content of the uranium.

A minimum value of the specific energy consumption for the extraction of uranium from seawater may be:

$$J_{\text{sea}} = 100 \text{ TJ/Mg (mainly electricity)} \quad \text{eq D.19}$$

This figure is based on the rigorous study of ORNL 1974 [Q133], and includes the energy costs of plant construction, chemicals and operation and maintenance of a pump-fed plant at sea, as large-scale tidal plants do not seem feasible. The process needs powerful pumps and consumes large amounts of electricity. Regeneration of the eluant by steam stripping is very energy-intensive and may alone consume thermal energy in the order of 200 TJ/Mg U.

The authors of ORNL 1974 emphasize that the values they found are based on very optimistic assumptions, so the real values can be expected to be several times higher.

A rough estimate of the specific energy requirements including plant construction, chemicals, operation and maintenance, can also be made using the costs in dollars and e, the energy/gnp ratio. Using the cost estimates quoted in INFCE-1 1980 [Q226] (taken from an undisclosed 1978 US Department of Energy study) of 18400-23700 \$(2000)/kg and $e = 10.6 \text{ MJ}/\$(2000)$, we find:

$$J_{\text{sea}} = 195\text{-}250 \text{ TJ/Mg} \quad \text{eq D.20}$$

The above values are underestimates, because the chemical industry is more energy-intensive than average economic activity. The figures agree fairly well with the estimates of Mortimer 1977 [Q98] and ORNL 1974 [Q133].

In all studies, to our knowledge, extraction losses (see above) are not taken into account, so the cited figures are low estimates.

Table D.30

Theoretical gross energy production from 1 Mg natural uranium in the reference LWR (representative of current design), operational lifetime 30 years at mean load factor of 0.82.

Heat production, per Mg natural U	TJ/Mg	465
Gross electricity production per Mg natural U	TJ/Mg	149

Table D.30 does not include the physical process losses of uranium, nor the energy requirements of the processes needed to convert uranium ore into reactor fuel, to operate, maintain and refurbish the nuclear power plant and to sequester the nuclear waste safely (see also Part B).

The net energy production of an LWR nuclear power system is much lower than the theoretical maximum of 149 TJ/Mg, and may be in the range of about 70-100 TJ/Mg

natural uranium (see Part B). Equation D.20 shows that the energy requirements of the complete process might be much higher than even the theoretical energy production possible using uranium recovered from seawater.

From an energetic point of view, uranium from seawater using the titaniumhydroxide method cannot be considered an option for the global energy supply. This is confirmed by the conclusion of INFCE-1 1980 [Q226]:

“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.”

Polymer adsorption method

Based on the concept of Sugo et al. some parameters of a hypothetical uranium extraction plant can be deduced (see Tables D.25 - D.28). The plant has an assumed annual production capacity of 10000 Mg uranium from seawater.

The effective ‘ore grade’ of the adsorbent beds is $G = 0.025\% \text{ U}$ (250 g U per bed of 1000 kg). This value is quite near the energy cliff of about 0.02% U of conventional ores. If the yield of the recovery of uranium from the adsorbent is lower than the recovery yield from conventional ore of the same grade, the effective grade of the adsorbent may be significantly lower than 0.025% U. If, for instance, a recovery yield from conventional ore $Y = 60\%$ is assumed, and from the adsorbent $Y = 30\%$, which is not unrealistic (see Figure D.12), twice as much adsorbent has to be processed as conventional ore, to recover the same mass of uranium.

Although the adsorption process itself requires no energy input because the seawater is continuously refreshed by the sea current, handling of the adsorbent units and processing the materials to recover the uranium requires huge quantities of energy, equipment and materials.

The transport of the adsorbent to the shore and back to sea - in contrast with the transport required for ore, using trucks and belts - requires cranes, ships and other equipment over long distances, maybe hundreds of kilometers. Moreover, the energy requirements of all the processes which each adsorbent cage has to undergo, cited under Table D.28, should also be taken into account. The replenishment of adsorbent beds requires energy-intensive materials, such as stainless steel and the amidoxime polymer.

The specific energy consumption of the production of the adsorbent, a polymer with the amidoxime groups, may be high, because of its special chemical composition and the need for electron beams to bind the active groups to the polymer. For each Mg uranium recovered at least 25 Mg polymer adsorbent is consumed.

Given the above considerations, it seems extremely unlikely that the specific energy consumption of the method of Sugo et al. will be less than the specific energy content of the recovered uranium.

Conclusions

Two methods of recovery of uranium from seawater have been studied in enough detail to get an impression of their feasibility and specific energy requirements: the titaniumhydroxide adsorption method and the Japanese polymer adsorption method.

Neither of the discussed methods have been tested as a complete process chain. The sole empirical basis consists of small scale laboratory experiments. These experiments demonstrated nothing more than the technical feasibility of extracting uranium from seawater.

Both methods are beset with so many and such far-reaching uncertainties and unsolved problems, that the feasibility of large-scale operational facilities should be considered speculative.

Uranium derived from seawater neither by the titaniumhydroxide adsorption method nor by the polymer adsorption method, can be considered an energy resource. The specific energy consumption of the whole chain of processes almost certainly - and by a wide margin - will surpass the energy to be generated from the recovered uranium.

INFCE-1 1980 [Q226] concludes:

“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.”

Given the the very low concentration of uranium ions in seawater and the relatively high concentrations of a large number of other dissolved species, such as sodium, magnesium, sulphate and chloride ions, the mixing entropy of uranium ions in seawater is very high. To separate uranium from all other chemical species in seawater, large quantities of work (high-quality useful energy) are needed. This separation work may be far higher than the useful energy which can be generated from the extracted uranium.

If so, any research of extraction techniques of uranium from seawater would be futile.

It would be wise to first calculate the theoretical extraction work based on thermodynamic laws, before embarking on a large R&D effort aimed at extraction of uranium from seawater, and before publishing statements on uranium from the oceans as an almost limitless energy resource

D11 MOX fuel

A rough estimate of the additional energy potential offered by using MOX fuel can be estimated as follows.

Assume MOX is used in an LWR with an all-MOX core. Current LWRs cannot handle more than 30% MOX fuel.

The fissile Pu content in LWR fuel has to be higher than the U-235 content in conventional enriched uranium fuel. Assume the fissile Pu content the MOX fuel has to be 5% and reactor-grade Pu is 70% fissile, then the r-g Pu content should be 7.1%.

Here we assume MOX fuel contains 7% reactor-grade Pu.

One reload of 20.3 Mg (reference reactor, see Part B) with 7% rg Pu contains 1.421 Mg Pu, rounded 1.4 Mg.

Assume worldwide 250 Mg r-g Pu is available, that would be enough for $250/1.4 = 178.6$ reloads, rounded: 180 reloads.

The uranium in MOX fuel can be depleted U or natural U. Assume depleted U is used. Uranium recovered from spent fuel is too radioactive to be handled in fuel fabrication plants at the current state of technology. Apart from the high radioactivity, the high content of U-232, U-234 and U-236 has an adverse effect on the fuel quality in the nuclear reactor.

To produce 180 reloads $180 \cdot (20.3 - 1.4) = 3402$ Mg depleted U is needed.

How much natural uranium, to be used in the conventional enrichment chain, would be required to produce the same amount of reactor fuel?

One reload corresponds with 162 Mg natural uranium in this study (tails assay 0.2% U-235). In the current practice more than 180 Mg of U_{nat} is needed.

The production of 180 reloads of 162 Mg each requires

$$m(U_{\text{nat}}) = 180 \cdot 162 = 29160 \text{ Mg natural U.}$$

If 180 Mg per reload is required:

$$m(U_{\text{nat}}) = 180 \cdot 180 = 32400 \text{ Mg.}$$

This is less than $\frac{1}{2}$ year of uranium supply (about 67000 Mg/a in 2006).

So 1 Mg r-g Pu replaces: $29160/250 = 117$ Mg U to
 $32400/250 = 130$ Mg U

Conclusion

The contribution of today's Pu inventory to the nuclear energy supply is negligible. For the future an energy balance should be made: reprocessing of spent fuel is an extremely energy-intensive process. In addition MOX fuel fabrication may consume much more energy than conventional fuel. The energy balance should include the dismantling of the reprocessing facilities.

D12 World uranium outlook

Physical/chemical viewpoint

Uranium deposits

Deposits of uranium minerals (and of other scarce elements) in the earth's crust are formed only when 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 at one site, reducing at another site), complexing agents to carry the element in solution and other agents to finally precipitate the mineral.

It should be noted that the same mechanisms that form uranium mineral deposits, also are able to destroy earlier formed deposits.

The probability that an ore deposit will be formed and will survive until today 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 D.13. 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 are plotted on a logarithmic scale.

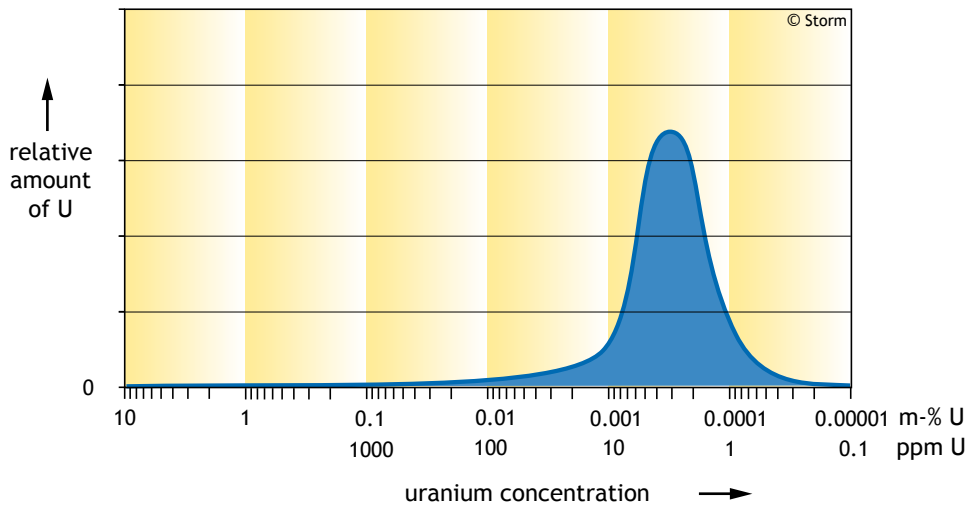


Figure D.13

Hypothetical log-normal distribution of uranium in the earth's crust, if such a distribution were to apply to uranium. Source: Deffeyes & MacGregor 1980 [Q281].

Note that the horizontal axis has a logarithmic scale (decreasing uranium concentration) and the vertical axis a linear scale. The area under the curve represents the total amount of uranium in the earth's continental crust. The peak is at the average crustal abundance of 2.8 ppm.

Inventarising and aggregating all known uranium deposits in the world, Deffeyes & MacGregor 1980 [Q281] found a global abundance distribution as in Figure D.14. They

considered the odds small that a major category had been systematically underestimated. According to these authors the diagram of Figure D.14 appears to support the hypothesis of a single log-normal distribution as illustrated in Figure D.13. However, the rich ore deposits in Canada (grades larger than 3%) were not included in their publication. In Figure D.14 these deposits have been added by the authors of this study.

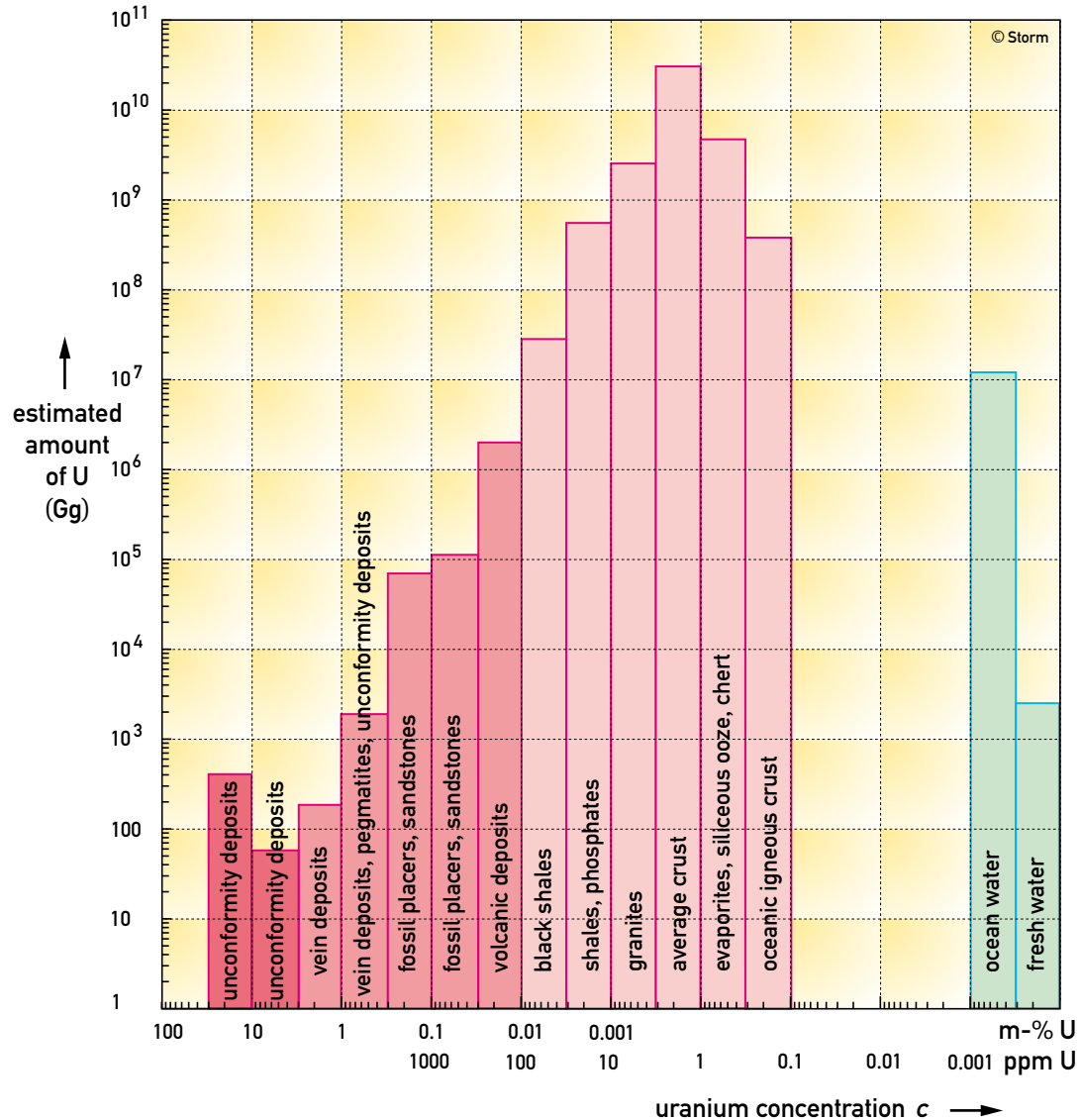


Figure D.14

Distribution of uranium in the major geological reservoirs, according to Deffeyes & MacGregor 1980 [Q281]. The height of each bar represents the estimated total amount of uranium at the given concentration. It should be noted that both the horizontal and the vertical axis have a logarithmic scale, whereas the vertical scale in Figure D.13 is linear. The rich deposits in Canada, with grades of more than 3% uranium were absent in the original publication of Deffeyes & MacGregor, and have been added by the authors of this study (the two dark shaded bars at the left). The medium shaded bars represent the reservoirs from which uranium is extracted in the current practice, down to a grade of 0.01% uranium.

However, Deffeyes & MacGregor point out that no rigorous statistical basis exists for expecting a log-normal distribution of uranium in the earth's crust. It is not possible to

determine the distribution of uranium deposits directly. Uranium exhibits a complex range of geochemical behaviour and a wide variety of different kinds of economic deposit. Other possible distributions would be a bimodal one, with two peaks, or a multimodal one, with more peaks in the distribution curve.

The study of Deffeyes & MacGregor 1980 [Q281] is still considered representative, as the authors of the study MIT 2003 [Q280] base their view on the prospects of the uranium supply on this study of Deffeyes & MacGregor.

No uranium minerals can be formed on the sea floor, due to the lack of suitable chemical conditions. However, dissolved uranium ions are present in seawater in very low concentration (see Part D10).

The diagram of Deffeyes & MacGregor says little about the mineralization of uranium, the presence of ores and the recoverability of the uranium from the various geological reservoirs. As pointed out in Part D3, the mineralization limit of uranium is of crucial importance in the recovery of uranium from the earth's crust.

Unlike all other metals, uranium is exclusively used as energy source (setting aside military purposes), not as construction material, chemical or ornament. This implies that the criterion for extraction should be based on the energy required for the extraction of the uranium from the deposit. If the energy consumed in the production of one kg of uranium is more than the energy which can be generated from the same amount, the uranium cannot be considered an energy source. Since the net energy obtainable from uranium depends on the nuclear system used to convert the nuclear energy into electricity, the criterion is system-dependent as well.

Table D.31

Abundance of uranium in various geologic reservoirs. Uranium is one of the scarce elements and is about as common in the earth's crust as tin or zinc

occurrence	Concentration (ppm = g/Mg)	reference
high-grade ore, 2% U	20 000	WNA75 2007 [Q85]
low-grade ore, 0,1% U	1000	WNA75 2007 [Q85]
Chattanooga shales (average)	60	Boyd 1980 [Q156], Burnham et al. 1974 [Q136]
copper-, gold ores (by-product)	50 - 500	UIC34 2003 [Q86], NAC 1982 [Q53]
phosphate ores	10 - 300	INFCE-1 1980 [Q226], Bergeret 1979 [Q47]
Conway granite	12 - 15	NEA/IAEA 1978 [Q48]
granite (average)	4	WNA75 2007 [Q85]
sedimentary rocks (average)	2	WNA75 2007 [Q85]
average continental crust	2.8	WNA75 2007 [Q85]
seawater	0.0034	ORNL 1974 [Q133]

Mortimer 1979 [Q118], Mortimer 1980 [Q122] and Bowie 1975 [Q49] conclude that 50 ppm ($G = 0.005\% \text{U}_3\text{O}_8$) is the minimum grade for uranium-bearing rocks to be considered ores. Brinck 1975 [Q55] takes a cut-off grade of 200 ppm.

One of the results of the energy analysis in this study, the energy cliff (see Part G) point

to a cut-off grade of between 0.02-0.01% U_3O_8 .

Geologic resources are distributed unevenly over the surface of the globe and they are concentrated in the outermost part of the earth's continental crust. Mechanisms that concentrate uranium operate most effectively on or near the earth's surface. Ore bodies tend to get poorer and harder to mine downward (Cook 1976 [Q344]). This phenomenon is, for instance, observable in the current practice at Ranger and Olympic Dam in Australia and is also illustrated by Figure D.15 (which is identical to Figure D.1 in Part D2).

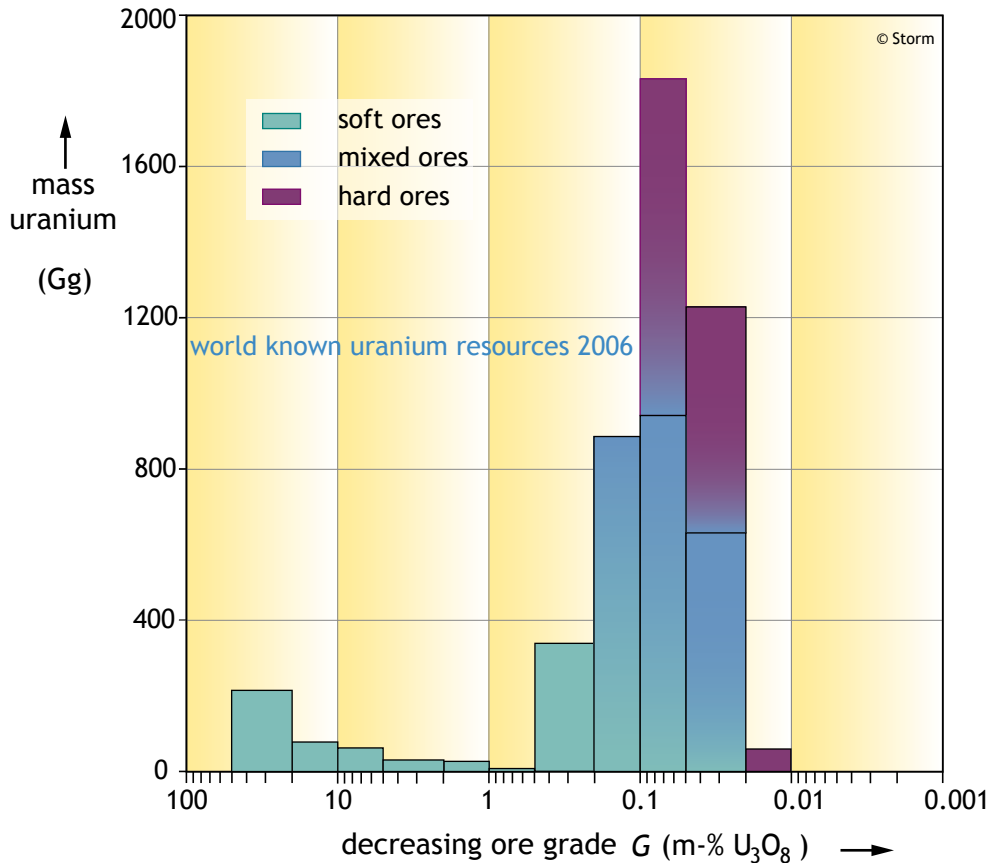


Figure D.15

The distribution of the world known uranium resources as function of the ore grade. Uranium resources tend to be larger at lower ore grades and lower grade ores tend to be harder. This diagram is identical to Figure D.1.

Below grades of 0.5% U_3O_8 the distribution in Figure D.15 appears to support the log-normal distribution as mentioned by Deffeyes & MacGregor 1980 [Q281] and the observations of Cook 1976 [Q344].

The relatively large resources at grades of $> 1\%$ U_3O_8 - located in the Athabasca Basin in Canada - may suggest a bimodal distribution of uranium in the earth's crust. We return to this topic in the next section.

Figure D.10 shows the empirical relation between production costs and ore grade; the diagram is based on data of all operating uranium mines outside China and the former Sovietunion in 1982 (NAC 1982 [Q53]). The total amount of these resources was 1.8 Tg

uranium. Unfortunately, we could not update Figure D.16, because recent data from the Nuclear Assurance Corporation are not free. Anyway, the diagram gives a good impression of the most occurring ore grades.

Notable aspects are, among others: the rough relationship between ore grade and production costs, the large spread in production costs and the use of the same price classes as today (40, 80 and 130 \$/kg U). Although the production costs in itself are not the information from Figure D.16 needed in this study, it may be interesting to note that the reported value of, for example, Ranger of 11.22 \$/lb U_3O_8 in 1982 would correspond with 29.17 \$/kgU (1982), about 61 \$/kgU in \$(2006).

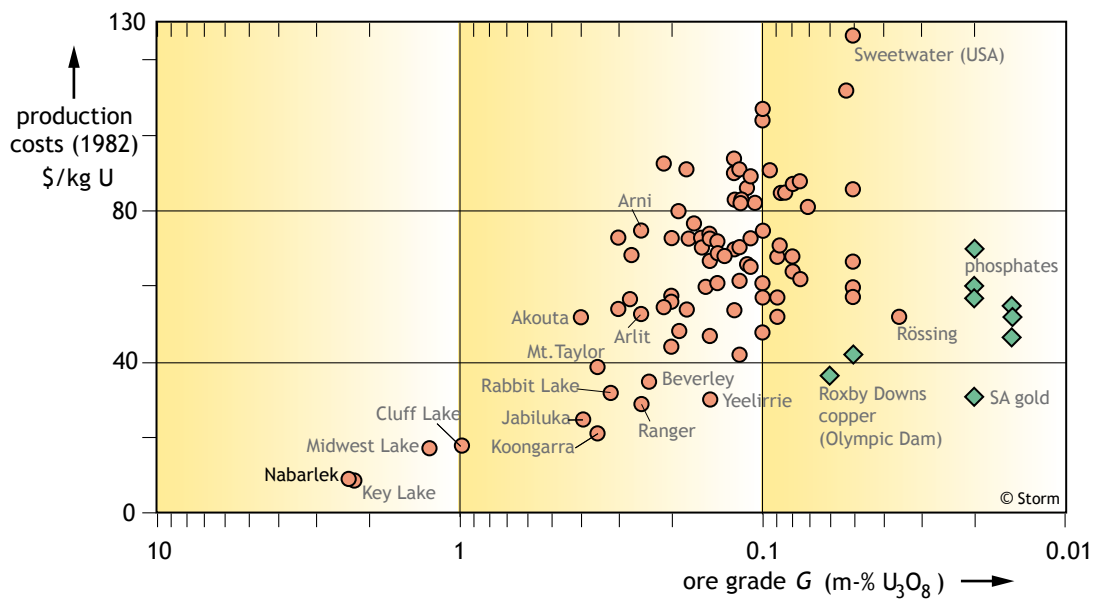


Figure D.16
Ore grade and production costs of operating uranium mines of the world, except China and the former Sovietunion in 1982. Based on data from NAC 1982 [Q53]. The diamonds represent mines where uranium is extracted as by-product of copper, gold or phosphates.

Figure D.16 shows that most uranium mines known in 1982 have ore grades in the range of 0.08 -0.4% U_3O_8 . The same holds true for 2006, as is illustrated by Table D.2. This is not to say that the largest resources are in that grade range. The currently known resources in the former Sovietunion and China have grades of 0.2% U_3O_8 or lower (see Part D2).

Since 1982 few new deposits are added: in Canada relatively large ore bodies with high ore grades (up to 20% U_3O_8) came into production (McArthur River and Cigar Lake, estimated at about 344 Gg uranium), and there are some new discoveries in Brazil (256 Gg uranium). Some mines from the list in 1982 now are closed or mined out (e.g. Nabarlek and Yeelirrie), but most are still in operation today. The main differences in the total known resources in 2006, compared with those in 1982, are the inclusion of resources not included in the 1982 list (e.g. China, Russia, Uzbekistan, Kazakhstan). In addition the size of some known resources in Australia, particularly Olympic Dam, have been greatly upgraded during the past years.

Striking is the absence of known ore bodies with grades in the range of 0.5 - 1.0% U_3O_8 . Today only one deposit with a grade in this range is reported: Key Lake with 0.254 Gg at 0.53% U_3O_8 . In 1982 Key Lake reportedly contained some 58 Gg uranium at a grade of 2.27% U_3O_8 , so the 2006 data may refer to a downgraded remainder of the original Key Lake deposit.

Also notable are the small number of deposits with grades of 1% U_3O_8 or higher. Since 1982 a few are added to the list, all located in one geologic province, the Athabasca Basin in Canada. Nabarlek has been mined out.

Prospects of new discoveries

Likely new uranium resources will be found in the future. Higher uranium prices will stimulate more exploration and more exploration might lead to new discoveries. However, the uranium deposits that are the easiest to discover and are of the highest quality - easiest accessible (physically and chemically) and mineable - are already known.

The question whether uranium has a normal or a bimodal distribution in the earth's crust is far from academic. Apart from the downgraded Key Lake deposit, no uranium deposits are ever reported in the ore grade range of 0.5 - 1.0% U_3O_8 . If uranium were to have a normal crustal distribution, significant discoveries in the grade range of 0.5 - 1.0% U_3O_8 could be expected. Up until now no such evidence has been reported.

From a geological point of view the chances of finding new, large deposits of the same high-quality as those of the Athabasca Basin seem small. No indications on the existence of such deposits have been published up until now. Not every ore deposit in the Athabasca Basin is easily accessible anyway, given the problems with Cigar Lake. Even if a new rich one would be discovered of the size of the Athabasca Basin deposits, it would add only 6-7 years of high-quality uranium supply to the current world demand.

Most likely new finds of significant uranium deposits will have lower energy quality than the currently known deposits of the same ore grade, due either to greater depth, longer transport distances, smaller ore bodies, harder rock or other causes. Lower quality means more energy consumed per kg extracted uranium. So yet-to-be discovered uranium deposits likely will lie closer to the edge of the energy cliff (see Part G).

Summarized:

- No uranium deposits have ever been reported in the grade range of 0.5 - 1.0% U_3O_8 (apart from the downgraded remainder of the Key Lake deposit).
- The easiest discoverable and easiest recoverable uranium resources are already discovered and almost all of them are already in production.
- During the past decades no evidence has been published on the existence of major new uranium deposits of the same energy quality as the currently mined deposits, that would significantly extend the world uranium supply.
- From a geological point of view the chances of finding major high-quality deposits seem not large. Which parts of the earth are so poorly explored that a second Athabasca Basin or a second Olympic Dam could be overlooked? May be Antarctica?
- The chances of finding new uranium deposits increase with decreasing ore grade, decreasing ore body size and accessibility.

Economic viewpoint

At the uranium industry the classification system of uranium resources usually is based on two parameters: costs, however defined, and confidence in the estimates of the ore deposits. Cost categories, in fact market prices, are for example: up to 40 US\$/kgU, up to 80 US\$/kgU and 80-130 US\$/kgU. Confidence categories used by OECD NEA and IAEA are: Reasonable Assured Resources (RAR), Estimated Additional Resources (EAR) and Speculative Resources. RAR of the lowest cost category are often called Reserves.

The NEA classification is not generally adopted; some important uranium-producing countries use different classifications, e.g. Australia, South Africa and Canada, which together hold more than 60% of the known resources. In 2006 the NEA and IAEA in their 'Red Book' [Q90] adopted a new classification system. To people outside the nuclear world the issue of uranium resources and their importance for the future energy supply is becoming increasingly complex and opaque.

The production costs of the uranium are related with the ore grade and ore body size (see e.g. Brinck 1975 [Q55]). But also non-physical factors contribute to the costs, such as economic and political factors and business interests. The rise of the uranium prices during the last year are an example of such non-physical factors.

Some authors, e.g. WNA-75 2007 [Q85], Wikdahl 2004 [Q285], MacDonald 2001 [Q286] and MacDonald 2003 [Q287] expect new large and rich discoveries when exploration will be resumed intensively. Their views are based solely on economic considerations and by analogy with the oil industry: more exploration will yield more known resources.

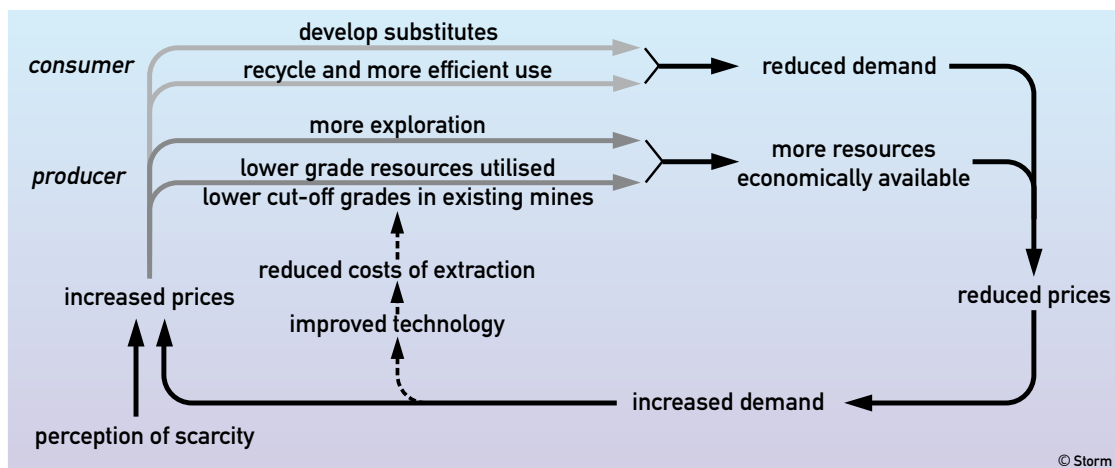


Figure D.17

An economic view on mineral resources, i.e. uranium resources, according to WNA-75 2007 [Q85]

In the economic view as illustrated by Figure D.16 mineral resources are virtually inexhaustable. In effect the diagram is based on non-physical quantities, which are not unambiguously quantifiable. Physical boundaries, such as an energy input limit per unit product, are absent from this model.

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