

TECHNOLOGY CHARACTERISATION FOR CERAMIC AND INORGANIC MATERIALS

Input data for Western European MARKAL

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Framework of the study

This report is a result of the MATTER project 'MATERials Technologies for CO₂ Emission Reduction', ECN project number 7.7018.

Abstract

This report is a product of the MATTER project (MATERials Technologies for CO₂ Emission Reduction). The report discusses the MARKAL model input parameters for the production and waste handling of ceramic and inorganic materials. These data encompass the energy and materials balance of processes, cost data and data regarding greenhouse gas emissions. Both the processes that are currently applied and improved processes based on new technologies are discussed. Apart from the process data, the report contains information regarding the current production volumes and materials application in Western Europe. These combined data will be used for the integrated energy and materials system modelling.

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SUMMARY

This report is a result of the MATTER project (MATERials Technologies for CO₂ Emission Reduction). The project focuses on CO₂ emission reduction options that are related to the Western European materials system (the production and consumption of materials, material intensive products and waste materials). The analysis will be based on the results of MARKAL (MARKet ALlocation) model calculations. This report describes the model input data for 20 ceramic and inorganic materials. 37 materials production technologies are discussed. The energy and materials inputs and outputs of these technologies and their costs are analysed. Improved materials production technologies, options for increased materials efficiency and increased materials recovery and recycling are discussed. Apart from the process data, the report discusses the current production volumes and materials applications in Western Europe. These combined data represent part of the MARKAL model input. Data for other materials and data for products are discussed in separate volumes.

1. INTRODUCTION AND OVERVIEW OF THE ANALYSIS

This report is a result of the MATTER project (MATERials Technologies for CO₂ Emission Reduction). The project focuses on CO₂ emission reduction options that are related to the Western European materials system (materials, material intensive products and waste materials). The analysis will be based on the results of MARKAL (MARKet ALlocation) model calculations. This report describes the model input data for ceramic and inorganic materials. The following materials will be discussed:

- Ammonia
- Bricks
- Cement
- Ceramic products
- Chlorine
- Concrete
- Glass
- Gypsum
- Kaolin
- Nitric acid
- Phosphoric acid
- Potash
- Quicklime
- Sand-limestone
- Sodium carbonate (soda)
- Sodium chloride
- Sodium hydroxide
- Ornamental Stone
- Sulphur
- Titanium dioxide

The selection of materials is based on preceding analyses [1, 2]. Their relative importance is analysed in Table 1.1. The energy consumption data and emission data in Table 1.1 are discussed in Chapter 2. For coal, oil, and natural gas, the CO₂ emission factors according to the IPCC guidelines have been used (94 kg/GJ, 73 kg/GJ and 56 kg/GJ, respectively) [3]. For electricity and heat, estimates for the average European situation have been applied (100 kg/GJ and 70 kg/GJ, respectively). Process emissions (not related to fossil fuel use) have been considered for all processes where limestone (CaCO₃) is dissociated. For nitric acid, considerable nitrous oxide emissions have been considered and translated into CO₂ equivalents.

Table 1.1 *Energy and CO₂ balance of ceramic and inorganic materials (2000) (assuming 0.1 t CO₂/GJ electricity and 0.07 t CO₂/GJ heat), excluding transportation*

	Production [Mt pa]	Coal [GJ/t]	Oil [GJ/t]	Gas [GJ/t]	Heat [GJ/t]	Ele [GJ/t]	Process ⁴ [t/t]	CO ₂ [Mt pa]	Fraction [%]
Ammonia	10	-	-	31	-	0.5	-	17.9	7.2
Bricks	66	-	0.5	2	-	0.1	-	8.1	3.2
Cement	150 ¹	3.5	-	-	-	0.1	0.5	125.9	50.2
Ceramic products	8	-	-	8	-	0.5	-	4.0	1.6
Chlorine/NaOH	8	-	-	-	1.2	9.9	-	8.6	3.4
Concrete	800	-	0.1	-	-	0.1	-	13.8	5.5
Glass	25	-	-	5	-	-	0.2	12.0	4.8
Gypsum	23	-	0.1	-	1.0	-	-	1.8	0.7
Kaolin	4	-	0.5	-	1.0	-	-	0.2	0.1
Nitric acid	18	-	-	-	-2.1	0.03	1.5	24.4	9.7
Phosphor	0.2	33	-	10	-	45	-	1.6	0.6
Phosphoric acid	2.5	-	-	-	4.0	0.5	-	0.8	0.3
Potash	4.5	-	0.4	7.5	-	0.18	-	2.1	0.8
Quicklime	22	-	-	5	-	-	0.75	12.7 ⁵	5.1
Sand-limestone	40	-	0.19	-	0.63	-	-	2.3	1.0
Sodium carbonate 6	-	-	-	9.2	-	-	3.9	1.6	
Sodium chloride	15.4	-	-	-	4.0	0.1	-	4.5	1.8
Stone	18	-	-	-	-	1	-	1.8	0.7
Sulphuric acid	7 ²	-	-	-	-3.3	-	-	-1.6	-0.6
Titanium dioxide	1.3	-	25	-	2.0	1 ³	-	2.7	1.1
Total								247.5	

¹ Excluding blast furnace/fly ash cement fraction

² From pure sulphur, resulting in net energy production

³ For oxygen production

⁴ Includes inorganic CO₂ emissions and nitrous oxide emissions for nitric acid production (converted into CO₂ equivalents on the basis of a Global Warming Potential of 310)

⁵ Including CO₂ fixation during use

The total emission of 250 Mt for all these materials can be compared to the total Western European emission of approximately 3500 Mt. The materials and processes that are analysed in this report represent 7.1 % of the total Western European CO₂ emission. Table 1.1 shows that cement represents half of the total CO₂ emissions in this materials group, followed by nitric acid (actually N₂O emissions), ammonia, glass, quicklime, and concrete with a considerably lower significance. All materials that represent a CO₂ emission below 0.1% of the total Western European CO₂ emission will not be discussed in detail. Only one production technology is modeled for these materials. Cement and concrete (the most important cement application) are discussed in detail. These materials represent the major part of the CO₂ emissions in this materials group and a significant emission reduction potential exists.

The discussion in Chapter 2 focuses on current production technology, autonomous efficiency improvements and technologies that result in a significant reduction of CO₂ emissions. The selection of technologies (Table 1.2) is largely based on earlier studies about the same materials and on the analysis of the European production situation [4, 5].

Table 1.2 *List of selected materials production technologies*

Ammonia
- Reference + new KAAP/ICI/AMV process
- Advanced reforming/advanced reactors
- Reference + new KAAP/ICI/AMV process + CO ₂ removal and disposal
- Advanced reforming/advanced reactors + CO ₂ removal and disposal
Bricks
- Tunnel kiln
- Roller kiln
Cement
- Portland cement clinker
- Portland cement clinker + CO ₂ removal and disposal
- Portland cement
- Fly ash cement
- Blast furnace slag cement
- Activated slag cement
- Geopolymeric cement
Ceramic products
- Tile kiln
Chlorine/sodium hydroxide
- Membrane electrolysis cells
- Advanced membrane electrolysis cells
Concrete
- Ready-mix concrete production
- Precast concrete production
Glass
- Tank furnaces 100% primary, gas heated
- Tank furnaces 100% primary, electricity heated
- Tank furnaces 100% cullet, gas heated
- Tank Furnaces 100% cullet, electricity heated
Gypsum
- Drying process
Kaolin
- Mining, separation and drying process
Nitric acid
- Conventional production from ammonia
- Conventional with N ₂ O abatement technology
Phosphor
- Electric furnace
Phosphoric acid
- Wet production process
Potash
- Mining, separation and drying process
Quicklime
- Lime kiln
Sand-limestone
- Conventional production process
Sodium carbonate (soda)
- Light soda production process (Solvay)
- Heavy soda production process
Sodium chloride
- Leaching, separation and drying process
Sulphur
- Sulphuric acid from pure sulphur
Stone
- Quarrying and cutting
Titanium dioxide
- Chlorate process

For materials where only one process is modeled, improvements can be achieved through CO₂ emission reduction in electricity production and in heat production. Materials substitution and increased efficiency of materials use are other options to reduce the emissions in materials production.

The process system boundary is on the input side determined by the natural resource extraction (including production processes outside

Europe) and on the output side determined by the factory gate. Transportation of natural resources to the processor is included in the technology. Transportation to the retailer is included where this is applicable. Transportation to the consumer is not included in these technology characteristics. As a consequence, all materials costs refer to the fob price (free on board).

Chapter 3 focuses on chain management options. These improvements can be split into:

1. Increased materials efficiency
2. Materials recycling
3. Materials substitution

Materials substitution in products will not be discussed in this report, as this type of options can only be analysed on a specific product level. This type of options will be discussed in separate volumes concerning products.

The following options are discussed:

- increased fertilizer use efficiency (ammonia, phosphates, potash)
- light weight bricks
- cement recovery from used concrete
- high strength concrete
- light weight glass
- increased glass cullet recovery
- size reduction of sand-limestone bricks.

The production volume data show generally an accuracy of $\pm 10\%$, while the data regarding current energy efficiency are characterised by an accuracy ranging from 10% to 20%, depending on the process. Energy and materials efficiency data for future decades are even less reliable. Cost data should be considered as estimates, as no detailed cost analysis on a European scale has been performed. The cost data will in a later stage be checked on the basis of a comparison of MARKAL material and product shadow prices and market prices. Cost data can be adjusted if large differences occur.

2. TECHNOLOGY CHARACTERISATION FOR MATERIALS PRODUCTION

2.1 Ammonia

Ammonia is mainly used for production of fertilizers. Table 2.1.1 shows the ammonia balance for Western Europe.

Table 2.1.1 *Ammonia balance for Western Europe, 1994/1995 (Mt N) [6]*

	[Mt N]
Supply	
Nominal capacity	11.1
Continental supply capability	10.3
Net import	2.2
	12.5
Use	
Industrial	2.2
Fertilizer	9.7
Losses	0.6
	12.5

A further subdivision can be made, based on fertilizer types. The European fertilizer capacity is shown in Table 2.1.2 (note: urea is for 80% used for fertilizer production). Between 1988 and 1995, nitrogen fertiliser demand has decreased by 20%.

Table 2.1.2 *Nominal nitrogen fertilizer capacity in the EU, 1994 [7]*

	[Mt N]	[%]
Ammonia sulphate	0.75	7
Urea	2.78	25
Ammonium nitrate	2.46	22
Calcium ammonium nitrate	2.53	23
Ammonium phosphate	0.12	1
Compound fertilizers	2.40	22
Total	11.04	100

Ammonia is produced from hydrogen. 90% of the hydrogen is produced from light hydrocarbons. Partial oxidation of heavy oil products constitutes the remaining production capacity. Natural gas is the most important hydrogen feedstock (80% of the Western European production), but even heavier feedstocks up to heavy naphtha fractions can be used [8]. The minimum energy requirement (energy + feedstock) range from 27.3-29.4 GJ/t ammonia for the ICI-AMV process (3 plants in operation worldwide), 32.2 GJ/t for the Linde Ammonia Concept (LAC, 1 plant under construction) and 29.2 GJ/t for the Kellogg Advanced Ammonia Process (KAAP, 170 Kellogg plants in operation worldwide) [9]. Since 1986, 46% of all new

ammonia production capacity has been based on Topsoe technology. However, energy data for this technology have not been encountered.

The European average specific energy consumption (SEC) was in 1989 35.5 GJ/t [10]. This value is significantly higher than for the state-of-the-art plants because of the combination of significant efficiency gains in the last decade and the old age of the European ammonia production plants. Data in Table 2.1.1 suggest that the capacity utilization of existing plants is high. However further capacity expansion in Western Europe is less likely because of heavy competition from countries with cheap local natural gas reserves (Middle East). However advanced reactors with even lower energy consumption can in the next decade further decrease the fraction of energy costs in the product costs. This may enhance the competitiveness of the European production. Data in Table 2.1.3 show the model input for the reference technology. The closure of old ammonia plants and the investment in new energy efficient capacity between 1989 and 2000 has been included in these figures. As a consequence, the feedstock input per tonne ammonia in 2000 is only 31.5 GJ/t. The introduction of autothermal reforming has been included as autonomous energy efficiency improvement. In Table 2.1.4, the model parameters for an advanced production process are shown, where the energy consumption is lowered because large feedstock recycling flows can be avoided because of an advanced ammonia reactor design. Tables 2.1.5 and 2.1.6 show the characteristics of the same technologies including CO₂ compression and storage. This option can easily be added to the existing plants, where CO₂ is currently already removed in a gas purification step. One should add that the potential of this option is limited by CO₂ sales for other purposes. The extent of these sales is not known.

Table 2.1.3 *Production process data for ammonia (Reference+ new KAAP/ICI-AMV process including autothermal reforming) [5]*

	2000	2020	2050
Input			
Electricity [GJ/t]	0.5	0.5	0.5
Gas [GJ/t]	31.0	27.3	27.0
Output			
Ammonia [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	300		300
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	4		4
- Variable [ECU ₁₉₉₄ /t]	5		5
<i>including labour [ECU₁₉₉₄/t]</i>	5		5
Availability factor [-]	0.95		0.95
Life [yr]	25		25

Table 2.1.4 *Production process data for ammonia (advanced reforming/advanced reactors) [5, 11]*

	2000	2020	2050
Input			
Electricity [GJ/t]	-	0.5	0.5
Gas [GJ/t]	-	24.0	24.0
Output			
Ammonia [t]	-	1	1
Investment [ECU ₁₉₉₄ /t cap]	-	350	350
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	-	4	4
- Variable [ECU ₁₉₉₄ /t]	-	5	5
<i>including labour [ECU₁₉₉₄/t]</i>	-	5	5
Availability factor [-]	-	0.95	0.95
Life [yr]	-	25	25

Table 2.1.5 *Production process data for ammonia (Reference + new KAAP/ICI-AMV process including autothermal reforming) with CO₂ removal and disposal [5]*

	2000	2020	2050
Input			
Electricity [GJ/t]	-	1.0	1.0
Gas [GJ/t]	-	27.3	27.0
Output			
Ammonia [t]	-	1	1
CO ₂ storage [t/t]	-	1.5	1.5
Investment [ECU ₁₉₉₄ /t cap]	-	330	330
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	-	4	4
- Variable [ECU ₁₉₉₄ /t]	-	6	6
<i>including labour [ECU₁₉₉₄/t]</i>	-	5	5
Availability factor [-]	-	0.95	0.95
Life [yr]	25		25

Table 2.1.6 *Production process data for ammonia (advanced reforming/ advanced reactors) with CO₂ removal and disposal [5, 11]*

	2000	2020	2050
Input			
Electricity [GJ/t]	-	0.9	0.9
Gas [GJ/t]	-	24.0	24.0
Output			
Ammonia [t]	-	1	1
CO ₂ storage [t/t]	-	1.3	1.3
Investment [ECU ₁₉₉₄ /t cap]	-	370	370
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	-	4	4
- Variable [ECU ₁₉₉₄ /t]	-	6	6
<i>including labour [ECU₁₉₉₄/t]</i>	-	5	5
Availability factor [-]	-	0.95	0.95
Life [yr]	-	25	25

2.2 Bricks

Tunnel ovens represent the most widely accepted kiln type, gradually replacing older concepts. Larger kilns use less energy per tonne of bricks than small kilns. A gradual trend towards large tunnel kilns is assumed as autonomous development, resulting in a gradual decline of energy consumption. The energy consumption of a modern tunnel kiln (2.5 GJ/t) is approximately 20% lower than for the older kiln designs. A further improvement to 2.0 GJ/t seems feasible [12]. It is assumed that this improvement is part of the autonomous development to the year 2050.

Energy costs represent a significant fraction of the total brick production costs. As a consequence, the European average energy consumption does probably not significantly differ from the Dutch average. For example data for the UK indicate an average energy requirement of 2.5 GJ/t for tunnel kilns for the year 1990 (probably excluding the drying section, which requires approximately 1 GJ/t, [13]).

Apart from the kiln design, the brick type is an important variable that determines the energy requirements. Table 2.2.1 shows the process energy requirements for different brick types. The difference is accounted for by the different kiln temperature and the different kiln residence time. The bulk of the bricks consists of the "outside cladding" type. The brick quality is characterised by the design strength and the water absorption rate.

Table 2.2.1 *Process energy requirements for different brick quality types, Dutch situation, 1993 [12]*

	[GJ/t]
Outside cladding	2.9
Pavement brick	3.3
Inside cladding	0.7

Addition of biomass to the clay is one way to reduce the fossil energy consumption in brick production. However, the savings potential is small, while the product quality is affected. UK estimates indicate an energy saving potential of 1% [13]. Because of the small saving potential and the small significance of the brick industry, this option will not be modeled separately. Table 2.2.2 shows the model input data for the tunnel kiln.

The roller kiln is modeled as improvement option (Table 2.2.3). The energy consumption is lower because of a lower residence time and less materials intensive transportation systems for the bricks in the ovens. These transportation systems must also be heated. The roller kiln is currently already used for production of sanitary stoneware products.

The production on the basis of roller kilns saves 0.2-0.4 GJ/t, compared to the state-of-the-art tunnel kiln [14]. The process is especially suited to produce bricks that are not frost resistant. As a consequence, the potential is limited to 50% of the brick production.

Table 2.2.2 *Production process data for bricks (assuming a gradual introduction of tunnel kilns) [5]*

	2000	2020	2050
Input			
Natural gas [GJ/t]	2.0	1.9	1.9
Fuel oil [GJ/t]	0.5	0.2	0.1
Electricity [GJ/t]	0.1	0.05	0.05
Output			
Bricks [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	125	125	125
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	7	7	7
- Variable [ECU ₁₉₉₄ /t]	15	15	15
<i>including labour [ECU₁₉₉₄/t]</i>	15	15	15
Availability factor [-]	0.95	0.95	0.95
Life [yr]	20	20	20
Residual capacity	50	10	0

Table 2.2.3 *Production process data for bricks (roller kiln) [14]*

	2000	2020	2050
Input			
Natural gas [GJ/t]	1.6	1.6	1.6
Electricity [GJ/t]	0.1	0.05	0.05
Output			
Bricks [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	150	150	150
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	7	7	7
- Variable [ECU ₁₉₉₄ /t]	15	15	15
<i>including labour [ECU₁₉₉₄/t]</i>	15	15	15
Availability factor [-]	0.95	0.95	0.95
Life [yr]	20	20	20
Boundup [Mt pa]	5	30	30

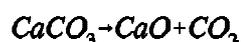
2.3 Cement

Portland cement clinker is produced by calcination of a mixture of clay and limestone (CaCO_3). The clinker contains approximately 65% CaO. 5% gypsum is added to the ground cement clinker in order to produce cement. The most widely used production process is the energy efficient dry process that has almost completely replaced the wet process in Western Europe. In the last decades, the pre-calcination technology has been introduced as energy saving measure. Another energy saving measure is an increased number of preheaters. Data are shown in Table 2.3.1. The temperature of the residual gases decreases accordingly, but the pressure loss over the drying section increases from 50 to 55 mbar. This implies additional energy use to overcome this pressure loss. Total power consumption for clinker production is approximately 100 MJe/t clinker (preheater+cooling+bypass, excluding grinding). In the European Union, the total specific energy consumption per tonne portland cement is currently on average 3.7 GJ/t [10]. Based on current trends, the gradual replacement of conventional dry kilns by dry kilns with precalcining and 6-step preheaters is modeled as autonomous development.

Table 2.3.1 *The energy consumption for cement clinker production in rotary kilns as a function of the number of preheaters [15]*

No. of cyclone preheaters	Fuel consumption [GJ/t]
4	3.38
5	3.10
6	3.00
Theoretical minimum	1.70-1.80

The bulk of the CO₂-emissions in cement production is however not related to the consumption of fossil energy. A significant process emission is caused by the calcination process (see Table 1.1):



This emission cannot be tackled with increased energy efficiency. However, replacement of limestone by other resources can solve the problem. This is illustrated in Table 2.3.2, where total CO₂ emissions (from fuel consumption and from calcination) are compared for different cement types. The data in Table 2.3.2 refer to four cement types based on different resources:

- fly-ash cement, based on residuals from coal based power production
- blast furnace cement, based on slag from iron production
- two types of activated slag cement, based on synthetic slag, fly ash, quicklime, and water glass (sodium silicate and potassium silicate).

Table 2.3.2 *CO₂ emissions for different cement types (includes process emissions + energy related emissions)* [16]

	[t CO ₂ /t cement]
Portland cement	0.86
Portland fly ash cement	0.67
Blast furnace slag cement	0.35
Activated slag cement I	0.55
Activated slag cement II	0.50

The composition of the cement types is shown in Table 2.3.3. Quicklime and pozzolanic mixtures without cement clinker pose also an important alternative. They are currently widely applied in Germany and in Italy. However the resulting concrete quality is water sensitive. Lime mortars will only harden if they are exposed to air, because the development of strength is based on the recombination of the calcium oxide with CO₂. Portland cement binders will also harden when immersed in water [17].

Table 2.3.3 *Composition of different cement types* [16]

Cement type ¹	PC [%]	PFC [%]	BFC [%]	ASCI [%]	ASCII [%]
Clinker	95	75	30	-	-
Fly ash	-	25	-	45	45
BF slag	-	-	65	-	-
Synth. slag	-	-	-	45	45
Quicklime	-	-	-	5	-
Water glass	-	-	-	-	10
Na ₂ SO ₄	-	-	-	5	-
Gypsum	5	-	5	-	-

¹ PC = Portland Cement
PFC = Portland Fly-ash Cement
BFC = Blast Furnace slag Cement
ASC = Activated Slag Cement (type I and II)

The comparison of different cement compositions is complicated because the features of the cement types differ. Especially the strength can differ. Higher strength cements allow structures to be less bulky. As a consequence, a comparison of the CO₂ emissions per tonne cement is not correct. In the following analysis, strength differences are not considered. High strength cement is discussed in more detail in Section 3.3.

Table 2.3.4 shows the European cement consumption, divided into cement types. The portland cement quality with less than 30% additives contains either blast furnace slag or fly ash. It is assumed that 50% is fly ash and 50% is blast furnace slag. The blast furnace cement contains on average 70% blast furnace slag. Based on these two assumptions, the blast furnace slag consumption is approximately 11 Mt and the fly ash consumption is approximately 5 Mt. This leaves ample room for further introduction of fly ash cement and blast furnace cement. Data in Table 2.3.5 represent the reference production process. The average energy consumption of 3.7 GJ/t for 1989 has been extrapolated for the year 2000. Fuel consumption of 3.1 GJ/t is assumed to represent the best available technology. The minimum value is based on Table 2.3.1 and on [18].

Table 2.3.4 *Cement domestic deliveries in Western Europe, 1993 (based on [19, 20])*

Type	Portland [%]	Portland+ additives ¹ [%]	BF [%]	Others ² [%]
Austria	86.6	8.1	2.1	3.2
Belgium	30.0	25.1	44.9	-
Denmark	54.0	20.7	25.2	-
Finland	61.6	29.4	-	-
France (1988)	28.0	60.0	4	8
Germany	76.6	3.2	12.9	7.3
Greece	26.0	73.0	-	1.0
Ireland	100.0	-	-	-
Italy	6.3	70.0	3.4	20.5
Netherlands	27.0	12.0	61.0	-
Norway	79.0	21.0	-	-
Portugal	99.0	-	0.4	0.6
Spain	19.4	66.6	1.0	12.9
Switzerland	93.1	6.9	-	-
UK (1988)	91.0	-	2	7
Total (est.)	70	20	5	5

¹ less than 30% additives

² mainly pozzolanic cement

The cement clinker production is well suited for combustion of waste fuels if a precalciner is installed (common in all modern clinker plants). The high temperature and the high pH prevent the generation of toxic compounds, while heavy metals are sealed in the clinker. Waste wood, waste paper, waste tyres, waste plastics and waste oils can all be incinerated in these

kilns. Data regarding the actual current energy mix for clinker production have not been encountered. Based on data for individual countries and plants, it is estimated that between 2 and 4 Mt waste is combusted (mainly waste wood and waste tyres), representing an energy value of 30 to 100 PJ, or 6-20% of the total energy consumption of the sector (mid 1990's situation) [21]. The model input data for the production processes are shown in Table 2.3.5-2.3.11.

 Table 2.3.5 *Production process data for Portland cement clinker [5, 22]*

	2000	2020	2050
Input			
Coal/gas/residuals [GJ/t]	3.5	3.3	3.1
Electricity [GJ/t]	0.1	0.1	0.1
Output			
Portland cement [t]	1	1	1
Inorganic CO ₂ [t/t]	0.51	0.51	0.51
Investment [ECU ₁₉₉₄ /t cap]	75		75
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	5		5
- Variable [ECU ₁₉₉₄ /t]	5		5
<i>including labour [ECU₁₉₉₄/t]</i>	5		5
Availability factor [-]	0.95		0.95
Life [yr]	25		25

Table 2.3.6 *Production process data for Portland cement clinker + CO₂ removal and separation (based on [23])*

	2000	2020	2050
Input			
Coal [GJ/t]	3.5	3.3	3.1
LT-steam [GJ/t]	3.5	3.2	3.0
Electricity [GJ/t]	0.5	0.5	0.5
Output			
Portland cement [t]	1	1	1
CO ₂ storage [t/t]	0.8	0.75	0.75
Net CO ₂ emission [t/t] ¹	-0.3	-0.25	-0.25
Investment [ECU ₁₉₉₄ /t cap]	75		75
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	80		80
- Variable [ECU ₁₉₉₄ /t]	15		15
<i>including labour [ECU₁₉₉₄/t]</i>	5		5
Availability factor [-]	0.95		0.95
Life [yr]	25		25
Boundup	0	20	75

¹ Net emission = process emission - storage = 0.5 - 0.8 = - 0.3 t/t (Emissions due to fossil fuel combustion are in MARKAL accounted for during the import or mining stage. Consequently, a net storage must be modeled for the clinker production process)

Table 2.3.7 *Production process data for Portland cement (based on [16])*

	2000	2020	2050
Input			
PC clinker [t/t]	0.95	0.95	0.95
Gypsum [t/t]	0.05	0.05	0.05
Electricity [GJ/t]	0.10	0.10	0.10
Output			
Cement [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	10		10
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	0.5		0.5
- Variable [ECU ₁₉₉₄ /t]	0.5		0.5
<i>including labour [ECU₁₉₉₄/t]</i>	0.5		0.5
Availability factor [-]	0.95		0.95
Life [yr]	25		25

Table 2.3.8 *Production process data for fly ash cement [16]*

	2000	2020	2050
Input			
PC clinker [t/t]	0.75	0.75	0.75
Fly ash [t/t]	0.25	0.25	0.25
Electricity [GJ/t]	0.15	0.15	0.15
Output			
Cement [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	10		10
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	0.5		0.5
- Variable [ECU ₁₉₉₄ /t]	0.5		0.5
<i>including labour [ECU₁₉₉₄/t]</i>	0.5		0.5
Availability factor [-]	0.95		0.95
Life [yr]	25		25

The production of fly-ash cement is limited by the availability of fly-ash. Per tonne of coal that is used for electricity production, 0.09 tonnes of fly-ash are produced. Total coal use for power production will in the year 2000 amount to approximately 5000 PJ. This amount equals approximately 15 Mt fly-ash, representing a potential cement production of 60 Mt. Total cement production is approximately 175 Mt per year. The comparison of both figures shows that the potential of fly-ash cement is limited. This conclusion is especially valid in a situation with CO₂ emission reduction, where coal will be replaced by gas or renewables for power production.

Table 2.3.9 *Production process data for blast furnace slag cement [16]*

	2000	2020	2050
Input			
PC clinker [t/t]	0.30	0.30	0.30
Blast furnace slag [t/t]	0.65	0.65	0.65
Gypsum [t/t]	0.05	0.05	0.05
Electricity [GJ/t]	0.29	0.29	0.29
LTH ¹ [GJ/t]	0.40	0.35	0.30
Output			
Cement [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	10		10
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	0.5		0.5
- Variable [ECU ₁₉₉₄ /t]	0.5		0.5
<i>including labour [ECU₁₉₉₄/t]</i>	0.5		0.5
Availability factor [-]	0.95		0.95
Life [yr]	25		25

¹ The drying of blast furnace slag requires additional energy

The production of blast furnace slag cement is limited by the slag availability. Western European iron production amounted in 1992 to 80 Mt. The slag production amounts to approximately 300 kg per tonne of iron [24], so 24 Mt slag is available. This represents a potential cement production of 37 Mt (29% of the total cement production). The bulk of this slag is currently still applied as crushed slag for sand piles or as silicate fertilizer. If the slag is granulated, it can be used for cement production. Data in Table 2.3.4 suggest an additional availability of 13 Mt blast furnace slag that is not yet used for cement. One must add that the introduction of new iron production processes like direct reduced iron production may reduce the slag availability (see [25]).

The production process data for activated slag cement are shown in Table 2.3.10. The synthetic slag is produced from a mixture of fly ash (480 kg/t slag), limestone (900 kg/t slag), and residuals from alumina production ('Red mud', 100 kg/t slag). The resources are heated in a fluid bed, where the limestone is calcined. The product is molten in an electric furnace at a temperature of 1500 °C and quenched to produce synthetic slag. This synthetic slag is mixed with fly ash, ammonium sulphate and slaked lime in a weight ratio 45:45:5:5.

Table 2.3.10 *Production process data for ASC I* [16]

	2000	2020	2050
Input			
Limestone [t/t]	0.41	0.41	0.41
Red mud [t/t]	0.05	0.05	0.05
Water [t/t]	0.02	0.02	0.02
Ammonium sulphate [t/t]	0.05	0.05	0.05
Fly ash [t/t]	0.67	0.67	0.67
Quicklime [t/t]	0.04	0.04	0.04
Coal/gas/residual [GJ/t]	1.55	1.55	1.55
Gas [GJ/t]	0.20	0.20	0.20
Electricity [GJ/t]	1.0	1.0	1.0
Output			
Cement [t]	1	1	1
Inorganic CO ₂ [t/t]	0.22	0.22	0.22
Investment [ECU ₁₉₉₄ /t cap]	100		100
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	5		5
- Variable [ECU ₁₉₉₄ /t]	5		5
<i>including labour [ECU₁₉₉₄/t]</i>	5		5
Availability factor [-]	0.95		0.95
Life [yr]	25		25

Table 2.3.11 *Production process data for geopolymeric cement (based on [26])*

	2000	2020	2050
Input			
Soda [t/t]	0.16	0.16	0.16
Sand [t/t]	0.84	0.84	0.84
Natural gas [GJ/t]	3.5	3.3	3.1
Electricity [GJ/t]	0.25	0.25	0.25
Output			
Cement [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	100		100
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	5		5
- Variable [ECU ₁₉₉₄ /t]	5		5
<i>including labour [ECU₁₉₉₄/t]</i>	5		5
Availability factor [-]	0.95		0.95
Life [yr]	25		25

Geopolymeric cement is based on the use of NaO and K₂O, similar to activated slag cement II in Table 2.3.3, that bind with silicates to water glass. The structure is similar to natural pozzolans and trass cement. Na₂O is produced from soda, Na₂CO₃. The production of soda requires 7.3 PJ heat and 0.3 PJ electricity per tonne [27]. Assuming 100% use of Na₂O, one tonne geopolymeric cement requires 0.16 tonnes of soda. The saving in CO₂ emission is based on the feature that 1 tonne of Na₂O can produce 5-10 times as much cement as one tonne of CaO, because three dimensional geopolymeric silico-aluminate structures are formed. However, the pH stability of this type of cement is considerably lower. A high pH value is important in situations where steel reinforcements are used, because steel is protected against oxydation in a basic environment. For this reason, it is thought that the application potential of this type of cement is limited.

2.4 Ceramic products

Ceramic products can be split into:

- floor tiles
- sanitary stoneware
- table stoneware.

The total energy consumption in this sector was approximately 120 PJ in 1992. The tile industry represents 74% of this energy consumption. The production of tiles is 840 million m², equal to approximately 7.5 Mt. Production data for sanitary stoneware, table stoneware, and china in weight units have not been encountered. Sanitary stoneware production was approximately 45 million pieces in 1995. A product subdivision is shown in Figure 2.4.1. Assuming an average weight of 25-50 kg, the total production of sanitary stoneware is 1-2 Mt per year. Table stoneware and china possess a lower relevance.

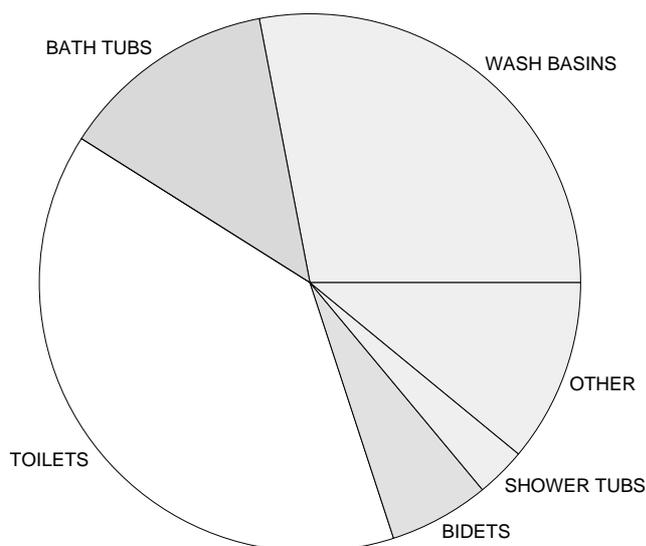


Figure 2.4.1 *Production of sanitary stoneware in Western Europe (no. of pieces) [28]*

The energy consumption per unit of product depend to a large extent on the product specifications (see Table 2.4.1).

Table 2.4.1 *Energy consumption per weight unit for different types of ceramic products [29]*

	[GJ/t]
White tiles, glazed	9
Red tiles, not glazed	6
Table stoneware	10
China	70
Sanitary stoneware	30

Only tiles will be modeled. The potential for increased energy efficiency in tile production is limited to approximately 15% [29]. These efficiency improvements have been taken into account for the model input data in 2020 and beyond. Because of the wide range in energy efficiency for different products, a more detailed approach is required for modelling of energy efficiency improvement options. The wide range of products and the low relevance from a CO₂ point of view makes such a detailed approach less relevant. Finally, labour costs in Table 2.4.2 are based on [30].

Table 2.4.2 *Production process data for tiles [29]*

	2000	2020	2050
Input			
Natural gas [GJ/t]	8.0	6.0	6.0
Electricity [GJ/t]	0.5	0.5	0.5
Output			
Tiles [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	500		500
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	25		25
- Variable [ECU ₁₉₉₄ /t]	600		600
<i>including labour [ECU₁₉₉₄/t]</i>	<i>560</i>		<i>560</i>
Availability factor [-]	0.95		0.95
Life [yr]	25		25

2.5 Chlorine/sodium hydroxide

NaCl is decomposed electrochemically to yield sodium hydroxide (Na(OH)) and chlorine (Cl₂). Hydrogen is generated as by-product. The consumption of chlorine and sodium hydroxide is analysed in Table 2.5.1. Total chlorine production amounted to 8.4 Mt in 1990 [1].

Table 2.5.1 *Consumption of chlorine and sodium hydroxide, Western Europe, 1994 [31]*

	Chlorine [%]	Sodium hydroxide [%]
Organic chemicals	37	29
EDC/VCM	35	-
Pulp and paper	1	13
Inorganic materials	13	27
Water treatment	4	4
Miscellaneous	10	27 ¹
Total	100	100

¹ 3% food; 5% soap; 4% metals production; rayon production; neutralisation of waste acids

Three production processes are currently applied. The mercury process and the diaphragm process are gradually replaced by the membrane electrolysis process. Table 2.5.2 shows these processes as a fraction of the total Western European production capacity for 1992.

Table 2.5.2 *Western European chlorine capacity, 1992 [32]*

	Production fraction [%]	Energy consumption [GJe/t Cl ₂]
Mercury process	66	11.8
Diaphragm process	26	10.0
Membrane process	8	8.6-9.2 ¹

¹ For current densities of 3.0 kAmpere/m² and 4.0 kAmpere/m², respectively

Data for the electricity consumption (expressed per tonne chlorine) are also shown in Table 2.5.2. These data include only the electrolysis process (salt production, Cl₂ gas compression and Na(OH) concentration are excluded). The product quality is not equal for the three electrolysis processes. The mercury cell results in a 50% sodium hydroxide concentration, the diaphragm cell results in a 11% sodium hydroxide concentration, the membrane cell results in a 33% sodium hydroxide concentration. Sodium hydroxide is generally sold in a 50% concentration or as solid pellets. The steam consumption for concentration of the sodium hydroxide solution to a 50% concentration amounts to 2.25 GJ/t NaOH (2.54 GJ/t Cl₂) for the diaphragm cell and 0.52 GJ/t NaOH (0.59 GJ/t Cl₂) for the membrane cell. For all three processes, the chlorine is generally purified through liquefaction and evaporation before it is sold (energy consumption: 0.24 GJe/t).

Energy requirements for the membrane process are determined by the current densities and the cell characteristics (especially the cell overvoltage). Current yields (for Na(OH)) are in the range of 95-98%, so the potential for savings is limited (theoretical maximum 100%). The cell voltage is currently 3.47 V, while the theoretical minimum is 2.19 V. Two improvements can reduce the overvoltage. One option is improved membranes to lower the membrane ohmic drop from 0.4 to 0.15 V. This option can be incorporated in the existing membrane electrolysis cell technology. The other important option is reduction of cathode overvoltage from 0.58 to 0.10 V. This is possible if a nickel coated cell (nickel cathode) is used. This adjustment has another advantage: higher Na(OH) concentrations in the electrolysis cell are possible if nickel is used as cell material instead of steel, so the steam for concentration of the Na(OH) product can be saved.

Table 2.5.3 and Table 2.5.4 show the model input data. The reduction of the membrane overvoltage is included in the reference technology, where the electricity consumption decreases from 11.0 to 9.0 GJ/t Cl₂ (data refer to bipolar cells, current density 4 kA/m²). The data for the year 2000 in Table 2.5.3 represent an average for mercury cells, diaphragm cells and membrane cells. It is assumed that membrane cells will completely replace the other two types because of lower costs and reduced environmental impacts. The nickel coated membrane cell is modeled as improvement option (Table 2.5.4). Data include Cl₂ purification.

Table 2.5.3 *Production process data for reference electrolysis cells (based on [5])*

	2000	2020	2050
Input			
Electricity [GJ/t]	11.0	9.0	9.0
Steam [GJ/t]	1.0	0.5	0.5
NaCl [t/t]	1.7	1.7	1.7
Output			
Chlorine [t]	1	1	1
Caustic soda (50%) [t NaOH/t]	1.13	1.13	1.13
Hydrogen [GJ/t]	3.38	3.38	3.38
Investment [ECU ₁₉₉₄ /t cap]	750	750	750
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	100	10	100
- Variable [ECU ₁₉₉₄ /t]	90	90	90
<i>including labour [ECU₁₉₉₄/t]</i>	90	90	90
Availability factor [-]	0.95	0.95	0.95
Life [yr]	25	25	25
Residual capacity [Mt/year]	6	0	0

Table 2.5.4 *Production process data for advanced membrane electrolysis cells (based on [5])*

	2000	2020	2050
Input			
Electricity [GJ/t]		7.9	7.5
Steam [GJ/t]		-	-
NaCl [t/t]		1.7	1.7
Output			
Chlorine [t]		1	1
Caustic soda (50%) [t NaOH/t]		1.13	1.13
Hydrogen [GJ/t]		3.38	3.38
Investment [ECU ₁₉₉₄ /t cap]		1100	1100
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]		100	100
- Variable [ECU ₁₉₉₄ /t]		90	90
<i>including labour [ECU₁₉₉₄/t]</i>		90	90
Availability factor [-]		0.95	0.95
Life [yr]		25	25

2.6 Concrete

Concrete is produced from a mixture of cement, sand, gravel and water. The mixture solidifies due to the reaction of silica and calcium hydroxide. The total primary energy consumption per tonne ready mix concrete is approximately 0.1 GJ/t [33]. Approximately 0.006 GJ electricity is used, the remaining energy consumption consists of oil products and natural gas. The upstream energy consumption (and even more the upstream CO₂ emissions) are 10-20 times more important than the emissions in concrete production, depending on the cement content of the concrete. If the concrete is reinforced, the energy content of the steel reinforcement dominates the energy balance (and the cost price) [34].

The concrete strength quality depends also on the cement content. The average cement consumption for ready mix concrete has been derived from the producers statistics. The cement content of 12% is relatively low, compared to the cement content of prefabricated building products (see Table 2.6.2).

Market data for concrete are given in Table 2.6.1. Approximately 25% of the total cement consumption is accounted for by the precast concrete industry, where the concrete is formed into its final shape in an industrial production process. The precast element is transported to the construction site [35]. The precast concrete industry is more dominant in Northern European and Central European countries because of the climatological conditions: ready mix concrete cannot be processed at low temperatures. Precast is also more dominant in densely populated regions due to the smaller transportation distances, making prefab more economical than in rural regions.

The precast concrete industry is more labour intensive than the ready mix concrete industry. The labour costs represent 25-40% of the total product costs in the precast industry (based on [36]). In the ready mix industry, labour costs represent a much smaller fraction of the total costs. However, additional labour costs occur during the building stage.

Model input data for ready mix concrete and prefab concrete are shown in Table 2.6.2 and 2.6.3. One must add that these data are a very simple representation of a complex reality with many concrete qualities. For example reinforcements are separately modeled as material input into constructions.

Table 2.6.1 *Western European concrete market data [34]*

	[Mt concrete pa]
Ready mix concrete	600
Prefabricated concrete	180

Table 2.6.2 *Production process data for conventional ready mix concrete [37]*

	2000	2020	2050
Input			
Electricity [GJ/t]	0.006	0.006	0.006
Diesel [GJ/t]	0.03	0.03	0.03
Heavy fuel oil [GJ/t]	0.03	0.03	0.03
Natural gas [GJ/t]	0.03	0.03	0.03
Cement [t/t]	0.12	0.12	0.12
Sand and gravel [t/t]	0.80	0.80	0.80
Water [t/t]	0.09	0.09	0.09
Output			
Concrete [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	50	50	50
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	1	1	1
- Variable [ECU ₁₉₉₄ /t]	10	10	10
<i>including labour [ECU₁₉₉₄/t]</i>	10	10	10
Availability factor [-]	0.95	0.95	0.95
Life [yr]	25	25	25
Residual capacity [Mt/year]	200	0	0

Table 2.6.3 *Production process data for conventional prefab concrete products (excluding reinforcements, estimates based on [33, 34])*

	2000	2020	2050
Input			
Electricity [GJ/t]	0.006	0.006	0.006
Diesel [GJ/t]	0.03	0.03	0.03
Heavy fuel oil [GJ/t]	0.03	0.03	0.03
Natural gas [GJ/t]	0.03	0.03	0.03
Cement [t/t]	0.20	0.20	0.20
Sand and gravel [t/t]	0.70	0.70	0.70
Water [t/t]	0.10	0.10	0.10
Output			
Prefab concrete [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	50	50	50
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	1	1	1
- Variable [ECU ₁₉₉₄ /t]	40	40	40
<i>including labour [ECU₁₉₉₄/t]</i>	40	40	40
Availability factor [-]	0.95	0.95	0.95
Life [yr]	25	25	25
Residual capacity [Mt/year]	200	0	0

2.7 Glass

Glass is produced by the melting of a mixture of sand and other compounds at temperatures of 1400 to 1500 °C. The composition of resources for soda-lime glass production, i.e. the bulk of total glass production, is shown in Table 2.7.1. 1-2% of the total glass production consists of borosilicate glass and lead crystal with differing composition. These less relevant glass types will not be discussed in detail. Conventional container glass contains 70% SiO₂, 15% CaO, and 15% Na₂O [38].

Table 2.7.1 *Resources used in soda-lime glass production*

Sand	60-70
Na ₂ CO ₃ (soda) / Na ₂ SO ₄	15-20
Dolomite/limestone	10-20
Felspar	0-5

The glass production process can be split into:

- preparation
- melting
- forming
- post forming
- product handling.

More than half of the energy consumption in the glass production process is accounted for by the melting process.

The molten glass is formed into its final shape. On the basis of the final shape, glass production can be split into flat glass and container glass. The minimum energy requirement is 0.5 GJ/t for chemical reactions and 1.8 GJ/t for melting. The theoretical minimum depends to some extent on the glass type. Table 2.7.2 shows the minimum energy requirements for different glass types.

Table 2.7.2 *Theoretical minimum melting energy requirement for different glass types (no cullet addition) [39]*

	Melting energy [GJ/t]	Temperature [°C]
Flat glass, heavy soda resource	2.49	1300
Flat glass, light soda	2.57	1300
White container glass	2.66	1500
Green container glass	2.77	1500

The furnace design can be split into furnaces that use fossil fuels and furnaces that use electricity for melting. Fossil fuel fired furnaces are nowadays equipped with recuperators or regenerators. Approximately 35 to 45% of the total energy input for glass melting in modern regenerative furnaces is covered by the theoretical energy requirement, structural heat losses contribute up to 20 to 25% of the energy input and the losses due to the heat content of flue gases represent 25% to 35% of the energy input [40]. The final energy consumption of the electrically heated furnaces is

lower because energy losses associated with the flue gases do not occur. The required quantity and quality of the glass will primarily determine the choice of the melting furnace. A subdivision of glass production is shown in Table 2.7.3.

Table 2.7.3 *European glass production in 1992* [41]

	[Mt pa]
Container glass	15.3
Flat glass	5.7
Tableware	0.9
Glass fibre	0.4
Others	0.9
Total	23.2

The specific energy consumption depends to a large extent on the scale of production, which depends on the product type. Container glass and flat glass are produced in large ovens with a capacity of 100 kt per year or more. Tableware is mainly produced in tank furnaces with a capacity in the range of 25 kt per year. The furnace design is similar to the furnaces for container glass and flat glass. The energy consumption for these furnaces is approximately 17 GJ/t (Specific Energy Consumption, including melting, forming and annealing). They are included in one category with container glass and flat glass production. Some smaller pot furnaces are also used for production of tableware. The production of these pot furnaces is relatively small (10 kt pa), but their energy consumption is rather high (SEC 100 GJ/t). The quantity of glass melted in pot furnaces is likely to diminish in the future as more companies adopt automatic forming and continuous electric melting. The savings of this switch are in the range of 50-80%.

The melting energy for flat glass and container glass production in fossil fuel fired furnaces is in the range of 5-6 GJ/t. A survey for Germany, Austria and Switzerland indicates an average melting energy of 4.8 GJ per tonne container glass (variation 3.5-7.9 GJ/t) and 5.7 GJ per tonne flat glass (variation 3.4-7.8 GJ/t) for fossil fuel heated melting ovens in 1993 [39]. In these countries, approximately half of the ovens is heated with natural gas; the other half is heated with heavy fuel oil. Only natural gas heated ovens are modeled because of their lower CO₂ emissions.

For electric heated melting ovens in the same region, the energy consumption amounted on average to 4.2 GJe per tonne glass in 1993. For electric heated furnaces in the USA, electricity consumption is approximately 3.0 GJe/t [42]. The difference is to a large extent accounted for by larger ovens.

Increased recycling is one way to reduce the energy consumption, because the energy for the chemical reactions can be saved. As a general rule, 10% extra cullet results in a 2.5 - 3% reduction of the furnace energy consumption (from 5.2 to 4.0 GJ/t for the rang 0-100% cullet). Moreover, the production of soda is reduced. Soda production requires approximately 10

GJ/t. 10% extra cullets results in 1.0 GJ/t additional savings due to reduced soda production [43]. The current average glass recycling rate is in the range of 50%. Higher recycling rates seem feasible (see Section 3.5).

Other energy saving measures are:

- use of oxygen burners (especially relevant for small furnaces)
- increase of energy recovery through regenerative/recuperative burners
- preheat glass cullet
- increase of furnace size
- use of melting aids.

These measures are included in the autonomous efficiency improvements. The labour costs in this analysis are based on [44]. The model input data for the melting processes are shown in Tables 2.7.4-2.7.7.

Table 2.7.4 *Production process data for glass, gas fired tank furnace (100% primary) (based on [4, 39])*

	2000	2020	2050
Input			
Sand/felspar [t/t]	0.72	0.72	0.72
Limestone [t/t]	0.23	0.23	0.23
Soda [t/t]	0.26	0.26	0.26
Natural gas [GJ/t]	5.2	4.0	3.5
Output			
Liquid glass [t]	1	1	1
Inorganic CO ₂ [t/t]	0.21	0.21	0.21
Investment [ECU ₁₉₉₄ /t cap]	250		250
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	20		20
- Variable [ECU ₁₉₉₄ /t]	250		250
<i>including labour [ECU₁₉₉₄/t]</i>	250		250
Availability factor [-]	0.95		0.95
Life [yr]	25		25

Table 2.7.5 *Production process data for glass, electricity fired tank furnace (100% primary) (based on [4, 39])*

	2000	2020	2050
Input			
Sand/felspar [t/t]	0.72	0.72	0.72
Limestone [t/t]	0.23	0.23	0.23
Soda [t/t]	0.26	0.26	0.26
Electricity [GJ/t]	4.2	3.5	3.0
Output			
Liquid glass [t]	1	1	1
Inorganic CO ₂ [t/t]	0.21	0.21	0.21
Investment [ECU ₁₉₉₄ /t cap]	150		150
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	10		10
- Variable [ECU ₁₉₉₄ /t]	250		250
<i>including labour [ECU₁₉₉₄/t]</i>	250		250
Availability factor [-]	0.95		0.95
Life [yr]	25		25
Resid	4	0	0

Table 2.7.6 *Production process data for glass, gas fired tank furnace (100% cullet)*

	2000	2020	2050
Input			
Cullets [t/t]	1.05	1.05	1.05
Natural gas [GJ/t]	4.7	3.5	3.0
Output			
Liquid glass [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	250		250
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	20		20
- Variable [ECU ₁₉₉₄ /t]	250		250
<i>including labour [ECU₁₉₉₄/t]</i>	250		250
Availability factor [-]	0.95		0.95
Life [yr]	25		25

Table 2.7.7 *Production process data for glass, electricity fired tank furnace (100% cullet)*

	2000	2020	2050
Input			
Cullet [t/t]	1.05	1.05	1.05
Electricity [GJ/t]	4.5	3.0	2.5
Output			
Liquid glass [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	150		150
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	10		10
- Variable [ECU ₁₉₉₄ /t]	250		250
<i>including labour [ECU₁₉₉₄/t]</i>	250		250
Availability factor [-]	0.95		0.95
Life [yr]	25		25

After the melting process, packaging glass is shaped and annealed. Shaping, annealing, and additional processes require 0.52 GJ electricity and 0.42 GJ natural gas [45]. For floating glass, the energy requirements for annealing are in the range of 0.8-1.5 GJ/t depending on the glass thickness [46]. The potential for savings in this section is rather small. The energy consumption for post-treatment must be added to the energy consumption in Tables 2.7.4-2.7.7.

2.8 Gypsum

Gypsum can be split into three types:

CaSO ₄	anhydrous
CaSO ₄ ·½H ₂ O	hemihydrate
CaSO ₄ ·2H ₂ O	dihydrate

Large gypsum deposits can be found in most European countries, generally of the dihydrate type. This dihydrate can be converted into hemihydrate by heating.

Gypsum can be produced from different resources:

1. Mining of natural gypsum
2. Gypsum as by-product from flue gas desulphurisation
3. Gypsum as by-product from phosphate production
4. Gypsum as by-product from titanium dioxide production based on the sulphate process.

Most phosphate gypsum (a residue from sedimentary phosphate ore processing) cannot be applied in the building industry because of its radioactivity. Therefore, it will not be considered in more detail. The residues from titaniumdioxide production through the sulphate process are less relevant on the longer term because of the introduction of a new process route (see Section 2.19).

Total natural gypsum mining amounted in Western Europe to 22 Mt dihydrate in 1990 [1]. The main producers of natural gypsum are Spain (7.5 Mt) and France (5 Mt). The increasing use of by-product gypsum is not always included in statistics. The total amount of gypsum from desulphurisation units was approximately 8 Mt in 1996 (based on [47]). Approximately 10% of the gypsum production is directly applied for soil improvement. The bulk of the gypsum (23 Mt) is sold as hemihydrate. Approximately 9 Mt are used for cement production. The bulk of the gypsum (14 Mt hemihydrate) is applied as a building material, e.g. building blocks, wall elements, floor constructions, and as plaster material. In Northern Europe exists a tendency to use dry wall building methods requiring gypsum wallboard, while in Mediterranean countries wet methods requiring gypsum or lime based plaster are used [48]. By-product gypsum is currently promoted as filler in plastics and as coating in paper production.

The production process of gypsum is rather simple. Dihydrate is converted into hemihydrate by heating to 159 °C. Hemihydrate can be moulded. Water is added and the dihydrate solidifies. The energy consumption in gypsum production is mainly related to the conversion of the dihydrate into hemihydrate. Transportation of gypsum will also add to the energy consumption. Table 2.8.1 shows the model input data for gypsum.

Table 2.8.1 *Production process data for gypsum hemihydrate from dihydrate [16]*

	2000	2020	2050
Input			
Low temperature steam [GJ/t]	1.0	1.0	1.0
Transportation [t.km/t]	500	500	500
Gypsum (dihydrate) [t/t]	1.17	1.17	1.17
Output			
Gypsum (hemihydrate) [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	10		10
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	5		5
- Variable [ECU ₁₉₉₄ /t]	10		10
<i>including labour [ECU₁₉₉₄/t]</i>	10		10
Availability factor [-]	0.95		0.95
Life [yr]	25		25

2.9 Kaolin

Kaolin is a white type of clay that was originally used for production of china (hence the name "China clay"). Nowadays, kaolin is mainly used for paper production. Higher paper grades contain up to 30% kaolin. Kaolin is losing market shares to other white mineral fillers like calcium carbonate, talc, and gypsum. Total kaolin production in Western Europe amounted in

1995 to 4 Mt [48]. The UK produces approximately 70% of the total amount.

The production process consists of kaolin clay mining and subsequent upgrading (particle size separation) through flotation. The product is dried. Kaolin prices for paper coating and paper filling are in the range of 75-180 ECU/t [48].

Table 2.9.1 *Production process data for kaolin (based on [49])*

	2000	2020	2050
Input			
Low temperature steam [GJ/t]	0.5	0.5	0.5
Transportation [t.km/t]	500	500	500
Output			
Kaolin [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	10		10
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	5		5
- Variable [ECU ₁₉₉₄ /t]	10		10
<i>including labour [ECU₁₉₉₄/t]</i>	10		10
Availability factor [-]	0.95		0.95
Life [yr]	25		25

2.10 Nitric acid

Nitric acid (HNO₃) production amounted in 1989 to 18.3 Mt. This production corresponds with a 5 Mt ammonia consumption, i.e. 40% of the total ammonia consumption. The Western European application of nitric acid is shown in Table 2.10.1. The table shows that over 85% of the nitric acid is used for production of fertilizers.

Table 2.10.1 *Nitric acid consumption, Western Europe, 1994 [31]*

	[%]
Ammonium nitrate	54.8
UAN Fertilizers	2.3
Other fertilizers	29.4
Adipic acid	5.1
Nitrobenzene	1.7
Toluene diisocyanate	2.3
Other	4.4
Total	100.0

The remaining nitric acid is mainly applied for organic nitrations in the chemical industry. The reaction that converts ammonia into nitric acid is exotherm. As a consequence, the energy consumption for nitric acid

production is relatively small. The production is discussed separately because of the process emission of nitrous oxide (N_2O). This emission represents approximately 1.5 tonnes CO_2 equivalents per tonne HNO_3 (note that use of nitrobenzene for explosives and the use of adipic acid for nylon production are two other major sources of nitrous oxide, [31]). End of pipe technologies can be applied to reduce this emission. Current production technology data are shown in Table 2.10.2. The data in Table 2.10.3 are based on the assumption that more than 90% N_2O emission reduction can be achieved through catalytic conversion. The abatement technologies are still in a development stage. At higher concentrations (like in adipic acid production), nitrous oxide can be destroyed in the oxidation zone of flames. When concentrations of many tens of percents N_2O have to be treated, N_2O itself can be used as oxidiser. N_2O destruction of more than 99.9% can be obtained. In the case of nitric acid production, the latter approach seems most suited. In the calculations, a more modest emission abatement of 90% will be assumed.

Table 2.10.2 *Conventional production process data for nitric acid*
[57, 50, 51]

	2000	2020	2050
Input			
Ammonia [t/t]	0.28	0.28	0.28
Electricity [GJ/t]	0.03	0.03	0.03
Output			
Nitric acid [t]	1	1	1
Steam (25 Bar, 400 °C) [GJ/t]	1.5	2.1	2.1
Process emission [t CO_2 equiv./t]	1.5	1.5	1.5
Investment [ECU ₁₉₉₄ /t cap]	50		50
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	5		5
- Variable [ECU ₁₉₉₄ /t]	5		5
<i>including labour [ECU₁₉₉₄/t]</i>	4		4
Availability factor [-]	0.9		0.9
Life [yr]	30		30

Table 2.10.3 *Conventional production process data for nitric acid including N₂O mitigation technology (based on [51, 52])*

	2000	2020	2050
Input			
Ammonia [t/t]	0.28	0.28	0.28
Electricity [GJ/t]	0.03	0.03	0.03
Natural gas [GJ/t]	0.10	0.10	0.10
Output			
Nitric acid [t]	1	1	1
Steam [GJ/t]	1.5	2.1	2.1
Process emission [t CO ₂ equiv./t]	0.15	0.15	0.15
Investment [ECU ₁₉₉₄ /t cap]	55	55	55
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	5		5
- Variable [ECU ₁₉₉₄ /t]	5		5
<i>including labour [ECU₁₉₉₄/t]</i>	4		4
Availability factor [-]	0.9		0.9
Life [yr]	30		30

2.11 Phosphor and phosphoric acid

Phosphor is used for production of fertilizers, for detergents and for a number of chemical substances (pesticides, pyrotechnical applications etc.). The bulk is used for fertilizers. The use for other applications is decreasing. Especially the use of phosphor for production of detergents decreased in the last decade due to environmental constraints.

All phosphor is produced from phosphate ore that is imported. Two types of phosphor products can be discerned: pure phosphor and phosphoric acid. Phosphor is produced in its elementary form in electric furnaces. Total elementary phosphor production in Western Europe is estimated to be approximately 0.2 Mt (0.5 Mt P₂O₅ equivalents). Total phosphoric acid production is estimated to be 1.5 Mt P₂O₅ equivalents. The net import is estimated to be 2.2 Mt P₂O₅ equivalents.

The bulk of the acid is imported because of the lacking endogenous ore resources of sufficient quality. A phosphoric acid balance for Western Europe is shown in Table 2.11.1.

Table 2.11.1 *Phosphor fertilizer balance for Western Europe 1994/95*
[Mt P₂O₅] [6]

	[Mt/year]	
Supply		
Fertilizer acid supply capability	0.9	
Other phosphate supply	0.6	
Net import (adjusted)	2.2	
Total supply		3.7
Use		
Fertilizer phosphate consumption	3.7	
Total use		3.7

Phosphor is produced from phosphate ore that contains 30-40% P₂O₅. The ore is sintered at 1000°C, coke and gravel is added. The mixture is melted in an electric oven, where a reaction occurs:



The production of phosphor is a very energy intensive process because of high chemical energy requirements of the phosphor production. The lower heating value of phosphor is 27 GJ/t [27]. The bulk of this energy can be recovered when the phosphor is incinerated to P₂O₅. Process data for phosphor production are shown in Table 2.11.2.

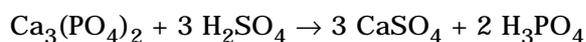
Table 2.11.2 *Production process data for phosphor* [27, 53, 54]

	2000	2020	2050
Input			
Electricity [GJ/t]	45	40	40
Cokes [GJ/t]	30	30	30
Gas [GJ/t]	10	10	10
Phosphor ore [t/t]	4.3	4.3	4.3
Gravel [t]	2.9	2.9	2.9
Transportation [t.km]	10000	10000	10000
Output			
Phosphor [t]	1	1	1
Residual gas [GJ/t]	30	30	30
Investment [ECU ₁₉₉₄ /t cap]	500 ¹		500
O&M cost			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	50		50
- Variable [ECU ₁₉₉₄ /t]	50		50
<i>including labour [ECU₁₉₉₄/t]</i>	25		25
Availability factor [-]	0.9		0.9
Life [yr]	25		25

¹ Estimated on the basis of steel EAF costs and phosphor price 3000 ECU/t

The phosphor is incinerated to produce pure P_2O_5 . Steam of 170 bar is produced as by-product (25 GJ steam/t P). The P_2O_5 is used for production of 90% H_3PO_4 . The advantage of the phosphor route is the higher purity of the phosphoric acid.

Production of phosphoric acid is dominated by the wet process:



Different production routes exist [55]:

- the one-step hemihydrate process (PER 2.3 GJ/t P_2O_5)
- the hemihydrate-dihydrate process (PER 5.3 GJ/t P_2O_5).

The dihydrate process includes an additional recrystallisation step for the gypsum by-product. The goal of recrystallisation is the removal of contaminants from the gypsum. The substitution of the hemihydrate process by the dihydrate process is no realistic option for energy saving because of the gypsum quality problems. Additional environmental measures may further increase the energy consumption.

The gypsum by-product of the phosphate production is not suited for other applications because of a high content of radium, a radioactive substance (see Section 2.8) [56]. MARKAL input data for phosphoric acid production are shown in Table 2.11.3.

Table 2.11.3 *Production process data for phosphoric acid* [55, 57, 58]

	2000	2020	2050
Input			
Electricity [GJ/t]	1.0	1.0	1.0
Low temperature steam [GJ/t]	5.0	4.0	4.0
Phosphor ore [t/t]	3.0	3.0	3.0
Sulphuric acid (99.8%) [t/t]	1.4	1.4	1.4
Transportation [t.km/t]	6000	6000	6000
Output			
Phosphic acid [t P_2O_5 equiv.]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	250		250
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	5		5
- Variable [ECU ₁₉₉₄ /t]	10		10
<i>including labour</i> [ECU ₁₉₉₄ /t]	10		10
Availability factor [-]	0.9		0.9
Life [yr]	25		25

2.12 Potash

Potash is mainly used as fertilizer. The potash balance for Western Europe is shown in Table 2.12.1.

Table 2.12.1 Potash balance for Western Europe 1994/1995 [Mt K₂O] [6]

	[Mt K ₂ O]
Potash fertilizer consumption	4.2
Net export	0.3
Technical potash use	0.4
Total production	4.5

Potash is produced by mining of natural resources. Ores contain between 9 and 30% K₂O equivalents, generally as potassium chloride (KCl). KCl is recovered by deep pit mining and by leaching of the salt sediments. European resources are concentrated in Northern Germany. The mines in France will be depleted in the next decade [59]. German production was 3.28 Mt in 1994 [48]. The potash demand will in the future be met by increased German production and by increased imports from Canada and from Russia. The German mines apply the deep pit mining method [60]. Leaching is more widely applied outside Europe because of the different microstructure of the salt deposits.

KCl is separated from the other substances in the ore by two methods. One method is recrystallisation, the other one is flotation. The separated KCl is dried. The drying process is the main energy consuming step. The energy consumption of the flotation process is considerably lower than the energy consumption of the recrystallisation process [61].

An interesting feature of the potash mining is the occurrence of CO₂ and CH₄ as gaseous by-products of the mining process. The emission is in the range of 8-25 m³/t ore, i.e. 30-100 m³/t K₂O [60]. The gas composition depends on the local conditions. Assuming 100% CH₄ (a high estimate), the methane emission represents a maximum of 6.4 Mt CO₂ equivalents per year¹. A more realistic specific CH₄ emission results in an estimate of 0.1-1 Mt CO₂ equivalents per year.

The data in Table 2.12.2 represent the production of KCl, based on the recrystallisation process. Energy consumption data for the recrystallisation process according to [61] are considerably lower. The difference may be explained by system boundaries in the latter reference where the mining process has not been considered. Given the energy consumption data for sodium chloride production (see Section 2.16), it is likely that the value in Table 2.12.2 is a better estimate. The flotation process represents an attractive production alternative with a lower energy consumption. However, this process cannot be applied for all ore types. Because of the limited importance of potash production, the flotation process is not modeled as improvement option.

¹ $4.5 \times 100 \times 1000 / 22.4 \times 1.6 \times 10^{-6} \times 20 = 6.4$, 4.5 Mt K₂O/year, 100 m³/t, 1000/22.4 × 1.6 × 10⁻⁶ t CH₄/t, Global Warming Potential CH₄ 20

Table 2.12.2 *Production process data for KCl (German situation, expressed per tonne K₂O) [57]*

	2000	2020	2050
Input			
Low temperature heat [GJ/t]	7.0	5.0	4.0
Diesel [GJ/t]	0.4	0.4	0.4
Electricity [GJ/t]	0.18	0.15	0.15
Transportation [t.km/t]	750	750	750
Output			
K ₂ O [t]		1	11
Investment [ECU ₁₉₉₄ /t cap]	50		50
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	5		5
- Variable [ECU ₁₉₉₄ /t]	5		5
<i>including labour [ECU₁₉₉₄/t]</i>	5		5
Availability factor [-]	0.95		0.95
Life [yr]	25		25

2.13 Quicklime

Lime is produced by decarbonisation of limestone (calcium carbonate, CaCO₃) to calcium oxide (CaO). CO₂ is generated as by-product. Calcium oxide (quicklime) is usually hydrated to form calcium hydroxide Ca(OH)₂. Calcium hydroxide is commonly called hydrated or slaked lime, and sometimes merely lime - which can be confusing as powdered limestone is often referred to in the same way.

Decarbonisation is achieved by heating limestone to temperatures above 1000 °C. 1 tonne of pure CaCO₃ yields 560 kg CaO and 440 kg CO₂. The 560 kg CaO can be converted into 740 kg Ca(OH)₂. In case dolomite (CaCO₃.MgCO₃) is used instead of limestone, the product mix contains MgO. The minimum energy requirement for CaO production is 3.20 GJ/t. For dolomite, the minimum energy requirement is 3.02 MJ/t. As a consequence, lime production from dolomite has a slightly lower minimum energy requirement [62]. These differences are not considered in the following analysis.

The energy efficiency of quicklime production is in the range of 40-50%. One should keep in mind that comparison of energy efficiencies of different producers may result in misleading conclusions, because the energy efficiency is to some extent determined by the product quality. Table 2.13.1 shows data for quicklime production in Western Germany in 1986. Rapid changes between technologies and increased energy efficiency have not occurred in the last decade, so the data are considered an adequate representation of the current situation. Because the bulk of the CO₂ emissions is caused by the limestone decomposition, the actual energy consumption is of secondary importance. Table 2.13.2 shows the model input parameters for quicklime production.

Table 2.13.1 *Oven types for quicklime production and their energy efficiency, 1986 [63]*

	Fraction [%]	Fuel requirements [GJ/t quicklime]
Shaft kiln	30	4.2
Counter/occurrent kiln	20	3.7
Ring shaft kiln	25	4.1
Rotary kiln		
- including preheating	17	5.5
- excluding preheating	8	6.9

Data regarding the slaked lime use in Western Europe have not been found. In Western Germany, the sales of quicklime amounted to 6.4 Mt in 1989 [64]. According to [1] the total Western European production amounted to 22 Mt in 1992. This implies an inorganic CO₂-emission of 20 Mt, approximately 0.6% of the total Western European CO₂ emission. However, a significant part of this CO₂ is again fixed when the slaked lime is subjected to air. Only in applications where lime is converted into glass-like materials (like in the steel and glass production processes) or into gypsum (in desulphurisation processes), this CO₂ fixation will not occur.

Quicklime is used for steel production, flue gas desulphurisation, soda production, in fertilizer production, in the glass industry, for sugar production, sand-limestone production, water treatment, and in the building industry. Data regarding the gypsum production in desulphurisation units (8 Mt per year [47]) indicate a quicklime consumption for desulphurisation of 3 Mt per year. Steel converter slag contains approximately 40% CaO. The production is approximately 0.1 t slag/t steel [65]. Total converter steel production amounted to 90 Mt, so total quicklime consumption amounted to 3.6 Mt². The production of sand-limestone requires approximately 65 kg CaO per tonne sand-limestone, resulting in a consumption of 2.6 Mt CaO per year. Sugar production in Western Europe amounted to 16 Mt in 1992/93 [66]. Lime requirements are 0.2 t/t sugar [67]. The total lime consumption for sugar production is 3.2 Mt per year. It is not clear to what extent this consumption is included in the lime production statistics, since most of this lime is produced at sugar production plants. It is assumed that approximately 5 Mt quicklime is used in the building industry, the remainder is predominantly used for fertilizer production. Based on this analysis, it is assumed that 50% of the quicklime recombines into CaCO₃. As a consequence, the estimated emission of CO₂ from inorganic origin is 10 Mt per year.

² $90 \times 0.1 \times 0.4 = 3.6$, where: 90 Mt steel production per year, 0.1 t slag/t steel, 0.4 t quicklime/t slag

Table 2.13.2 *Production process data for quicklime* [68]

	2000	2020	2050
Input			
Gas [GJ/t]	5.0	4.0	4.0
Output			
Quicklime [t]	1	1	1
Inorganic CO ₂ [t/t]	0.75	0.75	0.75
Investment [ECU ₁₉₉₄ /t cap]	300		300
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	10		10
- Variable [ECU ₁₉₉₄ /t]	5		5
<i>including labour</i> [ECU ₁₉₉₄ /t]	5		5
Availability factor [-]	0.95		0.95
Life [yr]	25		25

¹ Own estimates, based on data for cement kilns

2.14 Sand-limestone

Total Western European sand-limestone production is estimated to be approximately 40 Mt per year. Sand-lime bricks are produced by mixing CaO, sand and water in a ratio 0.065:0.935:0.053. The mixture is formed and heated with steam to a temperature of 180-205 °C at a pressure of 1.2-1.7 MPa for 4.5 hours. Next, the stones are sawn to their proper size. The input data for the production process are shown in Table 2.14.1.

Table 2.14.1 *Production process data for sand-limestone* [69]

	2000	2020	2050
Input			
Low temperature steam [GJ/t]	0.63		0.63
Heavy distillates [GJ/t]	0.19		0.19
Quicklime [t/t]	0.065		0.065
Sand [t/t]	0.935		0.935
Transportation [t.km/t]	200		200
Output			
Sand-limestone [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	100		100
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	5		5
- Variable [ECU ₁₉₉₄ /t]	5		5
<i>including labour</i> [ECU ₁₉₉₄ /t]	5		5
Availability factor [-]	0.95		0.95
Life [yr]	25		25

2.15 Sodium carbonate (soda)

Sodium carbonate or soda (Na_2CO_3) is a product that is widely used for detergent production (approx. 2 Mt pa), paper production (approx. 1 Mt pa), glass production (2.5 Mt pa), and for neutralisation of acidic compounds (eg in flue gas desulphurisation). Total production in Western Europe amounted in 1995 to 6.0 Mt [48]. The price of soda ranges from 125 to 200 ECU/t [27].

Sodium carbonate production is based on the Solvay process:



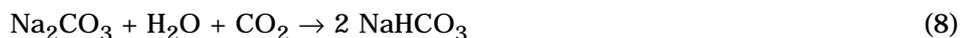
The process can be split into several stages:



An important fraction of the energy consumption is accounted for by the quicklime production (process 6, see Section 2.13). The theoretical minimum quicklime requirement is 0.53 t/t soda. In practice, the quicklime consumption is in the range of 0.60 t/t soda. The inorganic CO_2 emission of quicklime production does in this case not occur due to subsequent CO_2 fixation.

The process steps 2-5 and 7 require less energy. The calcination process (5) and the ammonia distillation require steam. Total energy consumption is approximately 12.2 GJ heat and 0.83 GJ electricity per tonne (including quicklime production [70]). The heat consumption can be split into 3 GJ high temperature heat for quicklime production and 9.2 GJ low temperature heat for the other processes (processes 2-5 and 7). Improved steam production (CHP), heatpumps and improved distillation columns pose a significant energy saving potential.

Soda is a basic substance that binds CO_2 :



A net CO_2 storage results. This storage is not considered in current CO_2 emission statistics (and is for this reason not considered in the MARKAL calculations). The basic properties of soda can also be applied to remove SO_2 from flue gases.

The production of natural soda is an improvement option compared to synthetic soda production. In the USA and in Africa, significant natural soda resources occur in former lakes and in underground sediments. The energy consumption for production is only 8.4 GJ/t [71]. However, transportation of this natural product to Western Europe implies high transportation costs (range 50-100 ECU/t). Moreover, the transportation

energy is in the range of 2-5 GJ/t, so the net saving is rather limited. This option is not further elaborated.

An alternative existing production process for soda uses caustic soda and CO₂:



This reaction is followed by reaction (5). The product has a somewhat higher density than the soda that is produced in the Solvay process (hence the names heavy and light soda). Both are however sold in the same markets. The energy consumption for glass production is slightly affected by the soda type that is used (see Table 2.7.2). This effect is not considered in the model calculations. Model input data for light and heavy soda production are shown in Table 2.15.1 and 2.15.2.

Table 2.15.1 *Production process data for light soda (Solvay process)*
(based on [27])

	2000	2020	2050
Input			
Low temperature steam [GJ/t]	9.2	7.0	6.0
Quicklime [t/t]	0.60		0.60
Salt brine [t/t]	1.10		1.10
Transportation [t.km/t]	200		200
Output			
Soda [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	100		100
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	5		5
- Variable [ECU ₁₉₉₄ /t]	5		5
<i>including labour [ECU₁₉₉₄/t]</i>	5		5
Availability factor [-]	0.95		0.95
Life [yr]	25		25

Table 2.15.2 *Production process data for heavy soda [27]*

	2000	2020	2050
Input			
Low temperature steam [GJ/t]	3.0	2.5	2.0
NaOH [t/t]	0.75		0.75
Transportation [t.km/t]	200		200
CO ₂ [t/t]	0.42		0.42
Output			
Soda [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	50		50
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	5		5
- Variable [ECU ₁₉₉₄ /t]	5		5
<i>including labour [ECU₁₉₉₄/t]</i>	5		5
Availability factor [-]	0.95		0.95
Life [yr]	25		25

2.16 Sodium chloride

Sodium chloride is produced by mining of fossil salt resources or by evaporation of sea water. The latter process is mainly applied in Southern European countries. The mining of natural resources can be split into deep pit mining (rock salt production) and leaching of salt caverns (solution mining). Solution mining is used for two product types: brine (a sodium chloride solution) and vacuum salt (a solid product).

Total Western European salt production amounted to 45 Mt in 1992 [1]. Brine production amounted in the EU to 15.2 Mt in 1993. The brine is mainly used for chlorine production (approx. 5 Mt pa) and for soda production (approx. 8 Mt pa). This production is not considered in the following analysis, because the main energy consumption occurs beyond the brine production. Total crystallised salt production capacity amounted to 40 Mt for the EU in 1993 [72]. Production of crystallised salt amounted to 19.3 Mt (50% capacity utilisation); apparent consumption amounted to 16.9 Mt. Rock salt accounted for 39.3% of the salt production (decreasing), sea salt amounted to 17.8% and vacuum salt amounted to 42.9%. Crystallised salt consumption can be split into chloralkalis with 48%; miscellaneous industries with 19%; food grade salt with 13%; and road salt with a 20% share of the total consumption. The energy consumption for evaporation of sea water in natural ponds is negligible. The energy consumption for rock salt production consists mainly of the energy for mining. No data have been encountered for the rock salt mining process; the production is modeled like salt leaching. The mining process, based on salt leaching, requires negligible amounts of energy. The bulk of the energy consumption is related to the brine concentration. Steam is applied for evaporation of the salt brine to produce vacuum salt.

In the Dutch situation, the salt leaching and brine evaporation consumes 4 GJ heat and 0.1 GJ electricity per tonne salt (1986 figures) [70]. These data have been extrapolated in Table 2.16.1, assuming a gradually increasing energy efficiency.

Table 2.16.1 *Production process data for salt (from brine) (based on [70])*

	2000	2020	2050
Input			
Low temperature steam [GJ/t]	4.0	3.0	3.0
Electricity [GJ/t]	0.1		0.1
Output			
NaCl [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	50		50
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	5		5
- Variable [ECU ₁₉₉₄ /t]	5		5
<i>including labour [ECU₁₉₉₄/t]</i>	5		5
Availability factor [-]	0.95		0.95
Life [yr]	25		25

The production of sea salt constitutes an option with lower energy consumption. The transportation costs for this salt from Southern Europe to Central and Northern Europe are however an obstacle for a major substitution in the current situation. For this reason, the option will be modeled as an alternative production option with an increased demand for transportation (Table 2.16.2).

Table 2.16.2 *Production process data for salt (from sea water), own estimates*

	2000	2020	2050
Input			
Diesel [GJ/t]	0.1	0.1	0.1
Transportation [t.km/t]	2000		2000
Output			
NaCl [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	5		5
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	5		5
- Variable [ECU ₁₉₉₄ /t]	5		5
<i>including labour [ECU₁₉₉₄/t]</i>	5		5
Availability factor [-]	0.95		0.95
Life [yr]	25		25

2.17 Ornamental stone

Total ornamental stone production in Western Europe amounted in 1992 to 18.0 Mt. The net export of finished stone products amounted in 1992 to 1.3 Mt [73]. 60% of the production is concentrated in Italy and Spain. The remaining production is spread over the other European countries with suitable rock resources. The processed materials are mainly calcareous rocks (marble, travertine, etc.) or siliceous rocks (granite, sandstone, porphyry, slate, etc.). Granites are mainly used for outdoor applications because of their weather resistance; marbles are principally used for indoor applications. Average labour costs, based on 190,000 employees, are in the range of 250 ECU/t [73].

The energy consumption is mainly accounted for by the stone cutting and the transportation process. The energy consumption for stone cutting will depend on the thickness of the slab. It is assumed that the consumption is 1 GJ electricity per tonne material. The main transportation is from Southern Europe to Central European countries. It is assumed that the average transportation distance is 1500 km.

Table 2.17.1 *Production process data for ornamental stone*

	2000	2020	2050
Input			
Electricity [GJ/t]	1	1	1
Transportation [t.km/t]	1500	1500	1500
Output			
Ornamental stone [t]	1	1	1
O&M cost¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	50		50
- Variable [ECU ₁₉₉₄ /t]	250		250
<i>including labour [ECU₁₉₉₄/t]</i>	250		250
Availability factor [-]	0.95		0.95

2.18 Sulphur and sulphuric acid

Sulphur is produced from mining and as by-product from metal ore processing (copper, zinc) and energy conversion processes (oil and coal in refineries and power plants etc.). Waste sulphuric acid originates mainly from refineries and the petrochemical industry. Mining is not applied in Europe; data for sulphur sources in the European Union are shown in Table 2.18.1. Total sulphur production amounted to 6.1 Mt in 1992 in Western Europe [1]. Total sulphuric acid production amounted in Western Europe in 1991 to 14.3 Mt monohydrate sulphuric acid (equivalent to 3.9 Mt sulphur, excluding Belgium) [74]. 65% of the total sulphur production is converted into sulphuric acid. This production process (sulphuric acid from sulphur) is modeled separately.

Table 2.18.1: Sulphur sources in Western Europe (EU data, 1990) [57]

	[%]
Claus process (desulphurisation)	50
Other sulphides (metal ores)	30
Waste acid recycling	20
Total	100

Approximately 28% of all sulphur is used for fertilizers, the remaining fraction is used for other applications like oil refining (approx. 1 Mt), phosphate production (approx. 0.6 Mt), titanium dioxide production, chemical pulp production, and the production of plastics and other synthetic organic materials [31].

Data for the sulphuric acid production from sulphur are shown in Table 2.18.2. The reaction is exotherm, steam can be recovered.

Table 2.18.2 Production process data for sulphuric acid from sulphur (based on [57])

	2000	2020	2050
Input			
Sulphur [t/t]	0.28	0.28	0.28
Transportation [t.km/t]	200	200	200
Output			
Sulphuric acid (H ₂ SO ₄ .H ₂ O) [t]	1	1	1
Steam [GJ/t]	3.3	3.3	3.3
Investment [ECU ₁₉₉₄ /t cap]	50		50
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	5		5
- Variable [ECU ₁₉₉₄ /t]	5		5
<i>including labour [ECU₁₉₉₄/t]</i>	5		5
Availability factor [-]	0.95		0.95
Life [yr]	25		25

2.19 Titanium dioxide

Titanium dioxide is a white coloured pigment. It accounts for 90% of the titanium demand. It is used for coatings (56%), paper (13%), plastics (20%) and other applications (11%). The price is approximately 1500 ECU/t. Total Western European titanium dioxide production capacity amounted to 1.3 Mt in 1996 [75]. Of this production capacity, 0.5 Mt was based on the chlorate process, 0.8 Mt was based on the sulphate process. The chlorate process is more recent and less harmful for the environment. The chlorate process will gradually replace the sulphate process. Because of this autonomous substitution, only the chlorate process is modeled.

The feedstock for titanium dioxide production is rutile. Rutile is mined or produced from ore (ilmenite). Western European ilmenite production capacity amounted in 1995 to 0.85 Mt [48].

The production process data in Table 2.19.1 do not include the production of rutile from ilmenite. The purification of titanium dioxide is also not considered. In [76], it is indicated that the energy consumption of both process steps is of similar importance as the actual production, resulting in a total primary energy consumption of 70 GJ/t titanium dioxide. In [49], it is indicated that the production of rutile requires 2.6 GJe/t and 0.3 GJ heat/t, while the transportation energy for the ilmenite ore adds another 4.0 GJ/t. Combining these data with the data in Table 2.18.1 results in a primary energy requirement of approximately 50 GJ/t. No separate purification is indicated. This lower figure will be used in the calculations.

Table 2.19.1 *Production process data for titanium dioxide, chlorate process (based on [49, 77])*

	2000	2020	2050
Input			
Titanium ore (rutile) [t/t]	1.1	1.1	1.1
Petroleum coke [GJ/t]	22.0	22.0	22.0
Electricity [GJ/t]	7.2	7.2	7.2
Low temperature heat [GJ/t]	2.0	2.0	2.0
Oxygen [t/t]	0.55	0.55	0.55
Transportation [t.km/t]	20000	20000	20000
Output			
Titanium dioxide [t]	1	1	1
Investment [ECU ₁₉₉₄ /t cap]	3500	3500	3500
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	200	200	200
- Variable [ECU ₁₉₉₄ /t] ¹	550	550	550
<i>including labour [ECU₁₉₉₄/t]</i>	550	550	550
Availability factor [-]	0.95	0.95	0.95
Life [yr]	25	25	25

¹ Rutile price 500 ECU/t [78]

3. OPTIONS FOR INCREASED MATERIALS EFFICIENCY

The potential for increased efficiency of materials *production* has been discussed in Chapter 2. The potential for increased efficiency in materials *consumption* is discussed in this chapter. Figure 3.1 shows the improvement strategies in the materials/product life cycle that will be discussed in this chapter. The other strategies involving inorganic and ceramic materials will be discussed in MATTER-reports for the product groups buildings and packaging.

The following discussion is organised per material. The analysis in this chapter includes options that enhance physical characteristics of the material (cement, concrete, glass) and that reduce the weight per unit of (semi-finished) product (bricks, concrete, sand-limestone). For fertilizers, increased efficiency of use is also discussed in this volume, because the application of such auxiliary materials is not discussed in a separate volume (Section 3.1).

Light weight bricks are discussed in Section 3.2. High strength concrete will be discussed in Section 3.4. Light weight glass packaging is discussed in Section 3.5. Reduced size of sand-limestones is discussed in Section 3.6. For other materials in the group of inorganic and ceramic materials, materials quality improvement seems to offer less potential.

Inorganic and ceramic materials show low recycling rates compared to metals and synthetic organic materials. The only exception is glass. The lack of high grade recycling is caused by the nature of the materials, by their application, and by the low price of these materials.

One must add that recycling of ceramic materials (substitution of natural gravel and sand) is widely applied, but this practice results in a significant downgrading of product quality, with little or no CO₂ savings. Regarding inorganic materials, fertilizers are partially recycled through fertilisation with manure.

Concrete can be recycled into its constituents gravel, sand, and cement (Section 3.3). Glass recycling can be increased through higher cullet recovery rates (Section 3.5). Clay bricks, sand-limestone bricks and prefab concrete elements can be re-used. Re-use is discussed in Section 3.7.

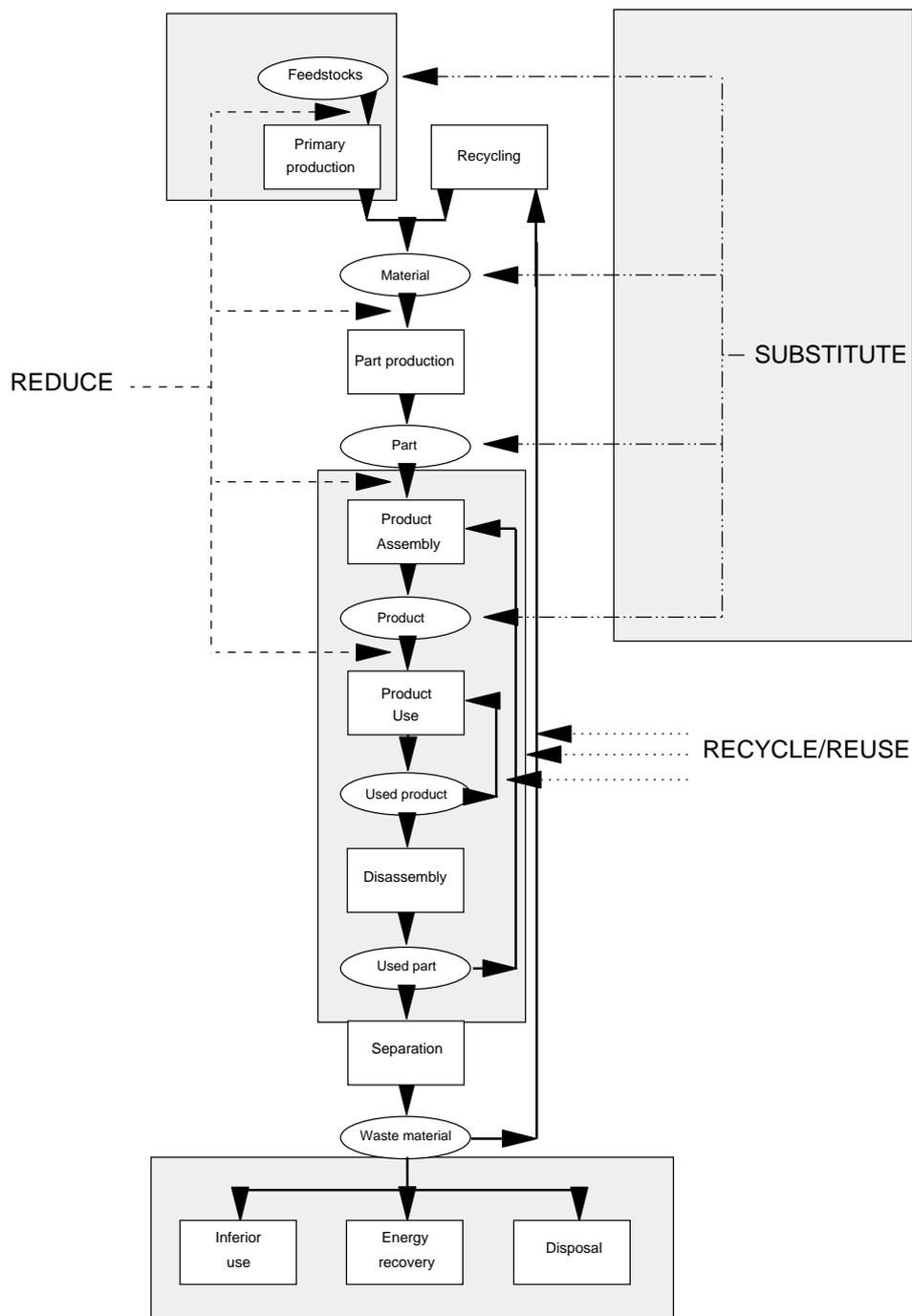


Figure 3.1 Options that are discussed in this chapter in relation to the life cycle (options in the shaded categories are not discussed)

3.1 Fertilizers

In [79], the following improvements in the fertiliser use are analysed:

- application of recommended fertilisation levels / increasing the use of manure
- improvement of the fertilizer spreader maintenance
- improvement of the fertilizer spreader geometry
- introduction of a fertilizer free zone at the borders of the fields
- application of row fertilization
- introduction of fertilizer need analysis
- introduction of nitrogen catch crops
- substitution of urea and CAN with calcium nitrate
- improvement of the fertilizer granule characteristics
- introduction of coated fertilizer granules.

These improvements apply to nitrogen fertilizers, but a number of these measures can also be applied for phosphor, potassium, and calcium fertilizers. The maximum saving potential of these measures is for the Netherlands in the range of 28 to 51%, depending on the soil type. The bulk of the savings (60-70%) is achieved through the application of recommended fertilisation levels/increased use of manure and through the improved fertilizer spreader maintenance.

The application of recommended fertilisation levels/increased use of manure has a relatively lower potential on the European level than on the Dutch level. The level of fertilizer consumption per hectare is still significantly lower (in 1988 European average 67 kg/ha vs. 250 kg/ha in the Netherlands) and the availability of manure is limited (contrary to the Netherlands with a huge manure surplus due to high cattle densities, see Table 3.1.1). In Europe (including Eastern Europe), the ratio of synthetic and manure N use for fertilisation is approximately 1:1 [80]. Separate data for Western Europe have not been encountered.

Table 3.1.1 *Cattle densities in the Netherlands and in Western Europe, expressed in number of animals per hectare cropland, situation 1990/1992 (based on [81])*

	Netherlands [1/ha]	Western Europe [1/ha]
Cattle	5.4	0.9
Sheep and Goats	2.1	1.3
Pigs	15.1	1.2
Chickens	113.0	9.2

Another substitution option that receives much attention is the development of nitrogen fixation techniques using 'engineered' bacteria or other organisms, called biological nitrogen fixation (BNF). Nitrogen from the atmosphere is directly converted into ammonia. Leguminous crops are able to extract nitrogen from the atmosphere because of their symbiosis with *Rhizobium*. This crop type can also be grown together with other crops in order to supply nitrogen [31]. This strategy was applied before introduction

of synthetic nitrogen fertilizers. It is still applied in many developing countries. A return to this strategy requires greatly enhanced nitrogen fixation systems. Current fertilizer replacement values for legumes when preceding a grain crop vary from 0 to 110 kg N/ha [82].

Nitrogen fixation also reduces the emissions of the greenhouse gas N_2O . Both synthetic and manure nitrogen fertilizers are partially converted into N_2O . Limited data indicate that organic nitrogen sources such as animal manures induce larger N_2O emissions per unit of N added to the soil than does mineral N [83]. It is unclear to what extent nitrogen from biological N-fixing crops is converted into N_2O . For Dutch grassland, the use of fertilisers increases the N_2O loss by 10 to 30 kg N/h per year. The wide range is accounted for by the soil type, groundwater level, and by the land use [84]. These emissions represent a CO_2 -equivalent of 5 to 15 t/ha per year. For other types of agricultural land use, the relevance is lower because N_2O emission levels are lower. The IPCC suggests a default value of 1% nitrogen fertilizer conversion into N_2O [85]. The CO_2 emission for nitrogen fertiliser production (assuming 400 kg N-fertilizer per hectare) is approximately 0.7 t/ha.year. A 1% conversion into N_2O suggests an emission of 2 t CO_2 -equivalents per year. The comparison shows the relevance of the reduced N_2O emission. If different fertiliser types result in different nitrous oxide emissions, this effect will dominate potential CO_2 emission reductions. However, data have not been encountered. Finally it is important to stress that biological nitrogen fixation reduces nitrogen leaching, another important environmental problem.

The use of nitrogen fertilisers is an inefficient process [31, 86]. Approximately 25% of the nitrogen in fertilizer is not consumed by plants: 10% is lost through run-off, 15% is lost through conversion by denitrifying bacteria. Not all nitrogen that is actually consumed by plants is useful: 30% of the total nitrogen fertilizers return to the soil in organic root, stem or leaf material that gradually decays. This nitrogen is partially lost for fertilisation. As a consequence of these losses, the actual end-use efficiency of nitrogen fertilizer is approximately 50%. However, there seems to be little opportunity to increase this efficiency.

In conclusion, the efficiency improvement potential for nitrogen fertilizer use for Western Europe is estimated to be in the range of 10-25% [87]. On the long run (>25 years), nitrogen fixation by micro-organisms poses an attractive option with a potential of replacing the whole nitrogen fertilizer consumption, i.e. 80% of the total ammonia production (see Table 2.1.1). Application is however still far away, and the success of the research is uncertain. However if such techniques can be developed successfully, they offer potentially lower fertilisation costs than synthetic nitrogen fertilisers. Significant secondary benefits due to reduced N_2O emissions and reduced nitrogen leaching can also be expected.

For other fertilizers, the saving potential is estimated to be in the same range like for nitrogen fertilizers. Fixation techniques cannot be applied because the fertilizer constituents do not occur in the atmosphere. Recycling of ashes from biomass combustion processes and recycling of sewage sludge pose to a certain extent alternatives from a nutrient

recycling point of view. These options have not been analysed in detail because the content of toxic compounds will result in high upgrading costs and will probably also result in a high energy consumption.

3.2 Bricks

The size of bricks varies within Western Europe. The weight load capacity of bricks is generally no limiting factor. As a consequence, perforated bricks are introduced on a large scale in order to replace conventional bricks. They have been used in the Southern European countries for a long time because these countries tended to use large clay blocks. These blocks cannot be made without perforations. Perforations of up to 60% are not uncommon in these blocks.

Perforations reduce the weight per brick. The weight reduction results in energy savings as the energy consumption per weight unit remains the same. This measure is not new in Northern and Central European countries, but is gradually introduced [88]. A general increase in the level of perforations up to 40% does not require any significant change in production methods, or any restrictive alterations in the design codes [89]. However, the mortar requirements increase if the perforation volume is increased in a vertical direction.

Table 3.2.1 *Current clay product consumption and optimised clay product consumption, Western Europe [90]*

Type	Density [t/m ³]	Current consumption [Mt/year]	Optimised consumption [Mt/year]
Facing bricks (<15% voids)	1.6	18	18
Solid bricks (<15% voids)	1.6	4	0
Perforated bricks (15-40% voids)	1.05	26	10
Hollow bricks (>40 % voids)	0.9	13	20
Pavement bricks (<15% voids)	1.6	5	5
Total		66	53

Table 3.2.1 indicates the saving potential for bricks with decreased density. The ultimate saving potential is estimated to be around 13 Mt bricks, representing a saving potential of approximately 20%. This figure should be considered as an upper estimate. UK estimates indicate a saving potential of 3% [13]. The costs for this transformation are negligible (there may even be cost reductions due to reduced clay consumption and due to energy savings). The mortar requirements increase by 40%. Mortar contains 24% cement. Assuming 1 cm of mortar per 8 cm brick height, the increased mortar use implies an increased cement use of 0.02 t cement/t bricks³.

³ $0.4 \times 0.24 \times 1/8 \times 2.6/1.6 = 0.02$, where 0.4 = 40% increased mortar requirement; 0.24 = 24% cement content of mortar; 1/8 = mortar volume per brick volume; 2.6/1.6 = mortar density compared to brick density

The CO₂ intensity of cement is significantly higher than for bricks (0.9 vs. 0.3 t CO₂/t). As a consequence, the CO₂ savings of light weight bricks are limited (provided Portland cement is used for mortar production). The effect of increased in cement use is less significant in case larger bricks are used. The model input data are shown in Table 3.2.2. These data are based on the input data for bricks in Table 2.2.1.

Table 3.2.2 *Substitution of average bricks by light weight bricks*

	2000	2020	2050
Input			
Natural gas [GJ/t]		1.9	1.9
Fuel oil [GJ/t]		0.2	0.1
Electricity [GJ/t]		0.05	0.05
Cement [t/t]		0.03	0.03
Output			
Light weight bricks [t]		1.2	1.2
Investment [ECU ₁₉₉₄ /t cap]		125	125
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]		7	7
- Variable [ECU ₁₉₉₄ /t]		15	15
<i>including labour [ECU₁₉₉₄/t]</i>		15	15
Availability factor [-]		0.95	0.95
Life [yr]		20	20
Boundup		20	50

3.3 Cement

Concrete can be split into its constituents cement, gravel and sand. This process is based on the dissociation of the bonding with microwaves and consecutive separation of the different fractions. The process is applied for stripping of concrete walls, where the outside 1-2 cm is removed. The concrete can be recycled into its constituting materials through heating to a temperature of 600 °C. However this process has only been tested on a laboratory scale. The main CO₂ emission reduction is achieved because the inorganic CO₂ emissions can be avoided. The cement yield is limited by the carbonation of cement that has occurred during the period the cement is used. The potential of the process is further limited by the availability of waste concrete (currently one quarter of the concrete production). As a consequence, the potential for emission reduction is limited to less than 15% of the total CO₂ emission for cement clinker production.

No process data have been encountered because of the laboratory scale tests. The data in Table 3.3.1 are based on own estimates.

Table 3.3.1 *Recycling process for cement from waste concrete, own estimates*

	2000	2020	2050
Input			
Concrete [t]	-	1	1
Electricity [GJ/t]	-	0.5	0.5
Natural gas [GJ/t]	-	2.0	2.0
Transportation [t.km/t]	-	100	100
Output			
Cement [t/t]	-	0.05	0.05
Gravel and sand [t/t]	-	0.75	0.75
Disposable waste [t/t]	-	0.10	0.10
Water (evaporation) [t/t]	-	0.10	0.10
Investment [ECU ₁₉₉₄ /t cap]	-	-	-
O&M cost ¹			
- Fixed [ECU ₁₉₉₄ /t cap.yr]	-	-	-
- Variable [ECU ₁₉₉₄ /t] ¹	-	250	250
<i>including labour</i> [ECU ₁₉₉₄ /t]	-	100	100
Availability factor [-]	-	0.95	0.95
Life [yr]	-	25	25

3.4 Concrete

High strength concrete (HSC)

Strength of concrete can be improved considerably. As a consequence, the weight of concrete building elements can be significantly reduced. High-strength concrete can be produced by using well defined and chosen ingredients and addition of strength improving ingredients. This includes:

1. Use of cement with a particle size distribution (PSD) between 2 and 20 μm (strength increase after 28 days of 9 to 22%, compared to conventional cement)
2. Use of coarse aggregates with other shape (more cubical, smaller maximum size) and other chemical composition (limestone, diabase) (strength increase after 28 days of 10 to 30%, compared to conventional cement)
3. Chemical admixtures (water reducers, 0.3 mass%) and superplasticizers (lignosulphonates, 0.2 mass%). The concrete strength can be tripled when both are combined.
4. Addition of mineral admixtures like flyash (60 kg/m^3) or silica fume (30-40 kg/m^3). Fly ash added alone to chemical admixtures can help achieve strengths up to about 83 to 100 MPa (strength increase of 200 to 300%). For higher strengths, additional silica fume must be used to achieve a cement paste that is as uniform and dense as required.

After 28 days, conventional concrete possesses a design compression strength of 28-30 kN/m^2 (design strength: strength to be used for design calculations, the actual strength is generally higher). The strength can be

increased to up to 69-125 kN/m² for high strength concrete [91, 92]. Higher values up to 690 kN/m² are even reported for laboratory tests.

The increase in strength is based on three effects [93, 94]:

1. A reduced water/cement ratio increases the strength (see Figure 3.3.1).
2. If smaller size particles are added, the layer between the sand and gravel particles and the cement particles is less porous. This results in larger bonding surfaces and reduced permeability of the bonding layer.
3. Fly-ash and silica fume possess pozzolanic properties, thus increasing the bonding strength.

Reduction of the water/cement ratio is not possible in conventional cement, because the concrete cannot be handled anymore. Small particles, chemical admixtures and superplasticizers increase the fluid character of the liquid concrete, so the water content (and the W/C ratio) can be reduced.

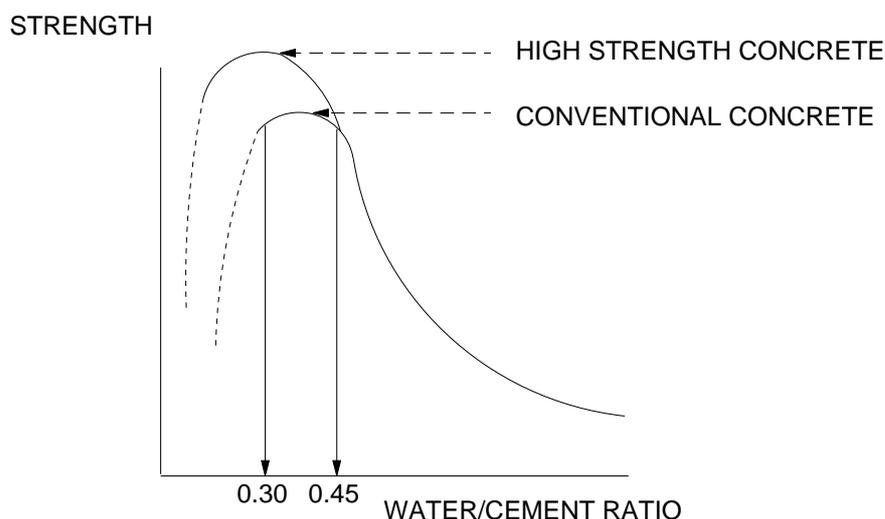


Figure 3.3.1 Concrete strength as function of the water/cement ratio

The cement content of high strength concrete is higher than for conventional concrete: 350-400 kg/m³ vs. 280-350 kg/m³. The reason is the easier handling of concrete at higher cement contents. Current research, for example at the technical university in Delft, aims for the reduction of the cement content for high-strength concrete to similar values as for conventional concrete.

For reinforced columns, a concrete column with a strength three times as high as conventional concrete (B105 instead of B45) only 2% steel reinforcement is required instead of 4% (a reduction by 50%), while the amount of concrete is reduced by 33% [95]. For bending loads, the gain is much smaller. The saving potential depends on the potential to adjust the shape of the beams. Calculations show that a prestressed concrete girder made out of concrete with twice the compressive strength reduces the weight by 30%, while the reinforcement is reduced by 25% [95]. The improved

materials quality does also allow more flexible building design. For floor elements, the use of high strength concrete results in an increased effective span by 40%. As less columns are required, the building space can be used more freely, which allows more flexible use of the building. This may significantly increase the economic life span of buildings. However such effects cannot be easily quantified, and will not be considered in the analysis. Currently a major problem regarding large scale application of high strength concrete is the cracking behaviour. Since the material is more brittle, larger cracks occur than in conventional concrete. These cracks do not diminish the strength of the material, but are not appreciated by the residents.

High strength concrete is already widely applied in skyscrapers, where it can significantly contribute to smaller columns and thus to larger floor areas. It is not used for the skyscraper floors, because its additional cost is not (yet) balanced by the cost savings for smaller columns. Its price is higher than for conventional concrete (a three times higher price per unit of volume, for a concrete with a three times higher strength for axial loads [96]). The most cost effective solution for high-rise buildings is ultra high strength concrete (>150 MPa) with only 1-2% reinforcement. For lower buildings, axial loads are not always the limiting variable. Noise insulation, fire insulation, or limitations of the production process may limit the potential for increased materials efficiency.

The costs of high strength concrete elements are determined by the costs of the additives and the cost savings for the sand and gravel use. Moreover, secondary effects are also beneficial (less foundation requirements, increased floor space etc.). Costs of the fillers are however high: silica fume 700 ECU/t and finely ground fly-ash 400 ECU/t [97]. Assuming a 10% content of fillers results in additional costs of 40-70 ECU/t. However, the costs of reinforcements are reduced by 25-50 ECU/t and the costs of sand and gravel are reduced by 10-20 ECU/t. As a consequence, the additional costs are small or negligible.

In the modelling study, it is assumed that the increased cement content of high strength concrete can be reduced to similar values as for conventional concrete. High strength concrete can on average contribute a factor 2 to the weight reduction of concrete columns with axial loads in a design with a constant amount of reinforcements, while the product part price increase (including material costs) is 25%. This application constitutes 25% of all concrete applications. For floor elements, it is assumed that the weight saving potential is 20%, while the material price increase is 25%. This application constitutes another 25% of all applications. These higher costs should be considered as upper estimates. The increased strength will result in net cost savings due to reduced transportation costs and reduced processing costs.

The potential contribution to CO₂ emission reduction of the increased use of high strength concrete is approximately 30 Mt CO₂⁴. Assuming additional costs in the range of 0-25 ECU/t conventional concrete and savings of 0.07 t CO₂/t conventional concrete, the emission reduction costs are 0-350 ECU/t CO₂.

3.5 Glass

Two options will be discussed for glass:

1. Light weight packaging glass
2. Increased glass cullet recovery.

The issue of packaging glass re-use will be discussed in the forthcoming MATTER-report about packaging.

Light weight glass

The bulk of the glass is used for packaging purposes. In the last decades, significant weight reductions have been achieved for the average glass packaging. In Germany, the weight of the state-of-the-art 1 l glass bottle has been reduced from 570 grammes in 1977 to 295 grammes in 1996, a reduction by 48% in 19 years [98]. UK sources indicate a smaller, but still important weight saving of 39% between 1960 and 1990 [99].

Glass strength is determined by the surface quality. The theoretical tensile strength and compression strength is generally 100 times higher than the actual strength due to surface deflections that serve as crack-initiators.

The main weight savings are achieved by the protection of the outside layer of the glass by coating materials like plastic or ceramic coatings. If similar technologies are applied for all types of glass packaging, a weight reduction by at least 25% seems feasible [38]. Annealing of flat glass products and glass bottles and jars with wide openings can increase the strength by a factor 4, resulting in a weight reduction by a factor 2. The annealing creates a surface tension that prevents crack propagation. The same effect can be achieved through the application of chemical hardening processes based on ion substitution. This can increase the strength by a factor 6-8 [38]. This technology is currently applied for optical glass and for oven glass. It is unclear to what extent these technologies can reduce the weight of packaging glass. Because of the high costs, the latter two options will not be modeled.

Increased glass cullet recovery

Current trends in glass collection in Western Europe are shown in Table 3.5.1. The recovery rates are rapidly increasing. Significant differences do however exist. The national recycling rates ranged in 1994 from approximately 30% in Southern European countries (Italy, Spain, Greece) to

⁴ $180 \times 0.5 \times 0.35 \times 1 = 30$, where 180 Mt cement/year; 0.5 = 50% of all applications; 0.35 = average weight saving; 1 t CO₂/t cement

84% in Switzerland (note: these recycling rates refer only to packaging glass) [100].

Table 3.5.1 *Current glass collection trends, Western Europe (including Turkey) [100]*

	[Mt pa]
1990	5.3
1991	6.0
1992	6.5
1993	6.7
1994	7.3

Two improvement options are discussed:

- increase packaging glass collection rates in Southern Europe
- develop technologies for recycling and recovery of flat glass.

Increased packaging glass recycling in Southern countries to the level in Central and Northern European countries will occur autonomously. An increase of packaging glass recycling to a level of 11 Mt per year should be feasible.

Recycling rates for flat glass are currently rather low. On one hand, the stock of glass in buildings is still increasing (approximately 80-90% net storage, 4 Mt per year). Apart from buildings, the other main application of flat glass are windows for cars and trucks (estimated 1 Mt per year). It is estimated that 2 Mt waste flat glass arise per year in Western Europe. Closed loop recycling for clear flat glass is technologically feasible, but the limited availability poses a barrier for large scale development of recycling schemes [46].

Total additional potential for glass recycling is estimated to be 5 Mt per year. This poses a CO₂ emission reduction potential of 2 Mt per year. Part of this potential is included in the autonomous developments. The colour of recovered glass can pose a limitation for recycling in certain countries: green and amber glass cannot yet be recycled into clean glass. However, techniques have been developed in the US to cover clear glass with coloured coatings that melt away when the glass is being recycled. If all glass were manufactured clear and coloured in this way, there would be no limit to the amount of cullet which could be re-used [101]. However, this would require a major adjustment of the existing glass making practice. This option has not been considered in more detail.

3.6 Sand-limestone

Sand-limestones are sold in different shapes and strength classes in the range of 4-60 N/mm², depending on the mixture of resources that is used. Most conventional types possess a strength of 12, 20, and 28 N/mm² [102]. The conventional size of the bricks depends on the building standards. In Germany, the thickness of regular sand-limestones (perpendicular

to the wall) is 115, 175, and 240 mm [102]. In the Netherlands, conventional thicknesses are 102 (or 104), 150, and 214 mm. A straightforward comparison of German and Dutch brick sizes is complicated by different strength standards. The difference suggests a materials saving potential of more than 10% in the German situation, compared to the Dutch standard sizes.

3.7 Re-use of building elements

Building elements like bricks, sand-limestone, and prefab concrete can be re-used. The life span of these building elements is generally higher than the building life span. For modern bricks, a theoretical life span of 1000 years seems feasible [103]. For sand-limestone bricks, the life span is still unclear because the material has only been used for about 100 years.

However the building material re-use strategy is significantly limited by the ratio of building production to building demolition of 4:1. As a consequence, re-use can cover at most one quarter of the building materials market. Moreover, the mortar bonding practice poses problems. This complicates re-use. Development of new bonding technologies will facilitate re-use. Due to the long building life span, results will only emerge after decades. As a consequence, only the re-use of existing building materials is considered in this study.

The availability of waste ceramic building materials and a cost estimate for their recovery is shown in Table 3.7.1. The recovery potential estimates are based on [34]. Total saving potentials for all three materials are estimated to be approximately 4 Mt CO₂.

Table 3.7.1 *Estimate of waste ceramic building materials arising and availability for re-use*

Material	Quantity [Mt pa]	Recovery potential [Mt pa]	Cost ¹ [ECU/t]	Cost [ECU/t CO ₂]
Bricks	15	5	100	300
Sand-limestone	10	5	75	750
Precast concrete elements	50	10	50	250

¹ Excludes reduced waste disposal fees and value of the recovered product

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