Process analysis of the Ranger mine

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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 referencesappear in the text.

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1 Introduction:

The objective of this paper is to validate the method used in the original study Storm&Smith 2008 [Q6] of estimating the energy consumption of the recovery of uranium from the earth's crust.

The method used in this study (Storm & Smith 2008 [Q6]) for calculating the energy consumption of the recovery of uranium from the earth's crust, has been derived from the study of Rotty et al. 1975 [Q95]. Rotty et al. in turn based their study on a thourough survey of a large number of uranium mines by the US Bureau of Mines in 1973. The Rotty study, adopted by the authoritative report ERDA 1976 [Q109], may be the most thourough and most reliable publication on the energy consumption of uranium mining up until today. Although the technology of mining and recovery of uranium has hardly changed since the 1970s, it seemed worthwhile to test this method by practical data from a currently operating large uranium mine. To this end the results of the Rotty method will be compared with the results of a in-depth process analysis of the Ranger uranium mine in Australia.

The Ranger mine has been chosen as study case for two reasons:

- Ranger is a large open pit mine with favourable conditions and one of the cheapest operating uranium mines in the world. If the Rotty method applies well at Ranger, we may assume it will apply at the world average uranium mine as well.
- In the open literature very little practical data on mining operations are available. Just enough data on the Ranger operations have been published by its owner to allow for a reasonably reliable process analysis.

Method

This paper describes the process analysis of the direct and indirect energy consumption of the Ranger uranium mine, based on physical and chemical data. The analysis starts from the data of the operations during 2005 as published by the owner of the mine, Energy Resources of Australia Ltd (ERA): ERA 2006 [Q320] and ERA-AR 2005 [Q321].

The ERA data give some clues on the direct energy consumption of Ranger. However, not all direct energy inputs are mentioned by ERA, as this analysis will reveal. The indirect energy inputs encompass the energy consumed in transports to and from Ranger and the energy embodied in material inputs, such as: equipment, spare parts, chemicals, lubricants and materials needed to construct and maintain the mine.

With two exceptions (explosives and sulfuric acid) ERA published no quantitative data on the material inputs of the Ranger mine. For that reason most indirect energy inputs had to be estimated via a physical/chemical analysis of the operations at Ranger and by implementing data from third sources.

2 Uranium production at Ranger in 2005 (ERA data)

Resources and grades

The data on the resources and grades of the ores at Ranger are not fully consistent.
Ranger Pit#1
mined out (May 1980 – Dec 1994)
20 million Mg ore mined, average *G* = 0.327%

60 million Mg waste rock + very low-grade mineralised material
So: the overburden ratio $S = 60/20 = 3$
In 'ERA history':
19.78 milion tonnes ore mined at an average grade $G = 0.321\%$
The question arises: which grade is the correct one?

Ranger Pit#3
Q320: 'Dec 2001: 22 miilion tonnes of ore in stockpile and in situ, at an average grade $G = 0.27\%$ U_3O_8, containing 54241 tonnes U_3O_8'
SvL: these figures are inconsistent
either: $22 \bullet 10^6 \bullet 0.0027 = 59400$ Mg U_3O_8
or: $54241/0.0027 = 20.09 \bullet 10^6$ Mg ore
Also possible: $Y = 54241/59400 = 0.913$
$\Delta = 59400 - 54241 = 5159$ Mg U_3O_8

All material is transported to a radiometric discriminator (scintillometer heads) to determine the destination: crusher, stockpiles or waste rock stockpile

By reason of the similarity of Pit#1 and Pit#3 of the Ranger mine, we assumed the overburden ratio of the currently mined Pit#3 to be the same as of Pit#1, that is S = 3.

Water

New water treatment plant complete by December 2005. 1.2 million Mg/a. or 7000 Mg/day pond water or 4000 Mg/day process water, combined mode 6000 Mg/day.
Pond water = rain water run-off from stockpiles and other areas of the mine.
Process water has been used in the treatment of ore in the processing plant and rtequires more intensive treatment [Q321].

4000 Mg/day => $4000 \bullet 365 = 1.46 \bullet 10^6$ Mg/a
6000 Mg/day => $6000 \bullet 365 = 2.19 \bullet 10^6$ Mg/a

Fresh water make up not given.

Operating time and load factor

"For most of the year, operations continued 24 hours, seven days a week." [Q321]
"... one major shutdown was replaced with a series of smaller plant shutdowns, allowing increased maintenance efficiencies ..." [Q321]
Photo p. 7 [Q321]: 4 large dump trucks + 3 smaller ones + large shovel.

Q321 p.6
Milling capacity per operating hour
Mill 1 capacity = 216 Mg/h
Mill 2 capacity = 107 Mg/h
sum = 323 Mg/h
1 year = 8760 h
If load factor $L_{100} = 1$:
Full year capacity m(ore) = $323 \bullet 8760 = 2829480$ Mg/year
Actually milled m(ore) = $2293 \bullet 10^3$ Mg

=> load factor L = 2293/2829 = 0.8105

=> Operating time = 0.8105•8760 = 7100 h/a

=> 1660 h not in operation = 1660/24 = 69 days/year
This time is needed for maintenance and repair and is due to planned and unplanned outages

Likely the dump trucks and excavators have also a mean operating time of 7100 h/a.

Primary data

Table 1
Primary data of the Ranger uranium mine, valid for 2005, as given by ERA [Q320] and [Q321]. The last two columns have been added by the author.

quantity	value	unit	source		
mass of ore milled	2.293	Tg (10^6 Mg)	Q321		
processing recovery	89.2	%	Q321		
mass of U_3O_8 (drummed) produced	5910	Mg	Q321		
product grade of drummed U_3O_8	98.7	% U_3O_8	Q321	83.7	% U
average ore grade Pit#3	0.27	% U_3O_8	Q320	0.23	% U
processing head grade	0.288	% U_3O_8	Q321	0.244	% U
cutoff grade	0.12	% U_3O_8	Q321	0.10	% U
number of dump trucks (Caterpillar) + 4 Caterpillar 777D	6 4		Q320 Q321		
excavator Hitachi EX-2500 + Hitachi EX 1200	1 1		Q320 Q321		
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		
electric generating capacity	28	MW	Q320		
average electric power load	~ 10	MW	Q320		
water treatment, not in 2005	1.5	10^6 m^3/a	Q321		
capacity haul trucks	90-135	Mg	Q320		

Basic data on diesel fuel and fuel oil

Diesel fuel
Energy content (LHV, lower heating value) J_{th} = 36.0 GJ/m^3 = 42.9 GJ/Mg
density d = 0,839 Mg/m^3
specific CO_2 emission γ = 75 g/MJ

Deduced data

Table 2

Deduced data of the Ranger uranium mine, valid for 2005, based on the ERA data (Table 1).

1 kg U_3O_8 contains 0.848 kg U

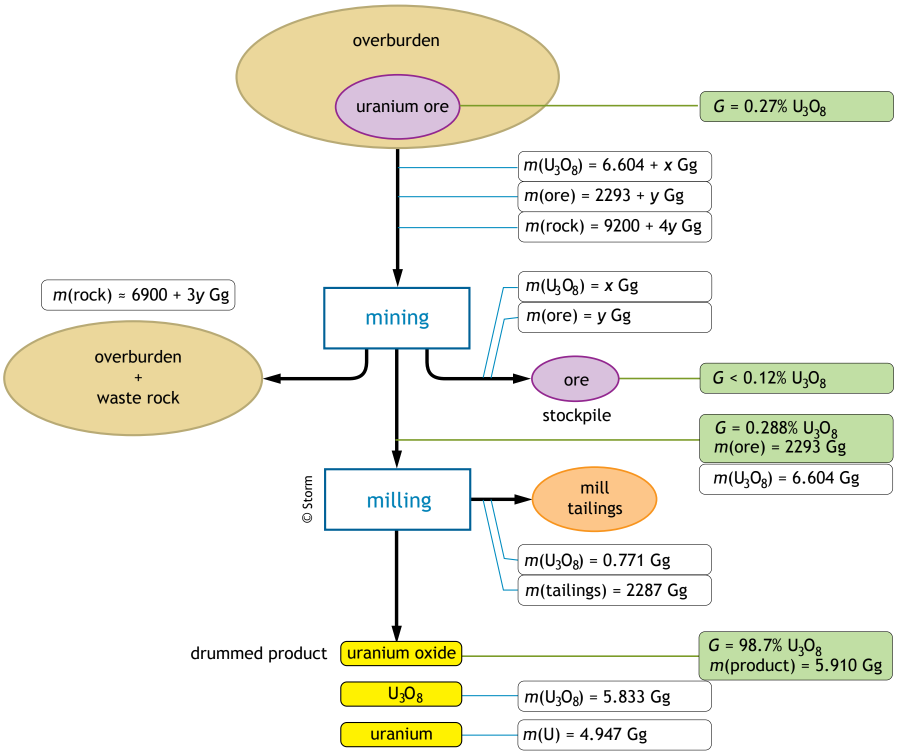
quantity	value	unit	remarks
ore mined at G = 0.288 % U_3O_8	2293	10^3 Mg	
mass of rock mined	9172	10^3 Mg	overburden $S = 3 + 1$ *)
mass of U_3O_8 in milled ore	6604	Mg	= 2293•10^3•0.288
mass of U in milled ore	5600	Mg	= 6604•0848
mass of U_3O_8 in drummed product	5833	Mg	= 5910•0.987
recovery yield	0.8833		= 5833/6604 slightly lower than ERA figure (0.892)
recovered U (in drummed product)	4947	Mg	= 5833•0.848
mass of U_3O_8 to tailings	771	Mg	= 6605 – 5833
electricity generated in 2005	315.36	TJ	= (10•3600•24•365)/10^6
diesel consumption, 50 Mg/day	18250	Mg/a	= 50•365
diesel consumption, 60 Mg/day	21900	Mg/a	= 60•365
stripping ratio S of Pit#3	3		assumed indentical to pit#1
operation time (load factor)	7100	h/a	see section 2.3
haulage distance	5	km	see section 2.8

* See mass balance in Figure 1. Also mined y Gg ore, to stockpiles, at G < 0.12% U_3O_8 cutoff grade plus 3y Gg overburden. These masses are not included in E balance of this paper. Mining and milling E input will discounted for when these stockpiles would be processed.

** S = mass overburden/ mass ore

From the ERA data a mass balance of Ranger can be deduced: see Figure 1. This mass balance is used in this analysis.

Mass balance

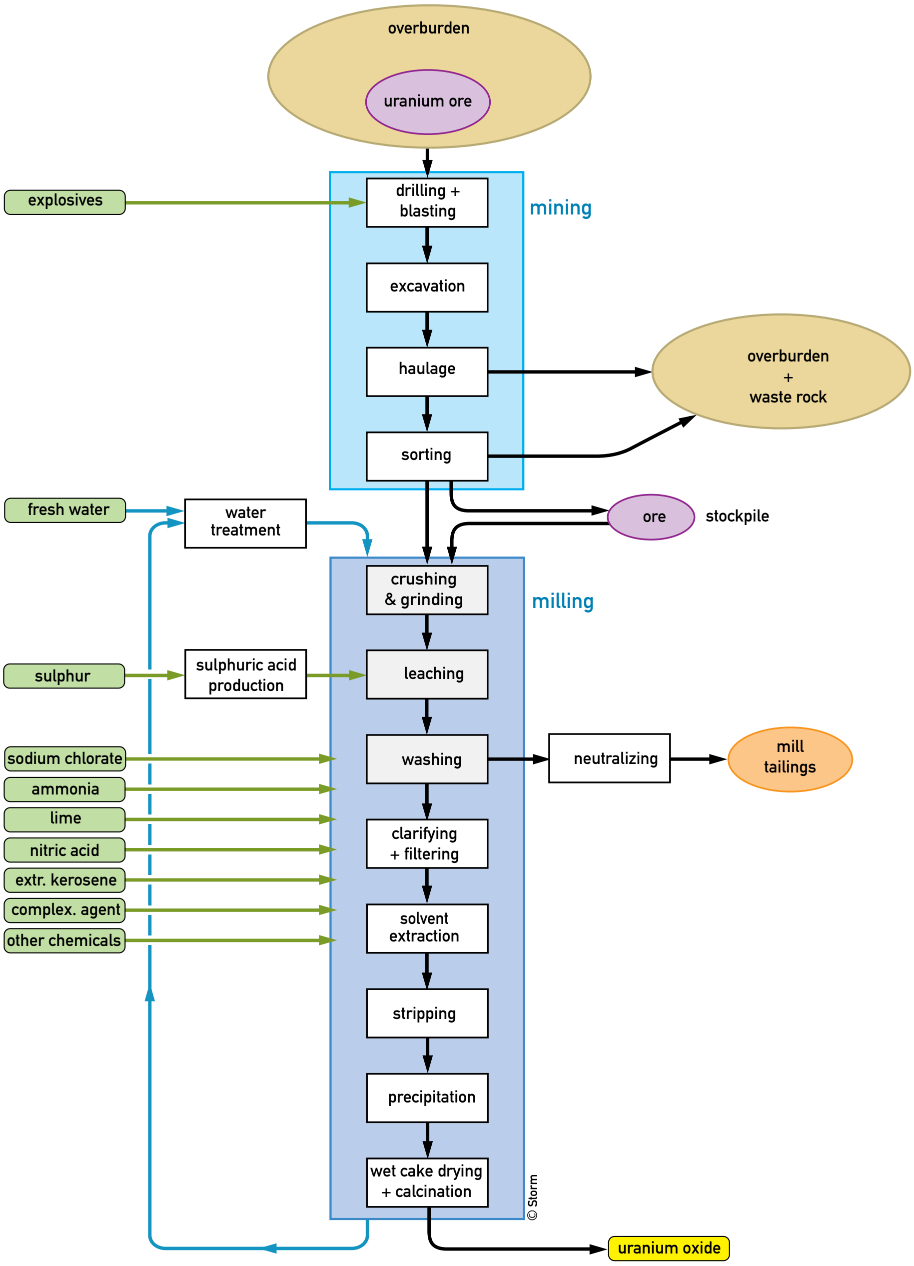


Ranger uranium mine mass balance 2005

Figure 1

Mass balance of the Ranger uranium mine in 2005. The figures in the green shaded boxes are data from ERA Q320 and Q321. The figures in the white boxes are directly deduced from the ERA data.

Flowsheet of Ranger



Flowsheet Ranger uranium mine

Figure 2

Flowsheet of the Ranger mine, based on data from ERA 2006 [Q320].

Mining site

Figure 3
The Ranger mine from space (source: Google Earth). From this photo the distance can be estimated over which the waste rock and ore has to be transported by dump trucks: the haulage distance. In this study we assume an average haulage distance of d = 5 km.

3 Mining

Embodied energy in equipment

Estimate of the operational lifetime of dump trucks

Example
Generally, a personal car with 400000 km on its counter has to be replaced. The number of its operating hours is:
at an average speed $v = 60$ km/h $T_{100} = 400000/60 = 6667$ h
if average speed $v = 50$ km/h $T_{100} = 400000/50 = 8000$ h

Assume mining equipment has an operational lifetime of some 20000 hours.
Assume in case of the Ranger mine T_{100} = 21300 hours or 3 full seasons (= 3 years). Assume that after 21300 hours the dump trucks and excavators have to be fully replaced. In practice the engines, for instance, may have to be replaced at shorter intervals.
Hydraulic excavators have a much shorter lifetime than draglines (which operate some 40000 hours), so 21300 hours may be a reasonable guess.

According to Mortimer [Q98]

Energy investment of 1-6 m^3 excavator	$E = 1.6 \pm 0.20$ TJ(th)
average 3.5 m^3	
Scaling up to 12 m^3 bucket:	$E = 4.8 \pm 0.6$ TJ(th)
(assume 3x (1-6m^3) and no scale effect)	
per year:	J(th) = 4.8/3 = 1.6 ± 0.2 TJ(th)/a
excavator with 6 m^3 bucket:	$E = 0.80 \pm 0.1$ TJ/a
10-15 m^3 pit truck	$E = 1.3 \pm 0.50$ TJ(th)
scaling up to 54 m^3 capacity (= 135 Mg)	$E = 4.68$ TJ
assume modest scale effect =>	$E = 4.5 \pm 1.7$ TJ
per year	J(th) = 4.5/3 = 1.5 ± 0.6 TJ(th)

Drilling and blasting

Direct E

Mortimer [Q98] p.233-234
surface mining $J_e = 0.1 - 1.0$ MJ/Mg
This study: $J_e = 0.6$ MJ/Mg
mechanical energy equals electric energy => $J_{mech} = J_e$

Indirect E

Mortimer [Q98] p.233-234
surface mining Jth = 1.8 – 20 MJ/Mg, excluding explosives
This study: Jth = 11 MJ/Mg
Embodied in drill rigs, drill bits and for maintenance

Indirect CO_2 emission

The specific CO_2 emission from the indirect thermal energy input, assuming it is supplied as oil, is calculated according to the equation: $m(CO_2) = J_{th}$ (MJ)•75 g/MJ [Q27].

=› $m(CO_2)$ = 11•75 = 0.825 kg CO_2/Mg rock

Ranger

direct E

Total mass mined m(rock) = 9.172•10^6 Mg

=› Emech = 9.172•10^6•0.6 = 5.503 TJ(e)

drill rigs powered by diesel engines, assume conversion efficiency = 40%

=› Jth = 0.6/0.40 = 1.5 MJ/Mg rock

Eth = Ee/0.40 = 5.503 /0.40 = 13.76 TJ(th)

diesel consumption V = 1.5/36.0 = 0.0417 L/Mg rock

m = 0.0417•0.839 = 0.0350 kg/Mg

direct $m(CO_2)$ = 13.76•75 = 1032•10^6 g = 1032 Mg CO_2

indirect E

Eth = 9.172•10^6•11 = 100.9 TJ(th), excluding explosives

indirect $m(CO_2)$ = 9.172•10^6•0.825 kg CO_2 = 7.567•10^6 kg = 7567 Mg

or: $m(CO_2)$ = 100.9•75 = 7567•10^6 g = 7567 Mg

Sum direct + indirect E

Jth = 1.5 + 11 = 12.5 MJ/Mg rock mined.

Explosives

Consumption of explosives at Ranger m = 0.25 kg/Mg rock. Commonly used explosive in mining consists of ammonium nitrate NH_4NO_3 mixed with fuel oil. Kind of explosives used at Ranger not disclosed. Assume equivalent to pure ammonium nitrate in this study.

Embodied E

Je = 15.35 GJ/Mg explosive

Jth = 31.00 GJ/Mg

Je + Jth = 46.35 MJ/kg R = 2.02

if 0.25 kg explosive /Mg rock:

Je = 3.84 MJ/Mg rock

Jth = 7.75 MJ/Mg rock

Je + Jth = 11.59 MJ/Mg rock R = 2.02

Ranger

Mass of consumed explosives

m(expl) = 9.172•10^6 Mg•0.25 kg/Mg = 2293•10^3 kg = 2293 Mg

embodied *E*

E = 46.35•2293•10^3 = 106.3•10^6 MJ = 106 TJ R = Jth/Je = 2.02

*E*e = 35.20 TJ

*E*th = 71.08 TJ

indirect $m(CO_2)$ = 71.08•75 = 5331•10^6 g = 5331 Mg

The explosives are imported. Electric component are kept separated in energy balance, according to the Q6 methodology.

Total energy input drilling and blasting Ranger 2005
Eth = Edir + Eindir + Eexpl = 13.76 + 100.9 + 71.08 = 185.7 TJ(th)
Ee = Eexpl = 35.20 TJ(e)
Ee + Eth = 220.94 ≈ 221 TJ R = Eth/Ee = 5.28
Total $m(CO_2)$ = 1032 + 7567 + 5331 = 13930 Mg

Excavation

Direct E

Excavator
Hitachi EX2500 bucket capacity 12 m^3 ≈ 30 Mg
shaft power 1.044 MW
Hitachi EX1200 bucket capacity 6 m^3 ≈ 15 Mg
shaft power 0.567 MW
total mass rock excavated
m(rock) = $9.172 \bullet 10^6$ Mg, during 7100 operating hours
per hour $m = 9.172 \bullet 10^6/7100 = 1292$ Mg/h
number of actions = 1292/45 = 28.7 actions/h = sum of Hitachi EX2500 and Hitachi EX1200 actions (likely the Hitachi EX1200 is not used for loading dump trucks).
That would mean about 4 minutes per action per excavator.
conclusion: both excavators operate at about full power during 7100 h/a
mechanical power P = 1.611 MW
mechanical energy E(mech) = $1.611 \bullet 3600 \bullet 7100 = 41.18 \bullet 10^6$ MJ = 41.18 TJ

mass of rock excavated: m(rock) = $9172 \bullet 10^6$ Mg
=> specific energy consumption
J(mech) = Je = $41.18 \bullet 10^6$ MJ/$9.172 \bullet 10^6$ Mg = 4.49 MJ/Mg rock

Assume thermal efficiency = 40%
=> J(th) = 4.49/0.40 = 11.23 MJ/Mg rock
direct $m(CO_2) = 11.23 \bullet 75 = 0.842$ kg CO_2/Mg rock

Ranger
=> total thermal energy Eth = 41.18/0.40 = 102.95 TJ = $102.95 \bullet 10^3$ GJ
direct $m(CO_2) = 102.95 \bullet 75 = 7721$ Mg

diesel consumption
$V = 102.95 \bullet 10^3/36.0 = 2860$ m^3
$m = V \bullet 0.839 = 2399$ Mg

specific diesel consumption
$v = 2860/9172 = 0.312$ L/Mg
$m = 0.312 \bullet 0.839 = 0.262$ kg/Mg rock

Indirect E

- Embodied E of equipment

excavator 12 m^3 bucket E = 4.8 ± 0.6 TJ(th) life 3 years, 21300 operating hours
annual J = 1.6 ± 0.2 TJ(th)/a
excavator 6 m^3 bucket E = 2.4 ± 0.3 TJ(th) life 3 years, 21300 operating hours
annual J = 0.8 ± 0.1 TJ(th)/a

Specific E
Assume 1/3 of the mass excavated by the small excavator (3.057 Tg) and 2/3 by the large one (6.115 Tg).
Embodied J(th) $= 0.8 \bullet 10^{12}$ J/$3.057 \bullet 10^6$ Mg =
$= 1.6 \bullet 10^{12}$ J/$6.115 \bullet 10^6$ Mg = 0.262 MJ/Mg rock

- Maintenance (shop, lubricants, spare parts, tyres, etcetera)

Mortimer [Q98] p.240, Figure B.2 energy inputs motor scraping of earth
direct E (diesel) 14.6 MJ(th)/m3 excavated earth
indirect E: 3.8 MJ(th) /m3 earth
=> fraction indirect E = 3.8/14.6 = 0.260
Mortimer used here primary energy units: electricity converted into fossil fuel, so R not known.

Assume the same fraction applies to excavators and to trucks
direct E J(th) = 11.23 MJ/Mg rock
=> indirect E (maintenance) J(th) = $0.260 \bullet 11.23$ = 2.92 MJ/Mg rock

Total indirect E
equipment Jth = 0.262 MJ/Mg
maintenance Jth = 2.92
sum Jth = 3.18 MJ/Mg rock
indirect $m(CO_2) = 3.18 \bullet 75 = 0.239$ kg CO_2/Mg rock

Ranger

Direct E
mechanical power $P = 1.611$ MW
mechanical energy E(mech) = Ee = $1.611 \bullet 3600 \bullet 7100 = 41.18 \bullet 10^6$ MJ = 41.18 TJ
thermal efficiency = 40% Eth = 41.18/0.40 = 102.95 TJ
diesel consumption $V = 102.95 \bullet 10^3/36.0 = 2860$ m^3
$m = V \bullet 0.839 = 2399$ Mg
direct CO_2 $m(CO_2) = 102.95 \bullet 75 = 7721$ Mg

Indirect E Eth = $3.18 \bullet 9.172 \bullet 10^6$ = 29.17 TJ
indirect CO_2 $m(CO_2) = 29.17 \bullet 75 = 2188$ Mg

Total energy consumption (thermal)

$E_{exc} = E_{dir} + E_{indir} = 102.95 + 29.17 = 132.12$ TJ(th) = 132 TJ(th) (rounded)
$m(CO_2) = 7721 + 2188 = 9909$ Mg

Haulage

Assume each of the 10 dump trucks has a shaft power of P = 1 MW. The larger trucks have a slightly higher power, the smaller ones slightly lower.

capacity dump trucks	6 small trucks payload 90 Mg
	4 large trucks payload 135 Mg

total payload 6•90 + 4•135 = 1080 Mg
average load = 108 Mg/truck
Total mass of rock hauled out of the mine m = 9.172•10^6 Mg, during 7100 h
average haulage per hour = 9.172•10^6/7100 = 1292 Mg/h
average number of trips = 1292/1080 = 1.20 trips/h per truck
or 0.836 h/trip => 50 minutes/trip
assume average speed loaded out of mine v = 15 km/h and haulage distance d = 5 km
loaded trip takes 20 minutes.
assume empty return trip downhill at maximum speed v = 50 km/h => takes 6 minutes
=> 24 minutes for loading, radiometric discriminator, unloading, fuelling, etcetra.
Assume 20 minutes at full power + 6 minutes at 25% power, 24 minutes idle

Direct E

Assume speed of loaded truck climbing out of the mine v = 15 km/h, at full power.
time to cover 1 km: t = 240 sec/km, P = 1 MW
=> $J_{mech} = J_e$ = 240 s/km•1 MW = 240 MJ/km
per Mg rock
$J_{mech} = J_e$ = 240/108 = 2.22 MJ/Mg.km
Empty return trip v = 50 km/h, or 72 s/km P = 0.25 MW
=> $J_{mech} = J_e$ = 0.25•72 = 18 MJ/km
Sum: per km haulage distance d = 1 km loaded + 1 km return empty
$J_{mech} = J_e$ = 240 + 18 = 258 MJ/km
per Mg rock: $J_{mech} = J_e$ = 258/108 = 2.389 = 2.4 MJ/Mg.km
This rounded value will be used throughout this study.

Assume the diesel engines have a thermal conversion efficiency of 40% (highest value attainable at the current state of technology)
J_{th} = 2.4/0.40 = 6.00 MJ/Mg.km
diesel V = 6.00/36.0 = 0.1667 L/Mg.km
m = V•0.839 = 0.1398 kg/Mg.km

Indirect E

- Embodied E of equipment (dump trucks)

Dump truck 54 m^3 capacity	E(th) = 4.5 ± 1.7 TJ (see above section 3.1)
annually	J(th) = 4.5/3 = 1.5 ± 0.6 TJ

Average load dump trucks m = 108 Mg
hauled mass during one season (operating year) m = 9.172•10^6 Mg, 10 dump trucks
=> m = 9.172•10^5 Mg/truck per year

Specific E
Embodied J(th) = 1.5•10^{12} J/9.172•10^5 Mg = 1.64 MJ/Mg rock

hauling distance d = 5 km, so:
J(th) = 1.64/5 = 0.328 MJ/Mg.km

- Maintenance (shop, lubricants, spare parts, tyres, etcetera)

Mortimer [Q98] p.240, Figure B.2 energy inputs motor scraping of earth
direct E (diesel) 14.6 MJ(th)/m^3 excavated earth
indirect E: 3.8 MJ(th) /m^3 earth
=> fraction indirect E = 3.8/14.6 = 0.260
Mortimer used here primary energy units: electricity converted into fossil fuel, so R not known.
Assume the same fraction indirectE/directE applies to haulage trucks
direct E J(th) = 6.00 MJ/Mg.km
=> indirect E (maintenance) J(th) = 0.260•6.00 = 1.56 MJ/Mg.km

Total indirect E
equipment Jth = 0.328 MJ/Mg.km
maintenance Jth = 1.56 MJ/Mg.km
sum Jth = 1.888 MJ/Mg.km
$m(CO_2)$ = 1.888•75 = 0.142 kg/Mg.km

Ranger

direct E
hauled mass of rock m = 9.172•10^6 Mg
haulage distance d = 5 km
mechanical energy = equivalent to electricity Je = 2.4 MJ/Mg.km
thermal conversion efficiency 40% Jth = 6.00 MJ/Mg.km
total Ee = Emech = 2.40•5•9.172•10^6 = 110.06 TJ
Eth = Ee/0.40 = 275.16 TJ
diesel V = Eth/36.0 = 275.16•10^3/36.0 = 7543 m3
m = V•0.839 = 6413 Mg
$m(CO_2)$ = 275.16•75 = 20637 Mg

Indirect E
Eth = 9172•10^3•5•1.888 = 86.58 TJ
$m(CO_2)$ = 86.58•75 = 6494 Mg

Total haulage
Ehaul = Edir + Eindir = 275.16 + 86.58 = 361.72 = 362 TJ(th)
$m(CO_2)$ = 20637 + 6494 = 27131 Mg

Summary of mining energy requirements

Table 3

Specific energy consumption of mining activities. Energy consumption per Mg mined rock. The figures of the direct energy input of excavation and haulage have been deduced in this study (Q6) from ERA data (Q320 and Q321) and from Q95 and Q98.

activity	direct *E* mechan.	direct *E* * thermal	indirect *E* electric	indirect *E* thermal	unit (mined rock)	reference
drilling	0.6	1.5	-	11	MJ/Mg	Q98
explosives	–	–	3.84 **	7.75 **	MJ/Mg	Q6, Q95, Q98
excavation	4.49	11.23	–	3.18	MJ/Mg	Q6, Q320, Q321, Q98
haulage	2.40	6.00	–	1.888	MJ/Mg.km	Q6, Q320, Q321, Q98

* If thermal the conversion ratio of the diesel engines $r = 40\%$

** Assumed consumption of explosives = 0.25 Kg/Mg rock

Table 4

Direct energy consumption at Ranger by the mining activities. Assumed overburden ratio S = 3 and haulage distance d = 5 km. Mass of mined rock m = $9.172 \bullet 10^6$ Mg. Assumed thermal conversion ratio of the diesel engines r = 40%. Excluding explosives

activity	mechanical direct *E* TJ	thermal direct *E* TJ	direct *E* diesel *V* (m^3)	direct *E* diesel *m* (Mg)	indirect *E* thermal TJ	dir*E* + indir*E* thermal TJ
drilling *	5.50	13.76	382	321	100.9	114.66
excavation	41.18	102.96	2860	2400	29.17	132.13
haulage	110.06	275.16	7643	6413	86.58	361.74
total	156.74	391.88	10886	9133	216.65	608.53

* Excluding explosives (see chemicals)

Table 5

Direct and indirect energy consumption at Ranger by the mining activities. Assumed overburden ratio $S = 3$ and haulage distance $d = 5$ km. Mass of mined rock $m = 9.172 \bullet 10^6$ Mg. Assumed thermal conversion ratio of the diesel engines $r = 40\%$. Mechanical energy equals electric energy.

activity	direct *E* electric TJ	direct *E* thermal TJ	indir *E* electric TJ	indir *E* thermal TJ	dir E+indir E thermal TJ	total CO_2 Mg
drilling	5.50	13.76	–	100.9	114.66	8600
excavation	41.18	102.96	–	29.17	132.13	9910
haulage	110.06	275.16	–	86.58	361.74	27131
subtotal	156.74	391.88	–	216.65	608.53	45640
explosives	–	–	35.22	71.08	71.08	6303
total mining	156.74	391.88	35.22	287,73	679.61	51943

Diesel consumption $V = E_{th}$ (GJ)/36.0 (GJ/m^3)
$m = V$•0839 (Mg/m^3)

Energy consumption of mining at Ranger
S = 3, d = 5 km). See also Table 5.

drilling + excav + haul direct E (th) = 391.88 TJ
indirect E(th) = 216.65
explosives embodied E(e) = 35.22
embodied E(th) = 71.08
sum mining E(e) = 35.22 TJ
E(th) = 391.88 + 216.65 + 71.08 = 679.61 TJ
Grand total: E(e) + E(th) = 714.83 TJ
R = Eth/Ee = 679.61/35.22= 19.3

Mass of mined rock $m = 9.172 \bullet 10^6$ Mg
Mass of mined ore $m = 2.293 \bullet 10^6$ Mg
=> Je + Jth = 714.83/9.172•10^6 = 77.94 MJ/Mg rock (depends on *d* and *r*)
Je + Jth = 714.83/ 2.293•10^6 = 311.74 MJ/Mg ore (depends on *S*, *d* and *r*)
= 0.312 GJ/Mg ore

Energy production per Mg natural uranium

One reload charge m(U) = 162.48 Mg
gross electricity production per relad E_e(gross) = 25.86 PJ
gross electricity production per Mg U J_e(gross) = 25.86/162.48 = 0.1592 PJ/Mg
= 0.1592/3.6 = 44.21•10^6 kWh/Mg

CO_2 emission of the ore mining

$m(CO_2)$ = 51943/2.293•10^6 = 5.66 kg CO_2/Mg rock
$m(CO_2)$ = 51943/2.293•10^6 = 22.65 kg CO_2/Mg ore
$m(CO_2)$ = 51943/4957 = 10.500 Mg CO_2/Mg U
$m(CO_2)$ = 10.500•10^6/44.21•10^6 = 0.2375 g CO_2/kWh

4 Discussion

The energy requirements of mining are determined by four main variables: the overburden ratio (= stripping ratio), the haulage distance, the thermal conversion efficiency of the diesel engines powering the mining equipment and the mass ratio of explosives over mined rock. Variations introduced by different types of mining equipment are left aside in this analysis.

A fifth important parameter is the hardness of the rock to be mined, e.g.sandstone or granite. As this factor of the mining energy consumption is difficult to quantify, it is ignored in this study. However, the variable rock properties may introduce considerable variations in the specific energy requirements of mining from mine to mine. Some consequences of the mining of harder rock are, among other:

- Higher wear of equipment, such as drill rigs and excavators, causing more time in the shop and a higher rate of replacement of components and higher use of consumables. These increased rates mean an higher direct and indirect energy consumptin per Mg rock mined.

- Higher specific consumtion of explosives: harder rock nrrd more explosives per Mg rock. The Ranger mine uses 0.25 kg explosives per Mg rock, which figure is used in this study as a world average. Likely there are mines with a significant higher explosives consumption. Underground mines have a higher explosives consumptions anyway.

The specific energy consumption of mining per Mg ore are given by the following equation 1:

$$\begin{aligned} J_{\text{mining}} &= (S + 1)\bullet\{(J_{\text{d+b}} + J_{\text{exc}} + d\bullet J_{\text{haul}})/r + J_{\text{d+b}}(\text{indir}) + J_{\text{exc}}(\text{indir}) + d\bullet J_{\text{haul}}(\text{indir})\} = \\ &= (S + 1)\bullet\{(5.09 + d\bullet 2.40)/r + 14.18 + \text{x}\bullet 46.35 + d\bullet 1.888\} \qquad \text{eq 1} \end{aligned}$$

J_{mining} [=] MJ_{th}/Mg ore

Here is:
$J_{\text{d+b}}$ = direct mechanical energy input of drilling and blasting (MJ_{e}/Mg ore)
J_{exc} = direct mechanical energy input of excavation (MJ_{e}/Mg ore)
J_{haul} = direct mechanical energy input of haulage (MJ_{e}/Mg ore)
S = overburden ratio (= stripping ratio)
d = haulage distance km
r = thermal conversion efficiency
x = mass ratio explosives over mined rock kg/Mg

In case of the Ranger mine $x = 0.25$ and equation 1 is reduced to:

$$J_{\text{mining}} = (S + 1)\bullet\{(5.09 + d\bullet 2.40)/r + 25.77 + d\bullet 1.888\} \qquad \text{eq 2}$$

The direct energy input is mechanical, equivalent to electricity, and is converted into thermal energy units by the thermal conversion ratio r. The indirect energy input – to produce equipment, drill bits, explosives, etcetera – is partly electric, partly thermal. The thermal and electric energy inputs are added to in above equation. As the thermal/electric ratio R is known ($R = 19.3$), both components can easily be calculated from the result.

In the calculations of the energy inputs of Ranger, the electric and thermal inputs are kept separate, according to the methodology followed throughout the Q6 study. The indirect electric inputs are not converted into virtual primary energy units, as is done in most other studies.

The figures used in this study (Storm & Smith 2008 [Q6]) for estimating the energy consumption of the recovery of uranium from the earth's crust, have been adopted from the study of Rotty et al. 1975 [Q95]. The Rotty study in turn has been based on a thourough survey of a large number of uranium mines, open pit as well as underground, by the US Bureau of Mines in 1973. This survey can be considered to be the most thourough and most reliable examination of the energy consumption of uranium mining and milling known up until today.

In the Q6 study we adopted the Rotty figures as a world average of uranium mining and milling, for the following reasons:

- The figures are based on the most reliable and extensive data base available up until today and reflect the average of a number of open pit and underground mines.
- The technology of mining and recovery of uranium has hardly changed since the 1970s.
- As the Rotty figures are valid for soft rock mining only, one may expect not to overstate the energy consumption of the world average mine. After all, the world uranium mines include many with hard rock.

The figures for mining from Rotty et al 1975 [Q95] are:

$J_{\text{mining}} = J_{\text{e}} + J_{\text{th}} = 1.056$ GJ/Mg ore $\qquad R = 8.0$

Found in this analysis:
Ranger $J_{\text{mining}} = J_e + J_{th}$ = 0.312 GJ/Mg ore $R = 19.3$

The specific energy consumption of mining at Ranger turns out significantly lower than the figure of Rotty, as expected for several reasons:

- The figures of Rotty are the average of a large number of underground and open cast mines.
- Underground mining is significantly higher energy-intensive than open pit mining.
- The ratio of waste rock to ore S varies widely among the surveyed mines and can be as high as 50.
- The haulage distances d may vary from one mine to another from a few kilometres to more than 200 km.
- Water pumping is not included in the Ranger figure, so the real figure may be higher.
- The specific consumption of explosives x may vary considerably from mine to mine. Underground mines, included in ther Rotty figures, have a higher explosives consumption than open pit mines with the same type of rock.

The differences of the value of R (the thermal/electric ratio) between Rotty and Ranger can be explained by the fact that underground mining is mainly performed by electrically powered equipment. This causes a lower value of R. A second factor is that in some open pit mines electric drill rigs and electric excavators are applied, with draw their energy form the local grid.

The fact that the Rotty figures are based on the survey of a large number real uranium mines was the reason to choose these figures as a world average in this study. Apparently the authors of the authoritative study ERDA 1976 [Q109] had similar considerations, as they also adopted the Rotty figures. Today there are still no better figures available in the literature, which could be applied as a world average.

Ranger has a large ore body, relatively rich ore ($G >$ 0.2% U), a low overburden ratio of $S = 3$ and a low haulage distance: we assumed $d = 5$ km. This distance is not disclosed by ERA and has been estimated from a satellite photograph. In addition we assumed a high conversion ratio of diesel engines (r = 40%), the highest attainable ratio at the current state of technology. In practice this may be lower, e.g. r = 30%. The large influence of this ratio is shown by Figure 4.
The Ranger mine is one of the cheapest producing uranium mines of the world, due to its favourable conditions and properties, so it is no surprise that the specific energy consumption per Mg ore at Ranger is lower than de world average of open pit mines, let alone underground mines.

The specific energy consumption figures in Table 3 are independent of the overburden ratio and the haulage distance.
By means of above equation 1 or 2 the conditions can be calculated at which an open pit mine similar to Ranger would have a specific energy consumption of mining as large as the Rotty figures. The graphs in Figure 4 show the large dependency of the mining energy on the three parameters S, d and r, with a fixed value of $x = 0.25$ (as at Ranger).

An overburden ratio of $S = 4$, a haulage distance of $d = 20$ km and a thermal efficiency of the diesel engines $r = 0.30$, for instance, can still be considered benign mining conditions in a global context.

Above analysis of the energy consumption of mining at Ranger demonstrate that the figures of Rotty will not lead to an overstated value of the global average mining energy consumption.

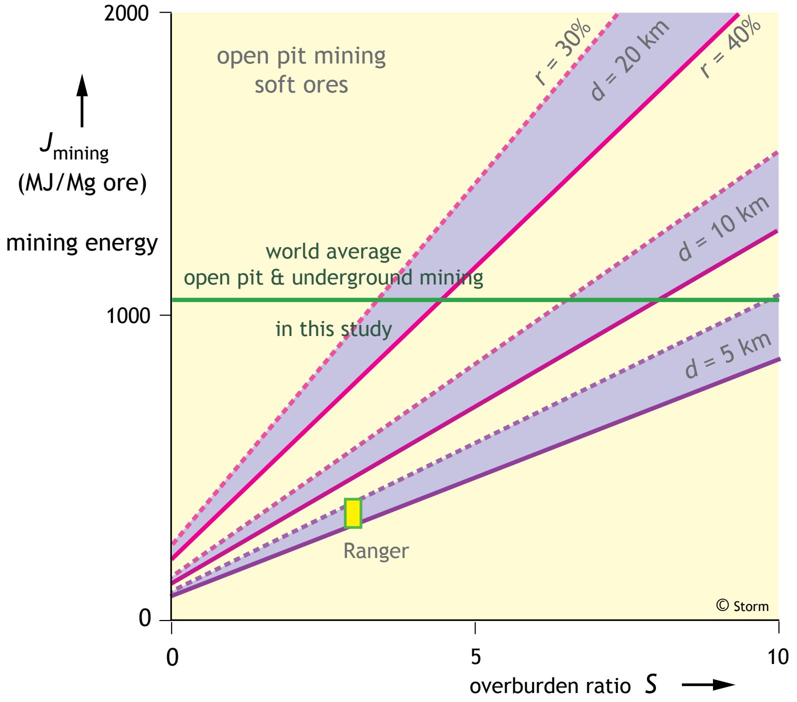


Figure 4
The mining energy consumption (MJ/Mg ore) as function of the overburden ratio S, at three different hauling distances and two different diesel engine efficiencies. In this diagram the explosives consumption is assumed x = 0.25 kg/Mg. At x = 0.50 kg/Mg the values of J_{mining} would increase by about 15% at short hauling distances (e.g. d = 5 km) to about 5% at longer hauling distances (e.g. d = 20 km).
There are mines with overburden ratios as high as 50. Hauling distances of up to 200 km are reported. Consequently the mining energy input at many mines in the world will be a multiple of the world average figure.

5 Ore processing

Crushing & grinding

Operating time, load factor
ERA-AR 2005 [Q321] p.6
Milling capacity per operating hour

Mill 1	capacity	= 216 Mg/h
Mill 2	capacity	= 107 Mg/h
	sum	= 323 Mg/h

1 year = 8760 h
If load factor *L* = 1:
Full year capacity m(ore) = 323•8760 = 2829480 Mg/year

Actually milled m(ore) = $2293 \cdot 10^3$ Mg
=> load factor *L* = 2293/2829 = 0.8105

Operating time = 0.8105•8760 = 7100 h/a

Direct E

SAG mills (semi-autogeneous grinding) generally draw 2689-22800 hp power (Engineering and Mining Journal), = 3.6 – 31.0 MWe, depending on size.
11.8 m mill draws 18 MWe, capacity probably some 800 Mg/h (Equinox Copper)
so specific power could be about 18/8 = 2.25 MW/100 Mg/h capacity
assume Ranger mill #1 draws $P = 4.9$ MWe
Ranger mill #2 draws $P = 2.4$ MWe
sum $P = P_{max} = 7.3$ MWe
year average $P_{av} = L \bullet P_{max} = 0.8105 \bullet 7.3 = 5.92$ MWe
Total electric E consumed by the mills during 7100 operating hours
$E_e = 7.3 \bullet 3600 \bullet 7100 = 186.6 \bullet 10^6$ MJ = 186.6 TJ(e)

Specific E consumption
m(ore) = $2293 \bullet 10^3$ Mg
Je = $187 \bullet 10^6$ MJ/$2.293 \bullet 10^6$ Mg = 81.55 MJ/Mg
This figure is used in this study.

Compare with Mortimer [Q98] p 252-254:
hard ores Je = 74.2 – 81.5 MJ/Mg ore
soft ores Je = 10.6 MJ/Mg ore

Indirect E

[Q98] p. 252
capital Je = 3.03 MJ/Mg ore
Jth = 22.72 MJ/Mg ore
balls & rods Je = 1.67 MJ/Mg ore
Jth = 15.15 MJ/Mg ore
sum Je = 4.70 MJ/Mg ore
Jth = 37.87 MJ/Mg ore
Je + Jth = 42.57 MJ/Mg ore
R = Jth/Je = 37.87/4.70 = 8.06
Maintenance not included.

Indirect CO_2 emission
The specific CO_2 emission from the indirect thermal energy input, assuming it is supplied as oil, is calculated according to the equation:

$m(CO_2) = J_{th}$ (MJ)•75 g/MJ [Q27].

=> $m(CO_2) = 37.87 \bullet 75 = 2.840$ kg CO_2/Mg ore

Ranger

Total E consumption crushing & grinding
direct E Ee = 186.6 TJ
This amount of electricity is generated on site by diesel-fuelled generators.

indirect E (imported materials and embodied E)
Ee = 4.70 MJ/Mg•$2.293 \bullet 10^6$ Mg = 10.78 TJ
Eth = 37.87 MJ/Mg•$2.293 \bullet 10^6$ Mg = 86.84 TJ

sum Ee + Eth = 97.62 TJ
R = 8.06
indirect CO_2: $m(CO_2) = 2.840 \bullet 2.293 \bullet 10^6$ kg = 6512 Mg
or: $m(CO_2) = 86.84 \bullet 75$ = 6513 Mg

Leaching

Acid leaching with sulfuric acid, pH 1.5, commonly occurs at ambient temperature.
Alkaline leaching at higher temperatures, 75-80°C. (Encyclopedia Brittanica)
pH = 1.5 => $c(H^+) = 0.032$ mol/L
$c(H_2SO_4) = 0.016$ mol/L = 0.016•98 = 1.6 g/L

Direct E

Mortimer [Q98] p 252-254
acidic ore:
J_e = 42.4 MJ/Mg ore
J_{th} = 230 MJ/Mg ore
$J_e + J_{th}$ = 272.4 MJ/Mg ore $R = J_{th}/J_e = 230/42.4 = 5.42$

alkaline ore
J_e = 9 – 190 MJ/Mg ore
J_{th} = 107 -376 MJ/Mg ore
Je +J_{th} = 116 – 566 MJ/Mg ore $R = J_{th}/J_e =$ 11.9 – 2.0
The values of acidic leaching are about the average of the alkaline figures.

Check
Assume volume diluted acid (leaching liquid) $V = 1\ m^3$ per Mg ore
Assume leaching occurs at 80 °C and the slurry has to be heated from 25 –> 80 °C, so:
ΔT = 55 °C = 55 K.
heat capacity aqueous solution c_p = 4.2 J/g.K
heat capacity solids, assume equal to that of copper sulfate: c_p = 120 J/mol.K
$M(CuSO_4)$ = 160 g/mol
=> c_p = 120/160 = 0.75 J/g.K
Heating the slurry ΔT = 55 K would consume per Mg ore
ΔH(solids) = 1•0.75•55 = 41 MJ/Mg
ΔH(solution) = 1•4.2•55 = 231 MJ/Mg
sum: J_{th} = 272 MJ/Mg ore

The above found value of J_{th} is nearly the same value as given by Mortimer.
Above approximation indicates that the values of Mortimer are plausible.
However, if the leaching is performed at ambient temperature, the thermal E input becomes zero. This analysis assumes a zero thermal input at Ranger.

Indirect E

Equal for acidic and alkaline ores
J_e = 1.5 MJ/Mg ore
Jth = 10.6 MJ/Mg ore
Je + Jth = 12.1 MJ/Mg ore

$R = J_{th}/J_e = 10.6/1.5 = 7.07$

Indirect CO_2 emission

The specific CO_2 emission from the indirect thermal energy input, assuming it is supplied as oil, is calculated according to the equation: $m(CO_2) = J_{th}$ (MJ)•75 g/MJ [Q27].

=> $m(CO_2) = 10.6 \cdot 75 = 0.795$ kg CO_2/Mg ore

Ranger

Ranger has acidic ore.

Energy consumption excluding embodied E in chemicals.

Probably the acid leaching at Ranger occurs at ambient temperatures, so no direct thermal E input is needed.

Direct E

Je = 42.4 MJ/Mg ore

Ee = $2.293 \cdot 10^6$ Mg•42.4 MJ/Mg = 97.2 TJ(e)

This amount of E, problably consumed by pumps and stirrers, is to be generated by the Ranger generators.

indirect E

Ee = $2.293 \cdot 10^6$ Mg•1.5 MJ/Mg = 3.44 TJ(e)

Eth = $2.293 \cdot 10^6$ Mg•10.6 MJ/Mg = 24.31 TJ(th)

sum Ee + Eth = 27.75 TJ

R = Eth/Ee = 24.31/3.44 = 7.07

indirect CO_2: $m(CO_2) = 0.795 \cdot 2.293 \cdot 10^6 = 1.823 \cdot 10^6$ kg = 1823 Mg

Embodied E in chemicals is separately addressed in section ‘Chemicals’.

Extraction

Two methods: by solvent extraction and by ion exchange. At Ranger the solvent extraction method is applied.

The solvent extraction method uses tertiary amines in an organic solvent (a special grade of kerosene). First the amines in the organic phase, R_3N(org), react with sulfuric acid:

$$2\ R_3N(org) + H_2SO_4(aq) \quad \rightarrow \quad (R_3NH)_2SO_4(org)$$

Then the amine sulfate extracts the uranyl ions from the aqueous phase into the organic phase. In case of the uranyl sulfate ion, the following reactions occur:

$$(R_3NH)_2SO_4(org) + UO_2(SO_4)_3^{4-}(aq) \rightarrow (R_3NH)_2UO_2(SO_4)_2(org) + 2\ SO_4^{2-}(aq)$$

Then the uranium ions are removed from the organic phase into an acid aqueous solution in the strip section. By adding ammonia, to neutralize the strip solution, ammonium diuranate $(NH_4)_2U_2O_7$ is precipitated. Excess water is removed in the thickener section and then in a centrifuge. In the calciner the ammonium diuranate is calcined at 800 °C to produce uranium oxide U_3O_8.

Direct E

Mortimer [Q98] p.252-254

solvent extraction		Je = 3.6 MJ(e)/Mg ore	Jth = 0
ion exchange	acidic	Je = 71.2 MJ/Mg ore	Jth = 0
	alkaline	Je = 266 MJ/Mg ore	Jth = 527 MJ/Mg ore

Indirect E

Mortimer [Q98] p.252-254
solvent extraction Je = 1.8 MJ(e)/Mg ore
Jth = 5.4 MJ/Mg ore

ion exchange Je = 1.8 MJ/Mg ore
Jth = 4.5 MJ/Mg ore

Solvent extraction. The specific CO_2 emission from the indirect thermal energy input, assuming it is supplied as oil, is calculated according to the equation:

$m(CO_2) = J_{th}$ (MJ)•75 g/MJ [Q27].
=> $m(CO_2) = 5.4 \bullet 75 = 0.405$ kg CO_2/Mg ore

Ranger

At Ranger the solvent extraction method is applied.
direct E Ee = $2.293 \bullet 10^6$ Mg•3.6 MJ/Mg = 8.25 TJ(e)
supplied by the Ranger generators

indirect E Ee = $2.293 \bullet 10^6$ Mg•1.8 MJ/Mg = 4.13 TJ(e)
Eth = $2.293 \bullet 10^6$ Mg•5.4 MJ/Mg = 12.38 TJ(th)

sum direct E + indirect E, excluding chemicals
Ee = 8.25 + 4.13 = 12.38 TJ
Eth = 12.38 TJ
Ee + Eth = 24.76 TJ
R = Eth/Ee = 1.00
indirect CO_2 emission: $m(CO_2) = 0.405 \bullet 2.293 \bullet 10^6 = 0.929 \bullet 10^6$ kg = 929 Mg

Drying wet cake and calcining yellow cake

Direct E

The last step of the uranium extraction process is drying the wet cake (the precipitate from the previous stage) to solid yellow cake, ammonium diuranate $(NH_4)_2U_2O_7$.
At Ranger drying is combined by calcining the yellow cake at 800 °C in an oil-fired multi-hearth calciner, into U_3O_8 (see flowsheet in [Q320]).

As no specific process data are available, this study roughly approximates the energy input of the drying and calcination process, as described below.

Water heat of vaporisation (at ≈ 100 °C) ΔH = 40 kJ/mol (Handbook of Chemistry & Physics, p. 6-10), or 2.22 MJ/kg

If we assume in yellow cake $U_3O_8 : H_2O = 1 : 4$ (this ratio is not given, but likely the ammonium diuranate precipitate is strongly hydrated)
=> per Mg U_3O_8 Jth = 4•2.22 = 8.88 GJ/Mg U_3O_8
Additional heat is consumed to heat mixture to 800 °C and to drive off the ammonia.

A very rough estimate of the energy to heat the yellow cake to 800 °C can be done if we assume the heat capacity of the yellow cake and uranium oxide to be of the same order of magnitude as of copper sulfate:
c_p = 120 J/mol.K
$m(U_3O_8)$ = 1 Mg
$n(U_3O_8)$ = 1/842 = 0.00119 Mmol
ΔT = 800 – 25 = 775 K
=> ΔH = 0.00119•775•120 = 111 MJ/Mg

The reaction enthalpy of the calcining process is not disclosed. In a rough approximation we assume it to be of the same order of magnitude as the reaction enthalpy of calcining calcium carbonate:
Je = 0.07 GJ/Mg
Jth = 8.4 GJ/Mg
Je + Jth = 8.47 GJ/Mg

Based on a above assumptions, a rough estimate the energy consumption of the last step, drying and calcining, would be:

drying	Jth = 8.88	GJ/Mg U_3O_8
heating to 800 °C	Jth = 0.11	GJ/Mg U_3O_8
calcining	Jth = 8.5	GJ/Mg U_3O_8
sum (rounded)	Jth = 17.5	GJ/Mg U_3O_8

This figure does not include the energy consumption of the thickener and centrifuge steps, probably electric, preceding the drying + calcining process.

Mortimer [Q98] p 252-254 gives as the specific energy consumption of drying (probably excluding calcination):
Jth = 175 – 190 GJ/Mg U_3O_8
These values are much higher than above approximation, and may be based on the drying of the wet cake (precipitate) without thickener and centrifuge.

Here we adopt the lower figure, deduced above:
Jth = 17.5 GJ/Mg U_3O_8
or Jth = 17.5/0.848 = 20.6 GJ/Mg U
assumed to include calcining at 800 °C.
As not all partial processes are included, this figure might be an underestimation.

Indirect E

Mortimer [Q98] p 252-254:
Jth = 3 – 4 GJ/Mg U_3O_8
or Jth = 3.5 – 4.7 GJ/Mg U
In this study the low value is used: Jth = 3.5 GJ/Mg U

The specific CO_2 emission from the indirect thermal energy input, assuming it is supplied as oil, is calculated according to the equation:
$m(CO_2) = J_{th}$ (MJ)•75 g/MJ [Q27].

=> $m(CO_2)$ = 3.5 GJ•75 = 0.2625 Mg CO_2/Mg U
(or $m(CO_2)$ = 0.263•0.848 = 0.2226 Mg CO_2/Mg U_3O_8)

Ranger

direct E
To produce $m(U)$ = 4947 Mg
Eth = 4947•20.6 = 102 TJ
$m(CO_2)$ = 102•75 = 7650 Mg

Required fuel oil to fire the calciner
combustion heat of fuel oil J_{th} = 36.00 GJ/m^3
density d = 0.939 Mg/m^3
V(oil) = 102•10^3/36.00 = 2833 m^3
m(oil) = 2833•0.939 =2661 Mg

Indirect E
Eth = 4947 Mg•3.5 GJ/Mg = 17.3 TJ

sum direct E + indirect E, excluding chemicals
Ee = 0 TJ
Eth = 102 + 17.3 = 119.3 TJ
indirect CO_2 emission: $m(CO_2)$ = 0.2625•4947 = 1298 Mg
or: $m(CO_2)$ = 17.3•75 = 1298 Mg
direct + indirect $m(CO_2)$ = 7650 + 1298 = 8948 Mg
specific $m(CO_2)$ = 8948/4947 = 1.809 = 1.81 Mg CO_2/Mg U
or: $m(CO_2)$ = (20.6 + 3.5)•75 = 1.808 = 1.81 Mg CO_2/Mg U

6 Embodied E in chemicals

Except of the sulfuric acid production, no data are available on the consumption of other chemicals by Ranger. The estimates below are based on stoichiometric considerations and on data from Rotty [Q95] and Mortimer [Q98].

- **Ammonia NH_3**

Embodied E Ammonia NH_3
Mortimer cites only direct E input, so we use Rotty et al. 1975 [Q95] p.57:
Je = 35.9 GJ/Mg
Jth = 50.7 GJ/Mg
Je + Jth = 86.6 GJ/Mg R = Jth/Je = 1.41

Usually ammonia is in several steps produced from methane, water and air, according to the endothermic sum reaction:

$$7\ CH_4 + 10\ H_2O + 8\ N_2 + 2\ O_2 \rightarrow 7\ CO_2 + 16\ NH_3$$

From this reaction equation follows a molar ratio of $n(CO_2) : n(NH_3) = 7 : 16$
and a mass ratio $m(CO_2) : m(NH_3)$ = 7•44 : 16•17 = 1.132

The chemical CO_2 production, accompanying the ammonia production

$m(CO_2) = 1.132$ Mg CO_2/Mg NH_3.

The specific CO_2 emission from the thermal energy input, assuming it is supplied as oil (could also be gas or coal, we take oil as a kind of average fuel), is calculated according to the equation:

$m(CO_2) = J_{th}$ (MJ)•75 g/MJ [Q27].
=> $m(CO_2) = 50.7•75 = 3.803$ Mg CO_2/Mg NH_3
The total specific CO2 emission is
$m(CO_2) = 1.132 + 3.803 = 4.935$ Mg CO_2/Mg NH_3

Ranger

Acid strip solution after the solvent extraction is neutralised with ammonia NH3 (flowsheet Q320) to let precipitate ammonium diuranate $(NH_4)_2U_2O_7$.
Calcining:
$(NH_4)_2U_2O_7$ –> 2/3 U_3O_8 + 2 NH_3 + H_2O + $^2/_6$ O_2
M = 624 M = 842 g/mol
1 mol U_3O_8 –> 1.5 mol $(NH_4)_2U_2O_7$ –> 3 mol NH_3

recovered $m(U_3O_8)$ = 5833 Mg
$n(U_3O_8)$ = 5833 Mg/842 g/mol = 6.93 Mmol

stoichiometric minimum
$n(NH_3) = 3•6.93 = 20.8$ Mmol
$m(NH_3) = 20.8•17 = 353$ Mg
In process excess NH_3 needed to neutralise acid solution, if we assume 2x stoichiometric ratio (may be a low estimate, as the strip solution is rather diluted in uranium ions)
=> consumed $m(NH_3)$ = 706 Mg
specific: $m(NH_3)$= 706/5833 = 0.121 Mg NH_3/Mg U_3O_8

Embodied E
Ee = 706•35.9 = 25.35 TJ
Eth = 706•50.7 = 35.79 TJ
$m(CO_2)$ = 4.935•706 = 3484 Mg CO_2

- **Nitric acid HNO_3**

Nitric acid is produced from ammonia according to the sum reaction:
$NH_3 + 2\ O_2$ –> $HNO_3 + H_2O$
M = 17 M = 63 g/mol
molar ratio $n(NH_3) : n(HNO_3) = 1 : 1$
mass ratio $m(NH_3) : m(HNO_3) = 17 : 63 = 0.270$

embodied E from NH_3 Je = 0.270•35.9 = 9.69 GJ/Mg HNO_3
Jth = 0.270•50.7 = 13.69 GJ/Mg HNO_3
direct E Mortimer [Q98] p.34 Je = 0.11 GJ/Mg
Jth = 12 GJ/Mg
sum: Je = 9.69 + 0.11 = 9.80 GJ/Mg
Jth = 13.69 + 12 = 25.69 GJ/Mg
Je + Jth = 35.49 GJ/Mg

$R = Jth/Je = 25.69/9.80 = 2.62$

The chemical CO_2 production from the NH_3 production is:
$m(CO_2) = 0.270 \bullet 4.935 = 1.332$ Mg CO_2/Mg HNO_3
The specific CO_2 emission from the direct thermal energy input, assuming it is supplied as oil, is calculated according to the equation:

$m(CO_2) = J_{th}$ (MJ)$\bullet$75 g/MJ [Q27]

=> $m(CO_2) = 13.69 \bullet 75 = 1.027$ Mg CO_2/Mg HNO_3

The total specific CO_2 emission is

$m(CO_2) = 1.132 + 1.027 = 2.159$ Mg CO_2/Mg HNO_3

Ranger

At Ranger HNO_3 is used in the stripping solution after solvent extraction. No data are disclosed by ERA.
If we assume that the stripping solution, from which the ammonium diuranate will be precipitated, contains the equivalent of 10 g/L U_3O_8 and has pH = 3, the amount of consumed HNO_3 can be roughly approximated.
$m(U_3O_8) = 5833$ Mg
=> V(solution) $= 5833 \bullet 10^3$ kg/10 kg/m^3 $= 5.833 \bullet 10^5$ m^3
pH = 3 => $c(HNO_3) = 1 \bullet 10^{-3}$ mol/L = 0.063 g/L = 0.063 kg/m^3
=> $m(HNO_3) =\ = 36.75 \bullet 10^3$ kg = 37 Mg, just to get the pH at 3.
In addition HNO_3 is consumed in the reaction of the organic complex from the kerosene into ammonium diuranate.
Above calculated $n(NH_3) = 20.8$ Mmol
$n(HNO_3) = n(NH_3) = 20.8$ Mmol
$m(HNO_3) = 20.8 \bullet 63 = 1310$ Mg
=> total mass of nitric acid consumed $m(HNO_3) = 1350$ Mg
This amount corresponds with a specific nitric acid consumption of:

$m(HNO_3) = 1350/4947 = 0.273$ Mg/Mg U

and a contribution to the CO_2 emission of:

$m(CO_2) = 2.159 \bullet 0.273 = 0.589$ Mg CO_2/Mg U

Embodied E
Ee = $1350 \bullet 9.80 = 13.23$ TJ
Eth = $1350 \bullet 25.69 = 34.68$ TJ
$m(CO_2)$ = $2.159 \bullet 1350 = 2915$ Mg CO_2

- **Ammonium nitrate NH_4NO_3**

Embodied energy calculated starting from NH_3
$NH_3 + HNO_3 \rightarrow NH_4NO_3$
$M(NH_3) = 17$ $M(HNO_3) = 63$ $M(NH_4NO_3) = 80$ g/mol
1 Mg NH_4NO_3 formed from: 17/80 = 0,2125 Mg NH_3 + 63/80 = 0.7875 Mg HNO_3
embodied E
Je(NH_3) = $0.2125 \bullet 35.9$ = 7.63 GJ/Mg
Je(HNO_3) = $0.7875 \bullet 9.80$ = 7.72 GJ/Mg
sum Je = 15.35 GJ/Mg
Jth(NH_3) = $0.2125 \bullet 50.7$ = 10.77 GJ/Mg
Jth(HNO_3) = $0.7875 \bullet 25.69$ = 20.23 GJ/Mg
sum Jth = 31.00 GJ/Mg
Je + Jth = 46.35 GJ/Mg

R = Jth/Je = 31.00/15.35 = 2.02

If assumed no direct E is needed to produce ammonium nitrate from ammonia and nitric acid, for it is an exothermic reaction, the CO_2 production is entirely due to the production of NH_3 and HNO_3.
The chemical CO_2 production from the NH_3 production is:
$m(CO_2) = 0.2125 \bullet 4.935 = 1.049$ Mg CO_2/Mg NH_4NO_3
and from the HNO_3 production:
$m(CO_2) = 0.7875 \bullet 2.159 = 1.700$ Mg CO_2/Mg NH_4NO_3
The total specific CO_2 emission is
$m(CO_2) = 1.049 + 1.700 = 2.749$ Mg CO_2/Mg NH_4NO_3

If Je + Jth is converted into primary energy units, as in the Mortimer study [Q98], then:
Jth = 3•15.35 + 31 = 77 MJ/kg
Mortimer [Q98] cites for an unspecified explosive:
Jemb = Je + Jth = 0.87 + 70 = 71 MJ/kg R = Jth/Je = 70/1 = 70

In this study we assume the embodied energy of explosives to be the same as of ammonium nitrate:
Je = 15.35 GJ/Mg
Jth = 31.00 GJ/Mg
Je + Jth = 46.35 MJ/kg R = 2.02
$m(CO_2) = 2.749$ Mg CO_2/Mg explosive

Ranger

Consumption of explosives in 2005: *m* = 2293 Mg
Embodied E explosives
Ee = 2293•15.75 = 35.20 TJ
Eth = 2293•31.00 = 71.08 TJ
Ee + Eth = 107.19 TJ
$m(CO_2) = 2293 \bullet 2.749 = 6303$ Mg

• Sulfuric acid H_2SO_4

Production capacity on site is 250 Mg/day. Not clear if full-load days are meant or the year average. Here assumed it to be the year average.
=> production $m(H_2SO_4) = 365 \bullet 250 = 91250$ Mg/a.
At Ranger in 2005 H_2SO_4 produced from imported elementary sulfur S_8.

$S_8 + 12\ O_2 + 8\ H_2O \rightarrow 8\ H_2SO_4$
M = 32 M = 98 g/mol
1 Mg H_2SO_4 from 32/98 = 0.3265 Mg S
=> $m(S) = 0.3265 \bullet 91250 = 29796$ Mg

H_2SO_4 consumption
milled ore $m = 2.293 \bullet 10^6$ Mg
$m(H_2SO_4) = 91250/2.293 \bullet 10^6 = 0.0398$ Mg/Mg = 40 kg H_2SO_4/Mg ore

U_3O_8 in milled ore $m(U_3O_8) = 6604$ Mg (Table 2)
$m(H_2SO_4) = 91250/6604 = 13.82$ Mg/Mg

Reactions during leaching (oxidizing and dissolving) not disclosed, so it is not possible to calculate the stoichiometric H_2SO_4 consumption.
A large excess of H_2SO_4 is necessary to keep pH at 1.5 in a large volume of liquid. Other chemical species in the ore may also consume H_2SO_4.

The production of from elemental sulfur is an exothermic process. The combustion heat of the sulfur is sufficient to run the H_2SO_4 plant. A part of the excess heat (steam) may be available for use outside of the H_2SO_4 plant, e.g. electricity generation (steam turbine)? But this electricity may be consumed internally by the H_2SO_4 plant itself.

$^1/_8\, S_8 + O_2 \rightarrow SO_2$ $\Delta H = -297$ kJ/mol SO_2 (reaction enthalpy)

if we write $^1/_8\, S_8 = S$, then:

$S + O_2 \rightarrow SO_2$

$m(S) = 1$ Mg => $n(S) = 1$ Mg/32 g/mol = 0.03125 Mmol

$\Delta H = 0.03125 \bullet 10^6 \bullet 297 \bullet 10^3 = 9.281$ GJ/Mg S

This amount of heat has been imported to Ranger embodied in the imported sulfur S.

Indirect E of construction, maintaining and operating the H_2SO_4 plant are not known and are ignored in this study.

- **Sulfur S**

Average production energy input of elementary sulfur S (Mortimer [Q98] p28-29):

Je = 1.5 GJ/Mg
Jth = 40 GJ/Mg

The specific CO_2 emission from the thermal energy input, assuming it is supplied as oil, is calculated according to the equation: $m(CO_2) = J_{th}$ (MJ)•75 g/MJ [Q27].

=> $m(CO_2) = 40 \bullet 75 = 3.000$ Mg CO_2/Mg S

Ranger

Imported mass of sulfur at Ranger in 2005: m(S) = 29796 Mg

Ee = 29796•1.5 = 44.69 = 44.7 TJ
Eth = 29796•40 = 1191.8 TJ
$m(CO_2)$ = 29796•3.000 = 89388 Mg

The combustion heat of the imported sulfur should be added to the energy consumption of the Ranger mine. This heat is used to produce sulfuric acid.

$m(S)$ = 29796 Mg

reaction enthalpy	Jth = 9.28 GJ/Mg S (see previous section)	
Total generated heat	Eth = 29796•9.28 = 276.51	TJ
per Mg ore	Jth = 276.51/2.293 = 120.59	MJ/Mg ore

Sum sulfur production + combustion heat:

Ee = 44.7 TJ
Eth = 1191.8 + 276.5 = 1468 TJ
Per Mg ore: Je = 44.7 TJ/2.293 Gg = 0.0195 GJ/Mg ore
Jth = 1468 TJ/2.293 Gg = 0.640 GJ/Mg ore

- **lime CaO**

CaO produced from limestone $CaCO_3$ by heating. The reaction enthalpy of calcining calcium carbonate is:

$CaCO_3 \;\; \text{–>} \;\; CaO + CO_2$ $\quad \Delta H = + 0.177$ MJ/mol (endothermal process)

M(CaO) = 56 Mg/Mmol
=› 1 Mg CaO = 0.01786 Mmol
=› $\Delta H = 0.01786 \bullet 0.177 = 3.16$ GJ/Mg

Mortimer [Q98] p34 gives as embodied E of CaO: Je = 0.07 GJ/Mg
Jth = 8.4 GJ/Mg
Je + Jth = 8.47 GJ/Mg

These values look quite reasonable, as the practical energy input of a chemical process is always much higher than the thermodynamical minimum.

Molar ratio $n(CO_2) : n(CaO) = 1 : 1$
mass ratio $m(CO_2) : m(CaO) = 44 : 56 = 0.7857$
The chemical production is $m(CO_2) = 0.786$ Mg CO_2/Mg CaO
The specific CO_2 emission from the thermal energy input, assuming it is supplied as oil, is calculated according to the equation: $m(CO_2) = J_{th}$ (MJ)•75 g/MJ [Q27].
=› $m(CO_2) = 8.4 \bullet 75 = 0.630$ Mg CO_2/Mg CaO
The total specific CO2 emission is
$m(CO_2) = 0.786 + 0.630 = 1.416$ Mg CO_2/Mg CaO

Ranger

Lime used to neutralise raw acid solution.
$CaO + H_2SO_4 + H_2O \;\; \text{–>} \;\; CaSO_4.2H_2O$ (gypsum)
M = 56 g/mol
Assume half of the amount of sulfuric acid has to be neutralised
=› m(H2SO4) = 0.5•91250 Mg
n(H2SO4) = (0.5•91250)/98 = 466 Mmol
=› n(CaO) = 466 Mmol
m(CaO) = 466•56 = 26096 Mg
Embodied E
Ee = 26096•0.07 = 1.83 TJ
Eth = 26096•8.4 = 219.2 TJ
$m(CO_2)$ = 26096•1.416 =36952 Mg

- **sodium chlorate $NaClO_3$**

In the leaching process the tetravalent U(IV) ions are to be oxidized to hexavalent U(VI) ions to make them soluble in aqueous solution. In uraninite, coffinite and brannerite the majority of the uranium ions are tetravalent.
Commonly used oxidants in uranium ore processing are sodium chlorate $NaClO_3$, manganese dioxide MnO_2, iron(III) sulfate $Fe_2(SO_4)_3$ and sometimes also hydrogen peroxide H_2O_2.
Assume Ranger uses $NaClO_3$.
$ClO_3^- + 3\,UO_2 + 6\,H^+ + 3\,SO_4^{2-} \;\; \text{–>} \;\; Cl^- + 3\,UO_2^{2-} + 3\,SO_4^{2-} + 3\,H_2O$
molar ratio: $n(ClO_3^-) : n(UO_2) = 1 : 3$
$M(NaClO_3) = 106.4$ $\quad M(UO_2) = 270$ g/mol
stoichiometric mass ratio:

$m(NaClO_3) : m(UO_2) = 106.4 : 3•270 = 1 : 7.61$

Encyclopedia Brittanica: typically 1.5 kg sodium chlorate per Mg ore suffice to oxidize the uranium ions.
$m(NaClO_3)$ = 1.5 kg
$n(NaClO_3)$ = 1500/106.4 = 14.1 mol

Mortimer [Q98] p.252: consumption = 1.2 kg/Mg ore (G = 0.2% U3O8)
This figure is a somewhat higher than Encycl Britt cites, when corrected for the higher ore grade at Ranger:
$m(NaClO_3)$ = 0.288/0.2)•1.2 = 1.7 kg/Mg ore

Embodied E
Mortimer [Q98] p252 Je = 21 GJ/Mg
Jth = 24 GJ/Mg
The specific CO_2 emission is calculated from the thermal energy input, assuming it is supplied as oil, according to the equation: $m(CO_2) = J_{th}$ (MJ)•75 g/MJ [Q27].
=> $m(CO_2)$ = 24•75 = 1.800 Mg CO_2/Mg $NaClO_3$

Ranger

Milled ore G = 0.244% U => 2.44 kg U/Mg ore = 2440/270 = 9.04 mol
Stoichiometrically about 3 mol $NaClO_3$ would be consumed, or some 0.32 kg/Mg ore, so a large excess of $NaClO_3$ seems necessary. A part of it may be consumed in reactions with other chemical species in the ore. Here assumed $m(NaClO_3)$ = 1.5 kg/Mg ore.

m(ore) = $2.293•10^6$ Mg
=> $m(NaClO_3) = 1.5•2.293•10^6 = 3.44•10^6$ kg = 3440 Mg
=> Ee = 21• 3440 = 72.2 TJ
Eth = 24• 3440 = 82.6 TJ
$m(CO_2)$ = 1.800•3440 = 6192 Mg

Summary of embodied E of chemicals

In Table 6 the energy intensities of the most important chemicals used at Ranger are listed. The specific CO_2 emissions were calculated in the previous sections. The electric input is assumed to be provided by the nuclear system itself via the grid (on global scale, in the long run).

Table 6

Energy intensities of several chemicals used at Ranger. The CO_2 intensity of each chemical is the sum of the CO_2 from burning fossil fuels (direct energy input) and the CO_2 from the chemical reactions involved in the production of a given chemical. The fossil fuels supplying the direct E input are assumed to have an average CO2 intensity of 75 g CO_2/MJ (equivalent to oil).

chemical used at Ranger	formula	Je GJ/Mg	Jth GJ/Mg	CO_2 Mg/Mg	remarks
ammonia	NH_3	35.9	50.7	4.935 *	from CH_4, H_2O, air
ammonium nitrate	NH_4NO_3	15.75	31.00	2.749 *	from NH_3 and HNO_3
lime	CaO	0.07	8.4	1.416 *	from $CaCO_3$
nitric acid	HNO_3	9.80	25.69	2.159 *	from NH_3, H_2O, O_2
sodium chlorate	$NaClO_3$	21	24	1.800	
sulfur	S	1.5	40	3.000	
sufuric acid	H_2SO_4	0.028	2.50	0.188	from S, Q95 p.55
extraction-grade kerosene	C_xH_y	?	?	?	
complexing agent	R_3N	?	?	?	
ion exchange resins		?	?	?	

* Includes CO2 from chemical reactions of the synthesis.

Table 7

Mass flows, embodied energy and (indirect) CO_2 emission of the chemicals consumed at Ranger in 2005 in ore processing, excluding and including explosives. Embodied energy and CO_2 emission of the explosives are included in the mining energy requirements.

chemical used at Ranger	chemical formula	mass Mg	Ee TJ	Eth TJ	Ee + Eth TJ	CO_2 Mg
ammonia	NH_3	706	25.35	35.79	61.14	3484
lime	CaO	26096	1.83	219.2	221.03	36952
nitric acid	HNO_3	1350	13.23	34.68	47.91	2915
sodium chlorate	$NaClO_3$	3440	72.2	82.6	154.80	6192
sulfur	S	29796	44.7	1191.8	1236.5	89388
sufuric acid	H_2SO_4	(91250)*	*	*	*	*
extraction-grade kerosene	C_xH_y	? **	?	?	?	?
complexing agent	R_3N	? **	?	?	?	?
sum excluding explosives		61388	157.31	1564.07	1721.38	138931
explosives		2293	35.20	71.08	106.28	6303
sum including explosives		63681	192.51	1635.15	1827.66	145234

* Here left blanc. The sulfuric acid is produced on site from imported sulfur. The energy required for the production comes from the combustion of the sulfur.

** No data are available.

7 Road transport to and from Ranger

Equipment, spare parts, fuel, chemicals and other consumables are transported from Darwin to Ranger, a distance of some 250 km. The product of the mine, uranium oxide U_3O_8, is transported back to Darwin.

Table 8
Quantities of materials transported to Ranger

chemical used at Ranger	mass Mg	see section	
diesel for electricity generation	20075	section 10	
diesel for mining equipment	9133	Table 4	
fuel oil for calciner	2661	section 5	
sum fuels	31869		
chemicals	63681	Table 7	including 2293 Mg explosives
other consumables	?		
equipment	?		
spare parts	?		
sum	95550+		

Energy consumed in transport

The mechanical energy input of transport to and from Ranger is calculated as follows.
Assume transport is done by road trains with a payload of 120 Mg and a shaft power of the engine P = 0.600 MW.
Loaded trip at 60 km/h => 60 s/km, at full power: P = 0.600 MW
=≥ J_{mech} = 60•0.600 = 36.0 MJ/km
empty return trip at 90 km/h => 40 s/km, at half power P = 0.300 MW
=≥ J_{mech} = 40•0.300 = 12.0 MJ/km
Per km hauling distance (with empty return trip)
J_{mech} = 48.0 MJ/km
per Mg payload:
J_{mech} = 48.0/120 = 0.400 MJ/Mg.km
Assume thermal conversion ratio diesel engines r = 0.40
=> J_{th} = 0.400/0.40 = 1.00 MJ/Mg.km (empty return trip included)

m = 95550 Mg, one way hauling distance *d* = 250 km, so:

E_{th} (dir) = 95550 Mg•250 km•1.00 MJ/Mg.km = 23.89 TJ

Indirect E

Mining dump trucks Jth = 1.888 MJ/Mg.km
Assume less wear at road trucks. A rough estimate is
J_{th} (indir) = 1.0 MJ/Mg.km
E_{th} (indir) = 95550 Mg•250 km•1.00 MJ/Mg.km = 23.89 TJ

sum direct + idirect energy input: J_{th} (dir+indir) = 2.0 MJ/Mg.km
specific CO2 emission:
$m(CO_2)$ = 2.0•75 = 150 g/Mg.km

E_{th} (dir+indir) = 23.86 + 23.86 = 47.72 TJ
$m(CO_2)$ = 47.72•75 = 3579 Mg
If we assume an average payload of a road train of m = 120 Mg, then the annual number of road transports would be: N = 95550/120 = 797
or 2.18 at average, 2-3, transports a day.

8 Summary of ore processing energy requirements

Table 9
Specific energy consumption of ore processing activities. The figures of the direct energy input have been deduced in this study based on data from ERA (Q320 and Q321) as much as possible, supplemented with data from other sources, see previous sections. The CO_2 emissions are due to the thermal energy inputs only.

activity	direct J_e	direct J_{th}	indirect J_e	indirect J_{th}	unit	CO_2 kg/Mg
crushing & grinding	81.55	–	4.70	37.87	MJ/Mg ore	2.84
leaching	42.4	–	1.5	10.6	MJ/Mg ore	0.80
extraction	3.6	–	1.8	5.4	MJ/Mg ore	0.41
sum	127.55	–	8.00	53.87	MJ/Mg ore	4.05
drying + calcining	?	20.6	?	3.5	GJ/Mg U	1808

Table 10
Direct and indirect energy consumption at Ranger by the ore processing activities in 2005. A number of energy inputs are not included, see text. Explosives are included in mining, see Table 6.

activity	direct *E* el TJ	direct *E* th TJ	indir *E* el TJ	indir *E* th TJ	indirect Ee + Eth TJ	sum *E*dir + *E*indir TJ	CO_2 Mg
crushing & grinding	186.6	–	10.78	86.84	97.62	284.22	6513
leaching	97.2	–	3.44	24.31	27.75	124.95	1823
extraction	8.25	–	4.13	12.38	16.51	24.76	929
drying + calcining	?	102	?	17.30	17.30	119.3	8948
subtotal A	292.05	102	18.35	140.83	159.18	553.23	18213
chemicals	–	–	157.31	1564.07	1721.38	1721.38	138931
E content sulfur	–	276.51	– *	– *	– *	276.51	– *
transport	–	23.82	–	23.82	23.82	47.64	3573
subtotal B		300.33	157.31	1587.89	1745.20	2045.53	142504
total A + B	292.05	402.33	175.66	1728.72	1904.38	2598.76	160716

* Included in chemicals

Specific energy consumption of ore processing at Ranger

mined ore $m(\text{ore}) = 2.293 \bullet 10^6$ Mg
uranium production $m(\text{U}) = 4947$ Mg
sum electric input, direct +indirect $E_e = 292.05 + 175.66 = 467.71$ TJ
sum thermal input, direct +indirect $E_{th} = 402.33 + 1728.72 = 2131.05$ TJ
sum $E_e + E_{th} = 467.71 + 2131.05 = 2598.76$ TJ

processing excl chem $J_e + J_{th} = 553.23 \bullet 10^6$ MJ/$2.293 \bullet 10^6$ Mg $= 241.27$ MJ/Mg ore
chemicals $J_e + J_{th} = 1721.38 \bullet 10^6$ MJ/$2.293 \bullet 10^6$ Mg $= 750.71$ MJ/Mg ore
energy content of sulfur $J_{th} = 276.51/2.293 = 120.59$ MJ/Mg ore
transport $J_{th} = 47.64/2.293 = 20.78$ MJ/Mg ore

Total ore processing $J_e + J_{th} = 241.27 + 750.71 + 120.59 + 20.78 =$
$= 1133.34$ MJ/Mg ore $= 1.133$ GJ/Mg ore
$R = E_{th}/E_e = 2131.05/467.71 = 4.56$
per Mg U: $J_e + J_{th} = 2598.76$ TJ/4947 Mg $= 0.5253$ TJ/Mg U

Not included in above figure are the energy inputs of:

- fresh water supply
- treatment of process water and pond water
- embodied energy of the extraction chemicals, kerosene and complexing agent
- direct and indirect energy of several partial processes of the ore processing chain, e.g. sulfuric acid plant, thickeners and centrifuge
- waste management
- construction of the mine

The energy input of each of these items may be of minor importance, but jointly the inputs may be a significant contribution to the overall specific energy consumption.

Energy production per Mg natural uranium

One reload charge $m(\text{U}) = 162.48$ Mg
gross electricity production per relad $E_e(\text{gross}) = 25.86$ PJ
gross electricity production per Mg U $J_e(\text{gross}) = 25.86/162.48 = 0.1592$ PJ/Mg U
$= 0.1592 \bullet 10^9/3.6 = 44.21 \bullet 10^6$ kWh/Mg U

CO_2 emission of the ore processing

processed ore $m(\text{ore}) = 2.293 \bullet 10^6$ Mg
uranium production $m(\text{U}) = 4947$ Mg
total CO_2 emission $m(CO_2) = 160716$ Mg

per Mg ore $m(CO_2) = 160716/2.293 \bullet 10^6 = 0.07009$ Mg/Mg ore
per Mg uranium $m(CO_2) = 160716/4957 = 32.49$ Mg CO_2/Mg U
per kWh $m(CO_2) = 32.49 \bullet 10^6/44.21 \bullet 10^6 = 0.73485$ g CO_2/kWh

9 Discussion

According to Rotty et al. 1975 [Q95] the average specific energy consumption of the ore processing at the surveyed mines is:

$J_{milling} = J_e + J_{th} = 1.062$ GJ/Mg ore $\quad R = 7.0$

According to above analysis the figure at Ranger is:

$J_{milling} = J_e + J_{th} = 1.133$ GJ/Mg ore $\quad R = 4.56$

Note that the Ranger figure may be a low estimate, because a number of energy consuming processes are not included, see previous section. The real figure may be significantly higher, probably somewhere between 1.2 and 1,3 GJ/Mg ore.

The Ranger mine is one of the cheapest producing uranium mines of the world, due to its favourable conditions and properties, such as:

- Ranger has soft ore. The processing of hard ores will take considerably more energy.
- Ranger has a relatively high ore grade ($G = 0.23\%$ U). Lower ore grades mean milling larger quantities of rock to a finer mesh and the exctraction will consume larger volumes of leaching and extraction liquids. Both factors increase the specific energy consumption.

For that reason one would expect the specific energy consumption per Mg ore at Ranger to be lower than the average figure found by Rotty et al.

The fact that the Rotty figures are based on the survey of a large number real uranium mines (in the USA), was the reason to choose these figures as a world average in the original study Storm&Smith 2008 [Q6].

This analysis demonstrates that the Rotty figures in fact are a low estimate. The world average figures should be significantly higher than the Ranger figure found in this analysis, which in itself is a low estimate.

In addition to the two factors regarding Ranger mentioned above, there are other factors giving rise to this statement, such as:

- Smaller mines have larger fixed energy input, for its construction, the processing plant and equipment.
- The world average includes alkaline ores. The leaching of alkaline ores takes much more energy than acidic ores (as at Ranger), due to the elevated temperatures (60-80 °C) and the consumption of chemicals with a high embodied energy.
- The drying and calcining of the wet cake to U_3O_8 may take more energy than estimated in this analysis.
- The transport distances of the supplies to the mine vary over a wide range and may be thousands of kilometers in some cases. More remote uranium mines have longer supply routes and consequently have a higher energy consumption.

10 Electricity at Ranger

Data from ERA [Q320]:
electricity generating capacity $P_{max} = 28$ MW maximum
Electricity also for township Jabiru.
5 x diesel generators + 1 steam turbine
The average load is $P_{av} = 10$ MW
Consumption of destillate fuel m(diesel) = 50-60 tons/day

diesel consumption
50 Mg/day = 18250 Mg/a
60 Mg/day = 21900 Mg/a
assume average consumption $m(\text{diesel}) = 20075$ Mg/a
=> $V(\text{diesel}) = 20075/0.839 = 23927\ \text{m}^3/\text{a}$
heat content (LHV) $J_{th} = 36.0\ \text{GJ/m}^3$
density $d = 0.839\ \text{Mg/m}^3$
thermal E from diesel $E_{th} = 23927 \bullet 36.0 = 861.38$ TJ

$m(CO_2) = 861.38 \bullet 75 = 64603$ Mg

electricity generated with $P_{av} = 10$ MW => each year $E_e = 315.36$ TJ/a

thermal conversion efficiency $r = 315.36/861.38 = 0.366 = 36.6\%$

The thermal conversion efficiency of the generators at Ranger is not disclosed by ERA. A value of $r = 40\%$ is the best achievable at current state of technology, so the figure of 36.6% seems plausible.
We may conclude that diesel import quantities, as stated by ERA, are almost exclusively used for electricity generation. A minor part may be used as fuel for the cars of the inhabitants of the township Jabiru.

Electricity consumption at Ranger

Estimated in this analysis:

crushing & grinding	186.6	TJ
leaching	97.2	TJ
extraction	8.25	TJ
sum	292.05	TJ

This corresponds with $P_{av} = 292.05 \bullet 10^6/(3600 \bullet 24 \bullet 365) = 9.26$ MW year average.

drying & calcining	?
water treatment and water pumping	?
township Jabiru	?
sum = 315.36 – 292.05 =	23.31 TJ (

This corresponds with $P_{av} = 10 - 9.26 = 0.74$ MW year average.

The unknown electric inputs above may be provided by the steam turbine, which in turn could be powered by the excess heat of the sulfur combustion (sulfuric acid plant) and/or by the waste heat from the calcining process (oil fuelled).

A rough estimate of the electricity consumption by the the inhabitants of the Jabiru township can be made as follows. Jabiru has some 1500 inhabitants. Assume an electricity consumption of 1500 kWh/yr per person.
E $= 1500 \bullet 1500 = 225 \bullet 10^4$ kWh/yr $= 225 \bullet 10^4 \bullet 3.6$ MJ/yr =
$= (225 \bullet 10^4 \bullet 3.6)/365 \bullet 24 \bullet 3600 = 0.257$ MW average.

If this figure is right, some 0.5 MW from the diesel generators would be consumed by the water treatment, calcining process and other processes in the mine,
The electric component of the indirect energy input of Ranger, embodied in materials, chemicals and equipment, are not converted into a fossil fuel equivalents. These energy inputs occurred in factories elsewhere in the world.

11 Mining plus milling at Ranger

In the Storm&Smith 2008 [Q6] study the electric and thermal inputs are kept separated, to make the results independent on the local fuel mix. A number of uranium mines draw their electric power, or a part of it, from the regional grid. At Ranger the electricity is generated on site by diesel-fuelled generators. For that reason the results of this analysis are presented in two ways:

Method 1	according to the general methodology, followed throughout the [Q6] study
Method 2	reflecting the actual situation at Ranger.

Summary

Table 11

Direct and indirect energy input of mining plus milling at Ranger in 2005, presented according to method 1. This table is assembled from Tables 5 and 10. The lower two rows refer to method 2: the actual situation at Ranger, with the direct electricity consumption generated by diesel generators.

item	direct *E* el TJ	direct *E* th TJ	indir *E* el TJ	indir *E* th TJ	sum dir+indir electric TJ	sum dir+indir thermal TJ	CO_2 Mg
mining	–	392	–	217	–	609	45640
explosives	–	–	35.2	71.1	35.2	71.1	6303
subtotal A, mining	–	392	35.2	288	35.2	680	51943
ore processing	292	102	18	141	310	243	18212
chem.+ S + transp	–	300	157	1588	157	1888	142502
subtotal B, milling	292	402	176	1729	468	2131	160716
total m + m (=A+B)	292	794	211	2016	503	2811	212659
Ranger (method 2)							
subtotal C, milling	–	1200	176	1229	176	2429	220563
total m+m (=A+C)	–	1592	211	2016	211	3609	272505

Method 2 – the actual situation at Ranger

The direct electric input from comes from diesel generators.

In the section 9 'Electricity at Ranger' the average thermal to electricity conversion ratio has been deduced: $r = 36.6\%$. Consequently the thermal equivalent of the electricity consumed in the ore processing is:

$E_e = 292.05$ TJ

=> $E_{th} = 292.05/0.366 = 797.95$ TJ

$m(CO_2) = 861.38 \cdot 75 = 59846$ Mg

Substitution of these numbers in Table 11 gives the lower two rows of Table 11.

Specific energy consumption

Specific energy consumption of the uranium recovery at Ranger (mining + milling):

method 1 (see Table 11)

$J_{m+m} = (E_e + E_{th})/m(\text{ore}) = 3313.57/2.293 \bullet 10^6 = 1.445$ GJ/Mg ore

$J_{m+m} = (E_e + E_{th})/m(\text{U}) = 3313.57/4957 = 0.670$ TJ/Mg U

= 670 GJ/Mg U $R = E_{th}/E_e = 5.6$

method 2 (see Table 11)

$J_{m+m} = (E_e + E_{th})/m(\text{ore}) = 3819.47/2.293 \bullet 10^6 = 1.666$ GJ/Mg ore

$J_{m+m} = (E_e + E_{th})/m(\text{U}) = 3819.47/4957 = 0.772$ TJ/Mg U

= 772 GJ/Mg U $R = E_{th}/E_e = 17.1$

Energy production per Mg natural uranium

J_e(gross) = 25.86/162.48 = 0.1592 PJ/Mg = 0.1592/3.6 = $44.21 \bullet 10^6$ kWh/Mg U

See section 8.

CO_2 emission of mining + ore processing

method 1

$m(CO_2) = 212659/2.293 \bullet 10^6 = 0.0927$ Mg/Mg ore

$m(CO_2) = 212659/4957 = 43.0$ Mg CO_2/Mg U

$m(CO_2) = 43.0 \bullet 10^6/44.21 \bullet 10^6 = 0.972$ g CO_2/kWh

method 2

$m(CO_2) = 272505/2.293 \bullet 10^6 = 0.1188$ Mg/Mg ore

$m(CO_2) = 272505/4957 = 55.1$ Mg CO_2/Mg U

$m(CO_2) = 55.1 \bullet 10^6/44.21 \bullet 10^6 = 1.246$ g CO_2/kWh

Table 12

Summary of the results (rounded): direct and indirect energy input at Ranger in 2005

activity	direct E $J_{th} + J_e$ GJ/Mg U	indirect *E* $J_{th} + J_e$ GJ/Mg U	sum $J_{th} + J_e$ GJ/Mg U
mining	79	65	145
milling	141	385	525
sum	220	450	670

Discussion

In the Q6 study the following specific energy input of mining and milling are adopted as as the average of the world uranium mines (see Part D4 of Q6):

soft ores: $J_{m+m} = 2.33$ GJ/Mg ore $R = 3.0$

hard ores: $J_{m+m} = 5.55$ GJ/Mg ore $R = 0.64$

In de section 'Summary of the mining energy requirements' we explained the reasons why the world average mining energy input is expected to be considerably higher than at Ranger. In the section 'Summary of the ore processing E requirements' is explained why the same holds true for the ore processing.
Important variables determining the energy input per Mg uranium recovered are the ore grade G and the recovery yield Y of the mining + milling.
Y = mass of recovered U/mass of U in ore (*in situ*).

Extraction of U from a given ore at low yield takes less energy than at higher yield from the same ore. The processing of a low-grade ore takes more energy than the processing of a higher-grade ore at the same yield. For that reason we introduced the following equation 2 to calculate the energy consumption of uranium recovery per Mg uranium as function of Y and G:

J_{m+m} (U) = $100 \bullet J_{m+m}(\text{ore})/Y \bullet G$ eq 1
Y = recovered fraction
G = grade in % U

For soft ores this equation becomes:
J_{m+m} (U) = $233/Y \bullet G$ (GJ/Mg U) $R = J_{th}/J_e = 7.5$ eq 2
for hard ores:
J_{m+m} (U) = $555/Y \bullet G$ (GJ/Mg U) $R = J_{th}/J_e = 0.64$ eq 3

The extraction yield (or recovery yield) itself is a function of the ore grade, at a given state of extraction technology. The [Q6] study applied an empirical relationship between Y and G, demonstrated by Figure 5. Above equations are based on this extraction yield relationship. By lowering the yield the energy consumption will be also lowered.

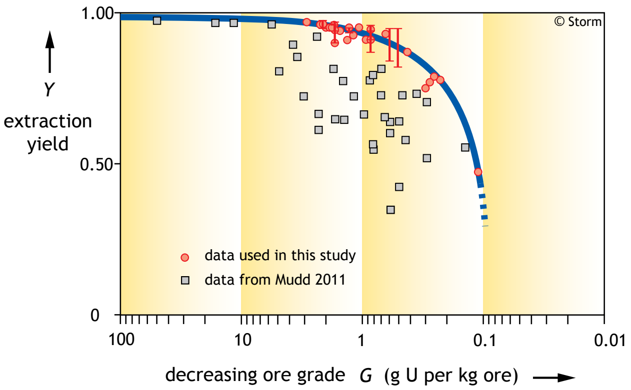


Figure 5
Empirical relationship between recovery yield and ore grade. For more details and references see report L21p21 *Uranium mining*. No distinction is made between soft ores and hard ores, although hard ores tend to let lower yields than soft ores at the same grade.

As Ranger has a soft ore equation 2 can be applied to calculate the energy consumption per Mg uranium.
J_{m+m} = $233/Y \bullet G$ GJ/Mg U
Ore grade in situ is $G = 0.23\%$ U (see Table 1)
The extraction yield is taken from the diagram of Figure 5: $Y = 0.98$. Substitution of the Ranger figure $Y = 0.892$ (Table 1) is not applicable here, as the equation is based on Figure 5.

J_{m+m} = $233/0.98 \bullet 0.23 = 1034$ GJ/Mg U $R = J_{th}/J_e = 7.5$
=> J_{th} = $(7.5/8.5)*1034 = 912$ GJ/Mg U

$m(CO_2)$ = 912*75 = 68.43 Mg CO_2/Mg U
$m(CO_2)$ = 68.43•10^6/44.21•10^6 = 1.5478 g CO_2/kWh

If calculated starting from the mill head grade G = 0.244% U
J_{m+m} = 233/0.98•0.244 = 974 GJ/Mg U $R = J_{th}/J_e = 7.5$
=> J_{th} = (7.5/8.5)*974 = 849 GJ/Mg U
$m(CO_2)$ = 912*75 = 64.46 Mg CO_2/Mg U
$m(CO_2)$ = 68.43•10^6/44.21•10^6 = 1.458 g CO_2/kWh

As expected the results from equation 2 are higher than the figures of Ranger, for reasons explained above. However, the differences are minor. Note that the Ranger analysis is not complete. Several energy contributions of the mining and milling activities are not included in the Ranger figures, due to lack of data.

Table 13
Comparison of the results of Ranger analysis with the results of the equation for the world average uranium mine

method	G ore grade % U	Y recovery yield	J_{m+m} e + th GJ/Mg U	R J_{th}/J_e	CO_2 g/kWh
Ranger method 1	0.244	0.892	670	5.6	0.97
Ranger method 2	0.244	0.892	772	17.1	1.25
equation 2 world average	0.244	0.98	974	7.5	1.46
equation 2 world average	0.23	0.98	1034	7.5	1.55

12 Conclusions

- The figures of the specific energy consumption of the uranium recovery at Ranger are slightly higher than the figures found by the method of Storm&Smith 2008 [Q6], which is assumed to hold for the average of the world's uranium mines.
  The Ranger figures as found in this analysis are low estimates in itself, due to an incomplete data set.

- At Ranger significantly lower values than the world average were to be expected, because the Ranger mine operates at the energy-lean end of the wide spectrum of the world uranium mines, owing to the favourable conditions of Ranger.

- This process analysis validates the equation applied in the [Q6] study to calculate the energy consumption of the uranium recovery.
  This means that the specific energy consumption of the uranium recovery at a world average uranium mine as calculated by that method, will not lead to an overrated value.

- Noteworthy aspect of above conclusion is that the energy consumption figures of mining and milling from the 1970s are still valid in 2005.

- The data provided by ERA on the energy consumption at Ranger concerns the diesel import for electricity generation only. Diesel fuel for the mining equipment (excavators and dump trucks) and fuel oil for the calcination of wet cake are not mentioned in the ERA publications.

- The indirect energy consumption at Ranger, embodied in chemicals, equipment, auxiliary materials and maintenance comprise some 2/3 of the total energy consumption per mass unit uranium recovered. These indirect energy inputs are not mentioned either in the ERA publications

- The mining energy input of open pit mining strongly depends on several variables:
  – overburden ratio (stripping ratio)
  – hauling distance
  – specific consumption of explosives
  – thermal conversion ratio of the diesel enigines
  – hardness of the rock (difficult to quantify)

- The ore processing (milling) energy input strongly depends on:
  – ore grade
  – hardness of the rock
  – mineralogy of the ore

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