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ANALYSIS

An estimation of the cost of sustainable production of metal concentrates from the earth's crust

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Abstract

An attempt has been made to estimate the value of present ores to future generations. The direct costs for the producer and the external environmental costs for the society of a sustainable production of ore-like metal concentrates are investigated. In a technical scenario, granite, granodiorite and basalt rock are mined, crushed and ground to give fine powders. The powders are acid leached and the extracted metals precipitated using hydrogen sulphide and sodium hydroxide to form the ore-like concentrates. All chemicals are produced in a sustainable way. Leaching efficiencies and use of energy-ware (ISO standard term for oil, coal, electricity, etc.) and chemicals are important factors in the cost estimation, as well as the number of metals mined at the same time. It is assumed that 10 different metal concentrates are produced at the same time. The technical scenario uses state-of-the-art technology. No new technical achievements are assumed. The estimated costs exceed current prices by orders of magnitude. In cost estimation for individual metals, the main uncertainty lies in the allocation of costs amongst the metals produced at the same time.

Keywords: Ore; Sustainable production; Metal; Cost estimation; Earth's crust

1. Introduction

When current technology is made more sustainable various choices have to be made, such as where to locate industries, what materials to use, product designs, etc. All these choices are between alternatives that can be described with respect to their impacts on various environmental or sustainability indicators. Methods such as life cycle assessment (LCA), environmental impact assessment (EIA) and risk assessment (RA) may be used and result in qualitative and quantitative descriptions of impacts on indicators. To interpret them, however, it is essential to have an idea of the magnitude of the economic consequences of a change in the indicators.

One type of indicator that has been particularly problematic to attribute a monetary value is the depletion of metal ore reserves. In the study

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presented here we try to increase the understanding of its value by estimating what it would cost to produce ore-like metal concentrates starting from a sustainable resource, such as common bedrock. The cost of metals would, of course, be much higher, and consequently, the use of metals would decrease, as there are substitutes in most applications. Considering the large amounts that are available in common bedrock (> 99.99% of most types of metal atoms in the upper crust are not in ore form—Skinner et al., 1986), it seems reasonable to regard mining from common bedrock as sustainable.

As far as we have been able to determine, no one has ever made this type of a study before. One reason for this is that there is no financial incentive for this type of process. The only extraction processes that have been studied involve low-grade ores or mining waste, but not bedrock.

2. Choice of technical scenario

The cost of sustainable production of metals is estimated from a technical scenario where concentrates of similar grade as presently available ores are produced in a sustainable way. The precautionary principle is applied in the sense that no new technology is invented. The scenario is based on current equipment and practices, but with the use of more sustainable material and energy sources.

The sustainable production of metal ores is assumed to be similar to a type of natural process, which once contributed to the creation of the reserves, i.e. weathering, leaching and precipitation of dissolved material. However, in economic systems, the processes have to be speeded up. Weathering is therefore replaced with mining, crushing and grinding. The leaching of rain or seawater is replaced with a more active leaching agent.

Examples of ore leaching processes are frequent in literature, but little is known about leaching rocks representing the ‘earth’s average crust’ for the production of metals.

When extracting metals from rocks for chemical analysis, hydrofluoric acid or salt melting is used to eliminate the silicon matrix. Such a process could be used for producing small amounts of metals. Skinner et al. (1986) assumes that a melting process would be needed and that it would cost several hundred dollars per ton of bedrock to extract valuable metals.

Leaching with microorganisms has also been mentioned and would probably be a more sustainable leaching process; but at this stage we know very little about such methods, and it is not considered possible to model such a method.

Leaching with strong acids is better known, but the extraction efficiency is probably less than would be obtained using hydrofluoric acid.

Tests with different strong acids made by Borg and Steen (2001) showed that hydrochloric acid (HCl) gave the highest average yield for the metals tested. As HCl seems easiest to produce in a sustainable way from electricity and sea salt, and as sodium hydroxide may be obtained at the same time, our main focus is HCl. The use of hydrofluoric acid has previously been investigated, and was not found to be competitive in comparison with leaching with strong acids (Steen, 1999).

3. The process

The process has been somewhat idealized and simplified. Most minor flows have been omitted and only those emissions and resource flows are included, of which the impacts are essential from an economic and environmental point of view. A process based on the use of HCl is described in Fig. 1. When leaching with acid some of the acid is neutralised by the metal oxides of the minerals. When neutralising to precipitate metals that do not precipitate as sulphides, some excess sodium hydroxide is left and exits the system.

The process contains the following steps, for which the emissions and resource use are summarised in Table 1 and operating and capital costs in Table 2.

3.1. Mining

Strip-mining is used because average bedrock is extracted. The waste, which is almost all of the

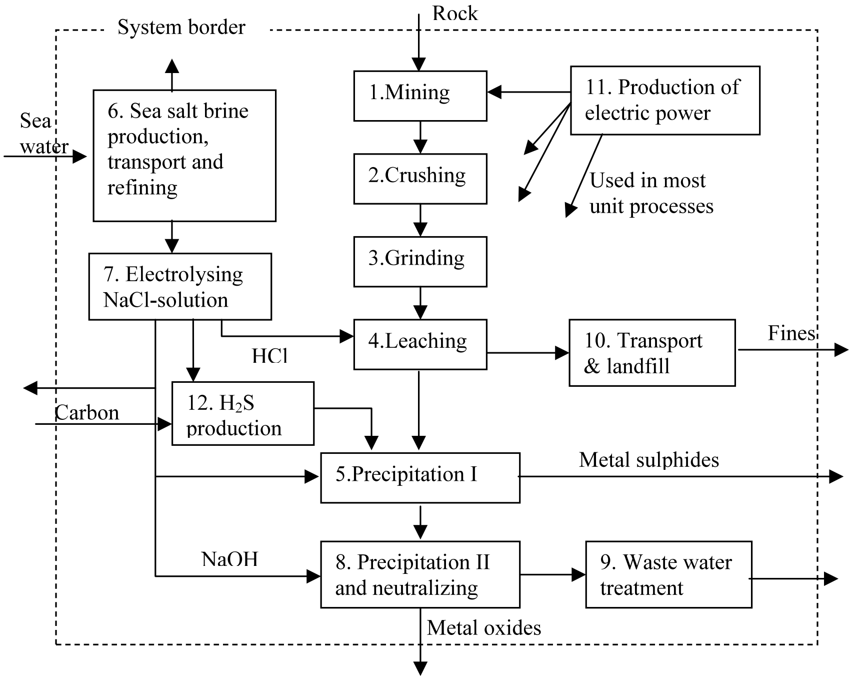

Fig. 1. A scenario for near-sustainable production of metal concentrates, similar to ores. *Technical system borders:* use of equipment for processing is included, but no production of capital goods. *Temporal system borders:* 2000 and onward. There is no time limit except for the valuations, which are today's values. *Spatial system borders:* global.

material, is returned to the mining area, and 20 years after the mining trees may be planted or the area may be used for other purposes. Mining consists of a number of subprocesses: drilling, blasting, loading, and hauling. The energy used is assumed to come mainly from biofuels and to a

Table 1
Emissions and resource use when processing 1 ton of average bedrock

Emissions and resource use							
Process step	PM10 (kg/ton rock)	NO_x (kg/ton rock)	CO (kg/ton rock)	H_2S (kg/ton rock)	Land ($m_2$year/ rock)	Fe (kg/ton rock)	Mn (kg/ton rock)
1	0.015	0.00802	0.016	0.008	0.4		
2						0.0002	2.3E−05
3						0.0225	0.0025
4							
5							
6	5E−05	0.0005					
7							
8							
9							
10							
11					56.0		
12							
Sum	0.0151	0.00852	0.016	0.008	56.4	0.0227	0.00252

Table 2
Summary of operating and capital costs (internal costs)

No.	Step	Use of materials (per ton bedrock)	Use of electricity (per ton bedrock)	Use of capital goods/ machinery	Cost (EUR/ ton bedrock)	Comments
1	Mining (includes strip-mining, drilling, blasting, loading, hauling and ground restoration)	0.5 kg ammonia dynamite and 0.0125 kg biofuel	0.07 MJ for dynamite production	Drilling equipment and wheel-loaders		
2	Crushing	0.0067 kg steel from wearing	2.58 MJ	Crusher	<5	
3	Grinding	0.5 kg Mn steel	140–400 MJ	Grinder	6	Total cost for Sections 3.1, 3.2 and 3.3
4	Leaching		0.1 MJ	Concrete pool, solar heaters, pumps, tubing	0.1	Cost for HCl is included in 7
5	Precipitation of sulphides		0.1 MJ	Pumps, tanks and screw conveyers	0.1	
6	Sea salt brine production	0.48–0.9 kg biofuel, 80–150 kg sea salt		Transport by ship, conveyers, wheel-loaders	1.4–2.6	
7	Electrolysing NaCl	47–89 kg HCl	1100–2070 MJ	Chlor-alkali electrolysing plant	8.2–15.6	Includes 6
8	Precipitation with NaOH		0.1 MJ	Pumps, tanks, screw conveyers	2.5	
9	Waste water treatment			Pool, pumps	0.0	
10	Transport and landfill			Wheel-loaders, trucks	0.1	
11	Production of electricity	Land use is 0.039 m^2 year/MJ		Dams, turbines		Cost, 0.05 EUR/kWh is accounted for in steps 1–10
12	Manufacture of H_2S	Carbon, sea salt			0.02	
				Total operating and capital cost	15–27	Weighted average: 15.9 EUR

lesser extent from electricity. Drilling is carried out with equipment using compressed air. Ammonium dynamite is used for blasting (consisting of ammonium nitrate, nitroglycerine, sodium nitrate and wood pulp) (Access Science Browse Encyclopedia, 2001). The use of explosives is about 0.5 kg/ton of rock (Malmberget, 1994). Approximately, 1 kg of natural gas or methane and 0.15 MJ of electricity are needed to produce 1 kg of ammonium dynamite. These figures are estimates from an LCA of N fertilizers (with a high content of ammonium nitrate) (Tillman et al., 1997) due to lack of more specific figures. The explosions generate about 32 g CO/kg of explosive, 16 g NO_x/kg of explosive, 16 g H_2S and 30 g PM10/kg of explosive. These figures are estimates from a manual from Environment Australia (1999). Loading is done using large diesel powered wheel-loaders. Approximately, 0.0025 kg of biofuel/ton of rock is used (assuming a 3 m elevation and 30% efficiency). Finally, the material is transported to the plant for crushing and grinding. This transport requires about 0.01 kg of biofuel/ton (assuming a 200 m transport distance, fuel consumption of 1 kg/km, and a vehicle capacity of 20 ton).

3.2. Crushing

LCA of a 454 ton/h of rock crusher was performed by Landfield and Karra (2000). LCA included the production of the crusher, use, and its end of life processes. Electricity was an average US mix. The authors concluded that 99% of most of the emissions and resources were caused by the use-phase and within the use-phase over 95% was caused by electricity consumption. The consumption of electricity was 2.58 MJ/ton, and of spare steel parts 0.0067 kg/ton. In our scenario the electricity is assumed to be produced by hydropower. The steel is a Mn alloy with 10% Mn. 70% is recycled with a 90% recovery and 30% follows the material flow to the leaching why the ore consumption will be 0.00021 kg Fe and 0.000023 kg Mn using Landfield and Karra's figures for the consumption of spare parts. Based on the current price of macadam from quarries, the cost for mining and crushing to 16/32 mm may be estimated to be 5 EUR/ton. However, in large-scale operations the cost may be reduced further.

3.3. Grinding

The energy consumption is dependent on how finely the material is ground. All the standard reference samples we tested with respect to leachability contained particles that were mainly less than 80 μm. According to Flanagan (1967), the following amounts of the reference samples used passed through a 200-mesh ($\sim$0.080 mm) sieve: granite G-2, 98.5%; granodiorite GSP-1, 96.1%; and basalt BCR-1, 99.3%. The amount coarser than 170 mesh ($\sim$0.09 mm) was: granite G-2, 0.6%; granodiorite GSP-1, 1.4%; and basalt 0.1%. The electricity used for grinding is estimated from formulas given by Perry et al. (1997) to be 400 MJ/ton. According to Perry et al. (1997), the wear of crushing and grinding equipment is almost as costly as its energy demand. This indicates a wear of the order of 0.5 kg/ton, which is about 100 times as much as for the crushing operation. Considering that the average concentration of Mn is approximately 0.5 kg/ton, a low Mn steel is to be preferred. If we use a 10% Mn steel and have a 50% recycling of the lining and ball material, we would still have at least a 95% efficiency as an average for recovery of Mn. Mn and Fe lost to wear is probably not lost completely because it will be dissolved in the acid and precipitated as oxides. The cost for grinding from 16/32 mm down to below mesh 200 (81 μm) is estimated to be a little more than twice the sustainable energy cost, i.e. 15 EUR/ton.

In large-scale mining and preparation of copper ores, a somewhat different picture of the costs emerges. Copper ore with an average Cu content of 1% or more or that contains other valuable metals in addition to copper (e.g. precious metals, nickel, cobalt) is mined underground, while ores with 0.5% Cu are mined by open-cast methods (Lossin, 2001).

For primary copper production, the overall energy consumption per ton of copper is about 45 GJ, about half of which is said to be consumed in mining and beneficiation. Present Cu prices are about 2 EUR/kg (Lossin, 2001). This indicates a cost for mining, crushing and grinding of around 1000 EUR/ton Cu, which at 0.5% Cu corresponds to 5 EUR/ton of ore and an energy demand of 125 MJ/ton of ore. As granite is harder than copper ore, the energy demand should be somewhat higher, but the work index for grinding of granite (14.39) is only slightly higher than that of copper ore (13.13). The work index is proportional to the energy demand (Perry, 1998). Considering a likely increase in sustainable energy cost in relation to current costs, a likely cost per ton rock for mining, crushing and grinding is about 6 EUR/ton and a likely energy demand is 140 MJ/ton.

3.4. Leaching

The fines produced in Section 3.3 are transported by belt conveyers to a relatively inexpensive leaching facility, where a process similar to current heap and dump leaching takes place. The pregnant leach solution contains many metal salts. It is assumed that 10 different metals are recovered from the solution. Allocation of emissions and resources is made equally to each metal. This means that the emissions, etc., from the system for each metal would be the same as if rock was used that contained only one metal but with 10 times its average concentration in the earth's crust. The leaching efficiency for some reference samples was studied experimentally by Borg and Steen (2001). The results, shown in Table 3, clearly indicate that a substantial part of the metals in bedrock may be extracted in a simple leaching process and that there is no need to completely dissolve the mineral crystals. The consumption of acid was relatively low, and agrees approximately with what can be expected from the amount and type of extracted material. For granite it was determined to be 47 kg/ton (mass of HCl), for granodiorite 57 kg/ton and for basalt 89 kg/ton. If weighted according to the relative abundance of granite (including gneiss, 55%), granodiorite (20%) and basalt (about 10% basalt-like rocks) the average consumption of acid would be 54 kg/ton of rock. The leaching is assumed to be carried out in a heap or dump leaching, which requires 0.1 MJ/ton for pumping. Heating to 50 °C is done by solar energy. The leaching cost is estimated to be very low, in the range of 0.1 EUR/ton of bedrock.

3.5. Precipitation I

The solubility of many metal sulphides is very low. If the leachate from 1 kg rock is 1 l, a recovery of metals from the leachate at various molarities of sulphide ions would be obtained as shown in Table

Table 3
Leaching efficiency obtained when extracting in 30% HCl for 24 h

	Granite		Granodiorite		Basalt	
	Content (g/ton)	Mean yield (%)	Content (g/ton)	Mean yield (%)	Content (g/ton)	Mean yield (%)
Al 27	81 496.8	9.9	80 703.0	14.5	72 182.3	5.6
Cr 52	7.0	100.8	12.5	62.0	17.6	10.2
Fe 54	18 824.5	86.1	30 335.0	80.1	94 139.0	44.6
Mn 55	260.0	90.9	331.0	81.8	1406.0	27.8
Co 59	5.5	73.3	6.4	76.9	38.0	35.5
Ni 60	5.1	44.9	12.5	51.0	15.8	24.2
Cu 65	11.7	79.9	33.3	80.2	18.4	(126.6)
Zn 66	85.0	90.5	98.0	85.1	120.0	62.9
Sr 88	479.0	5.5	233.0	8.5	330.0	5.5
Cd 114	0.0	22.5	0.1	50.0	0.1	33.7
Ba 138	1870.0	14.0	1300.0	11.3	675.0	2.7
Pb 208	31.2	22.1	51.3	65.3	17.6	38.4
Sn 117	1.5	76.2	6.3	72.9	2.6	58.3
W 186	0.2	27.6	0.3	22.3	0.4	63.4

First hour heated to 50 °C.

Table 4
Recovery of metals with sulphide precipitation at various sulphide concentrations

Metal ion	Solubility product of sulphide	Yield at 1E−6 M sulphide concentration	Yield at 1E−15 M sulphide concentration	Yield at 1E−22 M sulphide concentration
Ag^{+}	6.00E−51	100	100	100
Bi^{+++}	1.82E−99	100	100	100
Cd^{+}				
Co^{++}	7.00E−23	100	99.9835	0
Cr^{+++}				
Cu^{+}	2.26E−48	100	100	100
Cu^{++}	6.00E−37	100	100	100
Fe^{++}	6.00E−19	100	99.93299	0
Hg^{+}	1.00E−45	100	100	99.99925
Hg^{++}	2.00E−53	100	100	100
Ni^{++}	1.07E−21	100	99.92524	0
Mn^{++}	3.00E−14	99.99983	0	0
Pb^{++}	3.00E−28	100	100	95.55979
Pt^{++}	9.91E−74	100	100	100
Sn^{++}	1.00E−26	100	99.99999	48.3913
Tl^{+}	1.20E−24	97.36591	0	0
Zn^{++}	2.00E−25	100	99.99998	0
W				

Sulphide concentrations of 10^{-15} and 10^{-22} mol/l are obtained in a solution saturated with H_2S at a pH of 4 and 0.5, respectively. Solubility constants are according to Lide (1994).

4. The yield figures in Table 4 indicate that almost all of the extracted metals may be recovered from the dissolved state. In order to obtain a reasonably high concentration in the synthetic ore, it is important that the precipitate does not contain the most abundant metals, including Al, Fe, Na, Ca and Mg. Table 4 shows that FeS is not precipitated with solutions of sulphide concentrations of 1×10^{-22} M. The sulphides of the other abundant metals are more soluble than FeS and will also remain in solution at low pH. Energy consumption in the precipitation step is also low, about 0.1 MJ/ton of rock for pumping, clarifiers and screw conveyers. Total operating and running cost is estimated to be in the order of 0.1 EUR/ton of rock.

3.6. Sea salt brine production

Sea salt brine is produced in a traditional way from sun drying of seawater in Salinas. Transport by ship (1000 km) requires 6 kg oil/ton of salt. For granite about 80 kg sea salt/ton of rock is needed (47 kg HCl was consumed). For granodiorite and basalt the corresponding figures are 100 and 150 kg, respectively. Weighted according to the relative abundance of granite including gneiss (55%), granodiorite (20%) and basalt (about 10% basalt-like rocks) the average is 93 kg/ton of rock. This means that the biofuel oil consumed is 0.48, 0.6 and 0.9 kg/ton of rock. The weighted average for all bedrock types is 0.56 kg oil/ton of rock. The cost of imported salt today is around 17 EUR/ton (Bertram, 2000).

3.7. Electrolysing NaCl

NaCl reacts according to the formula $NaCl(aq) + H_2O \rightarrow NaOH + H_2 + Cl_2$. Theoretically, about 0.4 MJ/mol NaCl electric power is needed for the electrolysis. According to Boustedt (1994), the allocated use of electricity from cradle to gate is 9.43 MJ/kg of Cl_2 (including H_2) and 13.93 MJ/kg for NaOH, which equals 1.35 MJ/mol of NaCl. In Boustedt's investigation the energy demand varied substantially at different plants, and some results were only half the average. This leads to the conclusion that sustainable production of NaOH and HCl can be achieved at 0.8 MJ/mol of NaCl or 13.8 MJ/kg of NaCl. For granite leaching (mainly consuming HCl) this means 1100 MJ/ton of rock, for granodiorite 1380 and for basalt 2070 MJ/ton of rock. The production cost of 1 electrochemical unit (1 ton chlorine + 1.13 tone sodium hydroxide) depends up to 60% on the price of electricity. At 3.5 c/kWh, they are about 250 $/electrochemical unit (Schmittinger et al., 2001). At 5 c/kWh they would be about 175 EUR for 1 ton HCl and 0.13 ton sodium hydroxide. It is assumed that 1 ton NaOH is sold and that all H_2 is captured from the electrolysis and used for the production of HCl.

3.8. Precipitation II

Depending on how much of the acid is consumed in the leaching process, the NaOH produced in step 7 may be used to make the leachate basic. This will precipitate some hydroxides and oxides, such as Mn, Tl, Cr and possibly Fe. The exact precipitation procedure in steps 5 and 8 is not described here because it is of very little importance for the total environmental impact and cost. The energy used is assumed to be 0.1 MJ/ton of rock, mainly for pumping, clarifiers and conveying. The capital and operating costs are thus similar as for Section 3.5, i.e. 0.1 EUR/ton of rock.

3.9. Waste water treatment

Mainly an adjustment of pH with acid from process Section 3.7.

3.10. Transport and landfill of the remaining solids

The fines are brought back to the strip-mine area from which they were taken. The land use is included in the calculation in step 1. The transport is approximately the same as when the rock was brought to the leaching plant from the mine, i.e. the operation requires about 0.01 kg biofuel/ton and costs about 0.1 EUR/ton.

3.11. Production of electricity from hydropower

Hydropower is chosen as a sustainable power source, although solar power is a more likely alternative in the long run. The differences in cost and environmental impact are not expected to be significant. It has been demonstrated that mirror-based solar driven generators can produce electricity at a cost of 0.05 EUR/kWh. Hydropower is used in the scenario, however, as LCI's of this process are better known. To make the conclusions of the study valid for use of solar energy, too, the same cost per kilowatt hour is assumed for the hydropower and solar power (0.05 EUR/kWh). The main impact from hydropower comes from land use. 0.000358×60 m^2 year/kWh according to Vattenfall et al. (1996) for Luleälv, 1300 m^2 year/TJ according to ETH (1996), and 0.039 m^2 year/MJ are chosen as a scenario value.

3.12. Manufacture of H_2S

H_2S will be used to precipitate the desired metals. For granite, the use of H_2S will be about 0.05 kg/ton of rock if pH is raised to 4. For granodiorite and basalt the corresponding figures will be 0.07 and 0.06 kg. H_2S is produced sustainably from the remaining sulphate from seawater and from recycling of sulphur from the subsequent roasting of sulphide 'ores'. Sulphate is reduced with charcoal to form sulphides. The cost of H_2S is assumed to be 400 EUR/ton, which means a cost of 0.02 EUR/ton of rock.

The calculation of the total cost, T, when processing 1 ton of average bedrock is summarised in Table 5. T is defined as

$$T = C + O + E,$$

where C is the capital costs for the producing company, O the operating costs for the producing company (raw materials, energy, salaries, etc.), and E the external costs to the society due to environmental impacts on five safeguard subjects.

$$E = h + p + a + b + r,$$

where h is the loss in human health value through increased mortality and morbidity and through decreased welfare, p the loss in production capa-

Table 5
Calculation of total cost when processing 1 ton of average bedrock

Emissions, resources and process steps (EUR/ton rock)	External costs (EUR/ton rock)	Internal costs (EUR/ton rock)
PM10	0.5418	
NO_x	0.01815	
CO	0.0053	
H_2S	0.0397	
Land	2.56	
Fe	0.0218	
Mn	0.01423	
3		6
4		0.1
5		0.1
7		9.5
8		0.1
10		0.1
12		0.02
Sum	3.20	15.92
	Total cost	19.12

city value of ecosystems through decreased harvests, e.g. from toxic agents or land use practices, *a* the loss in abiotic resource value through mining and extraction of oil and gas, *b* the loss in biodiversity value through the extinction of species, and *r* the loss in recreational values, e.g. from landscape degradation and from impoverished flora and fauna.

External costs estimated by Steen (1999) are used. In this work, the values of human health, biodiversity and recreational values are mainly literature data on willingness to pay to avoid changes, and the value of ecosystem production capacity is estimated from market prices for producers and resource extraction from restoration costs. Restoration costs are determined in the same way as is described in this article, i.e. total costs, T, to produce a similar concentrate as the abiotic resource in question. The relations between emissions and impacts on the safeguard subjects were modelled through proven or postulated pathways described in the literature.

In Sections 3.1, 3.2, 3.3, 3.4, 3.5, 3.6, 3.7, 3.8, 3.9, 3.10, 3.11 and 3.12, the total cost, T, is determined for processing 1 ton of bedrock. The total cost per kilogram of metal is found by calculating how many tons of rock are needed to obtain 1 kg of respective metal. In Table 6, this has been done for the average concentration of the metals in the earth's crust, which determines how much bedrock is needed per mass of metal, using the leaching efficiencies from Table 3 and the acid consumption that was determined for granite, granodiorite and basalt. The weighted average cost for each element was also calculated using the relative abundance of granite (55%), granodiorite (20%) and basalt (10%) (Wedepohl, 1995).

In Table 6, the lowest costs are marked with bold figures. However, owing to the limited number of samples, it seems reasonable at this stage to use the weighted averages as a best estimate of the production costs.

4. Discussion

4.1. Sensitivity analysis

Some of the figures that play a significant role in the calculation of the cost for producing ore-like concentrates are uncertain. The cost for grinding is one such figure. It is not known which fineness is optimal. Energy consumption varies by a factor 2 between grinding that reduces the size to 60% passing through a 200-mesh sieve and grinding that reduces the size to 100% passing through a 200-mesh sieve. This corresponds to an uncertainty in the cost estimate of 1 EUR/ton of rock. The hydrometallurgical leaching process that exists today is fairly well developed for low-grade copper ore, but not for granite, granodiorite and basalt. The cost for sustainable hydrometallurgical processing was estimated to be 2.5 EUR/ton of rock for granite. An uncertainty of 1 EUR/ton seems likely. The major cost element was the production of HCl and NaOH (8.225 EUR/ton). This element and the others are to a large extent dependent on the consumption of acid and to some extent on the price of sustainable electricity. The consumption of acid is estimated to give an uncertainty of the order of 50% or 4 EUR. Commercially sustainable electricity production has been demonstrated at the level of 0.05 EUR/

Table 6
Allocated costs for the production of various metal concentrates

Metal	Abundance in earth's crust (g/ton)	Cost of 'ore' production from granite (EUR/kg metal)	Cost of 'ore' production from granodiorite (EUR/kg metal)	Cost of 'ore' production from basalt (EUR/kg metal)	Weighted average cost of 'ore' production (EUR/kg metal)
Cd	0.102	76 400	**38 800**	77 400	67 700
Co	11.6	**206**	222	646	262
Cr	35	**50.1**	91.1	745	141
Cu	14.3	**149**	172	186	159
Mn	527	**3.66**	4.59	18.2	5.60
Ni	18.6	210	**208**	586	254
Pb	17	467	**178**	408	392
Sn	2.5	**920**	1080	1830	1060
W	1.4	4540	6330	**3000**	4780
Zn	52	**37.2**	44.7	81.4	44.2

kWh using solar radiation and direct heating of oil, which, in turn, is used to generate steam for a turbine. The uncertainty in these estimations is approximately 2 EUR/ton. Together, this gives an uncertainty of 2.5 EUR/ton of rock.

There is an uncertainty in the figure for average metal concentrations in the earth's upper crust. According to previous figures, the average concentration of Cu was 60 ppm (Lide, 1994), while more modern estimates claim that 14.3 ppm is a better figure (Wedepohl, 1995). The leaching efficiency is determined on the basis of a relatively small number of certified reference samples, and it may be questioned how representative they are in terms of leaching efficiency and distribution of trace metals. When calculating the cost for a certain metal, there is great uncertainty involved in allocating the costs to different metals. It was assumed that concentrates of 10 different metals were produced at the same time and that the cost would be allocated equally to each, but there could either be more metals or fewer. The allocation principle is a subjective choice that may be disputed.

As the uncertainties mentioned above are relatively large, exchange rates between currencies and inflation effects has been accounted for in an approximate way. Cost estimates are of September 2001 and the exchange rate used is 0.9 $/EUR.

Despite all uncertainties, the magnitude of leaching efficiencies and production costs seems reasonable. The main reasons for this statement are twofold:

1) The technology used is fairly simple and well known.
2) The leaching efficiencies and low acid consumption obtained may be explained in terms of how crystals are formed in bedrocks. When cooling from a melted stage or re-crystallising under pressure, the trace elements do not fit into the well-ordered crystal grid and are forced out to the borders. When the material is ground to fine particles, cracks are predominately made along the borders between the crystals, and the trace elements are made available. This hypothesis is discussed in greater detail in Borg and Steen (2001).

4.2. Implications

It may seem premature to investigate a process that may not be used for several hundred years. Low-grade ores, deposits in ocean sediments and landfill sites for waste may be used before turning to bedrock. There are, however, two immediate consequences of the findings of this study.

Mental models including a picture in which metal resources will sooner or later be depleted may be repressed. There is no definite point where we will run out of metal resources. This study shows that it does not have to happen at all. The intrinsic properties of the metal atoms are a resource that is available 'forever'. Even if a certain metal may be replaced by another material in a certain use, there is a value in the degree of freedom it offers to have access to it. The cost estimation made is only dependent to a certain extent on the volumes of metal processed. Therefore, the cost estimation is approximately valid even if there is a major or total replacement of the certain metal.

The use value of the metal will shift from time to time. In this study there is no attempt to estimate this value or the future price of a metal concentrate.

There are a number of situations where the ore values determined in this study are needed, e.g. when performing green GNP accounting, LCA studies or making trade-offs in various projects or political decisions.

4.3. New technologies in the future

It may be argued that future technological developments will make the production of metals from bedrock cheaper. This may be so; in particular, biological methods may be developed to collect trace elements. There are, however, two reasons for using the estimates made here. One is the precautionary principle, which is often used in environmental policy. It is precautionary not to account for technology until it is developed. The second reason is that it is a basic problem to actually gain access to the metal atoms. This means that a certain amount of bedrock has to be crushed and ground in any present or future technology. Mining, crushing and grinding are some of the basic determinants in the cost estimate.

5. Conclusions

It is possible to produce metals from bedrock in a sustainable way and to estimate the costs. The cost of producing metal concentrates is very high compared to metal or ore prices today (Table 7). The current prices do not include external environmental costs and are not directly comparable to the cost for metal concentrates estimated in this study. It indicates, however, an upper limit for the economic value of the ore in ground, which has no external environmental impact costs associated with it. The present ore deposits and the metals in use today are therefore of very great value to future generations. This indicates a need to make recycling more efficient than today, to only use scarce metals when absolutely necessary, and to look for alternative sustainable production processes.

Table 7
Best estimates of costs for production of ore-like metal concentrates compared to some market prices of today for pure metals

Metal	Weighted average cost of 'ore' production (EUR/kg metal)	Today's price levels (EUR/kg metal)
Cd	67 700	0.5–16[a]
Co	262	8[b]
Cr	141	8[c]
Cu	159	1.7[d]
Mn	5.60	0.5[e]
Ni	254	6.8[d]
Pb	392	0.6[d]
Sn	1060	4.5[d]
W	4780	0.03–0.2[f]
Zn	44.2	0.9[d]

[a] Morrow (2001).
[b] Hodge and Dominey (2001).
[c] Chromite ore, Papp (2002).
[d] London Metal Exchange, February 2002.
[e] Matricardi and Downing (2000).
[f] Wolframite ore, Penrice (2001).

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