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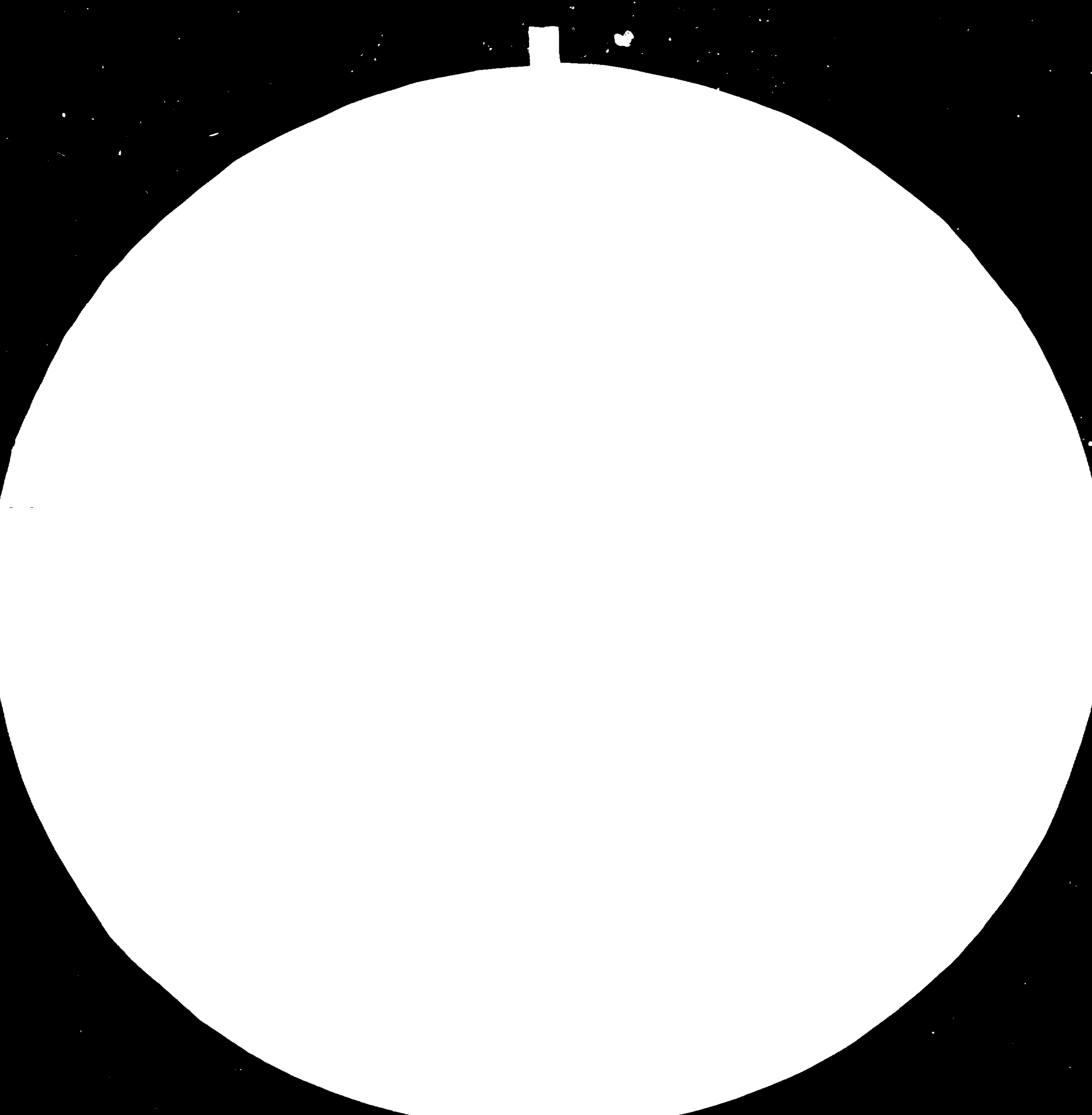

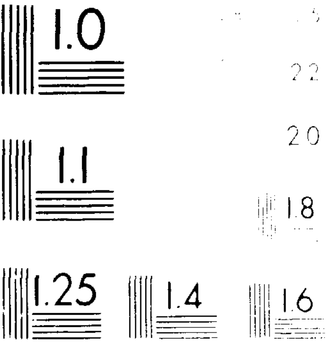
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10

STUDY ON ENERGY ASPECTS OF SELECTED INDUSTRIAL SECTORS: DIAGNOSIS, PROBLEMS, FURTHER DEVELOPMENT.

14015

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Energy aspects of selected industrial sectors

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1. MAJOR WAYS OF REDUCING ENERGY CONSUMPTION IN INDUSTRY

1.1 Introduction

Since man made energy technology his servant, to gain independence from his environment, many efforts have been undertaken to make better use of energy.

The current call for a more efficient use of energy is mainly due to the following factors:

- In many sectors of industry, energy costs are a significant production factor. To maintain competitiveness, the cost situation must be eased by optimising energy technologies.

- Countries that are poor in natural resources are forces to import energy which is a heavy strain on their current account and has caused foreign debts to rise. This financial burden can be eased and availability of energy improved if the sources of energy are diversified and technologies are selected carefully.

- Even countries with plenty of natural resources need to aim at a more efficient use of energy for political reasons and to economise the limited quantity of non-renewable reserves available.

- The steep rise in energy turnover in industrialised countries has caused a considerable load on the environment in the form of pollutants, heat and, very often also, noise nuisance. In some developing countries, there is the added risk of environmental damage caused by degeneration into steppe, as well as climatic changes due to the increased use of timber for energy production. A more efficient use of energy is therefore of vital importance to environmental protection.

1.2 Energy audits as a basis for a more efficient use of energy

An absolute prerequisite of all deliberations on a more rational use of energy is an analysis of the current state of affairs in energy. To provide a valid basis, such analyses, or audits, need to be based on tests and measurements, irrespective of whether they relate to individual plants, machines, or entire organisations. Large areas are covered in the energy balances of regions or countries, for instance. This kind of regional and national balance, however, remains inadequate as far as both quality and quantity of the data base is concerned because some data is based on guesswork rather than measured values.

Fig. 1-1 shows the example of the Federal Republic of Germany and the final energy demand by sectors and type of consumption.

It becomes clear that heat plays a dominating role. Space heating accounts for 33.5 per cent and process heat for some 32 per cent. Together they make up two-thirds of the total demand for final energy in the Federal Republic of Germany in 1982. Transport takes 23 per cent of the remaining energy, and some 10.5 per cent is used for meeting the demand of all kinds of drives in the industrial, domestic, commercial and agricultural sectors. The share of final energy used for lighting is over one per cent.

The charts on the right-hand side of the figure show, in absolute figures, the final energy consumption in the industrial, domestic, commercial and other sectors, as well as in transport. These are subdivided into shares relating to space heating, process heating, light and mechanical energy (electricity). This only goes to underline the dominating part played by space heating in the domestic and commercial sectors, at 77 and 53 per cent respectively, and process heating in the industrial sector at 73 per cent. At 6 per cent, electric light and mechanical energy (electricity) used in the domestic sector are of minor importance. In industry and in the commercial sector, however, where 16 and/or 25 per cent is consumed, electricity is a major source of energy. This is even more obvious in transport, which uses 99 per cent. With a view to obtaining tangible effects through a more efficient use of energy in industry, therefore, efforts have to be focussed on the dominating role of process heat in relation to mechanical energy, lighting and space heating. In order to be able to assess future trends it is necessary to analyse first the evolution of energy consumption so far.

Fig. 1-2 illustrates the development of industrial consumption of final energy, electricity and mineral oil in EC-countries. All countries show similar tendencies. The continuous rise until 1969 in the consumption of final energy is followed by some stagnation and/or a slight drop. A similar trend is observed in oil consumption. However, electricity consumption grew almost every year after 1969, although growth rates were lower than in the period before 1969 /1-2/.

The lower part of the Figure shows the degree of electrification e, which indicates the ratio between net electricity consumption and final energy consumption, and the share of oil p in total consumption. Due to static final energy consumption, despite growing demand for electricity, a great increase in the ratio of electrification has been noted in all countries under review after 1969.

1.3 Ways and means towards a more efficient use of energy

In all energy applications, including energy used in industrial production, there are basically five ways of optimising energy usage. These are

- avoiding undue consumption,

- reducing the specific consumption of useful energy in certain applications,
- improving efficiencies by cutting energy losses,
- recovering energy wherever meaningful in engineering and economic terms,
- making increased use of forms of energy that are permanently available.

1.3.1 Avoiding undue consumption

Undue consumption may be caused by

- idling of machinery and plants,
- overheating of space,
- excessive requirements placed on process parameters ('fear margins' on pressure and temperature),
- technical defects (eg leaks in a steam system).

In other words, undue consumption applies whenever the generation of net energy does neither optimise production nor create additional services or enhance comfort.

Some of the problems mentioned can be remedied by technical means, either by fitting appropriate controls or by improving the service and maintenance of the machinery involved. However, an even better savings effect is obtained by giving operating staff some fundamental physical and process engineering background, beginning with a cost appraisal of the various kinds of useful energy.

Light and its significance as the source of all organic life is normally overrated, at least inadvertently. This is also true of power and mechanical energy which, because of our associating it with muscle power, is also rated high. This contrasts strongly with heat which is extremely underrated, because we are only able to perceive temperatures rather than quantities of heat.

1.3.2 Reducing the specific net energy demand /1-1/

It is mainly technical means that can help cut the net energy demand in certain applications. This means that an optimum energy technology is also chosen for all secondary uses.
For instance, mechanical instead of thermal drying will cause the energy demand per kilogram of extracted water to drop to one-hundredth. Other examples include vacuum evaporation, pressure cooking, bonding instead of welding, non-cutting instead of cutting shaping.
A change of material can also lead to substantial reductions in energy consumption. For instance, the 4.4 MJ of energy needed to produce a 0.33-litre tin with aluminium top can be lowered by 30 per cent to 3.1 MJ for the same tin with a tin top.

The recovery of raw material, such as glass, paper, and metal, through classified waste disposal also reduces waste production and, hence, environmental pollution, plus the fact that it helps save energy.

The cumulative energy required to make a 'euroglass' bottle of 0.5 litre will drop by 2.5 per cent in power consumption if the share of cullet is increased from 17 per cent to 36 per cent. Simultaneously, fuel consumption will drop by 11 per cent, and the total final energy consumption will fall by 10 per cent /1-1/.

The energy requirement of heating and lighting in office blocks and production facilities can be reduced by improving both thermal insulation and lighting arrangement of rooms.

1.3.3 Improving efficiencies

Design improvements of plant and machinery and, in particular, the integration of modern control techniques will in many cases cut energy losses significantly. For the plant it means a rise in efficiency, for the production process a reduction of the specific energy consumption. Wherever technically feasible, all burners used in combustion systems should be fitted with waste gas analysers and a suitable control to adjust near-stochiometric combustion, because excess air lowers the temperature of the flame and raises waste gas losses. The rise in fuel consumption caused by excess air is a function of the waste gas temperature, as is shown in Fig. 1-3 /1-3/.

An important example of how efficiencies can be increased is the co-generation of electricity and heat. In co-generation, the energy losses are much lower than the separate generation of heat, and power produced in a condensing turbine, as is illustrated in Fig. 1-4 for th various methods of co-generating power and heat.

Especially the small and medium-sized power stations using gas-fired combustion engines are an important alternative to the conventional thermal power plants using back pressure steam or gas turbines.

Both energy losses and the electricity and heat efficiency ratio are mainly a function of the type and size of a plant, as well as the temperature level of the heat produced. The great differences that exist in electricity efficiency of the various types and its variations are in no way the only criterion to be applied to process selection, because the temperature level of thermal heat or process heat that can be reached in a system is as important as its actual requirement.

The ways in which co-generated heat and electricity is used in industry depends primarily on the amount and make-up of the energy required. Since 'co-energies', as the name implies, are generated at the same time, and can be stored to a limited extent only, it is of crucial importance to match the electricity and heat demand curves. The time course of electricity and heat demand, as well as the required temperature level, must be tested. Therefore, the requirement has to be analysed and the availability must be defined before all components of co-generation are compatible. Only then can a useful combination of all parts of the plant be assessed properly.

The principal source of conservation of energy lies in the advanced operating mode and in the optimum utilisation of existing energy technologies. In all energy sectors, a major part of the energy consumption is not so much as function of load, but one of size and type of the plant or machinery. This also means that the specific consumption drops as the load rises. Operating the plant or machinery to full capacity and avoiding downtime, be it in a tank furnace or a boiler, will improve the efficiency on a large scale.
Fig. 1-5 illustrates the situation of two chip driers, one being run with fresh air intake, the other one with recirculated air. Due to the different designs of both machines, there is a difference in the basic requirement and the rise in load-dependent energy input. This explains why the first machine shows reasonable energy consumption at a throughput of over 1,100 kg of water evaporation per hour, while it is better below that throughput for the second machine.

The degression of specific energy consumption with a rise in throughput becomes more and more obvious in modern plant and machinery because due to increasing mechanisation and automation, including control requirement, there is also a rise in the amount of energy that is independent of load. Therefore, plant with high shares of load-independent energy consumption need more power during downtimes. Therefore, it is essential to cut downtimes in such plants to make energy use more efficient. Whenever possible, controls should be automated so that undue idling can be avoided. Investigations at the Research Centre for Energy Utilization and Technology have shown that in machine tools about 30 per cent of the total energy consumed during one shift goes into downtimes.

It is also important to match the capacity of the installed equipment with actual needs. This applies in particular to drives which tend to be overdimensioned because the actual power requirements in individual production cycles are unknown. This means that they are not only unduly high in investment cost, but also in energy consumption. Matching the machine size with the production task can be done easily in single-purpose machines.

One way of reducing losses in overdimensioned three-phase current drives is to switch the electric motor from delta to star connection mode, on condition that the drive windings are designed for the main voltage and that the motor is loaded with less than 35 per cent of its rated load.

For machines operating at varying loads, such as conveyor belts and the like, load switches may be introduced to automate the switching from delta to star mode and vice versa. Machines that require very high ratings at the start but lower ratings during normal operation, such as pneumatic conveyors and centrifuges, lend themselves to starting on the delta mode and then switching over to the star mode after run-up /1-1/.

Many branches of industry employ drives of usually minor ratings for pumps, fans and blowers with a variable power supply. If such a drive runs on direct mains supply, the supply can be reduced only by the throttle and/or bypass control. The power used by the motor is hardly reduced. A significant amount of energy can be saved if the motor speed is controlled through frequency converters and voltage converters. In many cases, such additional components are economically justified (Fig. 1-6) /1-4/.

1.3.4 Energy recovery

In industry, energy recovery is always a synonym of heat recovery. The economical use of waste heat is possible whenever waste heat is concentrated, ie bound to one or several discrete mass flows (water, air, gases, solids), rather than occurring dispersed or scattered in the form of large-surface losses to the atmosphere. The higher the temperature, the less complicated and expensive heat recovery is.

Fig. 1-7 shows the temperature level and the kind of industrial production of waste heat in the Federal Republic of Germany in 1978. Almost half of the industrial waste heat is emitted in concentrated form and can therefore, theoretically, be used.

Wherever waste heat is to be recovered, the processes that produce waste heat need to be analysed in detail, and potential users of waste heat need to be identified.

The ideal match of both supply and demand is very rare, and therefore heat accumulators have to be used to bridge the time-lag between the two.

However, before trying to optimise the energy input of a system by recovering the waste heat it produces, it is recommended to minimise the yield of waste heat as far as is technically and economically feasible by appropriate design and operation. Whatever waste heat remains can be used according to the following priorities:

1. the waste heat produced in a manufacturing process should be recirculated into the same process wherever possible;
2. the waste heat produced in industrial plant should be recirculated

for use in the same plant wherever possible, and

3. only after exhausting the first two opportunities should the outside use of industrial heat be considered.

If the waste heat cannot be used directly in the plant, it can be converted into mechanical or electrical energy by a thermodynamic cycle process. At a temperature level of over 250° C of the waste heat, a conventional water and steam cycle process is used, whereas an organic medium (ORC Organic Rankine Cycle) is indicated at a temperature range of between 100 and 250° C /1-5/.

In densely populated areas it may also be worth checking if the waste heat can be fed into the district heating network.

Problems arise in heat recovery whenever the temperature level of the waste heat is too low, in which case heat pumps are used to exploit the heat flow. It needs to be noted, however, that there is a limit to the upper temperature level.

Industrial heat pumps are designed to apply one of the following principles:

- cold steam compression,
- absorption ,
- hot steam compression,
- steam jet.

Fig. 1-8 illustrates the respective switching diagrams. The main difference between the various principles is the kind of compression (thermal or mechanical) and the kind of process control (open or closed loop).

In mechanical compression, piston compressors (small feed volumes), screw compressors (small to medium feed volumes), or turbo compressors (medium to large feed volumes) are used. Both gas and electric motors are proven drives for these compressors.

An interesting combination between compression heat pump, electrical machine and combustion engine is shown in Fig. 1-9. In this kind of tandem plant, which uses 32° C-waste water for the generation of process heat in the form of hot water, three machines are arranged on a joint shaft. Three different modes of operation can be selected by actuating a clutch. Depending on the energy requirement, the sources of energy available and economic considerations, the system can be used as

- a gas heat pump,
- an electric heat pump, or
- a co-generation plant.

Table 1-1 lists the ratings and efficiencies of the various modes of operation.

Mode of operation	Energy input		Energy output			Efficiency
	Nat. gas	Electr.	Heat		Electr.	
			Motor	Heat pump		
	kW	kW	kW	kW	kW	
Gas heat pump	270	-	158	532	-	Heating capacity 2.6
Electric heat pump	-	98	-	503	-	Figure of merit 5.1
Co-generation plant	288	-	167	-	91	Efficiency 0.9

Table 1-1: Technical data of a tandem plant

1.3.5 Using renewable forms of energy

Renewable forms of energy are for the most part direct or indirect solar energy, such as the energy contained in wind and flowing streams, ground heat, and chemically bound biomass energy. In addition, there is geothermal energy flowing from the hot interior of the earth to the surface, energies from the gravitational attraction between Earth and Moon, and the waste heat from anthropogenic energy consumption. There are several ways of using this energy, and although some technical and economic constraints persist, these are likely to be overcome in the future.

It should be noted that, unlike fossil fuels, these sources depend upon technologies which are based on a growing need for auxiliary energy; and it is electric energy that comes into consideration most. Moreover, the space and material requirement per unit of power has to be higher than in conventional systems because the power densities of natural renewables are low.

There are only slight chances in the more developed industrialised countries to use renewable forms of energy for industrial production. However, the chances are better in many of the developing countries because of a more favourable climate and basically different conditions. In tropical and subtropical countries, in particular, solar energy could play a useful role in contributing to energy supply in agriculture and in some sectors of industry.

The following applications have been identified for solar energy /1-6/:

- production of low-temperature heat,
- direct generation of electricity through solar cells,
- production of high and very high temperatures by concentrating solar energy through optical systems.

The production of solar energy by photovoltaic systems or in solar towers or solar farms, will hardly be significant on a large scale because of the high capital expenditure involved. An interesting alternative to solar energy production is based on solar ponds and closed Rankine turbine cycles, using an organic working medium.

Possible applications of solar current are in places where other forms of energy are either too expensive or not available, and in special cases where electricity is needed to power and operate auxiliary systems of thermal processes which also run on solar energy.

However, it is simpler and more efficient to use solar energy in thermal processes, ie usually applications which are inconceivable in the climate of central Europe or those that are insignificant in Europe anyway. These are

- food refrigeration and drying,
- firing of building material in solar furnaces,
- sea water desalination,
- cooking in solar cookers (substituting wood as a fuel).

A wide application of solar energy in developing countries is impaired by high capital-intensity.

This is also true of wind energy although wind energy can be used favourably for powering small units, such as water pumps, corn mills, etc.

There are a number of processes (Fig. 1-10) /1-7/ that convert substances produced by biosynthesis (biomass) into energy. Mention should also be made of the production of biogas and fuel from biomass, in addition to the direct combustion of wood, for instance, which has caused deforestation and soil erosion in many regions.

The production of biogas on the basis of dung or plant waste is more or less possible in most countries. Compared with direct combustion of manure it has the merit of improving the quality of manure without keeping it from the farmland. Biogas can be used for cooking, heating, etc. in the domestic sector, and it can also be combusted in engines.

While the production and application of biogas is limited to rural areas, energy farming opens industrial applications. In Brazil, for instance, the production of ethanol from sugar cane has been developed on a large scale. The project consists of an industrial plant and a power plant which utilises the waste of the industrial plant to produce electrical energy. Ethanol can be used for blending or diluting conventional petrol or for combustion in a purpose-built car engine.

Due to the priority to be placed on the production of food and fodder, however, the wide-spread use of high-yield farmland for energy farming will be limited to countries which have gained a high degree of self-sufficiency in agriculture.

1.4 Problems and constraints of energy conservation /1-1/

As for anything else, there is a price to pay for energy conservation. Although there are many ways of meeting the energy demand more efficiently by making better use of existing energy systems, a lot remains to be done in the way of stimulating the information process, encouraging a re-thinking process on the part of the individual consumer of energy.

In view of the great differences in the cost-benefit-ratio of various options, each case of optimising energy usage has to be assessed on its own merits. Special attention is to be given to the cumulative effect of isolated action or measures taken in the same field. In general, priority should be given to measures that produce maximum savings even if they do not appear to be very spectacular or demand greater attention in the routine operation of the process concerned. This kind of inexpensive action alone can help save energy to an extent that makes costly and capital-intensive projects, such as heat recovery systems, superfluous.

Capital-intensive measures do not only lead to an increase in absolute figures; more important is the change of the cost structure and the shift towards growing overheads. This emphasises the need for reaching high load factors to make it profitable.

Moreover, technology is getting more and more complex in general, and its fundamental concepts are less and less clear to the layman, just as to the expert. Accordingly, repair and maintenance require increasingly qualified and specially trained staff and is therefore developing into an important factor.

Any assessment of the effect of energy conservation measures on the national economy presupposes relevant data and causes problems of practicality for that matter. All aspects relating to any measure taken to use energy more efficiently need to be viewed in such an assessment. These aspects are of an economic, ecological, social and human nature.

The fact that energy conservation has become the focus of heated discussions and debates emphasises the great challenge involved in pushing technical advance in this direction. However, in spite of all hostility towards technology and experts, unrealistic expectations are placed in 'overnight' improvements. More often than not, people believe that anything can be done with technology, basically, and that it is easy to develop an idea and to realise it within ever shorter periods of time, seeing it through to a well-tested, reliable, safe, maintenance-free and low-cost product;

ignoring the fact, of course, that technical advance takes time to mature from first conception to final demonstration. It is the mere need to carry out field tests, in addition to analyses and laboratory tests, that calls for a period of several, if not many, years.

The efforts undertaken so far in the field of technology and science have led to the commercialisation of a number of technologies and economical alternatives which reduce the demand of specific energy. Moreover, all realistic options should be taken simultaneously. There will be no way to 'eternal energy bliss', certainly, and urgent warnings should be raised against monolithic solutions.

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2. NON-FERROUS METALS

2.1 Aluminium

2.1.1 Production of aluminium from primary raw materials

Current aluminium production is based on bauxite as a raw material, which is first converted into alumina, or aluminium oxide. This is then reduced to metallic aluminium in a pyrometallurgical process and cast into ingots which are subsequently manufactured to semis and castings. Growing attention is paid to producing aluminium from scrap (see Chapter 2.1.2). Due to the energy-intensive processes used in the aluminium industry, low-cost energy has to be relied upon. Notes on the international cost situation in the aluminium industry are found in /2.1-23/.

2.1.1.1 Bauxite winning and alumina production in the Bayer process

Bauxite is almost exclusively used as a base material for the large-scale production of aluminium oxide. Bauxite is mined in open pits, then crushed, washed, screened, and partially dried; this used to be done at a temperature of up to 400° C. However, in most cases the drying stage is omitted.

Depending on the conditions, the specific fuel consumption in the drying process is 0 to 1.8 GJ per tonne of bauxite, with the specific electrical input being 7.5 kWh/tonne of bauxite. In all other stages of bauxite extraction, 7.4 kWh is consumed in electrical energy and 0.08 GJ in fuel for one tonne of bauxite /2 1-1/. The energy used for transporting bauxite to the oxide production plant has not been considered here.

The only process being used for the production of aluminium oxide today is the Bayer process /2.1-6/.

The Bayer process is based on the extraction of hydrated aluminium oxide from bauxite using a high-temperature solution of sodium aluminate, separation of the solid residue ('red mud') after cooling the suspension, partial precipitation of hydrated aluminium oxide from the oversaturated aluminium solution through seed crystallisation and recirculation of the solution after separation of the crystallised hydrated oxide. The system uses two physical and chemical properties of the Al_2O_3 - Na_2O-H_2O system, ie the fact that the solubility of the hydrated aluminium oxide in caustic soda solution is a function of temperature, and the metastability of oversaturated aluminium solutions. The resulting hydrated aluminium oxide is dehydratised to form aluminium oxide. A schematic of the Bayer process is shown in Fig. 2.1-1. The process engineering has been modified substantially in the past few decades, and different techniques have been developed in Europe and in the United States because of the different types of bauxite that were available in both areas. They were boehmite-bearing bauxites that were harder to digest in Europe, whereas in U.S.A. hydrargillitic bauxites were available that could be extracted at low temperatures.

The following description is an example of the continuous process which is applied with slight variations in modern European alumina plants. Bauxite is wet ground in ball mills or rod mills and then fed continuously by a piston diaphragm pump to the high-pressure digestion system. This system consists of a cascade of agitated autoclaves or a tube reactor, where an indirect pre-heating phase using flash evaporation, is followed by indirect heating up to the final temperature (up to 270° C), using high-pressure steam or salt melt. The resulting suspension, consisting of red mud and sodium aluminate solution, is cooled down to about 110° C through flash, with the vapour being used for pre-heating the bauxite suspension, as mentioned above, and the resulting condensate being used for washing the red mud.

Most of the energy in aluminium oxide production is consumed during the digestion stage. In the best case, energy consumption is 2.2 GJ per tonne of Al_2O_3. Modern plants with tube reactors have an input of 3.4 GJ/t Al_2O_3, although in practice it may be up to 8.7 GJ/t Al_2O_3 /2.1-2/. Heat input during extraction is mainly determined by the amount of solution used, its difference in temperature upstream and downstream of the extraction plant, and the type of heat transfer medium /2.1-2/.

As to the amount of solution, there is a characteristic difference between U.S.A. and Europe. In U.S.A. 15 m^3 of solution is used per tonne of Al_2O_3, in Europe it is 11 m^3. This difference in the specific solution volume, with all other conditions being equal, explains why U.S. plants use about 35 per cent more heat for heating the solution than that consumed by European plants.

The sodium aluminate solution is separated from the red mud in large-volume settling tanks or digesters. To remove the major share of adherent alkali, the red mud is washed again, filtered in Kelly filters or vacuum filters and then largely discarded. 50 per cent of the amount of bauxite is discarded in the form of red mud. Although it is possible to process red mud in order to recover its constituents (sodium aluminosilicate, iron oxide and titanium oxide), such process has not been found to be economically justified so far.

The excess solution from the digesters undergoes an indirect heat exchange with clarified solution (see below), and is then passed through a series of blow-off tanks. Here, at a temperature of 70 to 55° C, a major fraction of the hydrated aluminium oxide, which is dissolved in the oversaturated solution, is clarified, with hydrated aluminium oxide (hydrargillite, $Al(OH)_3$) being used as seeding. This has previously been separated on vacuum drum filters or leaf filters. A minor fraction of the separated hydroxide is washed again and fed to the calciner. The clarified solution (containing 40 to 50 per cent of aluminium oxide) is preheated through indirect heat exchange with oversaturated solution (see above) and recirculated into the bauxite mill, with approximately 85 kg/t Al_2O_3 being lost.

In the red mud separation stage and others, heat is consumed mainly in the form of radiation, amounting to 1.3 GJ/t Al_2O_3 or 2.1 GJ/t Al_2O_3. Systems that separate impurities from the solution, such as vanadin salt, soda, or oxalate, cause a heat consumption of 0.84 GJ/ t Al_2O_3 to 1.3 GJ/t Al_2O_3. The total energy requirement for losses and by-product separation are a clear function of the climatic location of the oxide plant; the mean total energy requirement in Europe is 2.7 GJ/t Al_2O_3.

After clarification, the aluminium hydroxide contains adherent moisture and chemically bound water, both of which have to be extracted before the electrolytic process. During calcining, a firing process, virtually all water is removed at a temperature of 1,100° C. Gas-fired recuperative rotary kilns which are up to 100 m long and 4 m in diameter are fed with aluminium hydroxide at the opposite end of the burner. While moving through the kiln, the aluminium hydroxide is dehydrated and calcined in the hottest zone. However, mostly fluidised-bed kilns are used today, which consume considerably less energy.
The specific fuel consumption of a fluidised-bed kiln is about 3.1 GJ/t Al_2O_3, which compares with about 5 GJ/t Al_2O_3 for the rotary kiln. A major share of the resulting oxide is processed in aluminium smelters. A minor share also present in the form of hydroxide is used in various applications in the chemical industry, in ceramics, etc.

Considerable improvements have been made in the past few years in the Bayer process. Particular mention should be made here of the change from discontinuous to continuous operation, the introduction of a tube reactor for the digestion of bauxite, and the vacuum drum filter for the separation of red mud, as well as the replacement of rotary kilns by fluidised-bed kilns for the calcination of hydrated aluminium oxide. These improvements have resulted in a considerable reduction of the amount of heat required in the process. Modern European plants require heat for the production of oxide in the order of less than 10 GJ/t oxide, while old plants used up to 24 GJ/t. The output of modern oxide plants is between 0.5 and 1 million tonnes of oxide per year, in some cases more /2.1-6/.

The electrical input is between 200 and 300 kWh/t Al_2O_3. Data for the Federal Republic of Germany in 1982 show a fuel consumption of about 14.3 GJ/t Al_2O_3 and an electrical input of 245 kWh/t Al_2O_3.

It becomes obvious that the energy input is subject to great variations, and that, therefore, the main energy-intensive stages present the greatest saving potential. While the energy demand in calcining cannot be cut further if a fluidised-bed kiln is used, all other energy-consuming stages being less significant, the only stages that vary most and therefore lend themselves to sustained energy conservation are the extraction and evaporation phase.

2.1.1.2 Other processes used for the production of aluminium oxide

In addition to the Bayer process, there are a number of other processes for producing aluminium oxide which, however, are insignificant in current practical application. Therefore, these processes are only listed below by name:

Processing of bauxite:
- sintering with soda,
- lime extraction,
- Hall process,
- Haglund process.

Processing of non-bauxite material:

- sulphite clay process,
- sulphuric acid digestion,
- Aloton process,
- Griesheim process,
- Nuvalon process.

At present, the development and demonstration of processes is pushed in the field of aluminium oxide production based on non-bauxite raw materials. These comprise, inter alia, clay, anorthoside, alunite, and dawsonite. Because of their availability in large deposits, caolinitic clays are of particular importance. The estimated energy input quoted in literature for the production of aluminium oxide on the basis of clay is 2.5 to 3 times that of the Bayer process /2.1-3/. Results of the latest studies suggest a factor of 2, however.

2.1.1.3 Electrolytic reduction of aluminium oxide

Virtually the only method used today for producing commercially pure aluminium (Al 99.9) is electrolytic reduction. The solubility of the aluminium oxide (melting point of about 2,050° C) in molten cryolite helps electrolyse oxide at a temperature of only 950 to 980° C. Some of the cryolite is consumed and needs to be replaced. Synthetic cryolite is used.

The electrolytic cells consist of carbon-lined tanks. Cathode lines are planted in the tank base. Cathodically separated liquid aluminium has a heavier specific weight than the remaining electrolyte and settles on the carbon bottom from where it is siphoned at regular intervals. From above, the anode that consists of carbon too, dips into the bath. It burns the oxygen released from the aluminium oxide, producing mainly carbon dioxide. In practice, two types of anodes are used. These are

a) the self-baking Soderberg anode which uses patent fuel, called 'loaves' of Soderberg paste (low-ash coal, containing 25 to 35 weight per cent of coal-tar pitch) that are filled into an aluminium jacket. In the heat of the cell they bake and form solid anode carbon;

b) pre-burnt discontinuous or continuous anodes of large carbon blocks (for instance, 1,900 x 500 x 600 mm, about 1.1 tonnes in weight) which are inserted from above.

Anode consumption is in the order of 460 to 480 kg C per tonne of aluminium /2.1-6, 18/. The energy required to produce anode carbon is about 10 to 11 kWh/kg, including non-energy consumption.

Electrolytic cells operate at over 150,000 A. They are connected in groups, one after the other, producing a system of long rows of cells. The operating voltage of a cell is 4 to 4.5 Volt which is much higher than the decomposition voltage of the alumina because voltage losses occur in various parts of the system.

There is a trend in electrolytic cells for larger sizes, higher intensities of electric current and lower cell voltages. Today, intensities of between 150 and 180 kA can be realised without problem, in some cases even 240 kA.

In the production of aluminium, both electrical and chemically bound energy is used. The latter is present in the form of carbon produced from anode consumption (daily electrode consumption about 1.8 to 2 m) and is a key factor in determining production cost and the entire operation of the process.

Schmidt-Hatting /2.1-4/ and G. Wilde /2.1-5/ have calculated the amounts of energy required in electrolysing aluminium.

The electrolytic decomposition of aluminium oxide to aluminium and oxygen at the usual temperature of about 950 to 970° C has a minimum energy demand of 8.7 kWh/kg Al, with the oxygen being separated at the inert anode at a pressure of 1 at. Out of this amount of energy, 6.6 kWh/kg Al is input in the form of pure electrical energy and 2.1 kWh/kg Al in the form of heat. This is to be added to the heat required for heating the alumina, which takes about 0.6 kWh/kg Al., which makes it a total theoretical energy demand of 9.3 kWh/kg Al.

Under common technical conditions, the carbon anode makes a theoretical contribution of about 3.7 kWh/kg Al, taking into account the energy required for its own heating (0.22 kWh/kg carbon ≙ 0.1 kWh/kg Al). Therefore, the minimum energy required to decompose aluminium oxide, using a carbon anode and a current efficiency of 100 per cent, is roughly 5.6 kWh/kg Al /2.1-6/.

For many decades, aluminium smelters have focussed their attention on the reduction of specific energy consumption in electrolysing aluminium. Thus, the specific consumption of electrical energy was cut from about 40 kWh (direct current)/kg Al in the first single anode or small cells using round anodes to 13 to 17 kWh (dc)/kg Al today. In 1980, the average consumption in the Federal Republic of Germany was at 16.8 kWh (ac)/kg Al /2.1-18/.

The specific consumption of electrical energy W_{el}/kWh/kg Al is calculated on the basis of two variables, ie the cell voltage U(V) and the current efficiency (= the relationship between actually and theoretically produced aluminium):

$$W_{el} = \frac{U}{0.3354 \cdot \eta}$$

The cell voltage was reduced from about 10 V in the past, to about 4.0 V today, and the current efficiency was increased from originally some 70 per cent to almost 90 per cent today. Therefore, the reduction in the specific consumption of electrical energy is largely due to the improved cell voltage, and to a lesser extent to the increase in current efficiency. Whereas the current efficiency can hardly be increased to more than the present 90 per cent in practical application, a further reduction is still possible in the cell voltage, which can be achieved mainly to optimising the cathode design, extending bus bar cross sections and reducing the current density in the cell. However, all these steps call for a high capital expenditure.

In terms of profitability, there is a limit to the specific consumption of electrical energy, which is in the order of 12 kWh/kg Al; this can be achieved, for instance, at a current efficiency of 91.5 per cent and a cell voltage of about 3.7 V /2.1-6/.

Fig. 2.1-2 illustrates the energy flow in electrolytic production of aluminium /2.1-15/. The amounts shown in this source differ within certain limits from other values given in other sections of the present study.

2.1.1.4 Electrolytic refining

In all applications where a higher degree of metal purity is required the three-layer electrolytic refining method (99.99 Al) has proven its worth for a long time. Recently, the liquation method has also come to be used. It produces metal of a purity of 99.96 to 99.99 per cent of aluminium.

The consumption of electricity in electrolytic refining is significantly higher. According to source /2.1-24/, it is 20 to 22 kWh/kg, and a typical value of the Federal Republic of Germany /2.1-18/ is 20.5 kWh/kg. It should be noted, however, that the only material to be used in three-layer refining is primary aluminium and/or alloys with very low foreign metal fractions.

2.1.1.5 Other ways of producing aluminium

Attempts have been made to develop new methods that could replace the electrolytic production of aluminium. They have led to a number of processes, including
- the Pechiney process,
- the Alcan-subchloride process,
- the Alcoa process, and
- the Toth process.

Because of poor profitability and because of problems in designing the process control, the development of the first two processes was given up. The Alcoa process is applied in a pilot plant.

The process which was further developed by Aluminium Company of America is based on the electrolysis of aluminium chloride which is dissolved up to 7 per cent in an electrolyte containing, for instance, 50 per cent NaCl and 50 per cent LiCl. Starting with bauxite, alumina is first produced in the Bayer process. This is turned into aluminium chloride, which is then electrolysed in tightly sealed cells. The objective is to start on the mineral itself, in which case the Bayer process would become superfluous.

The advantages of the Alcoa process are stated /2.1-8/ as follows:
- up to 30 per cent savings of electrical energy in relation to the current method;
- replacement of the expensive cryolyte by less expensive chemicals;
- elimination of the risk of fluorine emissions;

- low site area requirement;
- low operating cost.

In the Toth process, aluminium is obtained from aluminium chloride by reducing it with manganese. Manganese chloride is oxidised to manganese oxide, producing chlorine. The resulting manganese oxide is then reduced to re-usable manganese. The suggested method would require only very little electrical energy, but it appears to be very complex and is viewed with some scepticism by experts.

A survey among experts showed a success rate of only five per cent for the Toth process /2.1-9/.

2.1.2 Production of aluminium from secondary material

Remelted aluminium or aluminium from secondary material (new scrap, old scrap, chippings, dross) accounts for about 30 per cent /2.1-13, 14/ of all aluminium consumption in the Federal Republic of Germany. It is used almost exclusively for cast aluminium.

The average demand of electricity for the production of secondary aluminium from various types of scrap in Great Britain is 90 kWh per tonne of secondary aluminium; fuel consumption is 14.3 GJ/t secondary aluminium, and the primary energy requirement is 16.3 GJ/t secondary aluminium /2.1-11, p. 42/.

Other sources state a primary energy requirement of 13.9 GJ/t remelted aluminium /2.1-17/ and/or 4.2 GJ/t /2.1-18/.

The wide difference in quoted values is mainly due to the different equipment used in remelting plants; but it is also due to the difference in the quality of scrap used and the alloys to be produced.

2.1.3 Melting and holding of aluminium as a pretreatment stage

Depending on the two basic methods of producing aluminium (primary and secondary aluminium, also referred to as commercially pure and remelted aluminium) there is a need to add other elements in order to change the properties of the largely pure base material before it is treated further. A distinction is made here between wrought alloys and cast alloys.

Wrought alloys permit the addition of other elements in small quantities only. These elements are, for instance, Si, Fe, Cn, Mn, Mg, Cr, Zn, Ti. They are added to enhance the strength or the chemical resistance, among other things. Wrought alloys are used in the manufacture of semis or parts that are rolled or pressed.

Cast alloys, however, permit the addition of up to 13.5 per cent of silicon. Other elements can also be added in larger quantities than in wrought alloys.

In producing such alloys or aluminium castings, melting and holding aluminium liquid is a frequent process. It is usually done in two separate plants.

Theoretically, 1.17 MJ is needed to heat one kilogram of pure aluminium from room temperature to a casting temperature of 750° C. This requirement is composed of the amount of heat needed to get to the melting point (660° C), about 0.69 MJ, then the melting heat of 0.39 MJ, and the amount of heat required to heat the molten metal by another 90° C, which is 0.09 MJ. In many cases it is both technically feasible and economically justified to use a fuel-fired kiln for melting and an electric furnace for holding. A survey of common types of furnaces and firing is given in Table 2.1-1.

Recommended energy consumptions for various types of furnaces are shown in Table 2.1-2.

2.1.3.1 Crucible furnaces

Crucible furnaces are furnaces which collect the molten metal in a pot-shaped container. To remove the metal, the furnace tilts for pouring or the crucible is lifted out. Crucibles are made of clay graphite, silicon carbide, or cast iron.

Gas or oil-fired crucible furnaces are mainly used as smelting furnaces; however, they can also assume the function of a holding furnace. Modern automatic gas and oil burners guarantee trouble-free temperature control and great evenness of temperature distribution inside the furnace and in the molten mass. If both the design and the service and maintenance of the furnace are correct, the melt does not get into contact with the combustion gases. Pure melting furnaces are preferably tiltable crucible furnaces with built-in pots; the capacity is usually 350 kg, with the rate of melting being between 300 and 400 kg/h. In recent years, priority has been given to developing furnace engineering with a view to cutting energy consumption, improving the metal quality and reducing environmental pollution; and a high standard has been reached in these fields. Thus, crucible furnaces are now offered with recuperators for preheating the combustion air; they make up to 35 per cent better use of the thermal energy. However, they also require the use of high-quality insulating material, a favourable position of the burner, burner control, automatic temperature control, monitored ignition and combustion, in addition to the recuperator.

Crucible furnaces can also be heated by electric resistance (almost exclusively in holding operations), or by induction. In induction furnaces, the heat is generated directly inside the metal to be heated (see Chapter 2.1.3.3).

Basically, it can be stated that electrically heated crucible furnaces feature low melt losses or deads, ie about one per cent of the input weight. In fuel-fired furnaces, deads can be expected to be about two or three per cent.

2.1.3.2 Hearth furnaces and rotary kilns

Designs of potless furnaces are hearth furnaces (sometimes also called tank furnaces) and rotary kilns. Fuel-fired hearth furnaces can be used for both melting and holding aluminium.

A new application of this type of furnace is in storing and holding molten metal which is supplied by the aluminium smelter to the processor. The main features are a large bath surface in relation to the depth of the bath, and a flame which is usually directly above the surface of the bath.

A detailed description of the properties of this type of furnace is given in /2.1-19, 2.1-20, 2.1-21, 2.1-22/.

Rotary kilns are particularly suited for melting chips, drosses, and old scrap under a chemical cover. Their main features are good utilisation of fuel and high melting rates.

2.1.3.3 Induction furnaces

Unlike the resistance-heated furnace, this type of furnace produces heat directly in the metal to be heated. The input of electrical energy is transformed into heat through the effect of a magnetic field of an induction coil within the metal itself, in which case the metal to be molten acts as the secondary coil. The main advantages of induction heating are that inductive heat generation is more economical than resistance heating and the bath to be circulated and stirred causes the desired thorough mixing of the melt.

Other advantages of induction furnaces are high efficiency, the absence of waste gas losses, low deads of 0.5 to 1.0 per cent as a consequence of the small bath surface area and the mode of action, good temperature control, high melting rates, clean operation, no overheating of the furnace feed because the heat required in the metal is produced by induction. Basically, these furnaces operate on normal supply frequency of 50 Hz. The main furnace charge is ramming material. There are two types of induction furnaces, ie channel furnaces and coreless furnaces.

Induction channel furnaces

These furnaces have straight channels which can be cleaned easier with fixed tools than the curved channels of the past. The channel surrounds a built-in primary coil. Channel furnaces always require a closed secondary loop in the form of the liquid metal contained in the channel. Therefore, a liquid 'heel' must be maintained, ie a quantity of metal being up to half the furnace's capacity. The furnace must be started on liquid metal.

Channel furnaces are used as holding and casting furnaces. The channels may be either below the actual furnace chamber or at an angle on the side. When used as a holding reservoir in large-throughput operation, they are placed, in a kind of tower (with a capacity of 3 to 15 tonnes), between the melting and the casting shop. Because of the ease with which their temperature can be controlled, channel furnaces are particularly suited for use in quieting, or dead-melting, and in refining. Metal is discharged through a tap hole.

Coreless induction furnaces

In contrast to the channel furnace, the coil in the coreless furnace surrounds the furnace chamber, usually around its full height. The furnace can start with solid metal. Coreless furnaces are mainly used for melting. Recently, holding furnaces have also been designed without channels (short coil holding furnaces). Because of its plain and smooth shape, the furnace is easy to clean.

Preparation of the furnace is simple and rapid. The furnace can be used in intermittent operation without any problem, alloys can easily be changed. Metallurgically, these furnaces work well; both the mode of operation and the small size of the bath surface help avoid contaminations. The bath circulation can be adjusted, which is a good basis for adding alloy components, as well as for preventing the separation of components of the alloy because of differences in densities (separation by liquation).

2.1.4 Manufacture of aluminium semis

2.1.4.1 Production of sheet aluminium

Ingots of primary aluminium are molten to produce sheet aluminium. The scrap produced during the manufacture of strip can also be added to the melting.

The whole process of sheet aluminium manufacture comprises the melting process of starting material, continuous casting, a pre-heating stage, a hot-rolling plant, a two-stage cold rolling plant with an intermediate annealing stage, as well as trimming and conveying units. According to /2.1-11/, the manufacture of one tonne of 0.5 mm-gauge sheet aluminium requires 1,400 kWh of electrical energy.

2.1.4.2 Production of aluminium foils

Foil is produced by a continuation of the cold rolling process down to thicknesses of 0.005 mm.

Based on the above procedure on 0.5 mm-sheet, other cold rolling and annealing stages are added.

According to /2.1-11/, the production of one tonne of 0.02 mm-gauge aluminium foil requires 2,700 kWh of electrical energy, plus 17.4 GJ of energy in the form of fuel.

2.1.4.3 Production of aluminium sections and pipes

Various shapes of aluminium section and pipes are used in different applications, such as in car production and construction. They are made in extruders.

The whole process includes stages for melting, continuous casting, sawing, preheating, extruding, and final annealing. According to /2.1-11/, one tonne of semi-finished aluminium product takes 1,030 kWh of electrical energy and 16.5 GJ of energy in the form of fuel.

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2.2 Copper

2.2.1 Description of the process stages (see Fig. 2.2-1)

The raw materials for the production of copper vary according to type and origin. They therefore also determine to a large extent the process to be applied. About 60 per cent of all commercial copper is made from copper-bearing ores with a relatively low copper content of one to 2.5 per cent copper /2.2-1/. According to the mineralogical condition, a distinction is made between sulfide and oxide ores, which, usually, also contain fractions of other noble metals (Ag, Au, Pt) or base metals (Fe, Pb, Sn, Zn). For haulage reasons, copper mills are usually near the mine. The remaining 40 per cent of commercial copper is made from old metal. This includes directly remelted process scrap (recycled material) and mainly copper-containing scrap; about three-quarters of the copper consumed return to the smelter as scrap after a service life of about 30 years /2.2-2/. Due to this fact there is a rise in the share of scrap being used as a starting material in copper smelting.

There are basically two completely different categories of copper extraction. Most of the commercially pure copper, ie about three-quarters, is produced in pyrometallurgical processes which are based on first melting the raw copper, and then refining the resulting product. In contrast to the pyrometallurgical process, the hydrometallurgical process uses solving methods combined with electrolytic recovery.

Commercial copper is available in several grades, according to the degree of purity. With about 80 per cent, the major share is cathode-electrolyte copper with a copper content of at least 99.90.

In the subsequent remelting stage, various elements are added to obtain the mixture that is tailored to its future usage. Various copper alloys are produced, such as brass as a Cu-Zn alloy, bronze as an alloy of Cu and Sn, Al, or Pb.

In an appropriate casting process, wire bars, rods, or slabs are produced. As a rule, they are then shaped in a series of hot and cold working stages.

2.2.2 Mining and dressing of ores

About 80 per cent of copper ores is mined in open pits. Copper-bearing beds are blasted loose, and the ore is transported to a dressing plant. There, the rock is first crushed and finely ground. This is followed by an ore concentration process using a floatation method. During this process, the finely ground minerals or stones which are dispersed in water, deposit in gas bubbles, depending on their specific properties. They then appear at the surface in the form of charged froth. In this way, about 90 per cent of copper is obtained from rock. The concentrate, or fines, contains 15 to 40 per cent copper and 15 to 30 per cent iron, in addition to other metals that are present at lower concentrations.

Some U.S. American sources are available to provide some indication on the energy required in the mining and dressing of ores /2.2-1, 4, 5/. However, there are wide discrepancies in the figures presented. Table 2.2-1 lists the values /2.2-3/ relating to the fuel and electricity inputs in the various stages and are based on one tonne of ore containing one per cent of copper.

2.2.3 Pyrometallurgical processes

Sulfide Cu-concentrates are usually smelted in pyrometallurgical processes. In principle, these processes are based on progressive oxidation to separate iron and sulphur. The iron forms a slag, while the sulphur is emitted with the waste gas in the form of SO_2.

2.2.3.1 Standard process

The most important standard process applied until the 1960s comprised three stages, ie roasting, smelting, and blasting.

During roasting, the concentrate is oxidised to remove some of the sulphur and to obtain a quantitative ratio between Cu and S that suits the subsequent smelting stage. Depending on the required consistency of the material (either lumpy for smelting in a shaft furnace, or powder for smelting in a reverberatory furnace), sinter-roasting is done in Dwight-Lloyd sinter machines, or fine ore roasting is done in storey furnaces comprising between 7 and 14 hearths /2.2-6, 7/. The roasting temperature is about 800° C. Whenever the S-content of the initial concentrate is over 25 per cent, the exothermic oxidation processes make fuel-firing superfluous. In this case, the roasting is self-acting. In all other cases it may be necessary to use additional firing, usually coal dust. It is possible to produce steam for utilising the waste heat in a waste heat boiler.

In the following stage, the smelted Cu matte appears in a homogeneous phase and consists mainly of CuS and FeS. It has a temperature of 1,185° to 1,300° C. Above it is the slag layer which consists mainly of SiO_2 and CaO. The types of furnaces used are as follows: shaft furnace, reverberatory furnace, electric furnace.

The shaft furnace is used very little nowadays and is suitable for lumpy material and small throughputs in the range of 100 to 500 tonnes per day. The only fuel for shaft furnaces is coke, with the consumption being 12 to 15 per cent of the fines throughput, ie about 3.4 to 4.3 GJ/t feed.

The reverberatory furnace is the most widely applied smelting unit for Cu matte. It is suitable for treating finely powdered material at large throughputs of between 300 and 1,100 t/d. It is fuelled with coal dust, fuel oil, or natural gas, with the specific consumption being a function of the mode of operation of the furnace. If hot roasted fines are charged (at a temperature of about 500 to 650° C) the consumption is about 3.0 to 4.5 GJ/t, if wet unroasted fines are charged it is about 6.0 to 7.0 GJ/t relative to the quantity fed. Through a waste heat boiler about 30 to 45 per cent of the calorific value of the fuel input can be recovered for the generation of steam.

When low-cost electricity is available it may be profitable to smelt the Cu matte in an electric furnace. This is a low-shaft furnace with 3 or 6 Söderberg electrodes that dip into the slag floating on the melt. The major share of heat is generated through current passing through the slag, because the electrical conductivity of slag is much lower than that of the melt. One of the main advantages of electric furnaces is that the waste gas emission is significantly lower than in reverberatory furnaces relative to the feed quantity; and at more than 10 per cent, the SO_2 content is by one order of magnitude above that of waste gases emitted by reverberatory furnaces. This also accounts for the greater efficiency in scrubbing the gas emitted by electric furnaces /2.2-8/. Between 400 and 700 kWh is consumed for each tonne of feed.

The Cu matte obtained from the furnace is then overblown with air in a converter. Usually, the first matte is converted to contain 75 per cent Cu, and in a second stage it is converted to 97 or 99 per cent blister copper. No heating is required because the exothermic oxidation reactions liberate sufficient energy to maintain the required temperature of over 900° C. When low-Cu matte is converted there may even be an excess of heat which then requires the addition of adequate amounts of cooling scrap.

2.2.3.2 Flash smelting

This process combines the roasting and the smelting stage, with the heat released in the exothermic roasting reactions being available for smelting. /2.2-9/ states that about 10 per cent of the required heat input needs to be covered by fuel.

There are basically two process lines. In the 'Outokumpu process' the dried ore fines are charged to the smelting shaft together with preheated air. In the shaft the reactions occur. Both the matte and the slag are separated in a special settling hearth. The waste heat is used for preheating air and for generating steam, as the case may be. Its SO_2- content of 10 to 18 per cent suggests a favourable basis for making it into sulphuric acid.

The 'Inco process' uses technically pure oxygen instead of air. About 240 kg of pure oxygen are used per tonne of ore fines. Reactions take place in the horizontal gas flow. The waste gas consists of up to 75 per cent of SO_2. Because of unfavourable settling properties in both processes, slags are relatively rich in copper. Therefore, the smelting process is interrupted at regular intervals to reduce the CU-content of the slag by blowing in a special silica concentrate.

While the 'Inco process' did not survive because of the high consumption of technically pure oxygen and related costs, the 'Outokumpu process' has become widely applied in the 1970s.

2.2.3.3 Direct reduction processes

In the Brixlegg process, the sulphur contained in the concentrate is first removed to a mere one per cent in a fluidised-bed furnace. Then quartz and lime are added, as well as 8 to 10 per cent of reduction coal, and in a tank furnace with Söderberg electrodes it is smelted to 'black copper' with a Cu-content of 97 per cent. The consumption of electrical energy is higher than in matte smelting in an electric furnace. /2.2-8/ states a figure of 635 kWh per tonne of raw material (which corresponds to about 2.3 GJ).

Recent developments in the field of pyrometallurgical production of copper have led to the Cu-concentrates being directly processed to blister copper.

All processes have one feature in common, and that is the sequence of metallurgical stages in three reaction zones for smelting the charge under oxidising conditions, for converting molten sulfide into blister copper and slag through further oxidation, and finally, for smelting the copper-rich slag down to a residual lean content of about 0.5 per cent.

The differences between the various processes lie mainly in their process engineering concept. This may either be based on combining the three zones in one furnace, with the material and the gas being carried either in a counter-flow or in a joint forward flow, or else in a spatial separation of the zones in different furnaces which are interconnected.

2.2.3.4 A comparison of energy inputs in various pyrometallurgical processes

/2.2-12/ presents a comparative calculation of five different pyrometallurgical processes used to concentrate blister copper from wet ore fines:

- Conv.: conventional matte smelting in a reverberatory furnace;
- El. matte: matte smelting in an electric furnace;
- Brixlegg process: direct reduction in an electric furnace;
- Outokumpu process: flash smelting with air;
- Worcra process: direct reduction in one furnace unit, with low-copper slag being produced in the same process.

For the purpose of this comparison, a uniform basis has been established in relation to plant capacity (nominal throughput of 1,000 tonnes of fines per day) and capacity utilisation (100 per cent), for the composition of the concentrate feed, and for the location at a central European waterway.

Table 2.2-2 shows the various values relating to the energy consumption per tonne of concentrate feed. The figures refer to all stages of the respective process, as well as gas scrubbing and sulphuric acid production. Also considered was the recovery of waste heat and, whenever possible, its contribution to the production of electricity.

2.2.4 Copper refining

The blister copper produced in one of the pyrometallurgical processes contains between 1 and 10 per cent of impurities in the form of metallic elements, and oxygen. All elements which, because of their affinity with oxygen, volatilise or slag through selective oxidation, are separated through pyrometallurgical refining. In order to obtain copper of high electric conductivity, other impurities of Ag, An, Se, Te, Pb, Ni, etc. have to be removed through electrolytic refining. As a rule, blister copper is subjected to both refining processes.

2.2.4.1 Pyrometallurgical refining

This refining method consists of two periods. In the oxidation period the impurities are removed. First, the blister copper is melted (bath temperature of about 1,200° C) and then converted with air. The impurities that oxidise during this stage, form a slag unless they are dissolved in the Cu-bath or vapourise. The slag is removed from the bath at the end of the oxidation period. In the subsequent reduction period, the newly formed fraction of Cu_2O dissolved in Cu is reduced by dipping green and/or wet birch or beechwood, following which the bath is covered with charcoal or coke (pitch-toughening). The complete process, including pouring, takes about one day. The most commonly used size of reverberatory furnace is 20 to 400 tonnes. For smaller charges, either rotary kilns or drum furnaces with a capacity of 8 to 30 tonnes are used. They are fired with coal dust, oil, or gas. The specific consumption per tonne of refined Cu is about 3.0 to 3.7 GJ. About 40 per cent of that amount can be recovered from the waste gas with the aid of a waste heat boiler.

Added to that is the requirement of 12 to 15 beechwood logs of 8 to 12 m length for pitch-toughening a charge of 250 tonnes.

2.2.4.2 Electrolytic refining /2.2-10, 11/

In electrolytic refining, the copper to be refined is used as an anode and is cathodically separated by applying a voltage. The electrolyte is a solution of H_2SO_4 and $CuSO_4$. The impurities contained in the anode copper are not dissolved but settle in what is called the anode slime, provided they are more precious than Cu in terms of their electrochemical series. However, the less precious components, like the anode copper, dissolve in the electrolyte. They do not deposit on the cathode but settle as slime at the tank base, largely after hydrolysis or precipitation.

The share of H_2SO_4 in the electrolyte helps lower the ohmic resistance and improve deposition on the cathode. For the same reasons, the bath temperature is raised to 60 or 65° C. Since electric heat losses in the bath are insufficiently high to maintain the temperature, auxiliary heating by steam or hot water in an external heat exchanger is normally required. This external heating requires constant circulation of the electrolyte and leads to a narrowing of the difference in concentrations between the areas adjacent to anode and cathode. In so doing, the voltage drops caused by polarisation are kept low, which has a positive effect on the electrical input. However, it needs to be noted that the rotational speed of the electrolyte cannot be increased ad infinitum. If cycle times of the electrolyte are under one hour, there is no way of ensuring that the anode slime will settle at the base of the tank.

The central process parameter is the cathodic current density. The higher the current per square metre of cathode surface, the more intensive is the Cu-turnover from anode to cathode. Therefore, the production output of a plant of a given size increases with the current density applied.

On the other hand, higher current densities have their disadvantages, too. The bath potential needs to be higher because both the polarisation voltages and the voltage drops at the ohmic resistance of the electrolyte increase. This causes the specific consumption of electrical energy to rise. It is true to say that the energy required for the auxiliary heating is cut at the same time, but due to different rates being paid for different forms of energy higher current densities also involve higher energy costs.

Another adverse effect caused by higher current densities is the rise in co-precipitation of impurities at the cathode. This also means that the current density is limited by the required purity of the cathode copper. Moreover, a large part of the precious metals (mainly An and Ag) contained in the anode copper cannot be recovered from the anode slime which, again, has an unfavourable effect on the costs if the current density is increased.

Two factors have a positive bearing on the electrical input at a given current density (in addition to the effects on bath

temperature and electrolyte circulation. These factors are
- maximum purity of the anode copper, and
- a higher Cu-content of the electrolyte.

This goes to show that there is a complex interaction between the various factors. The optimum plant design has to be determined in each individual case, especially with a view to the price level of electrical energy. In the 1970s, the general trend in plant development was for higher current densities, with some of the increase in the specific consumption of electricity being compensated for by engineering and operating improvements.

Modern conventional plants operate at current densities of 250 A/m^2; the specific consumption of electricity is between 220 and 280 kWh per tonne of refined copper.

A significant increase in the current density to a maximum of about 400 A/m^2 can be achieved by applying PCR electrolysis (PCR = Periodic Current Reversal). In this process, the current is reversed for short periods, which has a positive effect on various factors. Some of the differences in concentration at the electrode surfaces are levelled out, thus limiting polarisation effects; passivation is avoided, and the deposition of impurities at the cathode is largely prevented. Established plants can be retrofitted with the PCR method, and plant throughput can be enhanced considerably. Thus, the specific consumption of electricity at the Montanwerke Brixlegg (Austria) rose to about 350 kWh/t Cu, with a rise of effective current density to roughly 300 A/m^2. The consumption of steam was still about 1.1 t/t Cu which corresponded to a requirement for auxiliary heating in the order of about 2.6 GJ/t Cu.

2.2.5 Hydrometallurgical processes /2.2-7, 13, 14/

Hydrometallurgical processes are mainly used to smelt oxide copper ores and burnt ore. However, they are also gaining in importance for processing sulfide ores because of environmental aspects.

The main steps of the process areas follows:

- Roasting, to turn copper into leachable compounds. A distinction is made between sulfatising roasting to turn CuO into soluble $CuSO_4$ and chlorinating roasting for processing burnt ores, in which case the iron contained in the burnt ores need to be made into insoluble compounds. Roasting is done in storey furnaces or fluidised-bed furnaces at temperatures of 500 to 600° C.

- Extraction, to separate completely Cu and other extractable metals from the gangue. Extraction at elevated temperatures improves the rate of dissolution and the yield, but it also causes additional cost for heating. Therefore, it is used preferably for the more complex ores which produce various metals and saleable by-products.

- Reduction of Cu enriched in the solution by
  - cementation, or
  - electrolysis.

In cementation, Cu is precipitated by adding iron scrap. Precipitation is accelerated by heating and circulating the solution. Cement copper, ie precipitated copper, contains usually 8 to 12 per cent of impurities, including ore particles, residual precipitants, insoluble components which have to be removed by elutriation or floatation. Refining is required in the subsequent stage.

To distinguish electrolytic reduction from electrolytic refining, the former is usually also referred to as electrolytic recovery. In this process, anodes consist of insoluble material, such as hard lead. By applying a voltage of 1.9 to 2.3 V per cell, Cu^{++}-ions are precipitated from the electrolyte in the form of cathode copper. The lean copper solution is then recirculated to the extraction stage, apart from some elutriated quantities.

In the course of the extraction process, a number of elements accumulate in the electrolyte which interfere with electrolytic recovery. Fe^{3+}-ions reduce the current efficiency to a very large extent, thereby raising considerably the specific consumption of electricity. Cl^--ions and NO_3^--ions speed up anode wear and favour corrosion of metal equipment. Other metallic elements of the electrolyte may deteriorate notably the quality of cathodically separated copper.

There are two ways of overcoming these problems:

- Either the cathodic current density is restricted to a range between 85 and 150 A/m^2, which used to be done until the early 1970s. Based on normal current efficiency between 75 and 90 per cent, the specific consumption of electricity of such plants is usually between 2,000 and 2,200 kWh/tonne of cathode copper.

- Or the copper contained in the solution is selectively made into an optimum electrolyte. This aim has been pursued since the early 1970s when solvent extraction came to be used. Solvent extraction denotes the separation of dissolved substances from liquid material, using liquid extractants. In hydrometallurgical Cu-production, solvent extraction offers a way of concentrating the low-copper solutions, or respectively, to isolate copper from heavily contaminated solutions. This in turn permits operation at much higher current densities (up to 350 A/m^2) in electrolytic recovery without impairing the purity of the resulting cathode copper. It is true to say that additional costs are involved for the design and operation of the solvent extraction plant, and the specific consumption of electricity is raised by about 8 to 10 per cent. However, much of the cost can be saved by employing much smaller electrolytic plants relative to the throughput. Solvent extraction, therefore, opens up new applications in electrolytic recovery which were of no interest before because of the high cost involved. This is particularly true of copper extracted from sulfide ores, where hydrometallurgical processes compete with pyrometallurgical processes, and whenever exacting criteria are applied to pollution control.

2.2.6 Casting and manufacture of semi-finished products

Melting and alloying of Cu is mostly done in crucible furnaces and hearth furnaces. They are heated by gas, oil, coke, or electricity. The specific energy consumption differs according to the casting temperatures and calorific capacities of copper and its alloys.

The energy efficiency of fuel-fired furnaces is 10 to 25 per cent for melting; in electric furnaces it is in the range of 50 to 75 per cent /2.2-15, 16/.

The type of furnace used is of particular importance. Channel-type induction furnaces have an inherently higher efficiency than induction furnaces of the coreless type. It is a disadvantage of the channel furnace that the channel has to be kept hot with a liquid 'heel'. Moreover, the mixing condition for the furnace charge is not as favourable as in the induction furnace of the coreless type with its typical bath circulation. There are also differences in the power/capacity ratio /2.2-17/:
- low-frequency crucibel furnaces: 200 kW/t,
- medium-frequency crucible furnaces: 600 kW/t,
- channel-type furnaces: 80 kW/t.

In induction furnaces, the rate of metal loss by oxidation, is only half or one-quarter that of fuel-fired melting furnaces. This is due to the even heat distribution and the relatively low temperature at the top of the bath in the induction furnace.

An additional bonus of the electric furnace is its low pollution and simple operation which help create a favourable working environment for operators. Since there are no combustion gases it is an easy thing to effectively clean the waste gases removed from the furnace chamber that carry the impurities contained in the charge.

Semi-finished products are usually manufactured in a number of hot and cold working stages. Depending on the formats to be handled, one of the following units is required for heating to a hot working temperature of 850 or 950° C /2.2-6/:
- oil or gas-fired chamber furnace, 15 to 50 per cent efficiency;
- resistance-heated furnace, 30 to 60 per cent efficiency;
- oil-fired, gas-fired or electric continuous furnace,the efficiency being usually higher than in chamber furnaces;
- induction holding units for ingots, slabs and billets; 40 to 60 per cent efficiency.

Finally, the following types of heat treatment are available for Cu and Cu-alloys:
- homogenisation of cast slabs and bolts at about 800 to 950° C in order to obtain a uniform structure for further processing;
- soft annealing above the recristallisation level at transient temperatures of 550 to 620° C in order to obtain an even, fine-grain structure;
- stress relief annealing at about 200 to 300° C in order to remove cold draw stress after cold working.

In general, these types of treatment can be handled either in batch-type furnaces or in continuous furnaces, depending on the size and shape of products to be heated and the throughput required. They are heated with fuel or electricity. For homogenising and soft annealing, salt-bath furnaces can also be used. They feature good sealing properties; however, their specific energy consumption is much higher than that of furnaces which operate on gaseous atmosphere or vacuum.

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2.3 Other non-ferrous metals

2.3.1 Nickel

A rough survey of the various methods of extracting nickel is given in Fig. 2.3-1. The group of sulfide Ni-ores appears mostly in Canada and the U.S.S.R. Its share of the total nickel output has decreased in favour of oxide-silicate Ni-ores which make up the major share of Ni-resources mined mainly in New Caledonia, Cuba, and Indonesia.

Usually, Ni is extracted from sulfide ores by pyrometallurgical methods, including the following stages /2.3-1, 2/:

- ore dressing: crushing, magnetic separation, or floatation;
- pre-roasting in a storey furnace, or in combination with sintering in a Dwight-Lloyd machine;
- smelting the ore to matte (15 to 24 per cent Ni) in a large reverberatory furnace at about 1,550° C (coke consumption 3.4 to 4.5 GJ/t charge) or in a water-jacketed shaft furnace (coke consumption 2.0 to 3.7 GJ/t charge) or in an electric arc furnace (electricity input about 860 kWh/t charge);
- overblowing to converter matte with about 70 per cent of nickel (no fuel consumed);
- further processing of converter matte: roasting and thermal reduction (partially under high pressure) according to various methods;
- electrolytic refining in Hybinette cells with a diaphragm between anode and cathode (electrical input about 2,700 kWh/t Ni) or in a sulfate-chloride process developed in the 1970s, which has an electrical input of only about 1,800 kWh/t Ni /2.3-3/.

In addition, there are ways of extracting Ni directly from sulfide ores using an ammoniacal leach with excess pressure as in the Sherrit-Gordon process, or without excess pressure as in the Inco process, or by sulphuric acid leach. The latter may also be combined with solvent extraction /2.3-4/. No mention of its energy input is made in literature, however.

Conventionally, Ni-extraction from oxide-silicate ores passes through a matte stage, with the coke required for smelting the matte in a water-jacketed shaft furnace being relatively high at about 8.5 to 11.5 GJ/t charge. Another 4 to 4.5 GJ/t charge are required for the preceding sintering stage. This is consumed in the form of coke and fuel oil. NiO, which is produced in the roasting stage, is then reduced. Instead of the reverberatory furnace, the electric furnace is now more widely applied for reduction. It uses graphite electrodes, and its electrical input is between 2,200 and 2,700 kWh/t Ni.

A number of other methods for direct reduction of oxide-silicate ores have either been developed or optimised. They include the Krupp-Renn method with a coke consumption of about 5.5 to 6.5 GJ/t ore, and the electrothermic direct reduction to FeNi in an arc furnace with electricity consumption being very much a function of the Ni-content of the ore. Data published in /2.3-2/ relate to two plants and state about 2,000 kWh/t ore containing about 10 per cent Ni and over 6,000 kWh/t ore containing only two per cent of nickel. Reduction coal in the form of coke is consumed in addition to graphite electrodes.

In the various processes nickel is obtained in the form of pure nickel, metallic nickel powder, nickel oxide, and nickel alloys. Only part of the pure nickel is processed as such; rather, like nickel oxide and nickel powder, it is used for melting nickel alloys.

The largest nickel user is the steel industry which requires large quantities of alloyed nickel for the production of stainless steel. Nickel alloys play a major role in the manufacture of aircraft, in space and rocket engineering, and in nuclear technology.

2.3.2 Zinc

Fig. 2.3-2 shows the most important stages of extracting and processing Zn. Extraction is mostly based on sulfide Zn-ores which are first concentrated through selective floatation and then roasted. This stage yields about one tonne of sulphuric acid per tonne of blende concentrate. Therefore, marketing outlets and ways of utilising the sulphuric acid (in fertilisers, for instance) may decide upon the siting and the profitability of zinc works.

At present, the following plants are mainly used for roasting:
- fluidised-bed furnaces without auxiliary heating for fine ore roasting in hydrometallurgical extraction (electrical input about 30 kWh/t blende);
- Dwight-Lloyd machines for sinter-roasting in pyrometallurgical extraction (fuel consumption 0.5 to 2 GJ and electrical input 40 to 60 kWh/t blende) /2.3-2, 5/.

The most important stage of extraction in terms of energy requirement is the reduction of ZnO, which may be either pyrometallurgical or hydrometallurgical.

In pyrometallurgical extraction of one tonne of vaporous zinc, and according to the endothermic reaction,

$$\mathrm{ZnO} + \mathrm{C} = \mathrm{Zn} + \mathrm{CO}$$

about 5.5 GJ of theoretical reaction heat is consumed, plus 185 kg C. These theoretical values are by far exceeded in actual practical application, as is shown in Table 2.3-1 /2.3-1, 2, 5, 6/. The fuel consumption quoted (most of it is in the form of coke, or sometimes special grade of coal) also include the amounts of coal dust required for reduction. The first three methods are non-electric, in the remaining methods electrical energy is used for generating heat (in the Sterling method through electric arcs, in the methods applied at Josephstown and Duisburger Kupferhütte through direct resistance heating). In 1978, electric reduction methods held a share of about 6 per cent of Zn-extraction worldwide, while non-electric methods accounted for about 12 per cent. However, good prospects for a wider application exist only for the process used at Imperial Smelting Co. (UK).

Most pyrometallurgical extraction methods produce spelter, or crude zinc, which is adequate for corrosion protection (zinc-plating), but inadequate for diecasting, its second most important application. For this purpose, crude zinc has to be refined to high-purity zinc of at least 99.95. To obtain this purity, fractional or plate distillation and condensation is performed in stripping columns. This requires a fuel input of 8 to 9 GJ and an electrical input of about 20 kWh/t high-purity zinc.

However, hydrometallurgical methods produce fine Zn that does not need subsequent refining. Cathodic separation of compact Zn from sulphuric acid electrolyte only succeeds if the negative normal potential of Zn^{2+} is exceeded by a higher overvoltage of H^+. In order to reach this objective at a satisfactory current efficiency of about 90 per cent, the electrolyte needs to be free of contaminants and impurities. Therefore, after leaching, where the roasting product ZnO is dissolved in sulphuric acid, the contained Fe is precipitated /2.3-9/. Following that, the solution is clarified through cementation with metallic Zn of Cu, Ni, Co, and Cd. The subsequent electrolytic stage requires high current densities of about 300 to 1,000 A/m^2 and a temperature of the electrolyte cooled to 30 to 40° C. The specific consumption of electricity in electrolytic recovery is normally around 3,100 to 3,400 kWh/t fine zinc /2.3-6, 7/, in some cases slightly less /2.3-8/.

The resulting Zn cathodes are then remelted at temperatures of up to 480° C max. (energy requirement in fuel-fired hearth furnaces about 1.2 to 2.0 GJ/t Zn, in induction furnaces 100 to 120 kWh/t Zn) and cast into slabs. In the 1970s, the hydrometallurgical Zn extraction came to be the most widely applied process worldwide. Its share is assumed to be over 80 per cent of total Zn production /2.3-3/.

2.3.3 Lead

Lead is extracted from ores containing lead in the form of lead glance, PbS. This is done by pyrometallurgical methods, ie the roast-reduction method, apart from some special cases where other methods are used. After crushing and fine grinding, the ore is concentrated through selective floatation to a sulfide concentrate of 60 per cent Pb on average. Then it is subjected to a sinter-roasting stage, which has no remarkable fuel consumption. In sulphur-rich ores pre-roasting may be required, in storey furnaces for instance, in which case 1.5 GJ of coal per tonne of concentrate have to be added /2.3-1/.

Crude lead (96 to 99.5 Pb) is obtained in reductive melting of the sintered material in the shaft furnace, with coke being used both as a fuel and a reductant. The specific consumption of coke is about 2 to 4 GJ/tonne of sintered material /2.3-1/.

The impurities contained in the work lead, or crude lead, are removed by refining. The most widely used method is pyrometallurgical refining which comprises a number of subsequent stages. These are /2.3-10/ as follows:

(a) removal of Cu, Ni, Co, by stirring the melt as it cools;

(b) removal of Te, Sb, As, Sn either by selective oxidation at 750° C in a reverberatory furnace (fuel consumption about 0.6 to 1.2 GJ/t Pb), or according to the Harris process, ie treatment with molten alkali salts at about 400° C (no remarkable fuel consumption because of exothermic reactions);

(c) removal of Ag by adding Zn on top of the cooling bath;

(d) removal of Zn which is done mostly in vacuum distillation at about 600° C nowadays;

(e) removal of Bi, depending on the Bi-content either by adding alkali salts and/or alkaline-earth metals or by electrolysis in a Betts process (specific consumption of electricity about 110 to 130 kWh/t Pb /2.3-1/).

/2.3-5/ states a specific fuel consumption in lead refining of 1.6 GJ and a specific consumption of electrical energy of 25 kWh/t Pb.

In highly industrialised countries, major shares of lead production are based on recycled scrap. This is either added to the charge of the reduction shaft furnace, or directly molten in refining.

The soft lead obtained in refining is molten and cast into ingots. For subsequent processing, the ingots are molten again to produce, for instance, alloys with Sb (hard lead) or Sn, which are then cast. The most favourable temperature for casting is between 250 and 400° C, depending on the alloy. Melting can be done in crucible furnaces or tank furnaces; they are gas or oil-fired, or electric. For electric furnaces /2.3-1/ quotes a specific consumption of electricity of 30 to 50 kWh/t Pb.

2.3.4 Tin

Fig. 2.3-3 shows the major stages in winning and processing tin. By far the most important tin-bearing mineral is the SnO_2-bearing cassiterite, which covers more than three-quarters of primary tin production. If the enriched concentrate has an S-content of over 0.5 per cent, it is roasted in hearth furnaces or rotary kilns, with the specific coal consumption being about 1.5 to 3.5 GJ per tonne of concentrate /2.3-2/. If some metal content is excessive, chloridising, oxidising, or roast-reduction may be indicated.

To obtain Sn-bearing concentrates from poor and/or complex starting materials, a volatilisation method is used. This is based on sulphurisation (if necessary) followed by re-oxidation. The fuel requirement in rotary kilns and/or shaft furnaces (coke only can be used!) is between 5.5 and 7 GJ per tonne of load. Tin is then found as SnO_2 in airborne dust which is clinkered in a rotating reverberatory furnace at a temperature of 900 to 950° C, in other words, it is clotting to form large particles which are then roasted. In addition to sulphur, As, Pb, and Bi are also partially removed.

Primary tin is won from its oxide by pyrometallurgical processes. Endothermic reduction at a temperature of about 1,200° C requires a theoretical 2.1 GJ of reaction heat, plus 150 kg C per tonne of Sn. Actually, the expenditure of reduction coal in the form of low-ash anthracite or low-temperature coke is up to 200 kg/t Sn. The actual energy consumption may be between 3.5 and about 20 GJ per tonne of tin in reverberatory furnaces which are the main type of furnace used. Consumption is a function of the size of furnace, the type of fuel used and the utilisation of waste heat for preheating the combustion air. For the short drum furnace and rotating short reverberatory furnace (which has a number of process engineering advantages) /2.3-2/ quotes a specific consumption of fuel oil of about 7 GJ per tonne of tin. More recent electric arc reduction furnaces consume about 1,000 to 1,300 kWh per tonne of tin, as well as 5 to 8 kg of self-baking electrode carbon.

The Sn-yield from the reduction of ore is only about 90 per cent if a certain iron content in the crude tin is not to be exceeded. The remaining tin is contained in the slag; it is processed in a reverberatory kiln at a temperature of 1,350 to 1,450° C, with about 350 kg of lime and 120 to 180 kg of reduction coal being added /2.3-1/ to obtain Sn. Due to the high temperature and the rather long processing time, the fuel consumed in firing is several times that of ore reduction in a reverberatory kiln (relative to the material input in either case). For both processes combined, ore reduction and slag reduction, in /2.3-2/ the normal fuel expenditure is stated to be about 8 to 9 GJ of fuel and about 220 kg of reduction coal per tonne of total Sn produced, provided the starting ore fines contain 70 per cent of Sn.

The impurities contained in the crude tin are mostly removed by pyrometallurgical refining or electrolysis. Pyrometallurgical refining is divided into two stages: liquation and poling. If the crude tin contains more than some per thousand of Fe, then the crude tin must be maintained in the liquation furnace at somewhat above its melting temperature of 232° C, with Fe and other impurities forming slag. /2.3-2/ quotes a fuel input in liquation of between 7 and 18 GJ per tonne of tin. Waste gases from the liquation furnace are used to heat kettles that contain the Sn melt at a temperature of 300 to 400° C. Air or steam is blown into the melt ('poling') to oxidise the residual Fe and other impurities which can be skimmed off.

If the crude tin contains large amounts of Ag, Bi, Pb, or Sb, no satisfactory refining effect can be achieved in pyrometallurgical refining. In this case electrolytic refining is applied. Crude tin, or primary tin, is used as an anode in a Na_2S-solution which is maintained at 90° C by additional steam heating. Refined Sn is separated at the cathode; its purity of 99.98 at the minimum. However, some 2.5 per cent of the anode Sn input remains in the slime together with other impurities that have been removed. By separate processes, Sn can be recovered from the bath slime. With a current efficiency of 95 per cent, the specific consumption of electricity in electrolytic refining is about 180 kWh per tonne of tin.

Refining produces cathode slabs which are melted and cast into ingots. Crucible furnaces and/or tank furnaces are used for casting, as well as for remelting and alloying. The specific consumption per tonne of molten material is between 35 and 65 kWh in electric furnaces, and 0.2 to 0.4 GJ for oil or gas-fired furnaces.

The main applications of tin include surface protection tinning (usually galvanising) and the manufacture of metal alloys.

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3. CONSTRUCTION MATERIAL INDUSTRY

3.1 Cement industry

3.1.1 Process description *

The production of cement is divided into the following stages:
1. raw material winning ,
2. beneficiation,
3. calcination,
4. grinding and mixing of cement.

There are two different processes, the wet process and the dry process. In the wet process, the solid material is first crushed, then wet ground in drum or ball mills, following which they pass on to bowl classifiers or screens in the form of slurry containing 30 to 40 per cent of moisture. The slurry is pumped to equalising tanks where the final blend and mixture is made which is then fed into the kiln. In the dry process, the material is crushed, and then the dry powdered material is filled into the kiln with water being added. All other stages are the same in both processes. Fig. 3.1-1 is a process flow diagram illustrating the manufacture of cement according to the two processes.

3.1.1.1 Raw material winning

Although the raw material used for making cement is the same for all types of cement, they may vary in composition depending on the finished product to be manufactured. These varied mixtures and the great number of additives and admixtures which are used to make an end product have created a wide range of commercial cement grades, each of which has its specific properties and applications. The raw materials required for cement production must contain usable quantities of lime, SiO_2, and Al_2O_3. Natural calcium deposits, such as limestone

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* Quoted almost literally from "Historical Trends and Future Projection for Energy Consumption in the Cement Industry" /3.1-1/

and coquina, supply lime. Natural argilliferous deposits, such as clay and slate deposits, contain both SiO_2 and Al_2O_3. Natural deposits of argillaceous limestone or marl (a calcarious clay) supply all three base materials, although possibly not necessarily in the required condition. In some places in the north-east of the U.S.A., a special type of limestone is quarried, called cement stone, which is used for making cement simply by heating it to a high temperature and grinding it afterwards.

3.1.1.2 Beneficiation

The material is transported from the quarries to the cement works where it is crushed in various crushers and then ground. Crushing is generally done in two stages: first, crushing into lumps of 15 cm in diameter, then fine crushing to under 2 cm in diameter. Before the materials are reduced further by grinding they are stored. In the wet process, which is normally used for clay and marl, ie materials that are rather wet in themselves, the materials are then filled into ball or drum mills where they are ground to a grain size of under 0.1 mm in diameter. Water is added during grinding to obtain a thin slurry. This slurry, which contains more than 30 per cent of water, is blended in storage tanks until ready to be taken to the kiln. In the dry process, the raw materials are filled into the mills in the desired composition and ground to powder. Often, the raw materials are dried before grinding, sometimes using waste gases, then mixed and stored in hoppers or silos.

3.1.1.3 Calcination

Rotary kiln process

The central process in cement making is calcining the raw material in a rotary kiln. The kiln is a long cylindrical, brick-lined furnace chamber that rotates slowly around its axle and is slightly inclined.

Today's commercial kilns of this type are 70 to 240 m long and up to 7 m in diameter.

After having passed through all preheaters, the raw material is fed into the top of the kiln whence it moves slowly in a counterflow towards hot gases and the bottom end, while the kiln is rotating slowly. The kiln is fired with gas, oil, or coal dust. While moving through the kiln, the temperature of the material rises to the level required for essential reactions to occur. The throughput of the kiln is monitored, and the temperature of the raw material is maintained at the level at which the raw material combines with lime, SiO_2, and alumina to form the component characteristic of portland cement. The typical dwell time is 3 to 5 hours, during which time the raw material heats up to 1,400 or 1,450° C at the lower end of the kiln (see Table 3.1-1). The calcined material leaves the kiln in the form of coarse lumps or grains of a diameter of between 0.15 and 1 cm.

The resulting clinker is cooled down, with the waste heat being utilised to preheat the combustion air. Various clinker coolers are commercially available. Some use a travelling grate, others are fitted with pipes close to the clinker discharge at the brim of the kiln. The clinker drops into the pipes and rolls inside them towards the outlet. Air is normally introduced into the pipes by induced draught. It is preheated and serves as combustion air.

As a rule, the daily output is 3,000 to 4,000 tonnes of clinker, but capacities of 6,000 to 8,000 tonnes per day are in no way rare /3.1-2/.

Shaft furnaces *

Shaft furnaces offer another way of firing clinker. Shaft furnaces consists of refractory-lined vertical cylinders of 2 to 3 m in diameter

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* Quoted almost literally from "Zementtaschenbuch" /3.1-3/

and 8 to 10 m in height. They are charged from the top either with pellets of raw meal and fine grain coal, or coke. In the somewhat enlarged top section, the material passes through a short sintering zone. It is then cooled by the combustion air blown in from the bottom and leaves the kiln in the form of clinker through a discharge at the lower end.

The daily output of shaft furnaces is under 300 tonnes of clinker at a heat requirement of 3.1 to 4.2 GJ/t. Therefore, they are only profitable in very small works, which also explains why their number is decreasing rapidly in the highly industrialised countries.

3.1.1.4 Grinding and mixing of cement products

The last step in the production line is grinding the clinker lump to fine powder. Normally, clinker is ground in two successive steps together with a small amount of gypsum (approximately 3 to 5 per cent in weight). The purpose is to control the curing time of concrete. Other substances are added to aid grinding, or else to give the cement specific properties, such as improved frost resistance. This kind of additive is also used in the concrete mix.

3.1.2 Energy consumption in cement making

At 90 per cent, by far the largest amount of energy in cement making goes into clinker firing. This is mostly done in rotary kilns at temperatures of 1,400 to 1,450° C. Therefore, the fuel consumed in firing has a direct bearing on the energy-intensity of the product.

This fact is clearly illustrated in Fig. 3.1-2. There is always a wide difference between actual and theoretical energy consumption in that actual consumption is usually double or, in the worst case, three times that of the theoretical input. * It is also explained by the type of process, whether a wet or a dry process is used. Since the water that is added in the wet process needs to be driven off either before the material is fed to the kiln or in the kiln itself, the wet process consumes 35 per cent more energy than the dry or semi-dry process. It is not always possible to convert the process because some of the raw material can only be treated in a wet process (Table 3.1-2).

There is reason to believe that the share of electrical energy is due to rise because more and more drives used in winning and dressing are electric. Moreover, more electrical energy is used in the dry process for crushing and grinding than in the wet process (Fig. 3.1-3).

A key factor is also the size of the works. Large plants which are using their capacities to the full, suffer smaller losses (see also Chapter 1.3.3), and they also tend to be able to afford energy conservation measures. The difference in the specific energy consumption is up to 16 per cent in the Federal Republic of Germany, for instance (Table 3.1-1).

Then, of course, the attitude towards energy consumption and different prices being paid for energy also explain why energy consumption varies between countries. The estimated world average is 8.28 GJ/t, in Malaysia it is 4.24 GJ/t, in India 8.08 GJ/t, in the U.S.A., where the wet method is applied more than in the Federal Republic of Germany, 9.09 GJ/t /3.1-7/.

* Quoted almost literally from "Möglichkeiten zur Energieeinsparung in der Zementindustrie" /3.1-4/

3.1.3 Energy conservation

Effective measures taken to save energy have helped cut significantly the specific consumption of energy in the past two decades (Fig. 3.1-4). In the past few years, consumption could not be reduced at the same pace so that in the medium term specific consumption can only be cut further at high capital investment in efficient plants. This cost, however, cannot be recuperated by the savings made in energy cost.

The common ways of reducing losses have not been applied to the process so far. For example, losses from the wall of the rotary kiln make up a major share of total energy lost. The drum has a jacket temperature of up to 350° C. It is true to say that the heat transfer is improved when the drum is insulated, but at the same time the temperature of both the metal jacket and the lining rises. This in turn reduces the mechanical strength and the life of the kiln. Moreover, there is the risk of the drum melting if the kiln is insulated and the lining fails. However, modern test facilities and monitoring of the jacket temperature seem to suggest that some kind of insulation can be considered.

The same kind of effect could be achieved by boxing-in the kiln. In this way convection heat losses can be minimised because high air velocities during wind are excluded.

By coating the kiln with aluminium bronze, for instance, radiation losses can also be cut. However, with minor radiation to the atmosphere, this kind of coating will make the temperature of the kiln jacket rise, which causes even higher convection and produces all the adverse effects mentioned before.

It becomes apparent that all external insulation is detrimental to operating safety. One way of overcoming this is to utilise heat losses rather than minimising heat transfer.

To this end, a recuperator is fitted at the top or on the side of the rotary kiln. It is heated by some of the radiation emitted by the kiln. Inside the recuperator flows a heat transfer medium, eg water which is heated. The key factors are the space between the recuperator and the kiln, and their respective temperatures. Some of the recuperator heat is released at its surface through free convection. However, this convection can be minimised by a top or side cover, or a screen.

Theoretical considerations have shown that a heat flow equal to 30 kg of fuel oil per hour can be saved with an optimised design and configuration of the recuperator, mean wall temperatures of around 250° C, an outside diameter of the kiln of 5 m, and a heat exchanger surface area or 200 m^2. This amount of heat saved can be utilised to dry coal or raw materials, or to heat buildings. The practical application of these theoretical considerations will, however, require additional research and development /3.1-5/.

Other ways of cutting the heat loss of the kiln consist in reducing the temperature inside the kiln or shortening the length of the kiln which would, however, presuppose an accelerated sintering stage. Both ways aim at altering the sintering pattern with the aid of additives. Mineraliser, as they are called, are added to the raw meal but may in some cases produce adverse effects, such as

- increased expenditure in clinker grinding,
- clinker firing problems,
- poor setting of the cement,
- delayed early strength,
- reduced final strength,
- environmental pollution through emissions of fluorine and sulphur compounds, possibly causing additional expenditure for emission control.

In view of the above, it is very difficult, indeed, to choose the best mineraliser for clinker firing because the quality standard, raw material base, technical facilities, and ecological aspects have to be taken into account.

So far test results have failed to suggest a rating of the applicability of various mineralisers in production. Most of the experimental work was done at laboratory or semi-commercial scale. Nothing is currently known of any ongoing demonstration project.*

Since presently available findings suggest that the use of suitable mineralisers opens a substantial energy saving potential, this issue should be the subject of further research and development work.*

Other than wall losses, the second loss factor is waste heat emitted from exhaust gases. In the 1950s, the cement industry first started to make direct use of this residual heat for preheating the raw materials.

Both operation and maintenance of the preheater are simple because it does not have any moving parts. At present, many different makes of preheaters are commercially available. They work in one or several stages which can be connected in parallel and/or in series. The underlying principle is a parallel flow, counterflow or cross-flow, with one feature being common to all three principles, ie the heat transfer through convection from the waste gas to the raw material or from the waste gas, via the contact surface, to the raw material.

In addition to all these energy saving methods, there is another way of minimising energy consumption by using the appropriate process, as indicated in Chapter 3.1.2. Table 3.1-2 shows that the wet process is poorer in terms of energy consumption. The 30 or 40 per cent (weight) of water which is added to the raw material during the dressing stage is to be driven off again, with energy being spent. However, mention should also be made of the fact that the moisture contained in local raw material may make it unsuitable for the dry process. In the Federal Republic of Germany only about 3 per cent of all works use the wet process.

* The section on modified sintering patterns has been quoted almost literally from "Möglichkeiten zur Energieeinsparung in der Zementindustrie" /3.1-4/.

3.1.4 Energy substitution *

Both high demand for cement and rationalisation which became imperative for economic reasons, caused cement firing plants to grow in the early 1970s. Upper limits of unit capacity are now about 3,000 to 4,000 tonnes per day and kiln. Larger kilns cause operating trouble, ie shorter lives of the refractory lining of the rotary kiln because of high mechanical and thermal stress. In an effort to realise even larger units, a new firing process is being developed in Japan and in Europe, providing for precalcination with secondary firing. In this process, the firing stage is divided into two, according to the two different temperature levels applied during dissociation and sintering. The primary firing, at the discharge end of the kiln, supplies only the heat flow that is required for sintering the neutralised material.

There are two ways of designing this type of secondary firing. First, the air required for combustion is guided through the rotary kiln to the combustion chamber, which requires the diameter of the kiln to be 20 per cent wider. In the second option, the combustion air is taken through a separate tertiary air duct to the combustion chamber (Fig. 3.1-5).
In the first case, where the air passes through the kiln during precalcination, the specific volume load rises from 2 t/m^3 to 3.3 t/m^3; in the second, where the air is ducted outside the kiln, it rises to 4 t/m^3 d. In spite of a higher volume load the specific cross-sectional load is clearly lower than in rotary kilns without precalcining stage. Therefore, the clinker output can be increased considerably when an existing kiln is fitted subsequently with a precalciner.

Many years of experience have shown that precalcining does not affect the cement quality; however, kiln operation becomes more balanced.

* Quoted almost literally from "Möglichkeiten zur Energieeinsparung in der Zementindustrie" /3.1-4/

Apart from the possibility of achieving higher outputs, secondary firing has also opened a way of saving high-grade fuel, such as fuel oil, gas, or coal, and of substituting it by low-grade fuel. A distinction can be made between diluent fuel and fuel substitute. Diluent fuels include coal and oil shale, mud coal and charcoal, whereas fuel substitutes consist of waste that contains some energy, such as domestic refuse, used oil, waste rubber (tyres), and active earth.

Low-grade fuel has a high ash content, low calorific value, and very often contains environmentally relevant components, such as sulphur, zinc, lead, chlorine, etc. The effects of heavy metals on clinker quality and pollution escapes a final analysis. However, it is certain that high sulphur and chlorine contents tend to disturb the operation of the kiln.

There are basically two good reasons for using low-grade fuels, especially fuel substitutes, for the precalcination of cement. First, no net calorific value is required because ignition is always ensured due to high meal and gas temperatures of 700 to 1,000° C. Secondly, many of the substances relevant to the environment are introduced into the process together with the fuel and are bound almost completely in the cement clinker without impairing its quality. What is more, the low-grade fuel can normally be used in the process directly, ie without pretreatment.

The use of precalciners opens another opportunity for substituting energy, that is solar energy. In the winter of 1983, first ideas of such a plant concept were presented. According to that concept, solar energy is deflected by plane mirrors, standing in a field next to the cement works, onto a concave mirror. This mirror is fitted on a pole high up above the works. It bundles the light and directs the beam through a window into the combustion chamber. The focal length of the mirror is large enough to place the focus exactly in the centre of the flame. Many of the questions that determine the realisation of the concept have remained unanswered so far /3.1-6/.

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3.2 Lime industry

The term lime normally denotes limestone, quick lime, or lime hydrate. These three substances can be produced in a cycle process. The building industry uses limestone ($CaCO_3$) as crushed stone in road building or as an asphalt filler. Quick lime (CaO) is used in the manufacture of lime-sandstone and gas concrete, as well as in soil stabilisation and for making mortar and plaster and external rendering, like hydrated lime ($Ca(OH)_2$).

3.2.1 Process description

Limestone is quarried in open pits. Drift mining is only profitable if very pure $CaCO_3$ can be worked which permits subsequent processing to high-quality products. After stripping the overburden, the matte is blasted loose. It contains clay and quartz impurities. The fly rock is transported by lorry to crushers where it is reduced in size in several stages. The most commonly used types are jaw breakers and rotary crushers. After breaking the rock to the desired size, all impurities in the form of sand, clay, and loam are washed out in various units with a view to obtaining quick lime of maximum purity. Then follows classification and screening into grades according to later use or sales demand. Over-sized grains are ground in mills.

3.2.1.2 Manufacture of quick lime

Technically, quick lime is manufactured by neutralising limestone at temperatures above 900° C. In industrialised countries, the matte is burnt in modern, largely automated kiln plants, whereas developing countries use the simpler shaft furnaces which are charged and emptied by hand.

Today's kiln plants consist of various types of shaft furnaces and rotary kilns. Rotary kilns are similar to the kilns used in the cement industry. They comprise a slightly inclined cylinder that rotates on its longitudinal axle at 0.5 to 2.5 revolutions per minute. In old type rotary kilns, that used to be 30 times longer than the internal diameter of the metal jacket, preheating is done in the upper end of the kiln. The specific energy consumption of these long kilns is relatively high at 6.6 to 8.4 GJ/t, with daily outputs of up to 560 tonnes /3.2-1/.

This type of kiln was further developed into short rotary kilns and separate preheaters. In other words, the preheating zone is no longer part of the rotary kiln but it is housed in a separate batch-type preheater. This makes the waste gas temperature drop. Heat consumption is about 5.0 to 8.4 GJ/t, with daily outputs of 200 to 1,000 tonnes of lime per day /3.2-1/.

Compared with shaft furnaces, the advantage of rotary kilns lies in the usage of common fuels, other than coke. Moreover, they permit changes in quick lime qualities and are particularly suited for small grain matte. Like in the cement industry, the kind of raw materials input and their moisture content determine the kind of process to be used, wet or dry. The quality of lime burnt in a rotary kiln is higher than that produced in other kilns.

In contrast to their significance in the manufacture of cement, shaft furnaces do have a reason of being in the lime industry. This is because investment costs are lower, and the shaft furnaces also consume less fuel.

The coke-fired or anthracite-fired shaft furnace has a round cross-section. It has no baffles. Solid fuel is mixed with limestone outside the kiln and is charged as a finished mix. Special provisions have to be made when charging the kiln to maintain a uniform blend of limestone and coke. For this purpose, charging buckets have been designed. Their diameter is identical with the shaft diameter, and their bottom flaps open to discharge the mix. In this type of kiln, all combustion air is entered as lime cooling air which makes the temperature very low at which the lime is discharged. Cooling air is taken in through a mushroom-shaped pipe. Common types of discharges are a rotating reamer and rams. This type of kiln has a capacity of up to 400 t/d, with a specific energy consumption of 4.2 to 5.0 GJ/t.

The annular shaft furnace contains a second cylinder in the kiln cylinder. It is usually fitted in the lower end of the kiln, and some of the flue gases and the cooling air is guided through this cylinder. The annular space between both cylinders is filled with the load to be burnt. Combustion does not take place in the load but in separate combustion chambers which are fitted at the outside of the shaft in two levels. The flue gases emanating from the combustion chamber enter the load through openings in the kiln jacket and are taken in a counterflow to the upper section of the shaft and in a parallel flow to the lower section where the material is. The grain size of the load to be burnt is 40 to 70 mm. Annular shaft furnaces with an internal diameter of 3.5 to 6.9 m of the outer cylinder burn 100 to 500 tonnes of quick lime per day, using liquid or gaseous fuels.

The parallel-flow-counterflow-regenerative furnaces consist of two or three shafts which are interconnected by channels in the upper third of the height of the shaft above the discharge. At its throat, each shaft is fitted with devices for charging the material to be burnt, a combustion air inlet, and a waste gas outlet. All shafts have a continuous discharge and a pressure-tight bunker underneath. The fuel is charged at about two-thirds the height of the shaft furnace above the discharge. Firing is done in cycles, with one shaft getting fuel and combustion air while waste gas is removed from the second shaft (in a two-shaft furnace) or the second and third shaft (in a three-shaft furnace). This cycle alternates. The burning shaft works in parallel flow, whereas the other one or two waste gas shafts work in counterflow. All shafts are fed with cooling air from below the discharges. This cooling air is not involved in combustion. The most important parameter in kiln control is the temperature of the flue gas in the overflow channel, which is about 1,000 to 1,100° C.

Three-shaft furnaces are mainly used for small-size matte of a grain size of 20 to 40 mm. Plants with an internal diameter of 2.5 to 4.0 m produce a throughput of 100 to 500 tonnes of quick lime per day. Oil or gas can be used as a fuel. *

The type of kiln described above has a specific heat requirement lower than that of other known systems; the reason being that the air used for combustion is preheated in a regenerator as part of the limestone load. Waste gas temperatures are therefore 60 to 140° C. The temperature cannot be lowered further because it would fall below dew point. The fired load has a temperature of 50° C. Another decrease in specific heat consumption can only be slight, and it will only be achieved at high capital cost.
In addition to the types of furnace discussed, that are used in the lime industry, there are a number of other types which, however, go beyond the scope of the present study.
The lime charge varies in its chemical and physical properties so that is is ground again after firing. This is done in tube mills or roller mills, with grinding aids, eg alcohols or amines, being added in some cases. Often, natural gypsum or molasses are added to adjust the slaking pattern.

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* The description of shaft furnaces has been quoted almost literally from "Wirtschaftlicher Brennstoffeinsatz" /3.2-1/.

3.2.1.3 Hydration of quick lime

Lime has to be hydrated to be used on the building site. This is referred to as slaking. While energy is liberated, the following reaction takes place:

$$CaO + H_2O \longrightarrow Ca(OH)_2 + E$$

There is a difference between the wet and the dry process.

In the wet process, which is only applied at building sites today, lime is mixed with water and is stirred with a pick. This liberates so much energy that the temperature rises to just below the boiling point. In some developing countries, lump lime is sumped in pits at the works. The lumps break and the quick lime does not have to be ground.

The lime industry applies a dry process exclusively now. Water or steam is added in the quantity required to bring about complete hydration of the quick lime. After complete slaking, a powder remains which can be bagged immediately. The throughput of modern hydrate plants is about 15 tonnes a day. It takes 320 kg of water of slake one tonne of lump lime.
Any condition or consistency can be obtained by adding more or less water.

3.2.2 Energy-intensity of the product

In lime production, by far the most energy-intensive stage (Fig. 3.2-1) is the firing of limestone. Winning, haulage, and dressing consumes an average 0.083 GJ of primary energy per tonne /3.2-2/, whereas a coke shaft furnace consumes 4.2 to 5.0 GJ/t. Annular shaft furnaces take 3.7 to 4.4 GJ/t, parallel-flow-counterflow-regenerative furnaces 3.6 to 4.0 GJ/t and rotary kilns 5.0 to 8.4 GJ/t /3.2-1/.

Taking into account the percentage distribution of process types, the following mean values are arrived at for the Federal Republic of Germany, for instance:

limestone	5.39 kWh/t	33 MJ/t
limestone powder	57.50 kWh/t	118 MJ/t
lump lime	31.90 kWh/t	4,961 MJ/t
pulverised quick lime	61.90 kWh/t	4,961 MJ/t
lime hydrate	27.70 kWh/t	3,759 MJ/t

/3.2-2/.

3.2.3 Energy saving potential

The thermal efficiency of lime furnaces reaches 84 per cent which puts them among the fuel-fired industrial furnaces with the highest heat utilisation. In modern furnace plants, therefore, more energy can only be saved at great capital investment. This fact emphasises the need to choose energy-saving types of furnaces whenever a new plant is projected.

The type of furnace that uses the highest amount of energy per tonne of lime produced is the rotary kiln, with or without preheater, with the highest losses being waste gases leaving the plant at a temperature of 250 to 350° C. These gases are always mixed with secondary air getting in at the interface between rotary kiln and preheater. If the plant were completely sealed, the waste gas temperature would rise to 430° C. This would not affect the balance sum of waste gas losses, because the overall amount of waste gas would be reduced accordingly. If the gas temperature is high, waste heat utilisation is favourable. If the furnace operates at an air deficiency, combustible components still enter into the preheater, combust with secondary air and contribute to pre-neutralisation. The degree of pre-neutralisation can be up to 40 per cent /3.2-4/.

The pre-neutralisation phase in the lime industry does not favour energy substitution by low-grade fuel because lime, unlike cement, does not bind the environmentally relevant substances contained in fuel. The lime quality would be largely affected which in turn would also cancel out the advantage of the rotary kiln, ie high purity of burnt lime. As to the reduction of wall losses occurring in the rotary kiln, that account for 90 per cent of all wall losses of the complete plant, the same action applies as described earlier in the section on the energy saving potential in cement making (Chapter 3.1-3).

All energy saving opportunities have been seized so far in shaft furnaces used in the modern lime industry. A case in point is the parallel-flow-counterflow-regenerative furnace. It is left with only two important sources of loss, ie waste gases and heat radiated from the furnace. The waste gases are 60 to 140° C hot when leaving the regenerative furnace. This waste gas temperature cannot be reduced further because it would get near their dew point which is 52° C.
The only way is to cut the amount of waste gas, which can be done by reducing the amount of cooling air. However, this would cause the temperature of the lime discharge to be higher than the original 50° C . If the amount of cooling air is adjusted to a level at which the lime temperature can be accepted by subsequent conveyors (100° C), then the heat saved from waste air amounts to only about 16 per cent of the added losses from a hotter discharge. It also raises the temperature inside the kiln and therefore causes greater wear of the brick lining. The question remains whether the risk of increased wear of the lining should be incurred in view of the slight savings that can be gained /3.2-5/.
Apart from the rotary kiln there are only a few minor areas where

savings can be made. They all vary according to the type of furnace and boil down to slight modifications in cycle times and air throughput.

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3.3 Brick and tile industry

The brick industry uses a great variety of starting materials and produces a wide range of products. Based on this fact and due to local conditions, a great number of technologies have been developed.

Products range from simple wall bricks to hollow brick boards, which is the latest ongoing development. It is a single storey-high brick element made in one piece that will be used in efficient building construction. The range of products also includes weather-resistant facing bricks, artificially aerated light-weight bricks, and a multitude of roof tile shapes.

3.3.1 Process description *

3.3.1.1 Raw materials

Raw materials are clay, argillaceous stone, and loam. They contain three important minerals, ie kaolinite, illite, and quartz, which ensure high-quality brick products. Various aggregates, such as sand, soda, phosphate, polystyrene foam or sawdust, are added to the starting materials. They alter the shaping or sintering pattern, or else they aerate the brick because they burn during the sintering stage.

The raw materials are mined in open pits, very rarely underground, and only if clay is mined as a by-product of coal, or if it can be mined by cheap labour from under a hard earth, or gobbing. The dominating procedure, however, is machine stripping and in some exceptional cases, manual stripping.

The raw materials are banked out at the works for a period of up

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* Data quoted from "Brick and Tile Making" /3.3-1/

to two years; during that time they weather, ie moisten thoroughly, and clayey substances swell evenly. Then raw materials and aggregates are ground and mixed. Depending on the type of raw material and the shaping process, water is added (up to 30 per cent) or extracted after mixing. To improve the quality of the material, the mix is then evacuated, ie all enclosed gas is removed. A thorough homogeneous mix is very important to ensure a long chain reaction during burning.

3.3.1.2 Moulding

The great variety of ceramic materials and the possibility of varying the water content also permit the use of a wealth of moulding and shaping methods. Although hand moulding is one of the oldest methods it is still applied, mainly in developing countries. The prepared mix is pressed by hand into a wet wooden mould which is open at the top and at the bottom. The frame is taken off and the brick is left to dry.

This method was superseded almost completely by mechanical methods, with current outputs of up to 30,000 bricks per hour. The machine presses the clay into automatically fed steel moulds or a chain of moulds. Only solid bricks can be made in this type of machine.

Other shapes can be made in an extruder press in which a continuous screw or a series of knives convey and force the raw material through the head of the press. Due to the rotation of the continuous screw, the clay is again blended thoroughly. A die at the head produces a column with grooves and holes. Pipes can also be produced on such an extruder press.

Based on the water content of the clay, the extrusion process is divided into two types, ie stiff extrusion, using clay of a moisture content of 12 to 20 per cent, and soft extrusion, with a moisture content of 20 to 30 per cent relative to the weight of the wet ware. Stiff extrusion produces wares of good dimensional stability. It is mainly applied in the U.S.A., Canada, Australia, and South Africa. In Europe, soft extrusion is used because bricks are more perforated, and pressures would have to be too high if stiff extrusion were used.

If products need to be made to even more exacting requirements, the dry press method is recommended. The clay then contains under 8 per cent of moisture, and a plunger presses it under high pressure into a steel mould.

3.3.1.3 Drying

After shaping and moulding, the brick is dried. Depending on the kiln used, the brick must only contain 0.5 to 6 per cent of residual water before firing. Convection is the only form of drying. During drying, the brick shrinks by as much as 5 per cent which explains why slow drying is essential so as to avoid cracks.

The oldest method of drying is hack where drying is performed in the open air and is very much a function of weather and time. The slow drying rate necessitates large spaces for stocking. This method is economically justified only where energy is expensive and labour cheap.

Large-size driers make use of the waste heat escaping from the kiln structure. They are placed above the kiln, sometimes at different layers. Although controlling is somewhat difficult, drying rates are shorter than in open-air drying. Drying is independent of outdoor weather. With the introduction of high-performance kilns, however, large-size driers have lost their significance.

Unlike large-size driers, chamber driers can be readily controlled because heat and air flows can be regulated in the chambers. They are periodically loaded and unloaded. Several chambers are required to permit quasi-continuous operation.

Purely continuous drying can be performed in a tunnel drier in which the green bricks move in batches through a heated tunnel. This creates a favourable basis for automation of the production process. The drying time is much shorter than in the chamber drier, operating costs are lower, but investment costs are higher. Tunnels are currently used in modern and efficient brick works.

The latest development is a high-speed drier in which the mouldings are overblown from all sides so as to avoid great differences in moisture contained in evaporating surfaces. Drying times can be reduced to under six hours.

3.3.1.4 Firing

After drying, the bricks are ready to be used in some applications. For the majority of applications, however, strength and weather-resistance have to be improved; this is done by firing. When the temperature starts to rise there is first a separation of water from the clay minerals. Up to a temperature of about 150° C, residual pore water evaporates, and above a temperature of 400 to 600° C the chemically bound water is released. In the temperature range up to 900° C, further gas-releasing reactions occur, such as the decomposition of calcium carbonate in $CaO + CO_2$. Moreover, carbonaceous components, such as humus and sawdust, oxidise. During the actual firing stage at a temperature of 1,050° C, the fine raw material particles are irreversibly bound to a solid through silicatisation.

The products are cooled to a temperature of about 20 to 40° C through heat exchange with the combustion air and preheating air being drawn to the cooling zone.

Firing is performed in a great variety of furnaces and types of furnaces, with some of their underlying principles being very old. For instance, many developing countries still use charcoal kilns. The fuel, eg peat or coal, is piled in alternating layers with mouldings and sealed with loam. Some fire holes are left open at the base of the pile, it is fired, and with heat being generated, successive layers of fuel are ignited. Only some of the bricks in the pile can be used after firing.

A distinction is made between three different types of kilns:
- intermittent kilns,
- semi-continuous kilns,
- continuous kilns.

In intermittent and semi-continuous kilns, running up the kiln, preheating, firing, and cooling make up one complete cycle. In semi-continuous kilns, flue gases are used for heating the next kiln. All types of chamber kilns figure in these groups.

Continuous kilns consist of one closed firing channel which is either ring-shaped or criss-crossed, within which the firing zone moves through a sophisticated firing system. Cooling is performed through heating of fresh air, and preheating is done by some of the hot exhaust gases. It is loaded and unloaded always in the coolest zone of the kiln. Annular and transverse arch (Fig.3.3-1) kilns belong to this type of continuous kiln.

Another kind of continuous firing is performed in tunnel kilns. Their firing is static, and the green bricks travel through the tunnel, including the preheating zone, the firing zone, and the cooling zone. Here again, the combustion air and some of the

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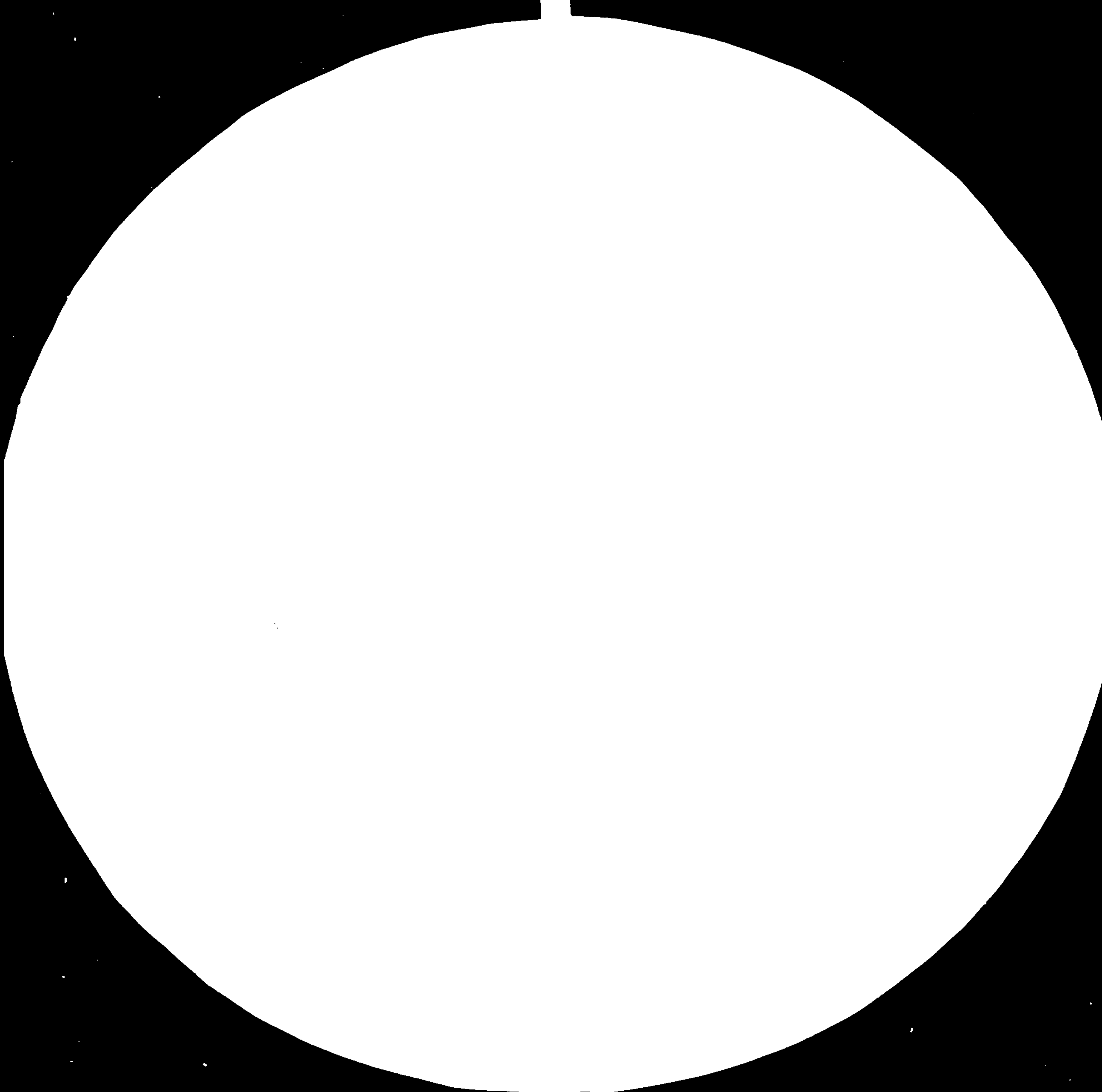

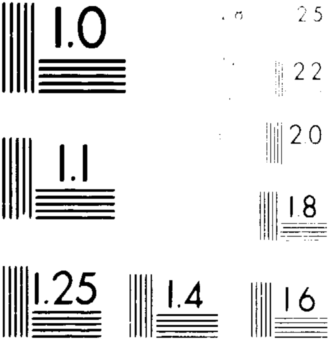
1.0
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preheating air is heated in the cooling zone. Moreover, some of the exhaust gases contribute to preheating.

The bricks are transported through the tunnel on tunnel cars or conveyor belts. The length of the firing channels in modern kilns is between 40 and 200 m, the width between 1 and 10 m, and the height up to 2.5 m. Tunnel kilns are fired with light crude oil, heavy fuel oil, coal, wood, peat, natural gas, or liquid petroleum gas. Tunnel kilns, when associated with tunnel driers, offer the most economical way of firing brick on a large scale because their firing rate is faster, energy efficiency is better, and they have the merit of automation. Moreover, the working environment is optimised because there is neither heat nor dust nuisance. Table 3.3-1 lists the time-firing curve.

3.3.2 Energy input

Due to the great variety of processes and the wide range of products, it is a practical impossibility to state a standard value for the primary energy input in brick making. For example, discontinuous kilns consume twice as much energy as tunnel kilns. One important factor that dictates the energy input of the drier, is its shape because that determines the water content of the clay; contents vary between 4 and 30 per cent. The energy distribution represented in Fig. 3.3-2, therefore, can only be taken as a mean value. Accordingly, brick production in the Federal Republic of Germany requires 2,799 MJ of primary energy per tonne of brick; for clinker it is 3,115 MJ/t, and for roof tiles 3,084 MJ/t /3.3-2/. The differences are due to the different temperature levels at sintering. Fig. 3.3-2 also clearly shows that drying and firing consume a major share of the energy input.

3.3.3 Energy saving potential

The largest amount of energy goes into drying and firing so that, therefore, the greatest savings can be made in these two stages. Due to the multitude of production processes there are many saving opportunities.

Mention should be made here of the most important opportunities: The choice of the moulding method has an effect on the water content of the clay. When converting to dry methods, energy can be saved in the drier. However, it is not always possible to pick the most economical process because the raw materials may be available with high water content only, or else the load on the press, especially when making extrusions with a high degree of perforations, may be too high.

Converting from intermittent to continuous operation helps avoid intermittent preheating of dead mass. In the continuous process, there are also more opportunities of utilising the cooling air. And the air that heats up in the cooling zone can be used as hot combustion air and as drying air. The utilisation of flue gases for drying and preheating, however, still creates problems because the impurities contained in the exhaust gases have a direct bearing on the quality of brick produced. That is why direct usage is to be excluded in most cases. The solution to the problem is a heat exchanger that does not only heat air but also water, and there are plans to connect brick works with continuous operation to the district heating network.

When converting to tunnel operation, there are also the losses involved in the tunnel car. These heavy iron cars transporting the ware through the tunnel kiln store high amounts of energy. Using flat conveyors instead of tunnel cars can cut the loss.

All kilns radiate heat through the kiln wall. Savings can be made by insulating the kiln. The insulation has to be applied at the inside because otherwise temperatures of the kiln jacket would rise too high. Driers, on the other hand, can only be insulated on the outside because the temperature involved is lower.

The air emitted from the drier has a temperature of 30 to 50° C and is very humid. If the energy contained in this air is to be utilised that can be done through either a space heating, or the energy is to be recycled to the drier input via a heat pump.

Clay always contains a certain share of combustible substances, for instance, humus components which release their energy to the green brick during combustion, ie firing. Sometimes a point is made of admixing foreign matter, like polystyrene pellets or sawdust. The underlying idea of adding such substances is that cavities form when additives are combusted without residue and that, therefore, the thermal conductivity of the bricks is impaired. Current research seems to suggest that the admixture of powdered lignite increases the energy contained in the clay mass to a level that meets fully the energy demand for drying and firing. These bricks have many small perforations which gives them a favourable heat transfer coefficient, and at the same time the mass of clay per fired brick is kept small. In addition to that, losses of flue gases are small because the heat is generated inside the stone and fire is only used to assist in the sintering.

3.3.4 Sand-lime bricks

Besides clay bricks, sand-lime bricks also play an important part in the building industry. Sand-lime bricks are wall bricks of silicate concrete. Their thermal capacity is greater than that of bricks, which makes sand-lime bricks a suitable building material in regions where temperatures differ widely between day and night. Less energy is needed to make sand-lime bricks. The raw materials are lime, silicious sand and water. Usually the sand, that accounts for 90 per cent of all raw materials used in sand-lime brick making, is quarried near the brick works, whereas lime has to be hauled over long distances. The raw materials are thoroughly blended and mixed. Good homogenisation is imperative to reach a high compressive strength. After mixing, the mix is conveyed to reaction silos where the fired lime hydratises completely within 2 to 3 hours. The material is mixed again and water is added to bring it to the required moulding moisture.

Moulding is done in mechanical or hydraulic presses. The green bricks are then loaded onto a car, cars are connected to trains that travel into 14 to 16-m long autoclaves. Inside the autoclaves, the bricks are cured for 4 to 8 hours under steam of 8 to 20 atmospheres and temperatures of 160 to 220° C. 75 per cent of the time spent in the autoclave is for curing, the remainder for preheating and precuring. Then the bricks are drawn from the kiln and packed.

The primary energy input is 871 MJ/tonne of brick /3.3-2/ which is distributed among individual process stages as shown in Fig. 3.3-3.

To make better use of the energy contained in the steam, it is ducted to the next autoclave after completing the curing process in one autoclave. The steam is re-used several times; first for curing, then for preheating, and finally for passing the residual energy to a condenser. The condensate, however, is not used again because it is too dirty to be used in the steam boiler. Fresh water consumption is 100 to 110 litres per tonne of bricks produced; curing takes 100 kilograms of steam per tonne. Where there is not enough water the condensate can be used for preparing the raw materials. In this way some of the energy contained in the condensate can be recycled to the process and thus shortens the preheating phase.

Almost 50 per cent of the amount of energy remains in the bricks when they leave the autoclave. This energy is emitted to the ambient air when the bricks cool off. As has been demonstrated in the clay brick and tile industry, this energy could be used for preheating the combustion air required for steam generation.

Since sand-lime brick making is a process that can be varied over a wide range of pressure and time, and since the steam temperature is not too high, it could also be run on solar energy.

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3.4 Flat glass industry

3.4.1 Process description

3.4.1.1 Raw materials

Glass is one of the oldest materials of mankind. The characteristic physical feature is its solidifying to an amorphous, rather than a cristalline state when cooling. Glass is an elastic, yet very hard material. The type of glass preferably used for manufacturing flat glass is soda-lime glass. It consists of 70 to 75 per cent of SiO_2, 4 to 10 per cent of CaO, 15 per cent of Na_2, the remainder including MgO and Al_2O_3 /3.4-1/.

For the main ingredients, raw materials need to be available in large quantities and at low cost. The most profitable sources of raw material are silica sand for SiO_2, lime for CaO and soda for Na_2. The most important impurities are A_2O_3 and Fe_2O_3. Iron oxide is a negative factor because it is responsible for the green colouring of glass.

Before melting, the raw materials are mixed; cullet, or scrap glass, can be added. A homogeneous mix is to be achieved with a view to the quality of the finished glass product.

3.4.1.2 The glass melt *

In melting, a distinction is made between discontinuous and continuous processes. The discontinuous method is represented by the age-old glass melting pot. Within a furnace, a fired clay pot holding 50 to 300 litres of melt is heated up to the melting temperature of glass. Then the mixed batch is inserted, melted, fined, and blended

* Quoted almost literally from "Werkstoff Glas" /3.4-1/

evenly. Small gobs of glass are gathered on a blowing iron, or all the melt contained in the pot is poured out. The glass is then formed through blowing, pulling, pressing, rolling or casting in a mould. A pot furnace can have several pots. This process is still profitable today wherever different wares are produced in small quantities.

The continuous method works with a melting end. This furnace compartment is built of refractory ceramic material (SiO_2, ZrO_2). Its size is determined by the throughput of the connected processing machinery. The raw materials are automatically weighed, mixed, and the mixed batch is fed continuously to the melting end. At the opposite end, the finished glass flows through a narrow outlet into the working end that feeds the processing machines.

The main fuels used in melting are natural gas or oil; electric furnaces and electrodes are not as common. In gas or oil-fired furnaces, the flames are delivered above the glass level and cover the surface of the melt which is between 2 and 400 square metres. Depending on the product to be made from the melt, the quantity of molten glass contained in the tank is in the range of 0.5 to more than 1,000 tonnes. The daily output of glass is between one tonne and more than one hundred tonnes. Temperatures at the charging end is some 1,500° C, in the centre of the tank in the refining zone some 1,450° C,and near the port to the working end some 1,350° C. This difference in temperature causes agitation in the tank that mixes the glass evenly.

In the charging zone,the raw materials start to react with one another. These reactions form a doughlike bubbly mass. During the refining stage that follows the bubbles rise to the top and liberate trapped gases (mainly CO_2). These bubbles mix and homogenise the glass. To accelerate the process, small quantities of substances like sodium sulphate or arsenic oxide and antimony oxide are added to the mix. At refining temperature they liberate many gases that rise in big bubbles entraining small bubbles known as seeds and carry them to the top of the melt.

Refining requires low-viscosity glass for the gases to be liberated. However, glass of such low viscosity cannot be processed, and therefore it is conveyed to a second tank, the working end. There, the glass is maintained at a temperature that gives it the viscosity required for subsequent processing.

4.3.1.3 Forming *

The molten and refined glass is usually processed to the finished product in the same glassworks. In view of the fact that the glass has the correct temperature (viscosity) when molten it would be uneconomic to let it cool and then heat it again at a later stage.

The first flat glass drawing machines were using the Fourcault method. In this method, a continuous flow of glass swells through a jet pressed into the glass melt and is pulled to the top in the form of a ribbon. This ribbon is cooled to a lower temperature and is then passed between two pairs of rollers that are part of the actual drawing machine. The shaft of the drawing machine serves as a cooling path. Above the machine, on what is called the cutting room, the glass ribbon is cut to the desired shape. Glass is always somewhat undulated when it comes off a metal plate; or, when a viscous glass ribbon is passed through two rollers, a plane-parallel sheet is obtained. However, the surfaces that have been in contact with the metal need polishing. The effect is favourable for giving the glass a texture or making wired glass; this is done by resoftening two sheets so that they can be united by rolling pressure over the intervening wire mesh.

The most modern method used in making sheet glass is the float-glass process. A continuous ribbon of glass is taken from the melting tank onto the surface of a pool of molten tin contained in a closed chamber, in which the temperature is kept constant. The glass ribbon is lighter than metal and floats on the metal bath. Inside the chamber, the glass is kept under an atmosphere of inert gas. As soon as the glass ribbon hardens it is conveyed on asbestos rollers through a cooling zone and can be taken off the roller belt after cutting. Float glass has an excellent surface quality that compares with the best glass produced in the more costly grinding and polishing process.

3.4.1.4 Cooling *

In all types of processes, the glass needs to be cooled slowly after forming. During cooling it undergoes some structural shrinkage at a certain temperature level. Since glass is a poor thermal conductor, some thin sections of the surface of thick glasses cool much faster. In wares that cool faster structural shrinking stops sooner. When the ware is cooled completely it is found that slow-cooling wares shrink more because they are exposed to structural shrinkage for a longer period of time. Internal stresses may develop. Most glass products become brittle because of these so-called frozen stresses. When cooling them to ambient temperature or room temperature they may even break. This explains why all glasswares, after forming, have to be heated to a temperature which is above the transformation point at which the internal stresses can be balanced through internal glass flow. This temperature level is not too high to deform the product. As soon as the internal stresses disappear the wares are cooled slowly and evenly to a temperature below the transformation point and then somewhat more rapidly to room temperature.

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* Quoted almost literally from "Werkstoff Glas" /3.4-1/

3.4.2 Energy consumption in flat glass production

To produce flat glass, energy is required in batch preparation, melting, forming, and cooling, with melting taking the largest amount of energy. The theoretical energy input is 2.94 GJ/tonne /3.4-2/. Due to high losses, up to 13 GJ/t is used in the Federal Republic of Germany /3.4-1/ for melting the mixed batch. Table 3.4-1 shows the total energy input in UK flat glass production. It should be noted that only 3.6 per cent of the melting heat is electrical energy. Accordingly, the total specific energy input is 11.87 GJ/t for the UK and 16.9 GJ/t for the Federal Republic of Germany, as quoted by /3.4-1/, and 18.4 GJ/t for the U.S.A.

3.4.3 Energy saving potential

Glass making has been a well mastered craft for a very long time. There have been no fundamental changes in glass making technology in the past few years. Some of the following opportunities for saving energy exclude one another. Therefore, each opportunity will have to be assessed on its own merits.

Over 80 per cent of the energy used in glass making is converted in melting furnaces. The major share of the energy input is emitted to the environment through the surface of the furnace in the form of waste gases.

The heat transfer through the furnace walls can be reduced by insulation. However, such insulation needs to be done in the inside, rather than the outside because otherwise the vault and, in particular, the steel foundations would be exposed to excessively high temperatures. Therefore, it is not suitable to apply an insulation to established furnace plants. Rather, it needs to be included in the early planning stage of a new furnace. By creating an even temperature distribution, however, the insulation causes the convection flows to decrease, which may lead to insufficient homogenisation of the glass melt.

Flue gases leave the furnace at a temperature of 1,200 to 1,400° C and more /3.4-2/. They can be used for preheating the combustion air when heat exchangers are employed. The heat exchangers may be recuperators or regenerators.

Recuperators are continuously operating heat exchangers with both media being separated from each other by a conducting wall. Because of the high exhaust gas temperatures involved in glass making, radiation recuperators are of particular importance. Since heat transfer through radiation is low at a temperature below 650° C, a possible remedy is to fit a convection recuperator downstream. The temperature of the combustion air that can be reached is below 800° C /3.4-2/.

Regenerators are discontinuous heat exchangers. They consist of a big chamber in which bricks are stacked as storage material that can be heated by flue gas. After about 30 minutes, the waste gas flow is deflected and combustion air is blown over the bricks. In this case the temperature of combustion air that can be reached is 1,100° C /3.4-2/. In order to ensure continuous operation, a second regenerator is needed so as to have one that heats up while the other cools down.

The furnaces are gas or oil-fired. Heat transfer occurs almost exclusively through thermal radiation. It is an advantage, therefore, to achieve bright flames. The flame should cover the entire surface of the melt but should not continue to burn at the stacks. Flames that have a poor coefficient of radiation, ie they do not 'light', as is the case with natural gas, can be brightened by adding oil.

Another way of making better use of the flame energy is to let the flame burn underneath the glass level. In that case the heat is converted inside the molten glass pool, the space above the melt remains cooler and the losses from hot waste gases are reduced. This entails two problems in that it requires a sophisticated burner design and a froth cover to spread on the melt.

The addition of scrap glass has also gained growing importance. At present, most of it is factory cullet, ie of the same composition; only a minor share is foreign cullet that is used. If one per cent of cullet is added 0.2 per cent of energy is saved /3.4-3/. In melting, the cullet acts as a solvent for the cristalline raw materials. It also improves the heat transfer and accelerates the melting process.

The thermal efficiency of the furnace is optimised in electric furnaces. Molten glass is electrically conductive, and therefore the heat that is generated when a voltage is applied can be utilised. A distinction is made between fully electric melts and auxiliary electric heating of fuel-fired tanks. The electrodes used consist of either graphite, molybdenum, platinum, or SnO_2, and some of them are internally cooled. They protrude into the melt from below or from the side, or they are fitted as boards at the pot wall so that heat is directly generated in the melt.
In the fully electric melt, the melting process is horizontal and similar to that in a shaft furnace. The unmolten mix isolates the melt near the top; this produces vault temperatures of 200 to 300° C /3.4-2/, and waste gas losses can be cut significantly. It also causes the consumption of final energy during melting to drop to 750 kWh/t or 1.2 MWh/t (2.7 to 4.3 GJ/t) /3.4-5/. Since electrical energy is very expensive, however, the type of process only pays in countries where power is cheap (Switzerland, Sweden, the U.S.A.).

In all glassworks, energy is added for cooling in order to avoid too rapid a temperature drop. The amount of energy supplied can be cut by providing a better insulation of both the cooling furnace and the air circulating in the furnace, combined with a new furnace design. Similar ways apply to preheaters and working ends that keep the glass at a workable viscosity until it is processed further. In the melting furnace, glass has a higher temperature and therefore a higher viscosity than that required in processing. Through good insulation and by lowering the processing temperature, energy can be saved in existing preheaters. Eletric preheaters require up to one third of the energy consumed in gas-fired furnaces.

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4. FOOD INDUSTRY

4.1 Food preservation

Industrial production of food has prolonged the time span between production and consumption, apart from the fact that more food is stored in general. This has led to a number of measures taken to protect food from deterioration.

The main causes of food spoilage are growth of yeasts, molds, and bacteria. Yeasts can grow under sealed conditions, molds grow on media containing carbohydrates and protein; some of them eliminate mycotoxins which are particularly dangerous. All kinds of food can be attacked by bacteria, some of which are a health hazard to man.

The most important methods of preservation are sterilisation, refrigeration, drying, pickling, sweetening and souring, as well as the use of chemical preservatives. In heat sterilisation, a distinction is made between pasteurisation and sterilisation. In the first, the food is heated to roughly 100° C which kills yeasts, molds and some types of bacteria; in the second, sterilisation, food is heated to 120° C to kill all spore-forming bacteria. Proteins are easier to digest for man after heat treatment, but the vitamins contained in food suffer considerably.

Storing food at 0 to 6° C is referred to as cold storage. All reactions are slowed down and storing periods of several days, eg for strawberries, up to several months, eg for oranges, are possible without deterioration of quality. Cold storage is suitable for fruits, vegetables, meat and fats.

If the temperature is lowered to -18 to -25° C this is referred to as freezing. Some microorganisms are killed at these temperatures, enzyme activity is mostly blocked, and trichinella is killed, for instance.

The food is first refrigerated to -2° C and then frozen at -40 to -50° C. Frozen foodstuffs have to be packed carefully to avoid dehydration during long storage periods, which may be 15 months /4.1-1/. A great disadvantage is the continued consumption of energy for low-temperature storage.

When food is dehydrated, this is referred to as drying. It is based on the fact that deterioration is inhibited at low moisture contents. Some foods are very sensitive to high temperatures, and therefore methods have been developed to ensure slow extraction of water, such as in fluidised-bed drying, froth drying, roller drying, spray drying and freeze dehydration. Solar plants are also suitable, as was demonstrated in the People's Republic of China, for instance /4.1-2/. Such plants, as was also shown in Crete, have the merit of dehydrating more rapidly than open-air drying and improving the quality of foods at the same time /4.1-3/. The dehydrated products that help save work, time, and energy in the home become more and more popular. They need to be rehydratised before consumption.

Dehydrated foods meet the consumer's need for convenience. This group comprises instant products which are dried powders that can easily be dissolved in hot or cold fluids. Precooked products and pre-prepared meals account for a growing share of the food industry; they are consumed in private homes, restaurants, hospitals, canteens, and schools. The trend for more convenience food has brought about a change in the food industry towards higher energy-intensities, although, of course, convenience food also saves energy that would otherwise be required for cooking at the ultimate consumer's home.

4.1.1 Fruits and vegetables

4.1.1.1 Fruits

4.1.1.1.1 Preservation in cans and glass jars

Since the middle of the last century, heat preservation in tins and jars has been the most important method of preserving fruits and vegetables. Today, mainly tin cans, coated or uncoated, are used. For reasons of process engineering, they are not as significant as glass jars for the preservation of foods.

For heat sterilisation only best-quality fruits can be used, fruits that have a good flavour and are not too ripe, not damaged or infected. The main fruits that are tinned are stone fruit, pears, pineapple, and apple sauce. Other fruits are less suitable for tinning; these are strawberries, raspberries, gooseberries, and blackberries.

Depending on the kind of fruit to be preserved, the first step is to wash the fruit and to remove any dirt, stalks, leaves, cores, pits, peels. In a next stage, fruits like apples, pears, peaches and peeled apricots are blanched, which is immersing them in hot water or steam. The fruits are normally uncooked when placed in the tin. Cooked fruit is tinned with sirup, which has double the concentration required in the final product. The fruits shrink as a consequence of osmosis. If water is used for filling up the tin then the fruits wilt. In general, tins are sterilised for 4 to 6 minutes at a temperature of 77 to 95° C /4.1-1/.

4.1.1.1.2 Frozen fruits

When freezing fruit, it is important to ensure slow chemical reactions and to inactivate microorganisms; however, not all enzymes or ferments need to be inactivated. Contrary to vegetables, where enzymes can be inactivated by blanching, this cannot be done with fruits. It very often causes the fruits to turn brown which can be prevented by adding ascorbic acid. Because of the enzymes that cannot be inactivated, fruits have to be frozen rapidly. Freezing is done at -30° C and takes about three hours /4.1-3/. The fruits should be as ripe as desired for consumption. Freezing is a better proposition than tinning for various kinds of fruit, such as strawberries. Less suitable are pears, grapes, light plums, light cherries and most of the tropical and subtropical fruits. As a rule, fruits are frozen in a sirup. They must be sealed to avoid dehydration. Frozen fruits are stored at a temperature of -18 to -24° C, which requires a continuous refrigerating chain from the producer to the consumer.

4.1.1.3 Drying of fruits

Below a certain water content, there is no growth of microorganisms. Only sound and ripe fruits are used that could also easily be eaten without being dried. Before drying, the fruits are washed and prepared in various ways, depending on the fruits. To prevent enzyme discoloration and parasitic attack, the fruits are sulphitated. Drying is still done in the open air and in the sun; even in sunny countries driers are also used . They are chamber driers, plate driers, cabinet driers, or tunnel driers. Operating temperatures are between 65 and 75° C; vacuum drying is particularly safe at 60° C /4.1-4/. The main fruits for drying are apples, plums, apricots, peaches, grapes, figs, and dates. The water content of the dehydrated fruits is 14 to 24 per cent, although some fruits are dehydrated to contain 3 to 5 per cent; this is to give the fruits good storing properties in tropical countries. After drying, the fruits can be pressed into blocks which increases transportability and storability.

4.1.1.2 Vegetables

4.1.1.2.1 Tinned vegetables

Vegetables must be of the best quality and appearance and should be prepared immediately after harvest. They are washed, prepared, some of them are cooked (for instance, celery), and then blanched. This inactivates the enzymes and removes undesired flavours and enclosed air. Moreover, the vegetables shrink and soften during this process so that their packing density grows. The remaining space is filled with brine and the tin is closed automatically.

Then follows sterilisation in autoclaves at temperatures of 108 to 120° C and a pressure of 2.5 to 3.5 bar /4.1-5/. Because of their high pH and the content of resistent soil bacteria, thermal processing of vegetables has to be more intensive than that of fruits. The dwell time in the autoclave depends on the kind of vegetable and is 6 to 20 minutes. Red cabbage is very heat-sensitive and is therefore preserved almost exclusively with vinegar which reduces its pH to below 4.5 /4.1-6/.

Unlike vitamins, proteins and carbohydrates are not affected by heat. With vitamins, at lot depends on the type of vitamin and how much of it is lost in sterilisation. For instance, the loss of vitamin B2 is 5 to 25 per cent, whereas it is 55 to 90 per cent for vitamin C. If tins are stored for several years, another 20 per cent of the vitamins are lost /4.1-1/. The shelf life of good tins stored at a temperature of 20° C is four years.

4.1.1.2.2 Frozen vegetables

The criteria to be applied to raw vegetables to be frozen are much more stringent than those applied to vegetables to be tinned because the condition of the raw vegetable has a direct bearing on the quality of the frozen product. Freezing is particularly suitable for beans, peas, peppers, Brussels sprouts, mushrooms, tomatoes, carrots, and various types of cabbage, radishes, lettuce, and whole tomatoes /4.1-1/. Blanching is reduced to a minimum because over- or underblanching would affect the quality. If the blanching time is too short, the vegetables discolour during storage and thawing; moreover, organoleptical properties are affected. After blanching, the vegetables are frozen in a plate or air freezer at a minimum temperature of -40° C; they are stored at -18 to -20° C /4.1-1/. Some frozen vegetables change their consistency. The storage life is normally one year, if properly stored at a minimum temperature of -18° C.

4.1.1.2.3 Drying of vegetables

Vegetables are always dried artificially. The raw vegetables are washed, peeled, scraped, and cut or diced, if required. Then they are blanched with hot water or steam for 2 to 7 minutes. This may be followed by SO_2 treatment. Drying is done at 55 to 60° C, and dehydration is taken down to a residual moisture content of 4 to 8 per cent.

Freeze dehydration produces high-quality products. Dehydrated vegetables require careful packaging because they are sensitive to light, air, and vapour. Some dehydrated products are compressed before packaging. The drying process causes a number of fundamental changes. First, the main components, protein, carbohydrates, and minerals are concentrated, with inevitable chemical reactions occurring. Fats are decomposed through oxidation and adversely affect both smell and taste of the dehydrated product. Some vitamins are also greatly affected. Existing volatile aromatics are largely lost. *

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* Quoted almost literally from "Lehrbuch der Lebensmittelchemie" /4.1-1/

4.1.1.2 Energy input in fruit and vegetable processing

In all stages of preparation, which are basically the same in all preserving methods, the interesting stage in energy terms is blanching. The energy input comprises energy consumed for heating and cooling the raw material, as well as the requirement for transport devices and transmission losses of both blancher and cooler. The energy requirement in this stage is between 0.74 and 1.98 MJ/kg, depending on the quality and the kind of fruit or vegetable preserved /4.1-7, 8/.

The energy required in sterilising cans or glass jars is 1.23 to 2.88 MJ per kilogram of sterilised fruit or vegetable /4.1-7, 8/, with the difference being explained by the different autoclaves used. However, most of the energy is not used in the actual food preparation stages but rather in can making. To produce one tin can requires 12.19 MJ/kg /4.1-7/, one glass jar requires 16.9 MJ/kg /4.1-7, 8/.

In deep-freezing, the energy required for packaging is 2.1 MJ/kg of packed food (in bags), and 3.8 MJ/kg of food in boxes /4.1-8/. The actual freezing process consumes 86 Wh/kg. Unlike tins, frozen foods require continuous refrigeration, hence permanent energy consumption. Freezers require 0.5 W/kg to 19 Wh/kg, cold-storage depots require 5.8 mW/kg to 35.8 mW/kg.

Dessication, or dehydration, of fruits and vegetables is the most energy-intensive method, consuming 2.1 to 4.77 MJ/kg, but due to its less energy-intensive packaging (1.5 to 2.99 MJ/kg) and storage (0 to 0.43 MJ/kg) /4.1-8/, it has the lowest energy input of the three preserving methods reviewed. This is illustrated in Fig. 4.1-1.

4.1.2 Fish

Fish is very delicate and sensitive to mechanical stress; it decomposes quickly which explains why fish is processed onboard the trawlers and/or parent ships. Fishes are killed, gutted and put either between ice at 0° C or quick-frozen. Whole fish is often protected against oxidation by coating it with an ice film. Deep-frozen fish has a mean energy input of about 83 MJ/kg , and semi-frozen, semi-refrigerated fish that is processed for subsequent hot smoking or tinning has an energy input of 76 MJ/kg. This accounts for 63 to 93 per cent of the total energy requirement in the processing of fish /4.1-8/.

One of the oldest methods of preserving fish is drying. This requires a low temperature, high air velocities and dry, germless air (like in Iceland or Norway, for instance). Drying brings the water content down to between 12 and 18 per cent /4.1-9/.

The smoking of fish is divided into cold smoking (2 to 4 days, at 18 to 26° C) and hot smoking. Hot smoking is applied to whole fish, emptied and deboned (2 to 4 hours at 100 to 120° C) /4.1-1/. It requires between 43 and 46 Wh of electricity per kilogram of fish and between 1.363 and 1.795 MJ of thermal energy per kilogram of fish. Packaging takes 2 MJ/kg /4.1-8/. This is based on wooden chip boxes containing 2.5 kg of smoked fish.

Marinades are fish products made of fresh, frozen, or pickled fish or fish sections which are made tender in a combination of vinegar, acids, salt, and spices, without being cooked. They come in juices, sauces, creams, mayonnaise or oil. They are perishable /4.1-1/. The energy input is 0.179 to 0.183 kWh of electrical energy per kilogram of marinaded fish and 0.324 to 0.933 MJ of thermal energy per kilogram. Marinades are often sold in glass jars which have a mean primary energy requirement of 25.1 MJ per kilogram of marinaded fish /4.1-1/.

German-type roast fish is made of fresh fish, frozen or deep-frozen fish or fish sections which are covered with bread crumbs and roasted, fried, or grilled. They are placed in a vinegar brine, sauce, or oil /4.1-1/. The energy input is 58 Wh of electrical energy per kilogram of roast fish and 3.4 to 3.8 MJ of thermal energy per kilogram. A common type of packaging is the 750-ml flat aluminium tin. This requires an energy input of 30.8 MJ per kilogram of roast fish /4.1-6/.

Tinned fish is fresh fish, frozen or deep-frozen fish or fish sections which are preserved by heat treatment in sealed packages or containers and can be stored without cooling for at least one year. These are fish products in their own juice, in brine, oil, sauces or creams, combined with other food, in the form of fish patê or fish balls, as starters and salads /4.1-1/. The packaging is normally an aluminium can or a tin can that requires 31.3 and/or 6.57 MJ of energy to make per kilogram of fish contents, based on the 200-ml German 'hansa'-can. The processing of the tinned fish then takes another 0.7 kWh of electricity and 5.7 to 6.3 MJ of thermal energy per kilogram of fish /4.1-8/.

Deep-frozen fish is usually only after-treated on land because the actual freezing is done onboard the ship. Frozen fish slabs are sawn into tablets, bread-crumbed, pre-baked and re-frozen. During pre-roasting, the fish finger warms up to -5° C. These stages of processing consume 192 Wh of electricity per kilogram of fish and 0.551 MJ of thermal energy per kilogram of fish fingers. This includes the energy required for storing the finished product for one month at a temperature of -28 to -30° C. The type of packaging used is plastic-coated folding boxes which require 3.9 MJ to make 300-g boxes per kilogram of fish fingers and 2.9 MJ/kg for 400-g boxes. It should be noted that the energy required for cooling or refrigeration has to be added if the product is stored for more than one month (see Chapter 4.1.1.3) /4.1-8/.

4.1.3 Meat

4.1.3.1 Meat products

Meat products are sold in a great variety. They comprise tinned meat, ham, bacon, sausages, patés, and extracts. According to an estimate of the German butchers' association, 60 per cent of all meat processing is handled by this trade. Tinned meat is, for instance, beef or pork in its own juice, corned beef, boiling sausages or roast sausages. The temperature reached during sterilisation of such tins is 121° C on average. Heating time depends on the size of the tin and its contents and varies between 47 and 130 minutes. Prolonged heating affects the flavour, texture and sensory appeal of the sterilised meat, and therefore, a trend has emerged for short-time high-temperature processes /4.1-1/.

To make ham, beautiful meat without tendons is used. It is cured for several weeks, ie treated with sodium nitrate or potassium nitrate and/or sodium nitrite, then rinsed, dried and smoked. Smoking times and processes depend on the kind of product. For instance, boiled ham has to be boiled after smoking. Bacon is salted, rinsed, dried, and cold smoked.

Sausages consist of minced muscle meat, offals, fat, salt, spices, and other additives. The minced meat is filled into tins or casings. Sausages are divided into raw sausages, boiling sausages, and cooked sausages. Raw sausages are, for instance, salami which consists of raw skeleton muscles, fat, and spices; all ingredients are minced, filled into casings, which are dried and smoked. In the case of cooked sausages, for instance liverwurst (German 'Leberwurst'), the starting material is cooked, minced and then filled in tins or casings which are then heated again (tins to 110° C and casings to 85° C /4.6-8/).

To make boiling sausages, for instance wienerwurst, all ingredients are thoroughly blended with water or ice. This mince is filled in casings which are then smoked at 75° C, or filled in tins that are sterilised at 110° C.

Patés are cooked meat products of a different style. They consist mainly of veal and pork, calf's and pig's fat; some are made of poultry or game, and all kinds are typically free of offals and low-quality meat.

4.1.3.2 Energy input in meat processing

The energy input is indicated for boiled, cooked, and raw sausages and is representative of all other processing of meat products. All processes start with slaughtering, comprising the killing of the animal, bleeding, scalding, plucking, eviscerating, and cooling. /4.1-7/ states an energy requirement of 0.52 MJ per kilogram of beef and 0.76 MJ per kilogram of pork.

Figs. 4.1-2, 3, 4 show the energy flows involved in making the individual products. Due to the different composition of beef, pork, and ingredients, the values for slaughtering vary. The type of tin chosen here is what is called a 3/4-preserve which is widely used today. Unlike the full preserve, the 3/4-preserve must not be stored at a temperature above 10° C. The difference in values is due to the fact that brine is added in the making of boiled sausages.

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4.2 Vegetable oils and fats

In chemical terms, fats and oils are the glycerin ester of fatty acids. More than 200 natural fatty acids are known, most of them being monobasic, even-numbered, straight-chain aliphatic carboxylic acid with 2 to 26 carbon atoms. Their degree of saturation varies. In addition, there is a less significant number of branched odd-numbered acids.

Fatty acids are divided into unsaturated and saturated acids. They are saturated when the carbon chains have no multiple bonds. Unsaturated acids, on the other hand, have one or several double or triple bonds.

Fats are often divided into drying and non-drying fats, due to their behaviour in atmospheric oxygen. Drying oils, for instance linseed oil, are able to absorb and polymerise oxygen, producing hard and resinous products.

Apart from carbohydrates and proteins, fats are a third group of nutrients, and at 37.7 kJ/gram, their energy is double that of carbohydrates or proteins. Dietary fats are either vegetable fats or animal fats. Synthetic fats are less suitable as a diet; they are produced by oxidation of paraffins to fatty acids, followed by esterification with glycerin.

4.2.1. Oil plants

Vegetable fats are divided into two main groups, ie fruit pulp fats and seed fats.

4.2.1.1 Fruit pulp fats

Fruit pulp fats form a small, but significant group of vegetable fats. The most important examples are olive oil and palm oil.

Olives are fruits of the olive tree, an evergreen tree with many branches; it grows up to 20 metres high. Olive trees are up to 700 years old; they grow mainly in the Mediterranean countries, as well as in California, Arizona, Argentina, South Africa, and Australia. Harvest is cumbersome and can hardly be mechanised. For the most part, it is processed to oil, with olive oil being a staple food in these countries.

Palm oil is obtained from the fruit pulp of the oil palm. It grows usually to a height of 20 metres. With 4,500 kilograms of total fat per hectare it is by far the highest yielding oil plant. The oil palm grows in a wide tropical belt in Africa, south-east Asia and Latin America.

4.2.1.2 Seed fats

All seeds contain at least some fat. However, only a few can be used for commercial fat production. Seed fats are divided into those produced from perennial and those from annual plants. Perennial oil plants are usually trees; their fruits are larger than those of annuals. Annuals are more profitable and more important than perennials because mechanical harvesting can be applied. Palms are perennials. The seeds of oil palms, described in Chapter 4.2.1.1, are dried (often in flue gases) and then exported.
The coconut tree or palm is up to 30 metres high and sets 25 fruits per year. The tree is not grown for more than 30 years on plantations for reasons of yield. It grows in all continents, except Europe. The fruit is ruptured, the pulp scraped out and dried. The dried copra is shipped in bags.

Babassu fat is the third kind of fat produced from palm fruits. Babassu trees grow in the area of the Amazonas river to a height of almost 10 metres. Babassu nuts are very hard; one palm tree produces about 125 kilograms of kernels. Palm oils and fats are important raw materials for producing edible fats.

The second type of fat obtained from the fruits of perennials is vegetable tallow. Characteristically, this type has a very narrow melting interval, in that it is hard at room temperature and melts completely at body temperature. It is therefore referred to as plant butter although it does not compare with butter.
The most important example of this group is cocoa butter which is produced from the seeds of the coconut tree cultivated in the entire tropical belt. The fruits contain up to 57 per cent of fats. Cocoa butter is one of the most expensive vegetable fats; it keeps extremely well and is mainly used in the chocolate and confectionary industry.

Annuals produce by far the largest share of vegetable fats for human consumption. They are all commercially grown, and in most cases mechanical harvesting is applied. Some of the most important annuals are discussed below.

One of the most significant annual is the sunflower, which ranks among the top oleiferous plants in Europe. The main countries are the Balkan countries, south Russia; outside Europe, the sunflower is also grown in China, India, Australia, America and Africa. Sunflower oil is a premium edible oil which is used in margarine and mayonnaise production.

One of the oldest European oil plant is rape, or colza. It is cultivated in Europe, Canada, Argentina, Japan, and Taiwan. Rapeseed oil contains, among other acids, erucic acid which is a major contributor to fatty degeneration of the heart muscle cells in humans. When this finding became first known in 1960, erucic acid-free varieties were bred. Because of its poor flavour, rape-seed oil can only be used as an edible oil after refining.

All cereals contain considerable amounts of oil in their germs. Maize oil, rice oil and wheat-germ oil are the most important industrially produced corn oils.

Soy-beans and peanuts are among the oil plants with many uses. Soy-bean, originally grown in China, is cultivated in the U.S.A., supplying all the western world. Soy-bean oil is used in margarine, in cooking, frying, and baking fats.

Peanut is cultivated mainly in Africa, India, China, South America and Indonesia. After pollination, the peduncle is thrust from the base of the flower towards the soil and fertilised ovules are carried well below the soil surface which has to be light and sandy. Peanut oil keeps exceptionally well, which makes it a popular edible and cooking oil.

Cotton-seed oil plays an important part worldwide. The quantities produced depend, of course, on cotton production. Cotton belts exist in the U.S.A., U.S.S.R., China, Brazil, India, and Egypt. Raw cotton-seed oil is dark red. After refining, it is mainly used in diets. If it is to be used as a table oil, it needs to be winterised (see also Chapter 4.2.3). The majority is processed into cooking , baking, or frying fat, or margarine.

4.2.2 Production of dietary fats

4.2.2.1 Production of fruit pulp fats

Oil fruits cannot be stored for a long time because of their high water content and activated enzymes, so that oil production has to start soon after harvest. This also explains the decentralised structure of the industry.

To produce olive oil, the olives and kernels are first crushed, then mixed and kneaded in malaxeurs, with salt being added. During this stage, the oil coagulates and forms major droplets. Then, the oil-bearing material is expressed several times and ducted into a container for clarifying the oil by settling. This is sometimes done in separators. Palm oil is produced in a similar manner but the fruit pulp is sterilised in steam as a first step, and the kernel is removed by cooking.

4.2.2.2 Rendering of seed fats

After harvest and transport to the oil mill, the oil seeds are cleaned, peeled and reduced in size. This is done by machine and aims at breaking the texture and the cells to make oil discharge easy during pressing. Depending on the type of seed, cylinder systems are used which contain one or several pairs of either smooth or riffled flaking rolls. Large-size seeds, like copra, are crushed first. Then follows the conditioning stage, ie heating in dry saturated steam and adjusting a certain moisture content which causes the remaining intact cells to rupture. Before actual pressing, the material is dried and heated. Pressing is done in several stages, with the press design being specific to the type of seed pressed.

Extraction is another way of obtaining oil from cells. In this method, a solvent is used to dissolve the oil, which is washed out and then separated from the solvent by percolation and distillation. The solvent is commercial hexane.

4.2.3 Processing of dietary fats

Only a very small share of natural fats is for direct human consumption after rendering or extraction. This includes the first few pressings of olive oil, sunflower oil and cacao butter. Most of it is not edible as raw fat. Its undesired and partly toxic components need to be removed and the fat itself needs to be modified according to subsequent use.

The first part of processing is refining, which is done in four stages. First, degumming, ie removing all mucilaginous substances with phosphoric acid and bleaching earth, and filtering at 80 to 90° C. This is followed by neutralisation, or deacidification, which determines the quality of the final product. Sodium hydroxide solution (caustic soda solution) can be used to remove free fatty acids from the mass. The free acids combine with the caustic soda solution to form sodium soap which is separated in the form of soapstock, ie a concentrated aqueous soap solution. To remove remaining soap residues, the fat is then washed and dried. This procedure can be applied to all raw fats.
A second method of neutralising the mass is distilling. However, it only works with raw fats containing at least three per cent of free fatty acids. The free fatty acids are vaporised from the raw fat and collected in a condenser. They can be used as industrial by-products.

Neutralisation is followed immediately by bleaching which removes undesired colour bodies, soap residue, heavy metal traces, auto-oxidation products, as well as residual amounts of phosphorus-bearing substances. In other words, bleaching is more than just brigthening of the colour. The resulting bleaching earth is extracted with hexane and then either discarded or burnt as a low-grade fuel.

The last refining stage is deodorisation, ie removal of flavours and odours. Complete removal of heavy and lipoid-solving compounds is performed in a vacuum steam distillation process.

Some edible vegetable oils separate solid components after long storage at under 5° C. These are usually glycerides and wax that impair the quality of the product. Such fats are cooled, and then solid components are removed. Sunflower oil, for instance, is also treated in this way.

The process is referred to as winterisation or destearination, and it follows refining. A major share of fats obtained from plants are liquid. However, only some of it is marketed as oil, and the remainder is hardened. The production of hard fats consists of catalytic hydration of liquid and/or low-melting fats, thereby being converted to higher-melting fats. The hardened products are not fully saturated because fully hydrated fats have a very high melting point which makes them hardly resorbable and unsuitable as dietary fats. Before hardening, the fats must be well cleaned. A common catalyst in the process is nickel; but copper is also used in spite of the fact that its life as a catalyst is shorter. The hydrogen required for hydration is normally produced in-house.

The fat is mixed with the catalyst suspension and hydrated under a hydrogen pressure of 1 to 3 bar. The process starts at a temperature of 150° C and continues until 200° C is reached. The reaction heat required to hydrate one tonne of fat is 0.966 MJ, nickel consumption is 20 grams per tonne, taking into account its re-use. After hydration, the fat undergoes re-refining to reduce mainly the nickel content. Hard fats are mainly used in the production of margarine, cooking, frying, and baking fats.

4.2.4 Margarine

Thanks to the advances made in the field of fat science, margarine has developed into a foodstuff of its own right. Depending on its use, a number of different qualities are marketed. The base materials of margarine are dietary fats, potable water and milk and/or milk powder.

In the production of high-grade vegetable margarine, water or milk powder is dissolved in water. The water should have a pleasant flavour and odour, it must be hygienically safe and have a pH of about 6. Its quality must be monitored continuously. A biological acidification process produces aromatic ingredients of the kind obtained in butter making. If these ingredients are insufficient additional flavourings are added to adjust the concentration. Emulsifiers are added in any case, helping the formation of a water-in-oil emulsion. Margarine consists of 80 per cent of fat and 20 per cent of water.

The emulsion is mechanically treated at a low temperature. The cooled mass is taken to a resting tube in which recrystallisation occurs. The resulting product is margarine that is ready for packaging.

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4.3 Energy input in the food processing industry (FRG)

The average specific consumption of energy in the food industry is higher than the general industrial mean, although there are wide variations in individual branches of industry. For instance, in 1981, oil mills used 865 tonnes of hard coal equivalent (HCE) per 10^6 DM worth of food, while the total vegetable fat production used 171.5 tonnes of HCE/10^6 DM. The figure for fruits and vegetables is 141.3 tonnes of HCE/10^6 DM, for fish 37.4 tonnes of HCE/10^6 DM, and for meat 97.6 tonnes of HCE/10^6 DM.
Between 1973 and 1981, the specific consumption of energy decreased by six per cent, in some sectors of industry by over 20 per cent. However, there are also industries which consumed more in 1981 than in 1973 which is explained by a growing production of energy-intensive products. At 1.7 per cent of the total cost, energy costs in the food industry are relatively low compared with the rest of the industry, although costs are rising here, too.

In 1981, the entire food industry consumed 6.1 million tonnes of HCE, which accounts for 5.9 per cent of the total industrial final energy requirement. In the last few years, there has been a clear shift away from fuel oil and coal towards gas and electricity, although heavy fuel oil is still the dominating form of energy besides gas.
In the period between 1973 and 1981, the share of fuel oil in energy input in the food industry dropped from 70 to 55 per cent, while gas rose from 11 per cent to 25 per cent and electricity from 9 to 13 per cent. Coal dropped slightly from 9 to 7 per cent. Table 4.3-1 lists the figures of individual branches of industry /4.1-7/.

5. WOOD PROCESSING INDUSTRY

5.1 Introduction

The UNIDO Study IS.437 entitled "The use of residues for energy production in the mechanical wood processing industry" has given a detailed account of the energy aspects involved in the wood processing industry. To avoid repetition, a discussion of the most important production processes and the energy-intensity of individual products is not considered in the present study. Rather, this study focusses on further considerations of a more efficient use of energy in some stages of wood processing, in particular wood drying. In addition to that, further information is provided on the energy production from residues and the potential of reaching energy self-sufficiency in the wood processing industry.

5.2 Wood drying

As a raw material, wood contains large amounts of water, ie between 40 and 200 per cent relative to the absolutely dry (kiln-dry) wood. To avoid shrinking of the finished product, the water content has to be reduced to between 6 and 14 per cent, depending on the application.

While open-air drying, or natural wood drying, is still employed in developing countries, it is rarely used in industrialised countries, at best for pre-drying. Open-air drying means leaving the cut timber in the open air to dry, without applying additional energy to promote the process. In central Europe, the moisture content can be reduced to 14 or 15 per cent in the summer, and 20 to 22 per cent in winter in this way /5-1/.

The increasing use of technical aids (artificial drying) is characterised by a number of advantages compared to open-air drying, in that they

- shorten the drying period,
- cut stocking cost and thereby reduce capital binding,
- accelerate turnover,
- help reach the correct final moisture content of the wood,
- reduce stress due to an even reduction of the wood moisture,
- kill molds and insects,
- avoid quality losses caused by discoloration, deformation and cracks.

However, large amounts of energy go into technical drying methods. In the Federal Republic of Germany, for instance, 60 to 70 per cent of all energy input in wood processing is currently consumed in drying processes /5-2/.

5.2.1 Convection drying

The most widely applied drying method today is convection or fresh-air drying. Fans draw in fresh air from the atmosphere; it is mixed with circulating air, then heated and blown over the wood to be dried, thereby removing moisture from the wood. Some of the air filled with vapour is ducted to the outside atmosphere and has to be replaced by fresh air. The principle is illustrated in Fig. 5-1.

The most important parameter applied to optimum drying is the drying climate. If, towards the end of the drying process, the climate becomes too dry, the air in the drying chamber can be wetted by sprinklers.

The drying plant, either drying chambers or drying tunnels, are available in metal or brick construction. They are fired by various forms of energy /5-3/.

Due to the continuing exchange of fresh air and waste air, considerable amounts of energy are spent in heating the fresh air in this process. Therefore, a number of concepts have been developed in the past few years to ensure that a more efficient use be made of energy.

One way of saving energy is to recover the waste heat contained in the flow of waste air. Depending on the type of media contained in the waste air, plate heat exchangers made of glass, aluminium or stainless steel are employed for preheating the fresh air flow. The amount of energy recovered is a function of the intake air temperature; according to manufacturers' information, an average of 10 to 20 per cent /5-4, 5/ can be recovered in a plate-type heat exchanger.

An interesting concept has also been suggested for computerising the combined heat-using system. In a plant, the drying chambers are interconnected by ducts, and the waste air of one chamber is used to heat the air in another chamber. A computer collects data on temperature and climate from the various chambers, processes it and controls the exchange of air /5-4/.

In convection drying, an important cost factor is the electrical energy required for circulating the chamber air. The volume of air needed to transport both humidity and heat decreases to a minimum level in the course of the drying process. For this reason, fans that run at a single speed and handle a maximum volume of air (when drying starts) draw too large a volume as the drying proceeds. This can be avoided by using fans with a variable speed motor. Table 5-1 shows how much electricity can be saved at various speed settings. Optimum speeds and correct air velocities in the drier can be computer-controlled or manually set. When computer-controlled, not only moisture content but other parameters relating to the chamber and specific wood properties can be considered as well /5-6/.

Convection driers can be fired by wood residue, as will be discussed in Chapter 5.3 below, but solar radiation may also serve as a source of energy for heating the fresh and/or circulating air inside the drier. Tests with solar driers have been carried out in the tropical climate of Puerto Rico, India, Madagaskar, as well as in central European climate. Compared with open-air drying, the final moisture content that can be reached with solar driers is significantly lower (down to 8 per cent). In mean moisture contents of under 30 per cent, the drying period can also be cut significantly. Since solar driers use energy for the fans only, the specific consumption of energy is by two-thirds lower than in conventional convection driers, although longer drying periods have to be accepted /5-7/.

5.2.2 Condensation drying /5-3/

Condensation drying is based on a closed circuit of drying air, with a heat pump being interconnected. Some of the air flow is cooled and dehumidified in the drying circuit, and the heat released, which is increased by the thermal equivalent of the compressor power, is recycled to the air flow. Additional fans ensure the air velocity required in the drier, which is 1.5 to 2 metres per second. The principle applied to drying wood with an air-to-air heat pump and partial dehumidification is shown in Fig. 5-1. The chamber design is basically the same as in convection drying.

Due to the fact that the air is circulated and evaporation heat is recovered heat is only required to cover radiation losses and leaks in the chamber. In relation to fresh-air drying, the primary energy consumption can almost be halved in condensation drying.

The energy flow diagrams in Fig. 5-1 illustrate this fact, using an example of drying 50-mm thick sawn oak timber from 50 per cent to

10 per cent of moisture content. In this case, the useful energy is purely the hot moisture evaporated from the timber.

Low-energy compressor power and low chamber temperature necessarily extend the drying period considerably. In the example given, the drying period doubles to 6 weeks in relation to fresh-air drying periods. In view of the fact that it is better to dry softwood and light hardwood more rapidly and at higher temperatures, condensation drying is more suitable for heavy wood.

5.2.3 Vacuum drying /5-1/, /5-3/

The law of physics that applies to vacuum drying is that a decrease in pressure surrounding the material to be dried makes the boiling temperature drop and the water transport velocity in the wood rise. In practice, two process variations are applied, both of which operate in the low vacuum range.

In the continuous method, heat is transferred to the wood through heating plates which are placed in between layers of timber. The plates are interconnected through flexible hoses with hot water circulating in the system. Some of the liquid extracted from the timber by evaporation precipitates on the drier casing and mainly on the cooling surfaces in the drier.

In discontinuous vacuum drying, vacuum drying periods alternate with normal pressure heating periods. This cyclical operation extends drying times. However, since there are no heating plates, the expenditure is reduced in terms of manpower, capital cost, and loading and unloading times.

Vacuum driers consist of horizontal steel cylinders with useful capacities of up to 20 cubic metres. Investment costs relative to the capacity are very high, and their control is also more complicated than in conventional driers.

The energy required in vacuum drying is divided into thermal energy for heating the timber, and electrical energy for powering the vacuum pumps and fans.

5.2.4 Special Processes

In special applications, a number of other timber drying methods are used, including
- infra-red drying,
- high-frequency drying,
- high-speed drying,
- high-temperature drying.

Within the general context of application, the significance of these methods is judged as rather low.

To utilise some of the specific merits of the above methods, combinations of several technologies are sometimes applied to individual drying stages.

A completely new process has been discussed for some time in the veneer and chipboard industry. Press driers that work at a pressure of about 35 N/cm^2 and a temperature of some 150° C use both mechanical and thermal drying methods. Finnish test results seem to suggest that a relatively favourable specific energy consumption can be expected of this method /5-8/.

5.2.5 Energy input in the most important drying methods

It is hardly possible to make a direct comparison between energy input data of various drying methods because of the multitude of parameters that are specific to individual types of timber, as well as to plants. The figures presented in Table 5-2 relate to drying 50-mm thick oak sawnwood of an initial moisture content of 50 per cent and a final moisture of 10 per cent /5-3/. Here again, it should be noted that these values may vary in absolute terms and in relation to each other.

5.3 Energy production from wood residues

In the timber trade, waste is produced in logging and processing in the form of bark, slabs, chips and sawdust. In the Federal Republic of Germany, 54 per cent of the total timber waste of 14 million cubic metres is used as feedstock in the paper and pulp industry, as well as in the chipboard and fibreboard industry. Some 14 per cent is dumped, and about 32 per cent is already being utilised for the production of energy /5-8/. In developing countries, the percentage of waste usage is much lower because of the absence of both chipboard and fibreboard manufacturing.

The fundamentals of energy production from residues have been covered in UNIDO Study IS.437. As a supplement, further demonstrated methods of utilising waste are described in the following sections.

5.3.1 Direct combustion

This is the most widely applied form of using logging waste. Both the size of waste and the calorific value, that varies a lot (Table 5-3), /5-9/ determine the type of charge and firing.

The fuel is introduced manually, via blowing devices (chips or dust), or mechanical conveyors (bark, lumpy wood). Frequently, units are fitted upstream for drying (using flue gas), or for crushing the fuel to be used. In the combustion chamber itself, oil or gas burners provide the temperature required for ignition.

The thermal energy from flue gas is often applied directly to thermal processes. Two examples illustrate the procedure:
Fig. 5-2 shows the setup of a solid-fuel-fired chip drying plant developed for the chipboard industry. This plant was designed and manufactured in Sweden and utilises all types of wood residues. Firing is

on a zone feed grate, with the material to be burnt passing through three zones , ie drying, burning, and burning-out zone. Flue gas from the combustion chamber is mixed with fresh air and recirculated flue gas; it is then fed to the drier via a separator at a temperature of 450 to 500° C. This basic concept can be integrated into established plants and produces 2 to 14 MW /5-10/.

Flue gases are also used directly in a bark and chip disposal plant at Wackersdorf, Germany. At full load, a drier dries 23.4 tonnes/h of bark and chips containing between 103.5 and 35.1 per cent of moisture. Some of the total material to be dried (ie 15.5 tonnes per hour) is used as fuel (2.3 t/h) in the plant itself. The remainder is pelletised, which makes it easier to store and to transport, apart from the fact that pellets have a higher calorific value than the wet material /5-11/.

Wood-fired hot water and/or steam boilers are used in the timber industry both for generating thermal energy (heating presses, drying kilns, etc.) and for producing power through steam engines and steam turbines. The piston steam engine, in particular, has become a well-proven power unit for generators. It has a modular design, with the ratings of the various cylinders varying between 30 and 220 kW, depending on the size. By coupling up to ten units, an individual adjustment is possible. The steam engine is profitable wherever all waste steam can be used for heating presses, drying chambers or operating spaces. It can also be used for directly powering processing machinery, in which case the investment costs are lower because no power generator or motor is required /5-12/.

Because of their power range, steam turbines are only profitable in the large companies of the timber industry.

5.3.2 Wood gasification

Another way of producing heat and power from low-grade wood residues, without passing through the generation of high-pressure steam in boiler plants, is wood gasification. The following example describes how a sawmill covers its own needs in an isolated operation.

It is the sawmill run by H. Schmaus at Morbach, West Germany, cutting 3,000 solid metres of round timber per year. They also process sawnwood on a small scale. Fig. 5-3 gives an overview of the existing wood gas power plant. The gas generator uses the system of rising gasification in a cylindrical refractory-lined shaft furnace. The system utilises small-size waste (bark, chips) of a moisture content of up to 100 per cent. The conversion of wood gas into electrical energy and heat is done in a 12-cylinder gas engine of the same type as used in unit thermal power plants. These engines have to be converted to using wood gas because the calorific value of wood gas is much lower than that of natural gas. The entire unit is designed for electrical operation at 130 kW and consists of a gas engine, a three-phase generator, cooling water and waste gas heat exchangers, and the electrical switching control. At full load, about 800 MJ of waste heat is produced per hour; it is used for drying and heating. At part load the plant is working at 55 per cent on average, and, due to its engineering principle, the gas generator produces only as much gas as is required; it therefore adapts to every load change. Condensates from gas scrubbing are treated in a mechanical and biological plant because it is impossible for environmental reasons to discharge phenolic waste water into the ground water. In view of the relative wetness of wood residues when turned into gas, the volume of condensate can be expected to be up to one litre per kWh produced /5-13/.

In the Federal Republic of Germany, block units are now available applying the above principle and producing 125 to 130 kW of electrical power. If higher outputs are required, several engines are combined in a modular system. In a mains-parallel mode of operation, this kind of plant can meet every demand for electricity or heat. Depending on the demand, an electronic control can either switch individual engines on or off /5-14/.

Wood gas generators that are suitable for direct drier firing can compete with wood-fired boiler plants. According to /5-5/, for instance, the investment cost of a wood gasifier connected with a wood drier of a capacity of 100 cubic metres is only half the cost of a corresponding boiler plant.

A more recent technology is fluidised-bed gasification. The system has been well proven in the U.S.A. although it needs to be of a certain size to be economically justified. The smallest plant demonstrated so far runs on 3.8 tonnes of wood residue per hour /5-15/.

5.4 Other opportunities for saving energy

More than in other industries, heat losses from ventilation play an important part in the wood processing industry. Exhausters used for dust, chips, etc., that have air throughputs of up to 100,000 cubic metres per hour, cause losses from ventilation heat four to five times as high as the heat losses from transmission /5-16/. This also explains why, in furniture making, the share of energy consumed for space heating and ventilation makes up almost 50 per cent of the total energy that goes into making a finished product /5-8/. To reduce the heat losses from ventilation, the waste air flow has to be reduced in the first place. This can be done in several ways:

- In many cases, the actual forced-draught volume flows are much higher than required. The extraction capacity can be adjusted to actual needs by retrofitting baffles or by changing the intake cross sections /5-8/.

- Exhausters are usually switched on in the morning and switched off at the end of the shift, whether or not the machines are running.

The problem can be overcome by coupling the electrical supply of the machine with that of the exhauster /5-17/.

- On-times can be minimised by decentralising exhausters and allocating them to smaller production units /5-16/.

- In fully or partly encapsulated machines, the intake air can be drawn directly from the atmosphere /5-16/.

If the above measures have been implemented or, if they cannot for lack of space, then heat recovery should be considered. Due to the amount of dust or solvents contaminating the waste air, however, it is sometimes difficult to recover heat in the wood industry. In many sectors of the industry, cloth filters are used for environmental reasons. They allow for some recirculation of air (up to 84 per cent). Heat exchangers can also be applied if their exchanger surfaces can be cleaned easily /5-18/.

To cut the space heating requirement, the following additional measures can be taken:

- maintaining reasonable room temperatures through suitable controls /5-8/;
- lowering room temperatures during non-working hours (at night and for the weekend);
- minimising volumes of heated spaces; separating production spaces from storage spaces /5-16/;
- improving thermal insulation of buildings.

Much energy is lost and escapes recovery in compressed air production. Due to leaking pipes up to 30 per cent of the compressed air escapes. Most of the drive energy conveyed to the compressors is converted directly into heat and is usually driven off by cooling fans. Suitable positioning of cooling ducts ensures redirection of the hot air to the atmosphere or recirculation into production and storage facilities during the cold season /5-16/. Waste heat from water or oil-cooled compressors can also be utilised for space heating and industrial water heating after passing through heat exchangers /5-19/.

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Type of furnace	Firing Oil	Gas	Resistance-heated	Induction-heated
Crucible furnaces	Melting and holding	Melting and holding	Holding	-
Hearth furnaces	Melting and holding	Melting and holding	Melting and holding	-
Rotary kilns	Melting of large pieces under a cover of salt	-	-	-
Induction furnaces with and without channel	-	-	-	Melting and holding

Table 2.1-1 Common types of furnaces and firing / 2.1 - 6 /

Type of furnace	Capacity kg	Melting-down power kg/h	Energy consuption per 100 kg of metall Melting[1)]
Crucible furnaces			
Gas or oil heated / M J /	100	90-120	670-730
	175	140-180	590-670
	250	180-220	500-570
Resistance - heated / kWh /	150	55-65	75
Induction - heated (line-frequency) / kWh /	1000	400-550	56-57
	3000	1000-1300	53-54
Hearth furnaces			
Gas or oil - heated / MJ /	2500	800	380-460
Resistance - heated / kWh /	3000	400	48

1) Depend from furnace design, raw material, operation conditions

Table 2.1-2 Final energy consumption of different types of furnaces /nach 2.1-6/

	Fuel	Electricity
Drilling		0.6 kWh
Explosives for blasting	14.2 MJ 1),2)	
Shovel loading		2.4 kWh
Hanlage	43.0 MJ 3)	
Crushing and grinding		15 kWh
Floatation	0.4 MJ 2)	
Total	56.7 MJ	18 kWh

1) Ammonium nitrate
2) Fuel oil
3) Diesel oil

Table 2.2 - 1 : Energy consumption for mining and concentrating 1 tonne of copper ore

	Conv.	Outokumpu	El. Matte	Brixlegg	Worcra
Electricity : / kWh/t /					
Total	316	416	720	932	432
- generated	230	330	-	210	360
- purchased	86	86	720	722	72
Fuel oil / GJ/t /	6.3	3.8	0.7	0.8	3.8
Coal / GJ/t /	-	-	-	2.1	-

Table 2.2 - 2 : Energy consumption for various pyrometallurgical methods of copper extraction, relative to one tonne of concentrate input

Process	Fuel input GJ/t Zn	Electricity input kWh/t Zn	Zn-yield %	Zn-purity %
Muffles	45..55	2..50	86..94	97.5..98.5
New Jersey	34	175	93	99..99.6
Imperial Smelting	30	400	92	.
Josephstown	24..26	2600	92	.
Sterling	10	3470	95	98.3
Duisb. K.-Hütte	9	3640	84 (+ 15	99.96 90)

Table 2.3 - 1 : Pyrometallurgical extraction of Zn

Table 3.1-1

Temperature cycle of clinker formation

Stage	Reactions	Temperature range
I Dehydration	(a) Evaporation of water that is not chemically or mineralogically bound (from raw meal, raw-meal grains, raw slurry)	Room temperature to about 120° C
	(b) Removal of adsorbed water from clay minerals and intermediate layers	100 - 200° C
	(c) Removal of hydroxyl groups from clay minerals	250 - 700° C
II Decomposition of $CaCO_3$ CaO formation	(a) Early stage of CO_2 splitting in contact with SiO_2, Al_2O_3, Fe_2O_3	over 550 - 600° C
	(b) Main reaction of CO_2 splitting by thermal dissociation, CaO_{free} 15 to 20 per cent	800 - 1,000° C very fast above 900° C (atmospheric pressure)
III CaO solids reactions	(a) see II a; formation of $CaO \cdot Al_2O_3$; $12\ CaO \cdot 7\ Al_2O_3$ and $2\ CaO \cdot SiO_2$; free CaO content is 2 per cent	over 550 - 600° C
	(b) Formation of more C_2S increases at 1,000° C max.; formation of C_3S; decrease of CaO_{free} and C_2S, rise of C_3S	600 - 1,000° C 800 - 1,000° C
	(c) C_2S lattice disorder, improved solubility in part-melt	> 1,400° C
IV Sinter reactions	(a) Part-melt in the presence of 2 per cent of alkali (max.)	over 1,280° C
	(b) First part of the melt at invariance (eutectic: 54 % CaO, 23 % Al_2O_3, 17 % Fe_2O_3 and 6 % SiO_2	1,338° C
V	Digestion of major mineral grains; steady-state	1,400 - 1,450° C

Source: Möglichkeiten zur Energieeinsparung in der Zementindustrie /3.1-4/

Table 3.1-2

Specific energy requirement

Type of energy required by process stage	Production method		
	Dry	Semi-dry	Wet
Thermal requirement			
Raw material preparation	0.73	0.57	
in GJ per tonne of clinker			
in GJ per tonne of cement	0.59	0.46	
Clinker firing			
in GJ per tonne of clinker	3.14	3.38	5.44
in GJ per tonne of cement	2.57	2.77	4.46
Electrical energy requirement			
Raw material preparation and clinker firing			
in GJ per tonne of clinker	0.14	0.11	0.08
in GJ per tonne of cement	0.12	0.09	0.07
Cement grinding			
in GJ per tonne of cement	0.25	0.25	0.25
Total thermal requirement per tonne of cement			
in GJ/tonne of cement	3.18	3.24	4.46
Electrical energy requirement per tonne of cement			
in GJ/tonne of cement	0.37	0.34	0.30
Total energy requirement per tonne of cement			
in GJ/tonne of cement	3.54	3.58	4.78

Source: Möglichkeiten der Energieeinsparung in der Zementindustrie /3.1-4/

Table 3.1-3

Energy input to dry process

Works size (cement output in 10^3 t)	Total energy consumption (10^6 GJ)	%-share of energy thermal	%-share of energy electrical	Specific energy consumption (GJ)
to 200	11.13	90.8	9.2	3.95
201 - 500	34.73	90.3	9.7	3.75
501 - 1,000	48.76	89.8	10.2	3.45
over 1,000	24.13	89.6	10.4	3.30
total and/or mean	118.75	89.7	10.3	3.54

Source: Möglichkeiten zur Energieeinsparung in der Zementindustrie /3.1-4/

Tunnel kiln zones	Temperature range °C	Spec. heating and cooling values C/h	Time h	Remarks and possible reactions
Heating zone	0 - 300	30	10	Residual moisture, pore water
	300 - 400	50	2	
	400 - 600	20	10	Water of Crystallization separation
	600 - 900	25	12	Decomposition organic, combustible materials, etc.
	900 - 1050	50	3	
Final firing zone	1050	-	6	Final firing time
Cooling zone	1050 - 650	60	5	
	650 - 450	12	16	
	450 - 30	52	8	

<u>Table 3.3-1</u> Tabular representation of time-firing curve /3.3-1/

Energy centre	GJ/tonne (melted)
Mixing	0.04
Melting and annealing	10.39
Cutting and temporary store	0.21
Warehouse	1.02
Auxiliary maintenance	0.09
Maintenance	0.04
Main services	0.02
Administration	0.03
Transport	0.03
Total	11.87

The following are the energy centres used for this process:

Energy Centre	*Description*
Mixing	Mixing room and raw material stores
Melting and annealing	Furnace, bath, lehr and compressed air
Cutting and temporary store	Automatic warehouse and immediate environs
Warehouse	Stockrooms, packing and despatch
Auxiliary maintenance	Sewing room, pot rooms, tackle shop and stores
Maintenance	All branches
Main services	Steam
Administration	General offices, R & D, laboratory, canteen, welfare and surgery
Transport	Yard traffic, office, locos etc.

TABLE 3.4-1

Energy use in Energy Centres per tonne of glass melted (Flat Glass)

	Fruit Vegetable	Fish	Meat	Vegetable fats
Energy consumption 1981				
1,000 tonnes HCE	230.7	25.6	390.1	239.6
%-chance 1973:1981 %	+ 5.4	- 6.2	+ 76.6	- 19.9
Fuel oil 1981	144.3	13.7	192.3	56.3
1,000 tonnes HCE				
%-chance 1973:1981 %	- 27.9	- 31.6	+ 34.3	- 55.1
Gas 1981	59.2	5.0	109.5	123.9
1,000 tonnes HCE				
%-chance 1973:1981 %	+ 717.6	+ 230.0	+ 310.5	- 8.1
electricity 1981	26.3	6.8	86.6	49.5
1,000 tonnes HCE				
%-chance 1973:1981 %	+ 36.9	+ 29.4	+ 130.0	+ 40.0
coal 1981	0.8		1.7	10.0
1,000 tonnes HCE				
%-chance 1973:1981 %	- 72.2		- 87.3	+ 171.9
%-share energy cost of total cost 1980	2.2	0.7	0.9	1.3
Specific energy consumption tonnes HCE/10^6DM net production value 1981	141.3	37.4	97.6	171.5

Table 4.3 - 1 Energy use in the food industry (FRG) / 4.3 - 7 /

Criterion	1 set speed	Pole switching 2 set speeds	Pole switching 3 set speeds	Frequency converter infinitely variable speed
r.p.m.	1	2 75 %, 100 %	3 50 %, 75 %, 100 %	ad lip actual: 50 - 100 % of nominal speed
Cost per unit incl. share of control cabinet	100 %	115 %	130 %	200 - 300 % dep. on no. of fans
El. saving against set speed	-	30 - 35 %	35 - 45 %	45 - 53 %
Pay - back period against one - down system	-	ca. 3 mon.	ca. 6 - 12 mon.	ca. 3 - 5 years

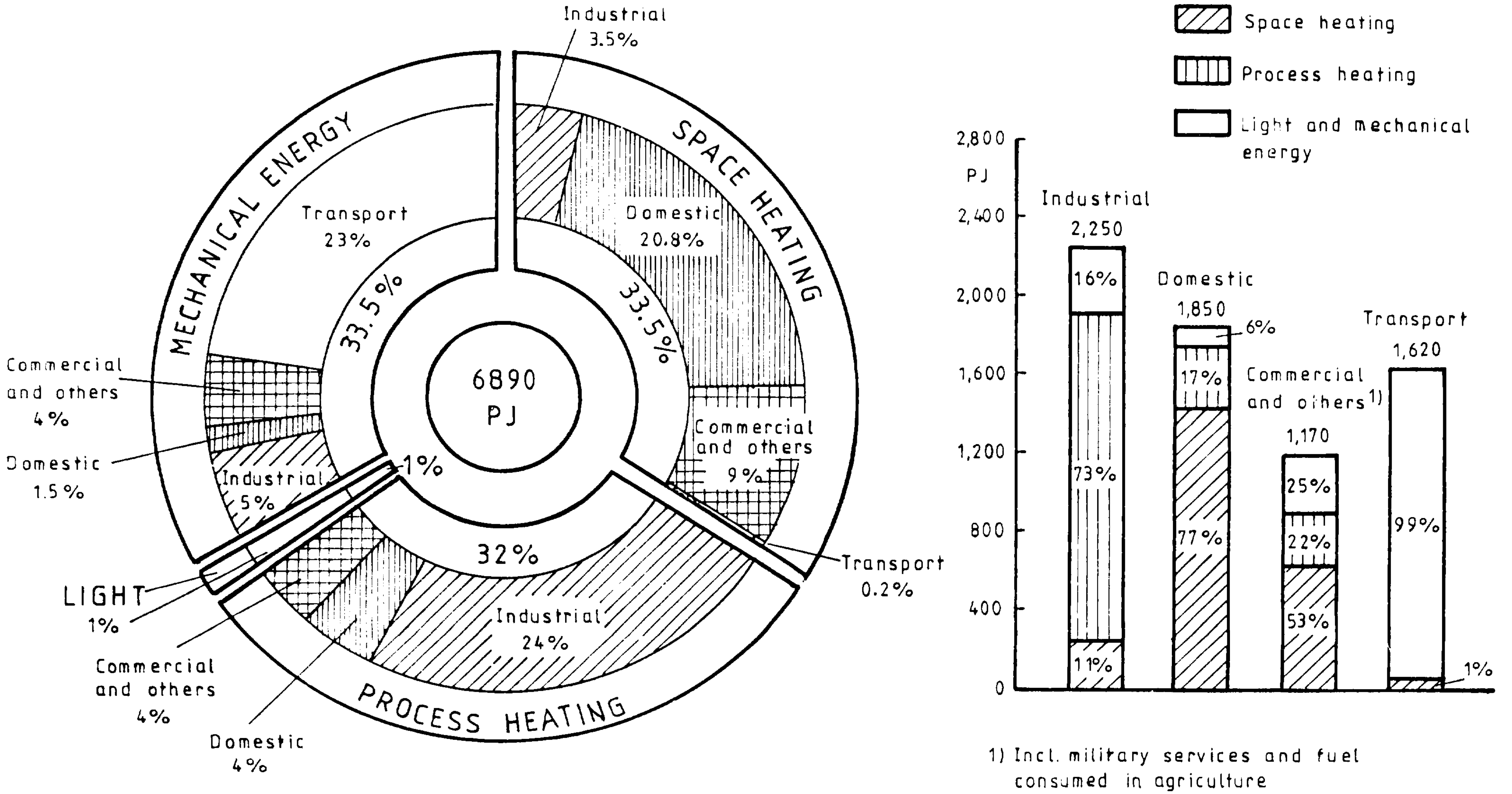
Table 5 - 1: Comparison between systems: fixed speed - pole switching - frequency converter / 5 - 6 /

Drying method	Specific energy consumption	
	Electrical energy / kWh/m³ /	Thermal energy / MJ/m³ /
Convection drying	67	6,851
Condensation drying	380	-
Vacuum drying, plate system	115	1,465
Non-plate vacuum drying with heat recovery	95	1,099

Table 5-2 : Specific energy use in various timber drying methods

Product	Humidity %	Calorific value
Grinding particles (chipboard)	8	17,9
Grinding particles (logs)	12	16,6
Sawdust (chipboard)	10	17,6
Sawdust (logs)	15	15,9
Sawdust	50	11,7
Sawdust	80	9,0
Millings	15	15,9
Coarse chips	15	15,9
Coarse chips	50	11,7
Coarse chips	80	9,0
Wood cuts, cur - seasoned	20	15,3
Wood cuts, green	60	10,7
Bark	60	10,5
Bark	100	8,4
Bark	150	6,3

Table 5 - 3: Calorific value of wood residues and bark of varying moisture

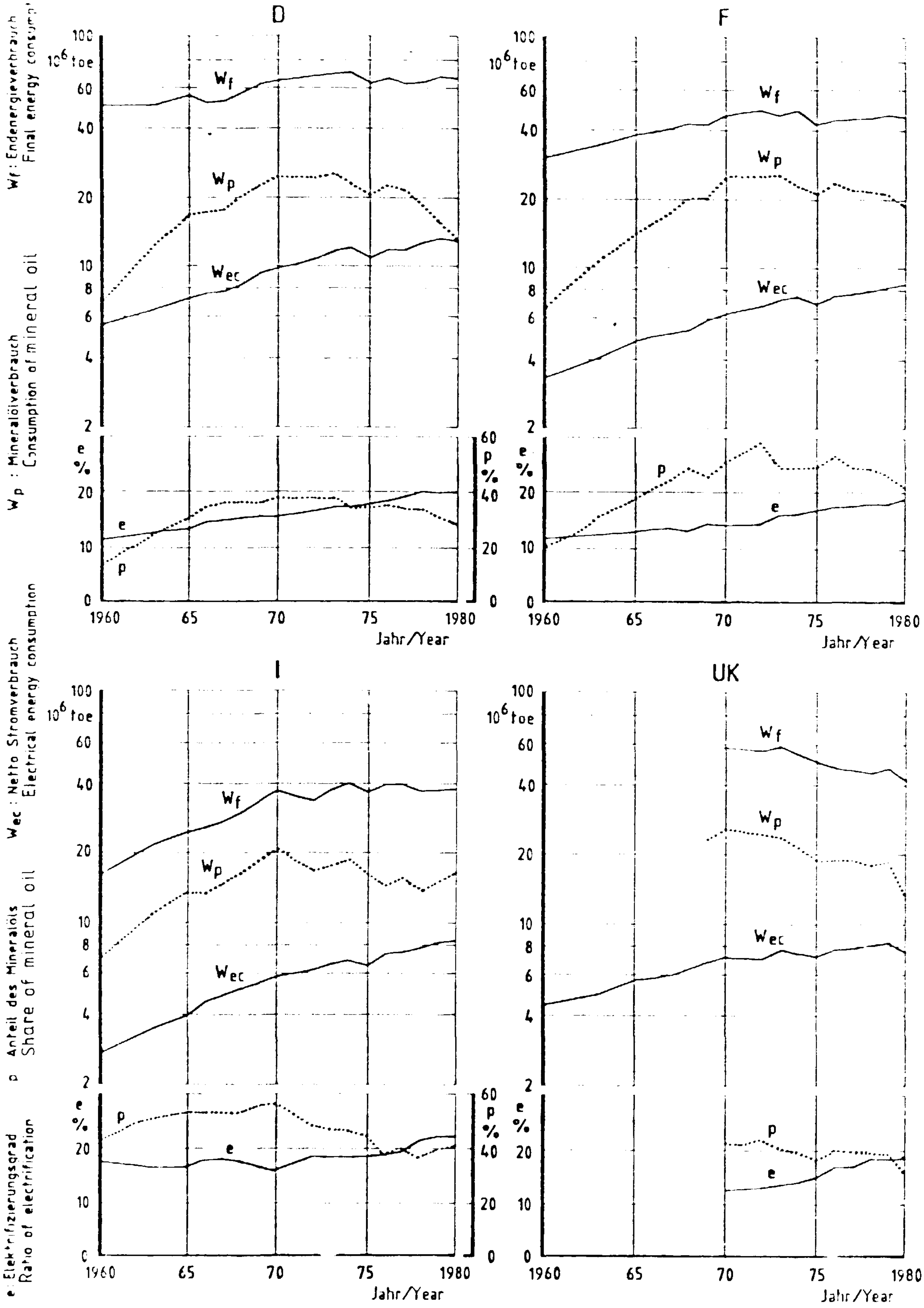


1) Incl. military services and fuel consumed in agriculture

Fig. 1-1

Final energy demand by consumer groups and types of requirement in the Federal Republic of Germany in 1982

0820
E 001 - 84

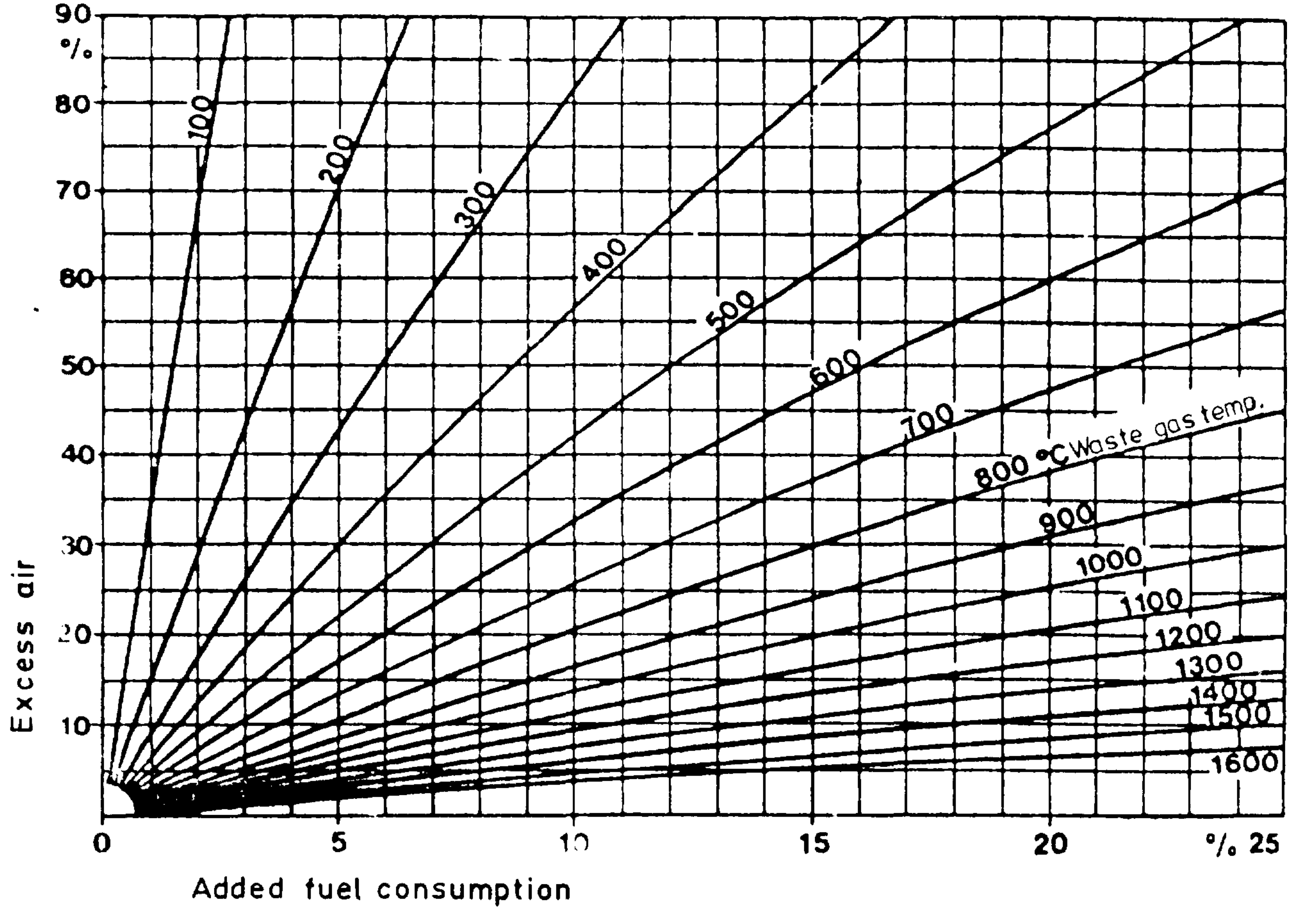


F f E

Energieverbrauch des Sektors "Industrie" D,F,I,UK

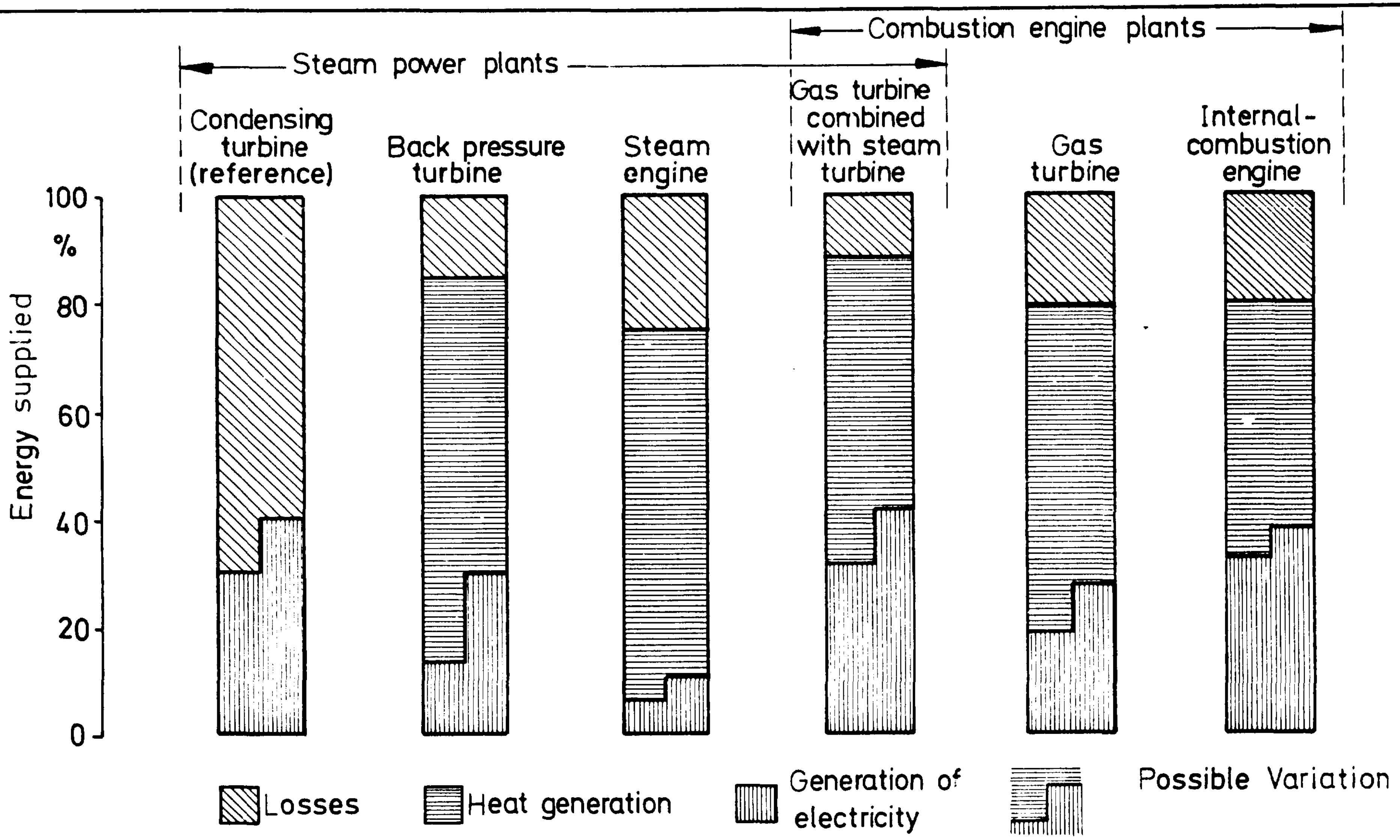
Energy consumption of industry as a sector, D,F,I,UK

Fig. 1-2



Added consumption of natural gas due to excess air

Fig. 1-3



Combined generation of electricity and heat: Energy balances of different types of plants

Fig. 1-4

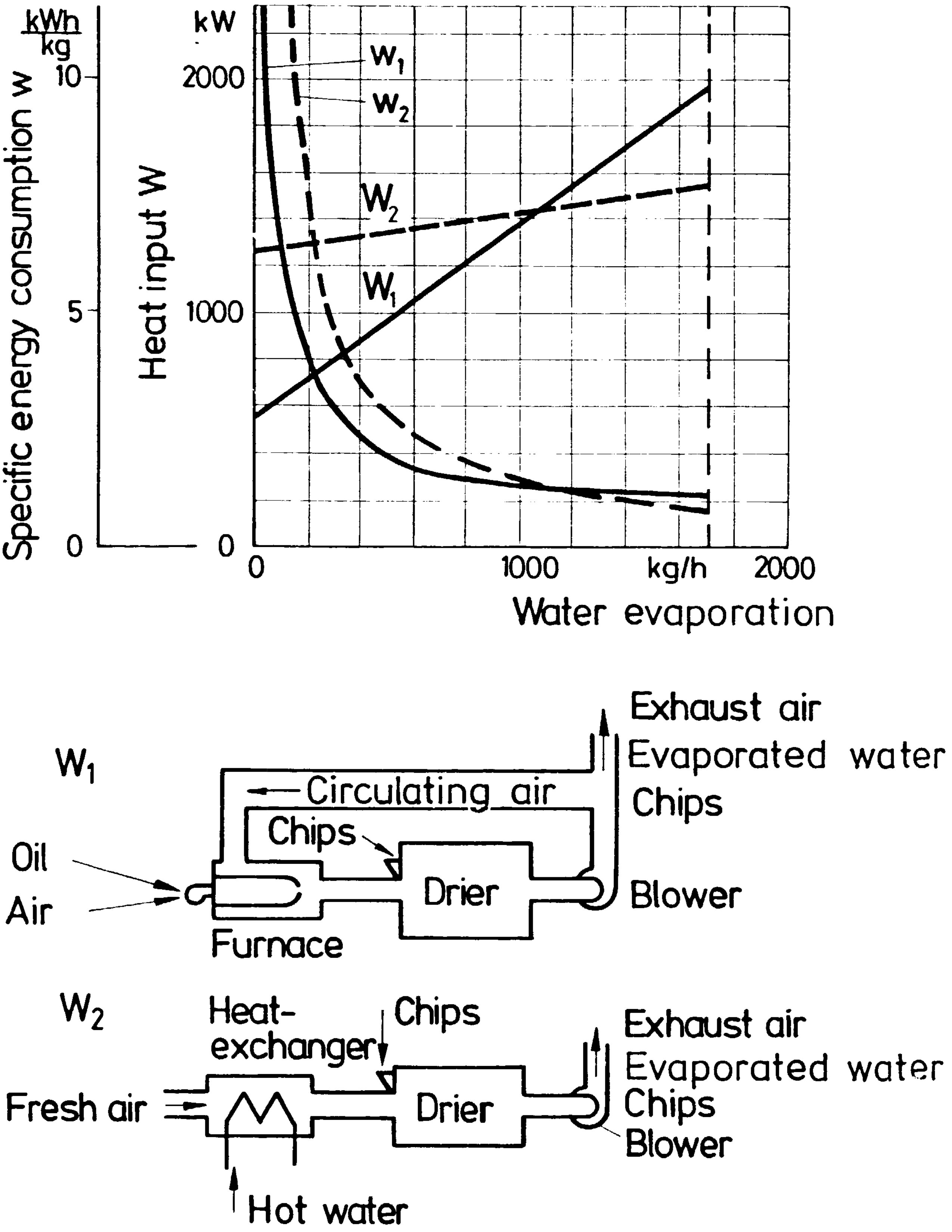
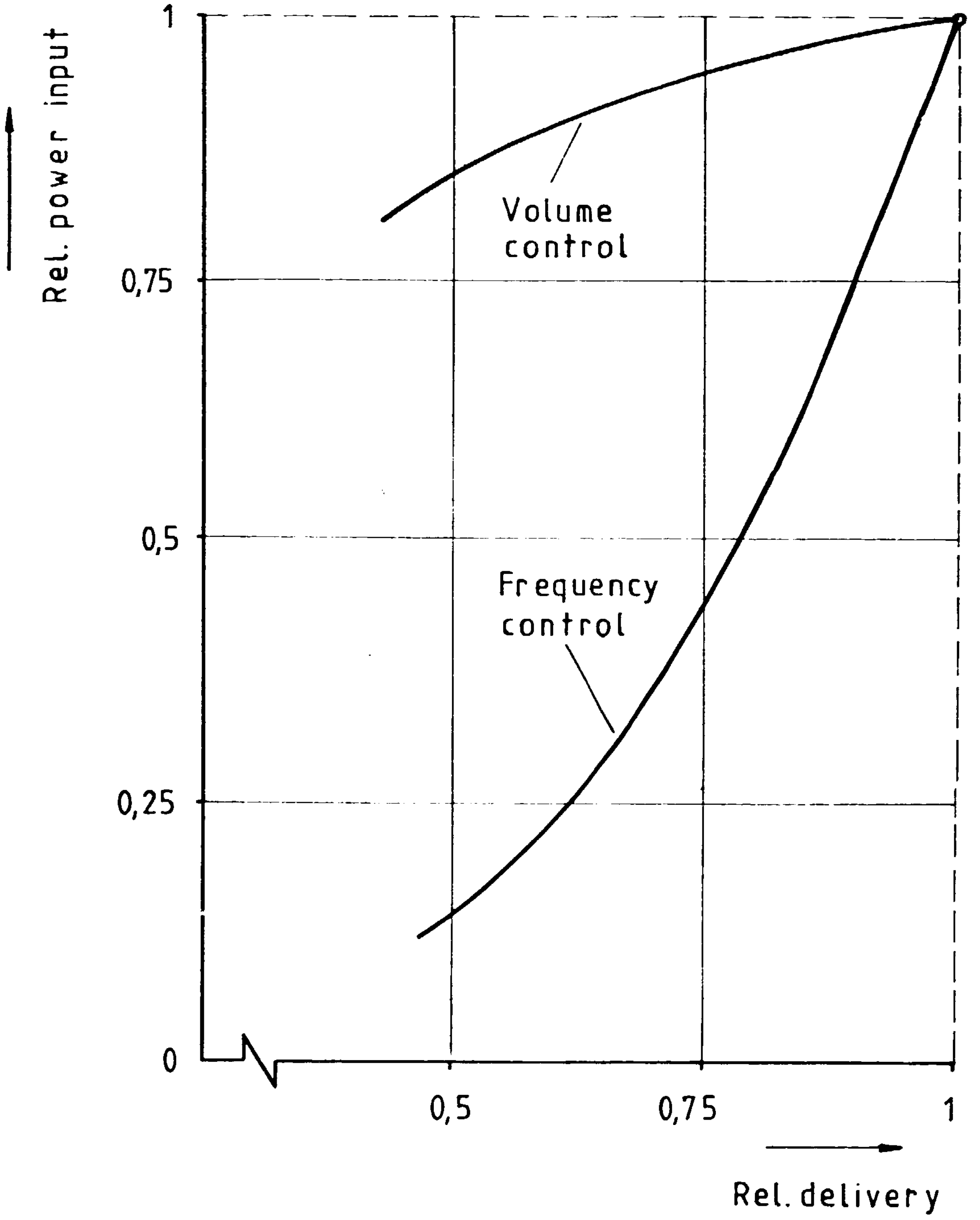


Fig. 1-5	Energy consumption of two chip driers	E018 0173 - 79



Power input of an electric pump

Fig. 1-6

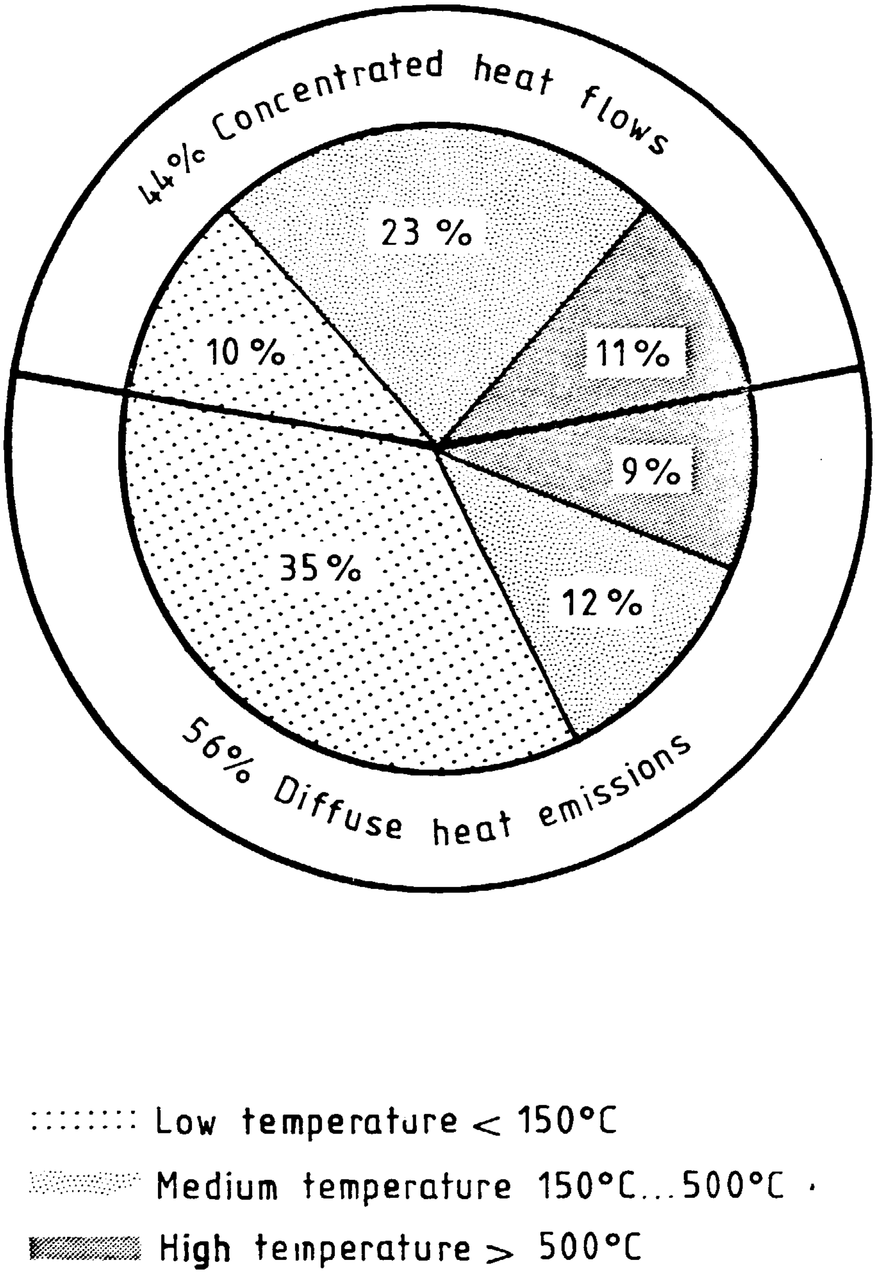
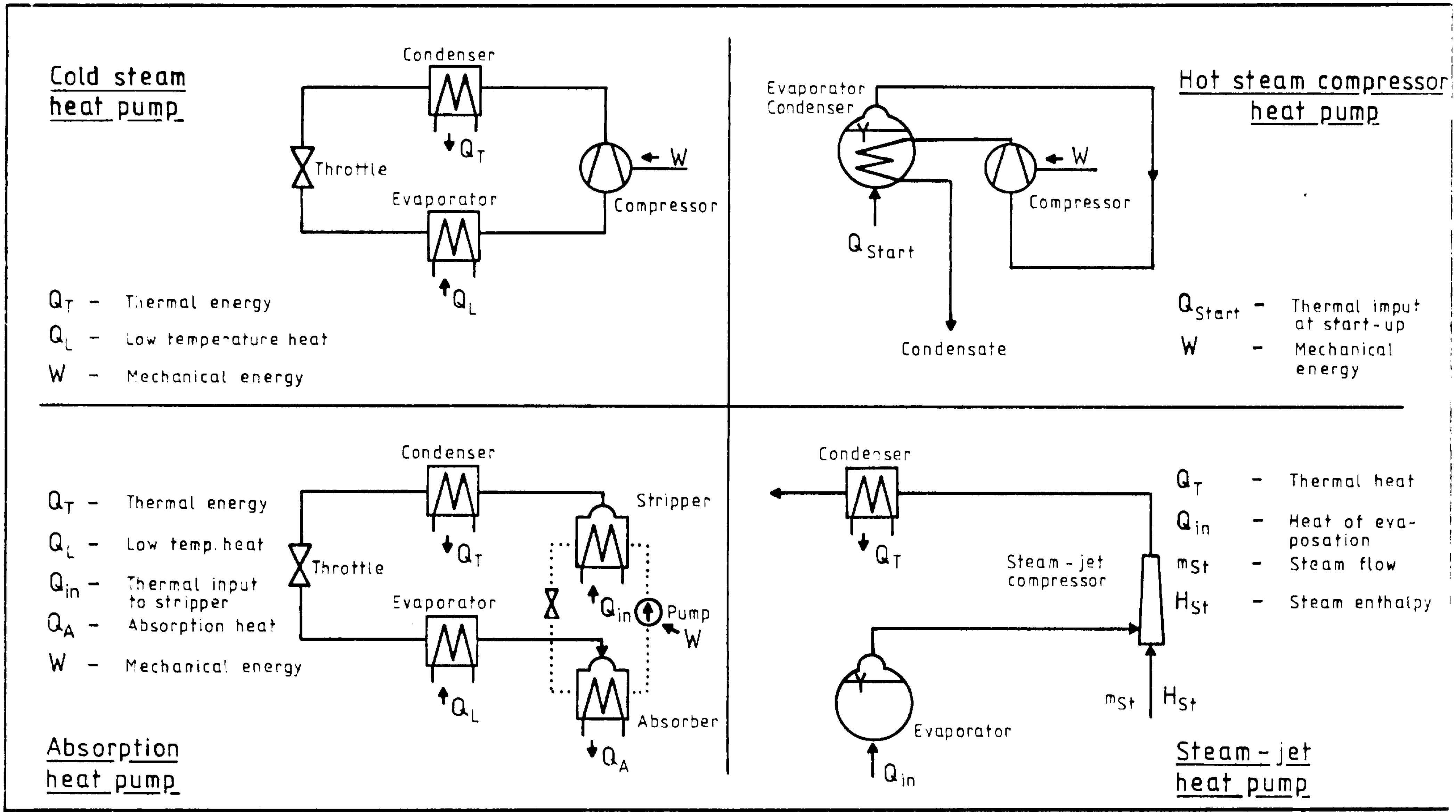


Fig. 1-7

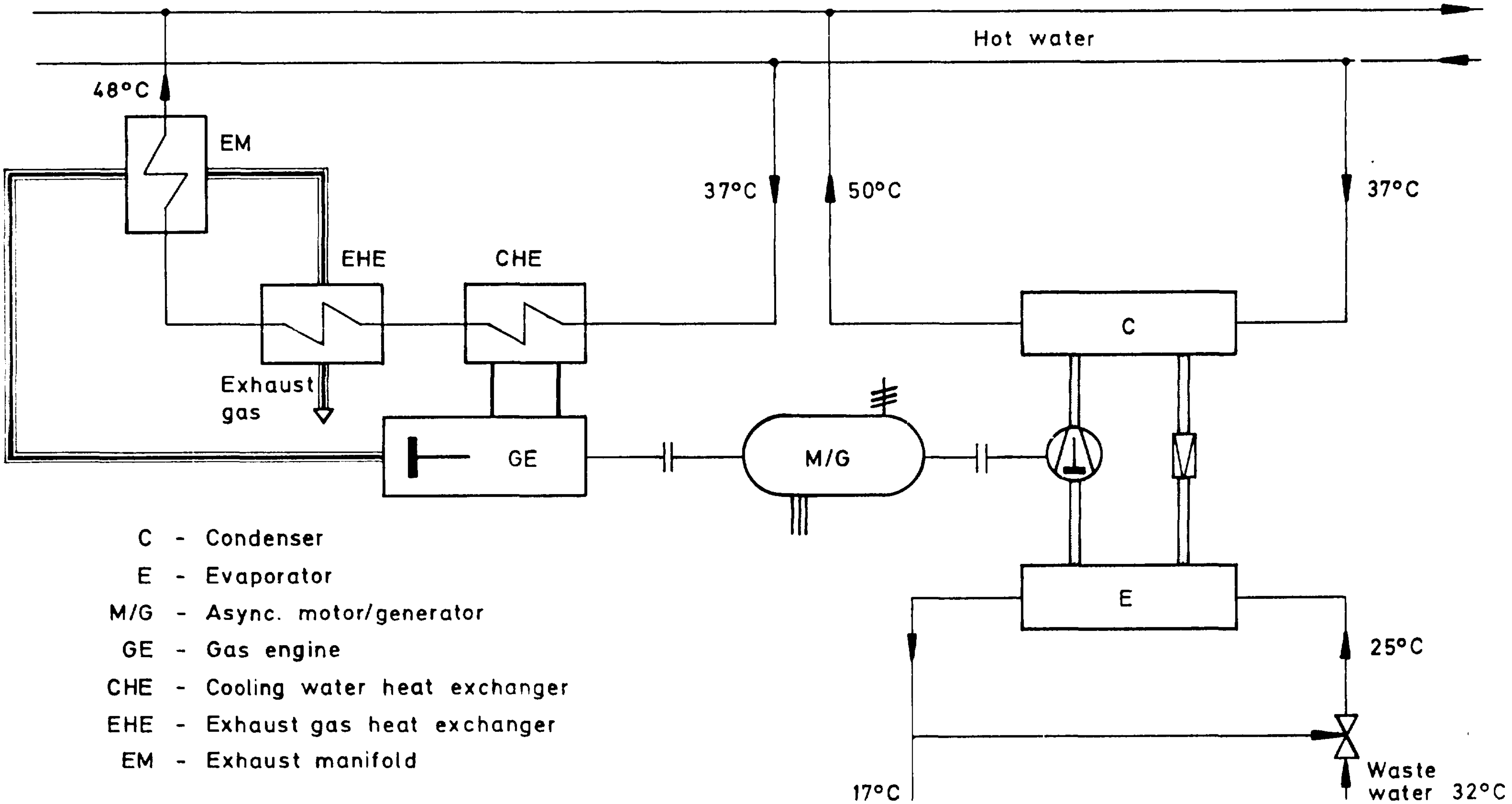
Waste heat in industry
in the FRG, 1978

E 058-83
1359



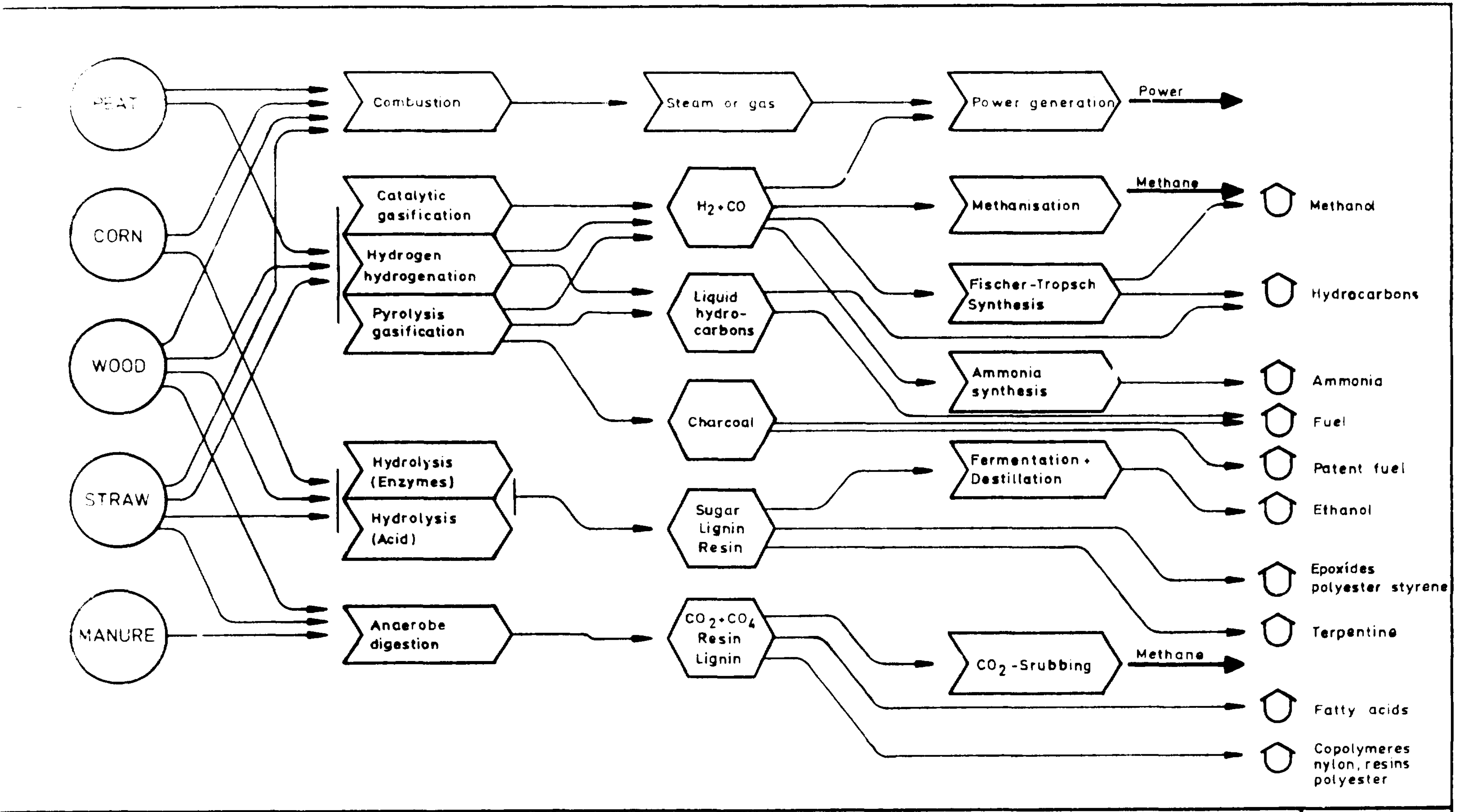
Industrial heat pump designs

Fig. 1-[illegible]



Switching diagram of a tandem plant

Fig. 1-9



Biomass conversion processes

Fig. 1-10

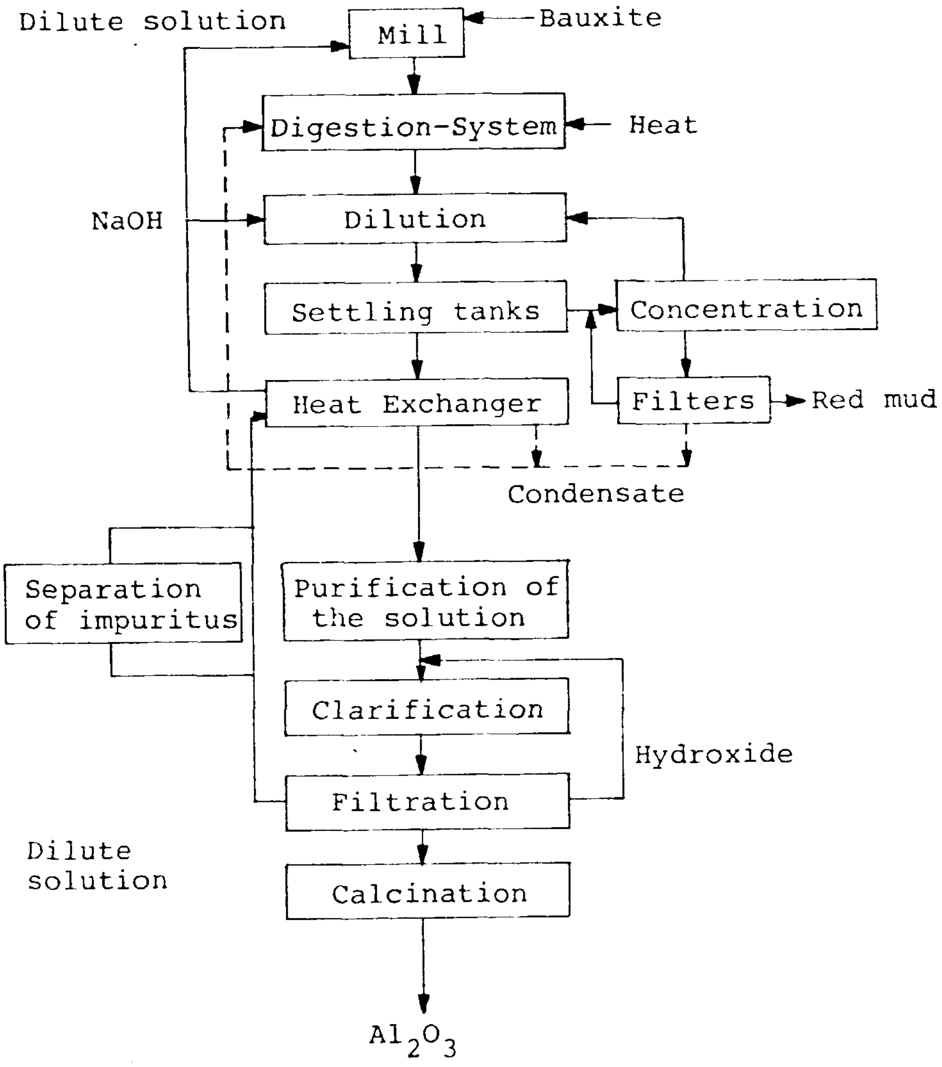


Fig. 2.1-1: Schema of the Bayer-process

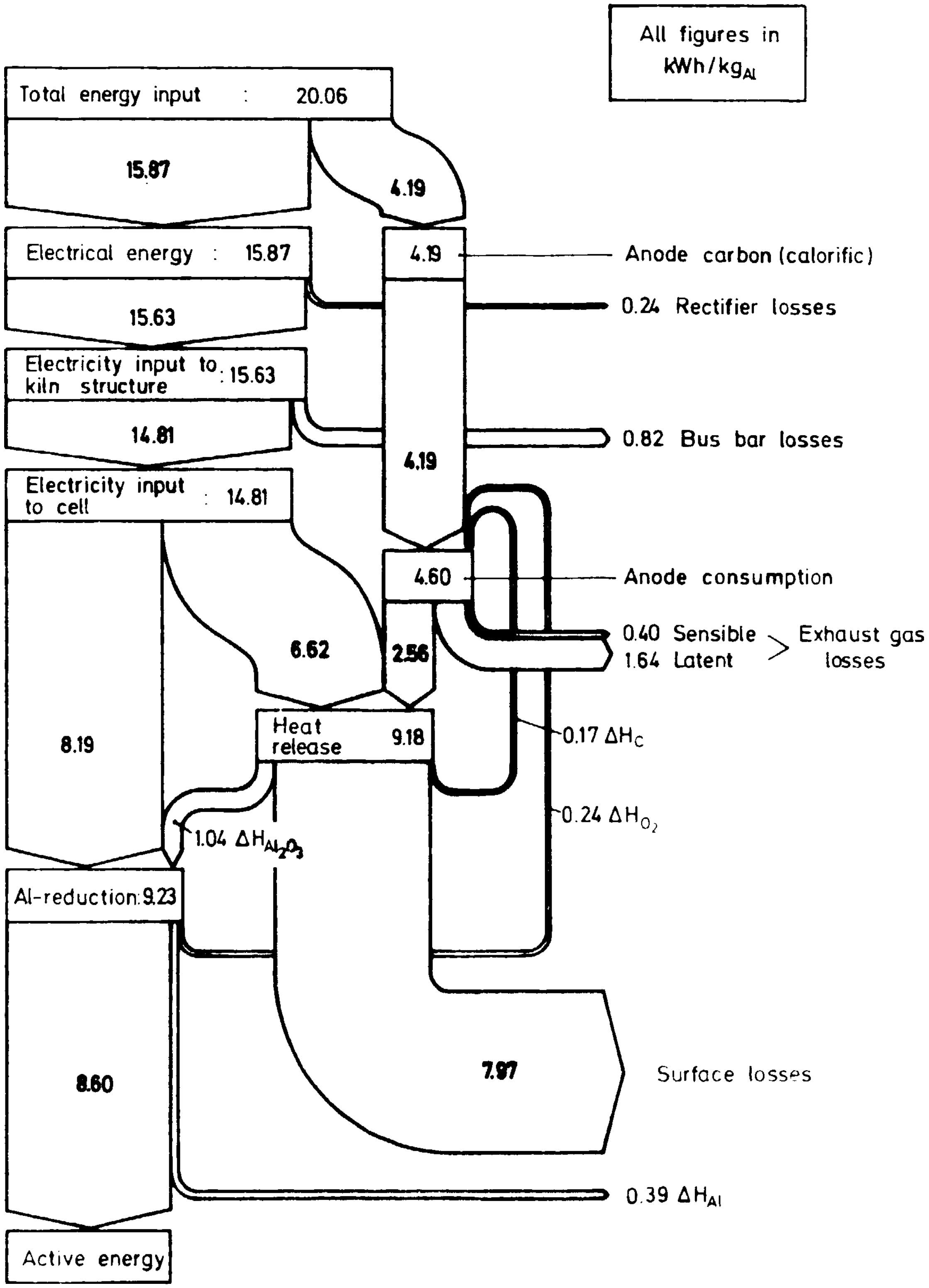


Fig. 2.1-2

Energy conversions in pyrometallurgical production of aluminium (simplified)

0987
79

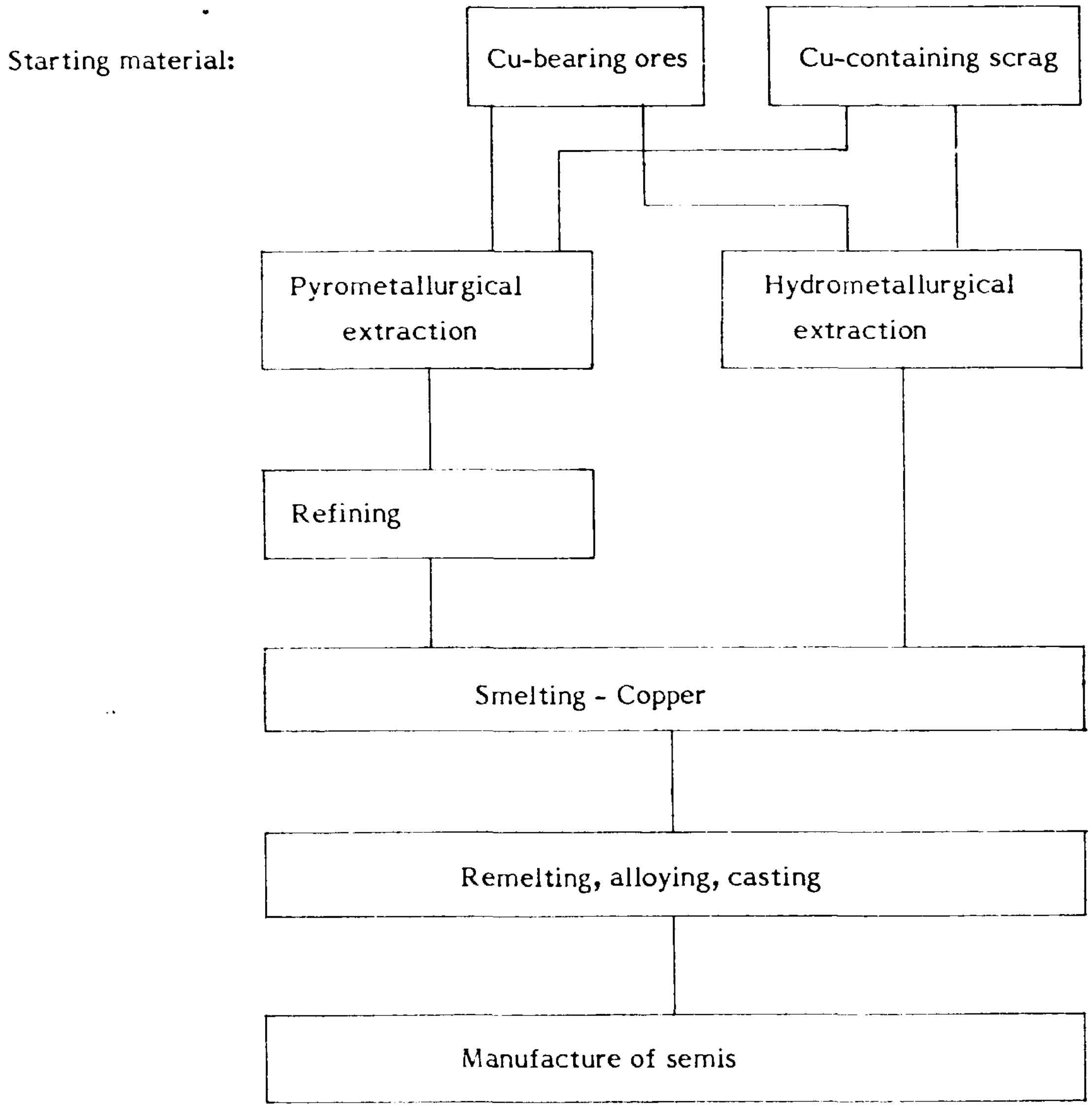


Figure 2.2 - 1 : Flow diagram for copper production

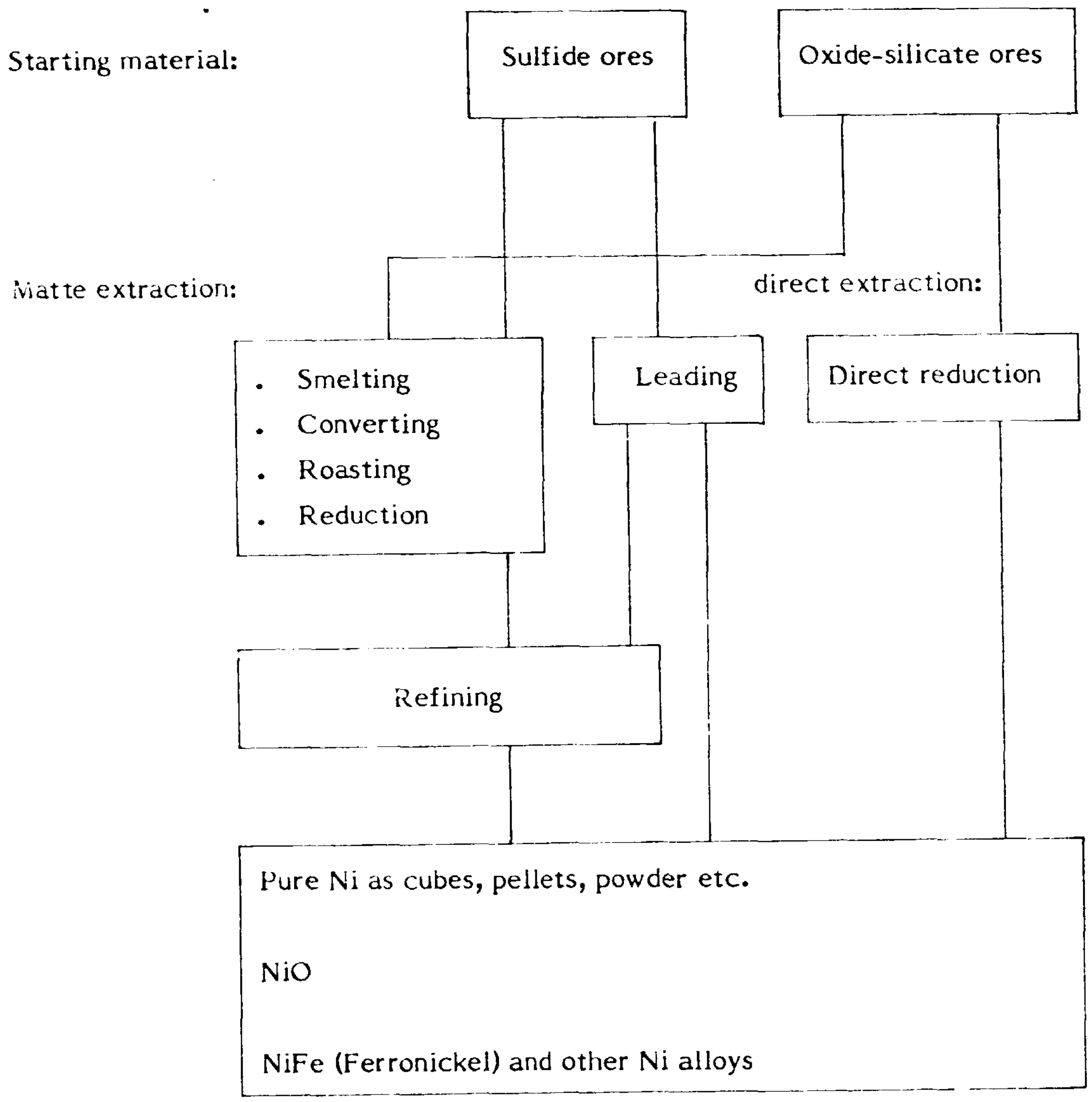


Figure 2.3 - 1 : Process flow diagram of Ni production

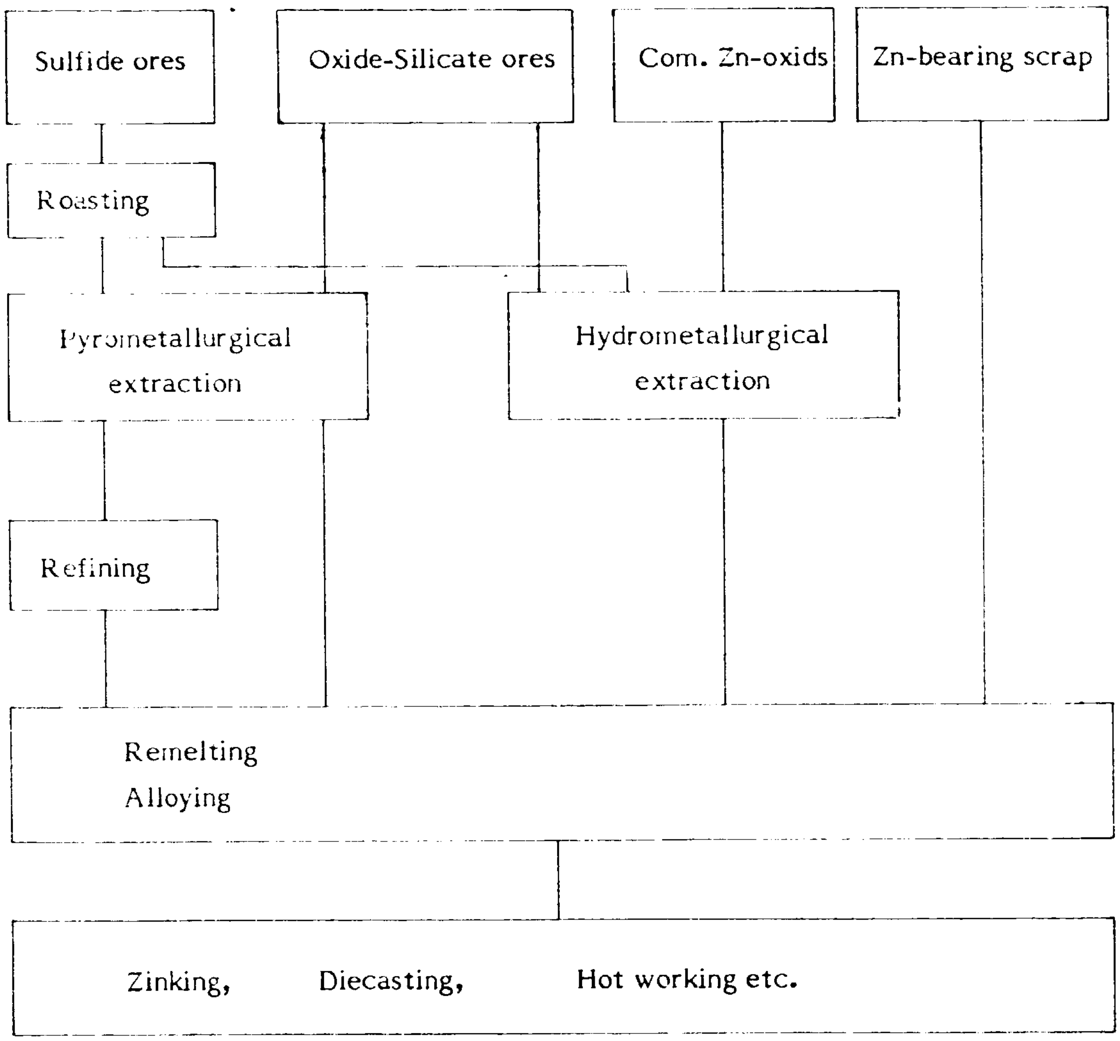


Figure 2.3-2 : Process flow diagram of Zn production

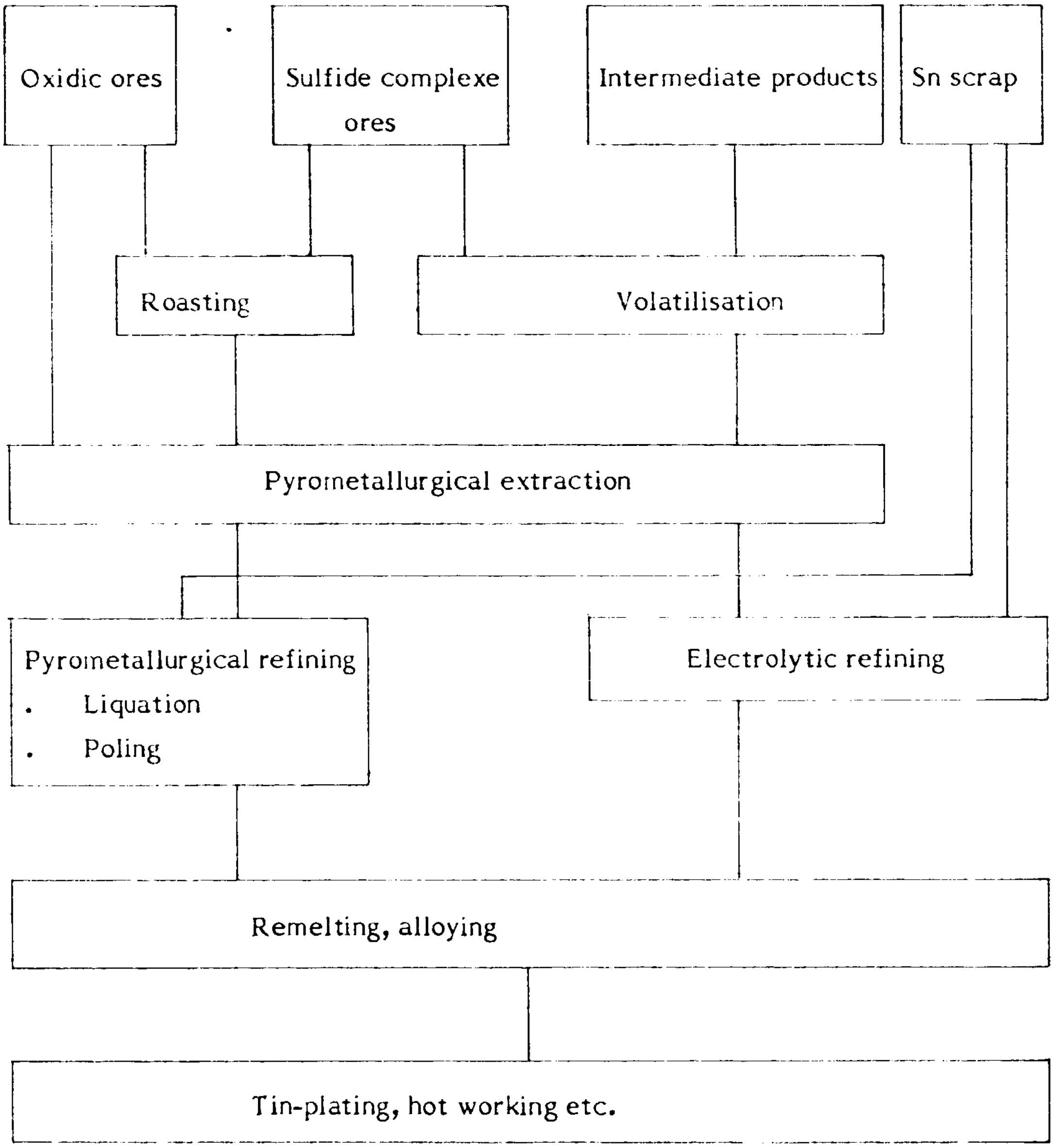
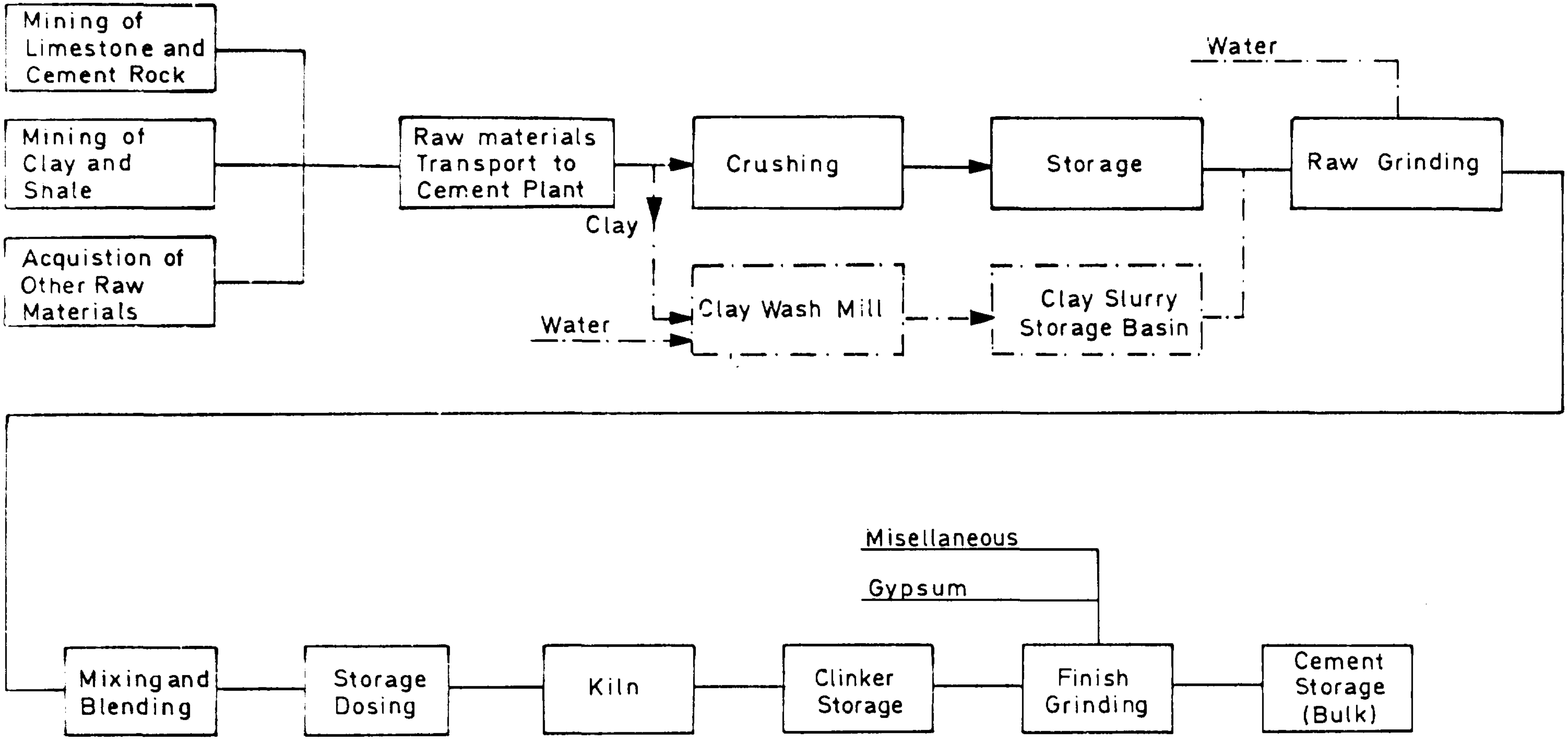


Figure 2.3-3 : Process flow diagram of Sn production

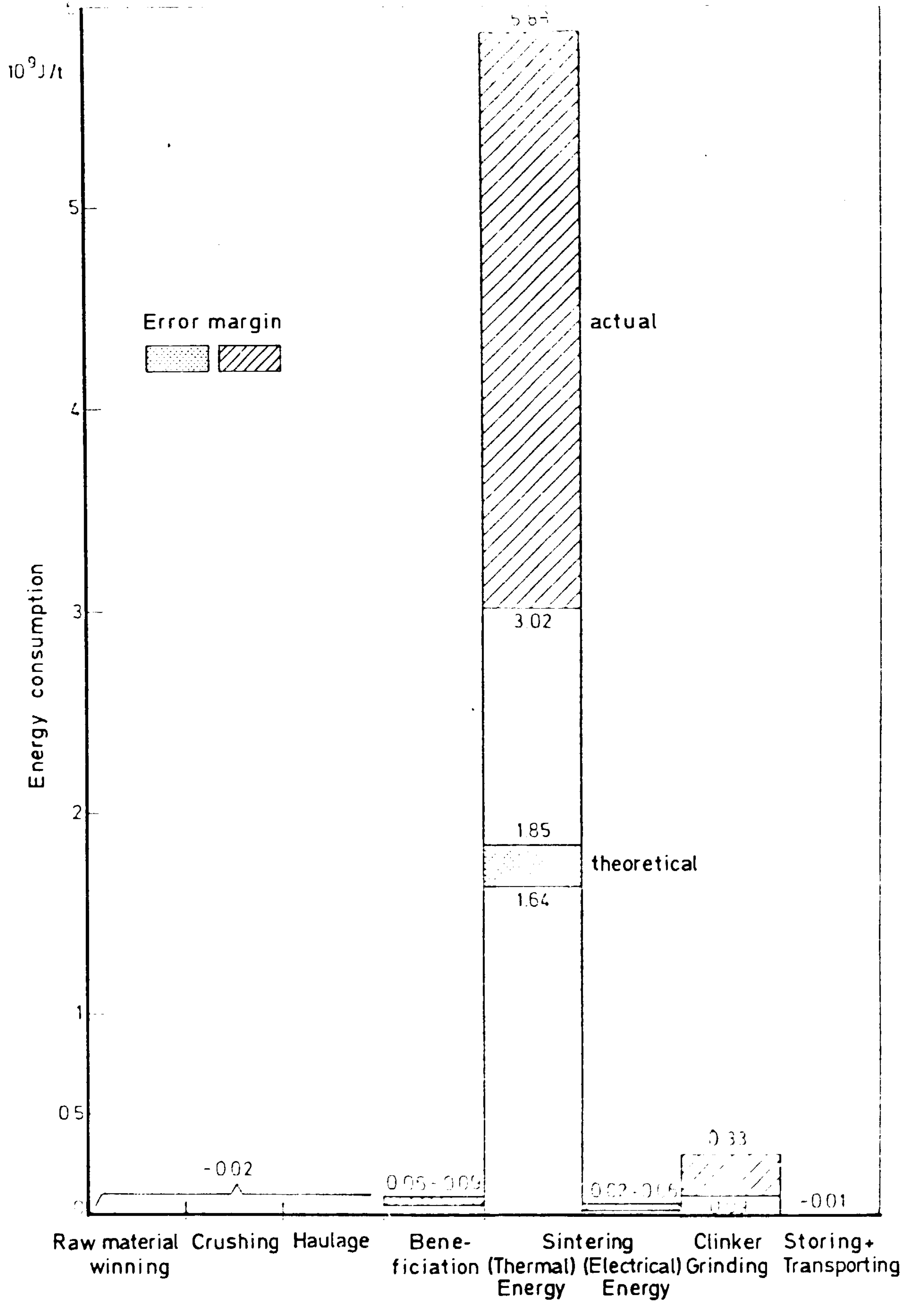


source : 3.1-1

FfE

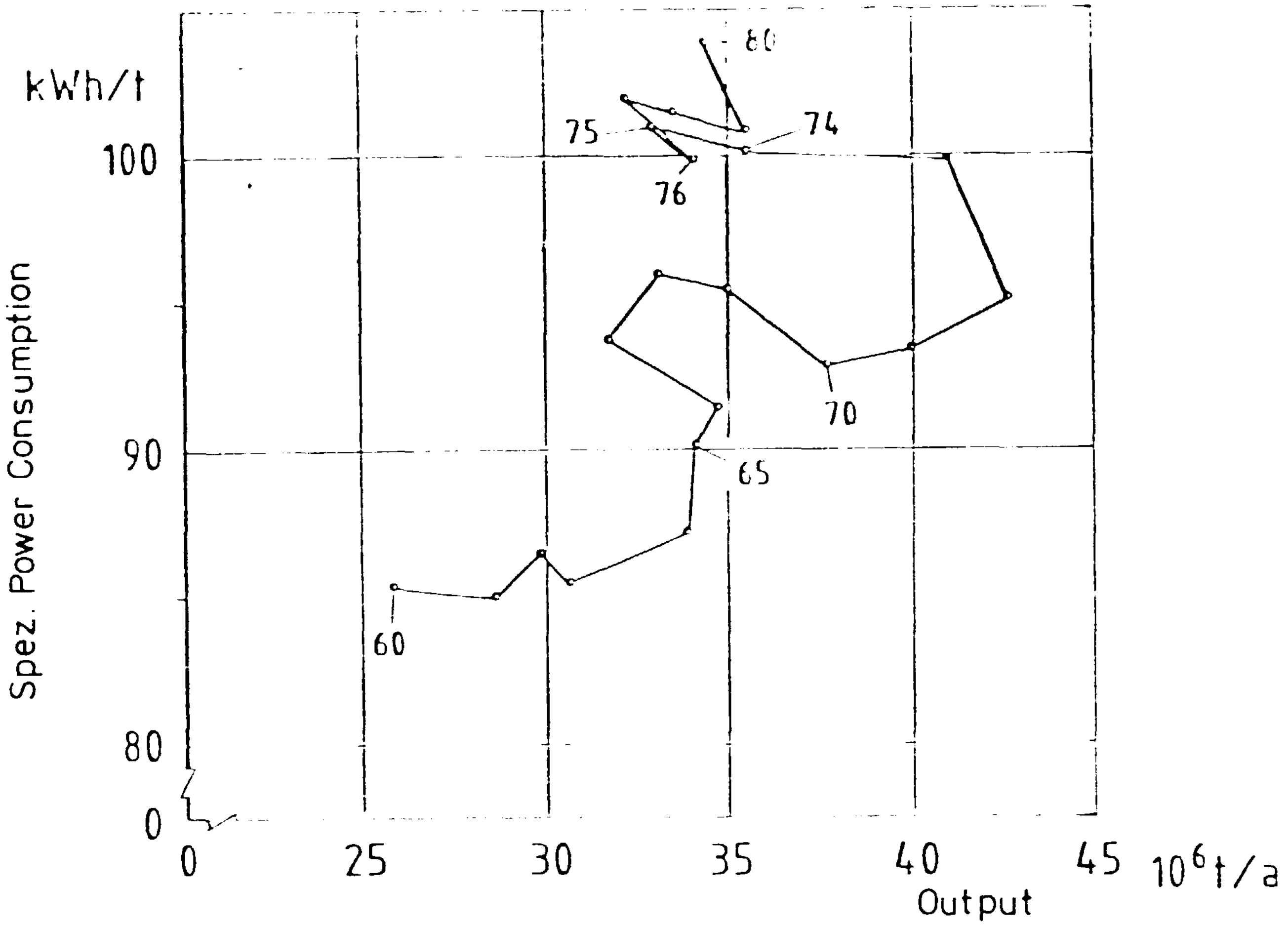
Portland cement manufacture flowsheet

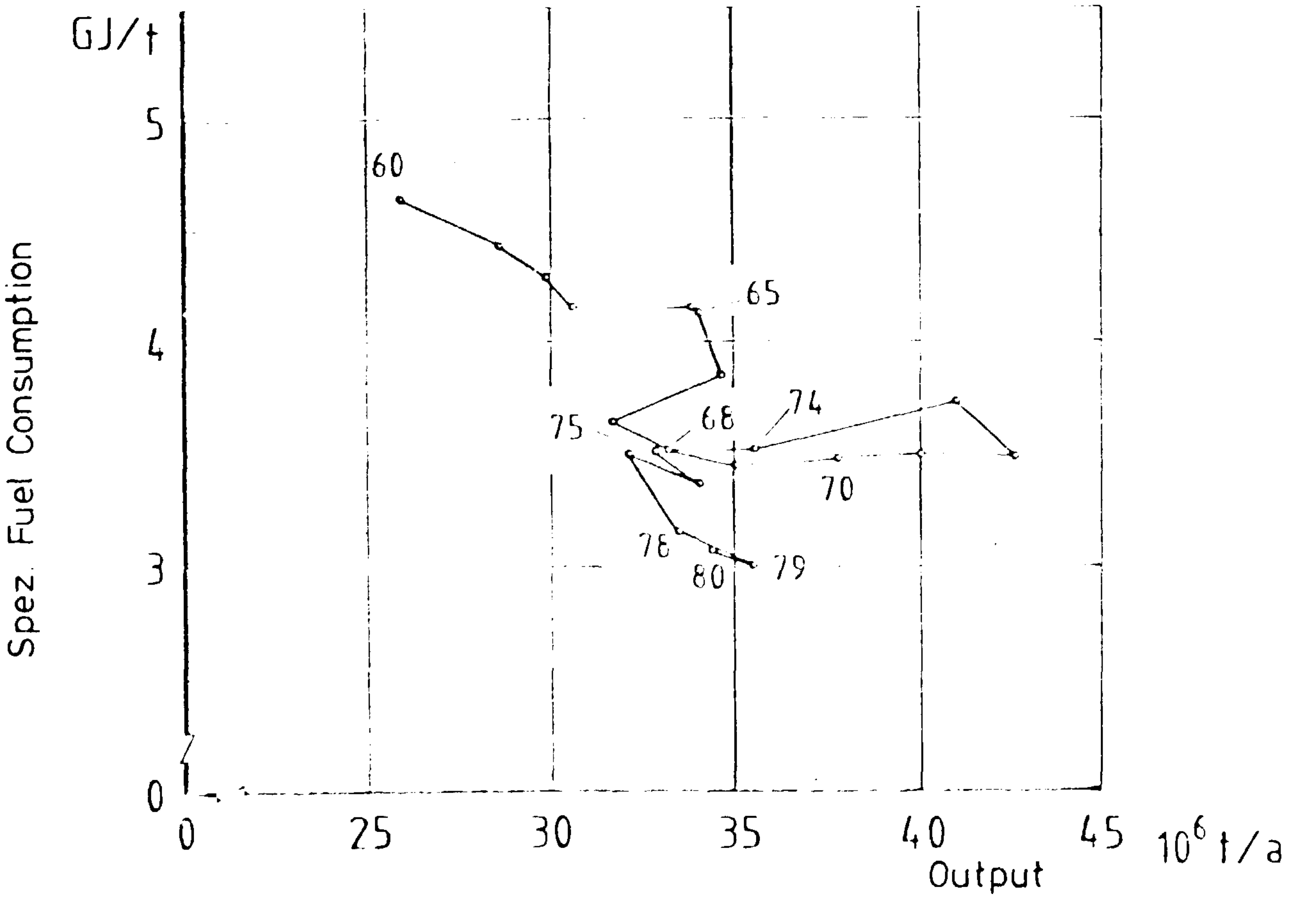
fig. 3.1-1



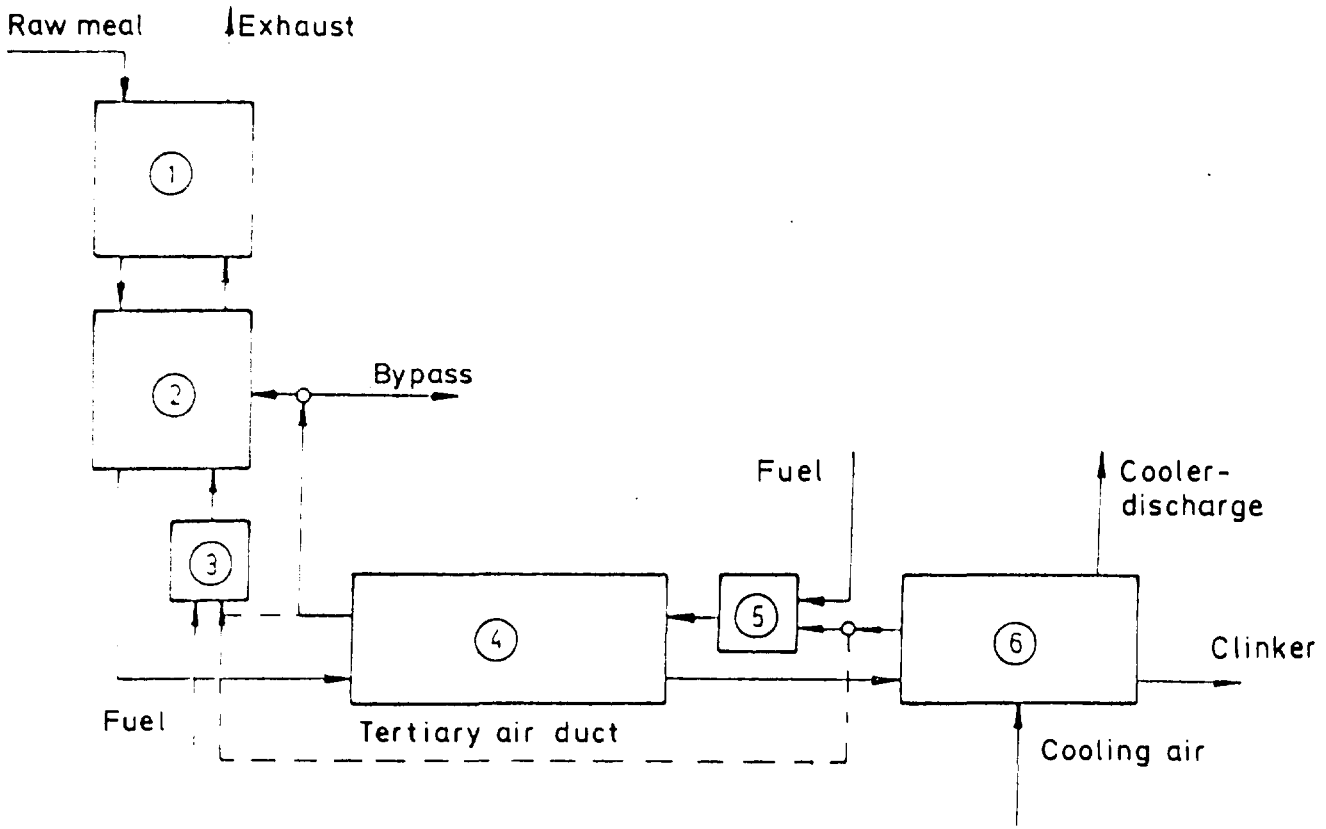
source: 3.1-4

FfE	Energy consumption diagram of Portlandcement making	fig. 3.1-2



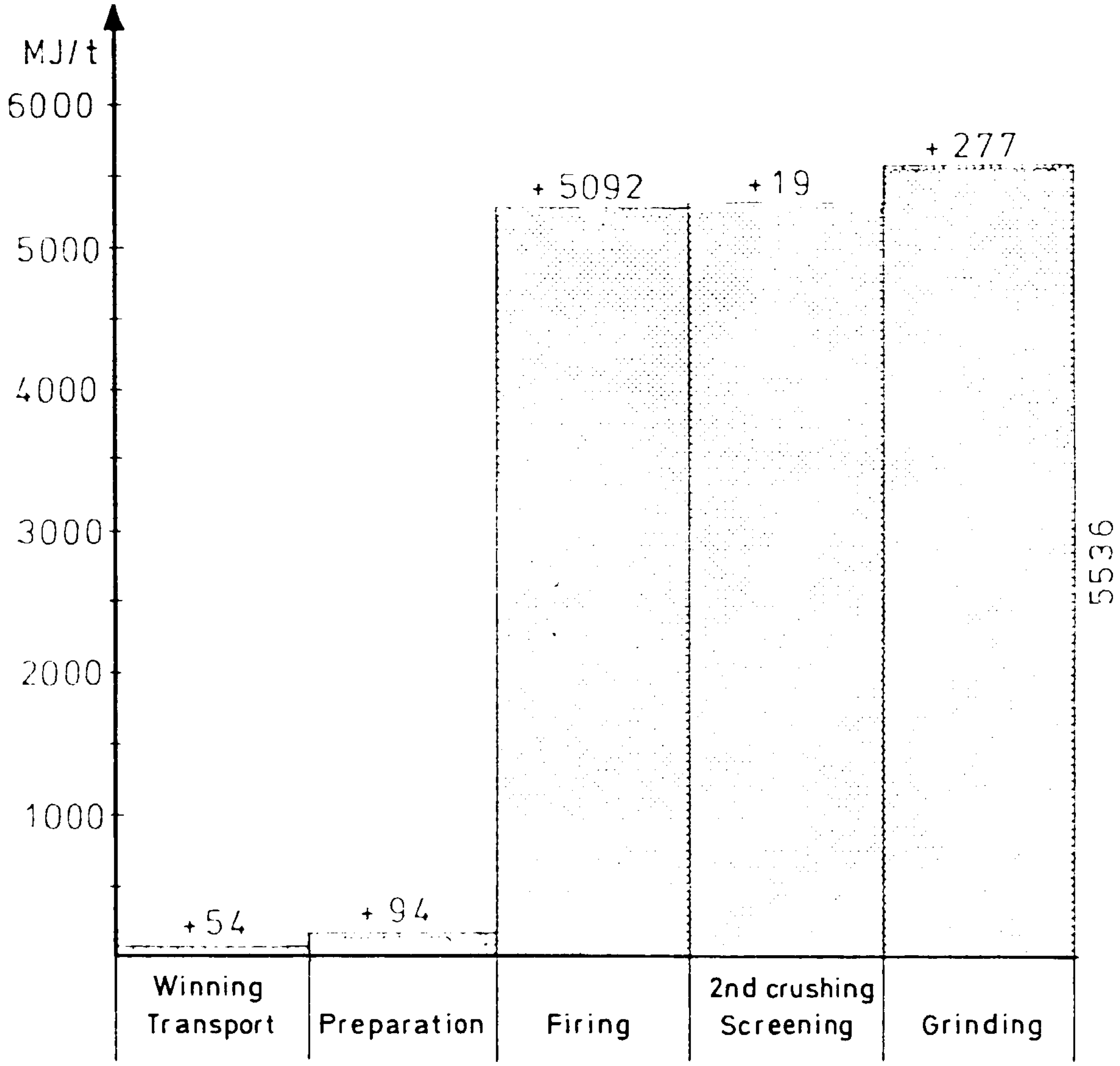


FfE	Spez. energy consumption of cement making	fig. 3.1-3 fig. 3.1-4



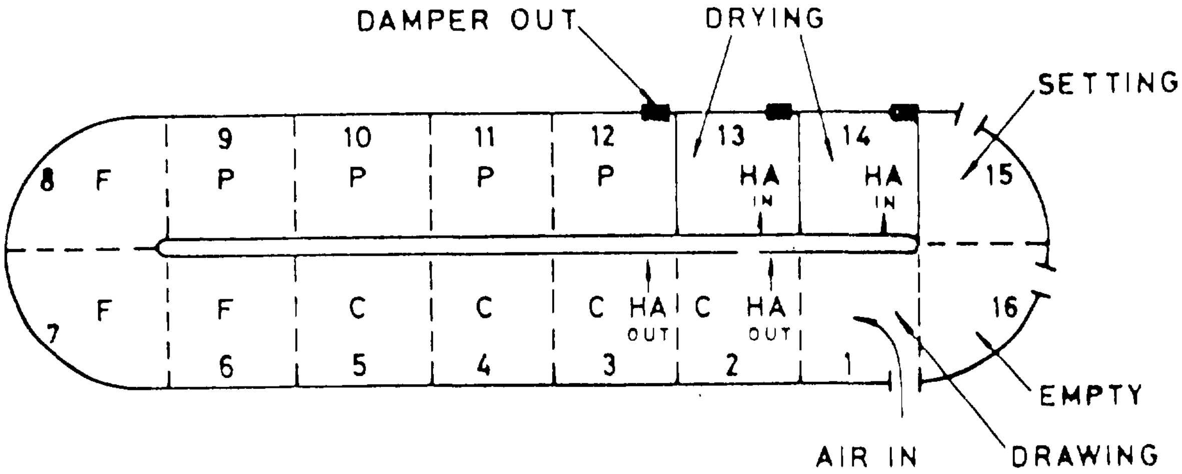
1 Preheater
2 Precalcining stage
3 Secondary Firing
4 Rotary Kiln
5 Primary Firing
6 Clinker Cooler

FfE | Firing of Cement with Precalcination | fig. 3.1-5



source: 3.2-2

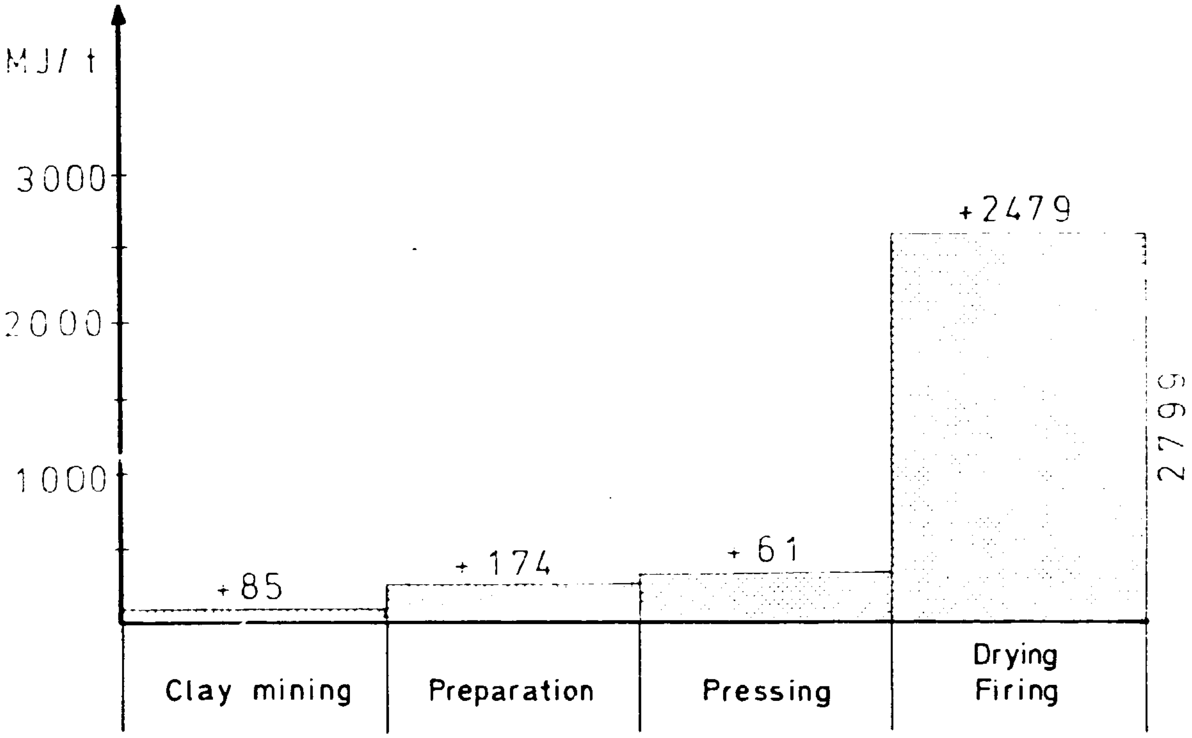
FfE	Primary energy consumption diagram of pulverised lime	fig. 3.2-1



C = COOLING P = PREHEATING
F = FIRING HA = HOT AIR

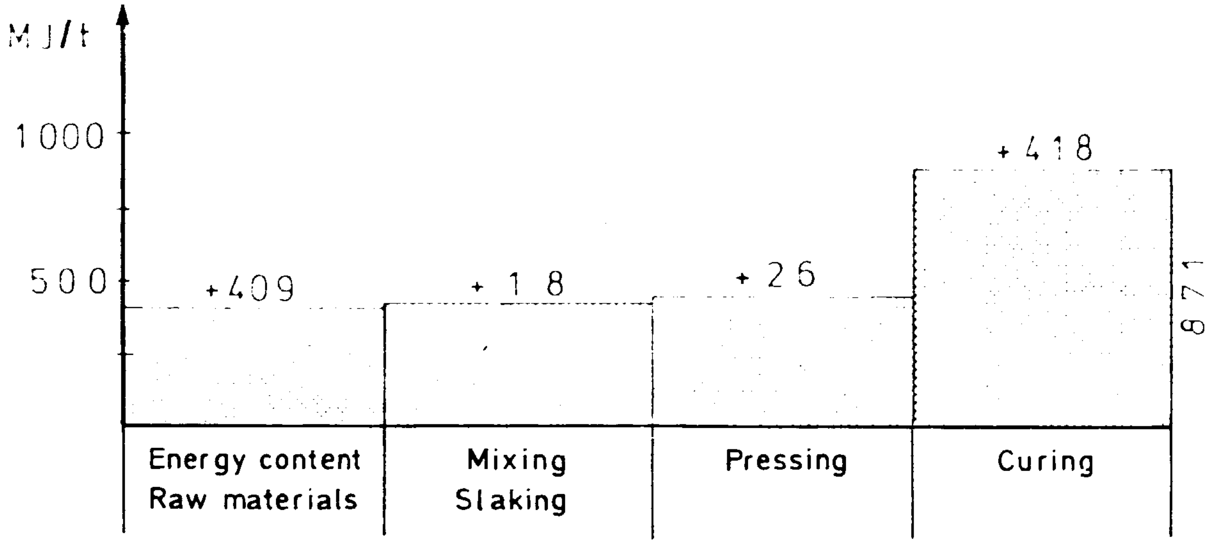
source: 3.3-3

FfE	Firing circuit on a 16-chamber Hoffman kiln	fig. 3.3-1



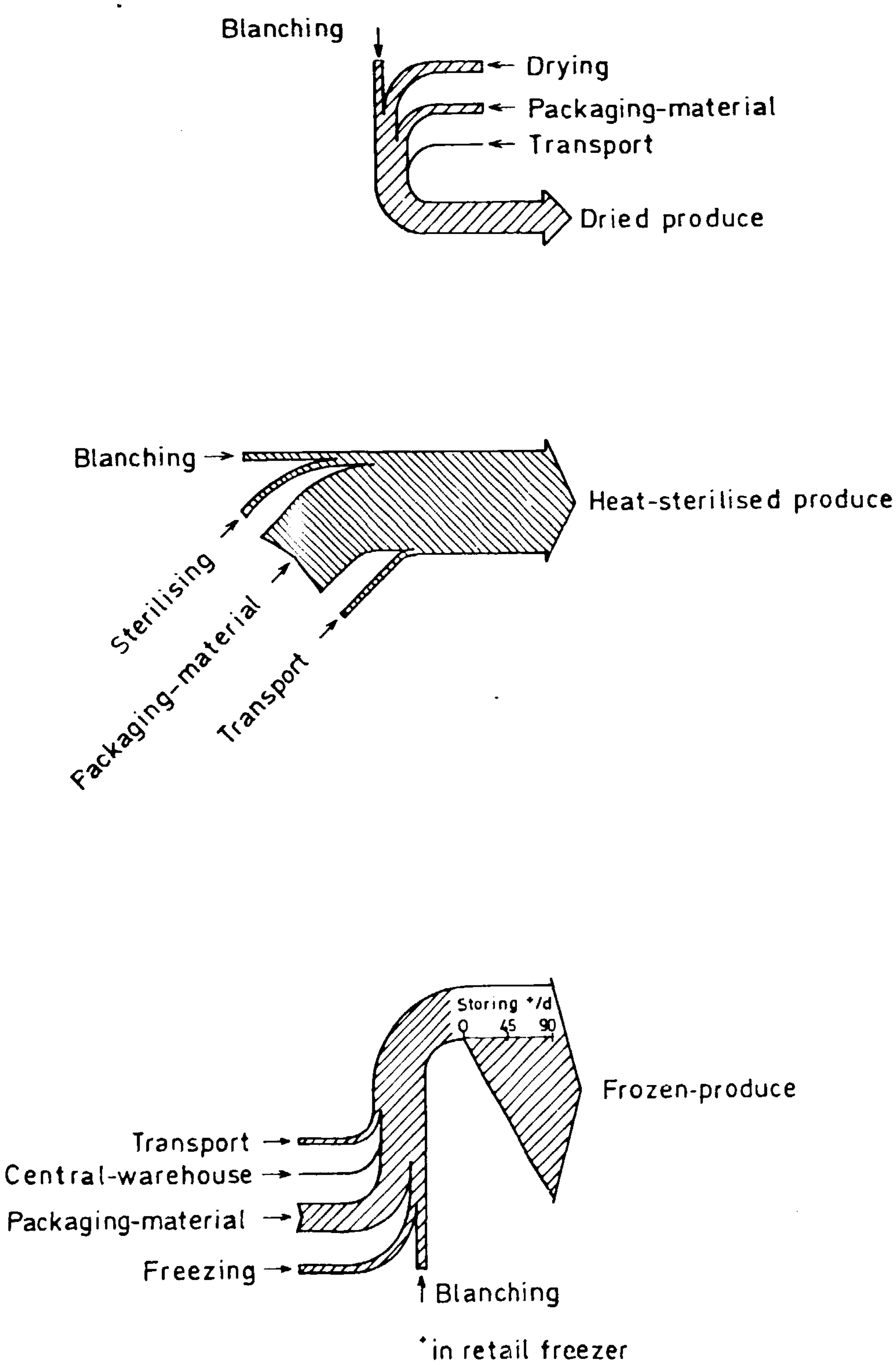
source: 3.3-2

FfE	Energy consumption diagram of brick making	fig. 3.3-2



source: 3.3-2

FfE	Energy consumption diagram of sand-lime brick making	fig. 3.3-3



source : 4.1-8

FfE	Energy used in producing and marketing processed fruit and vegetable produce	fig. 4.1-1

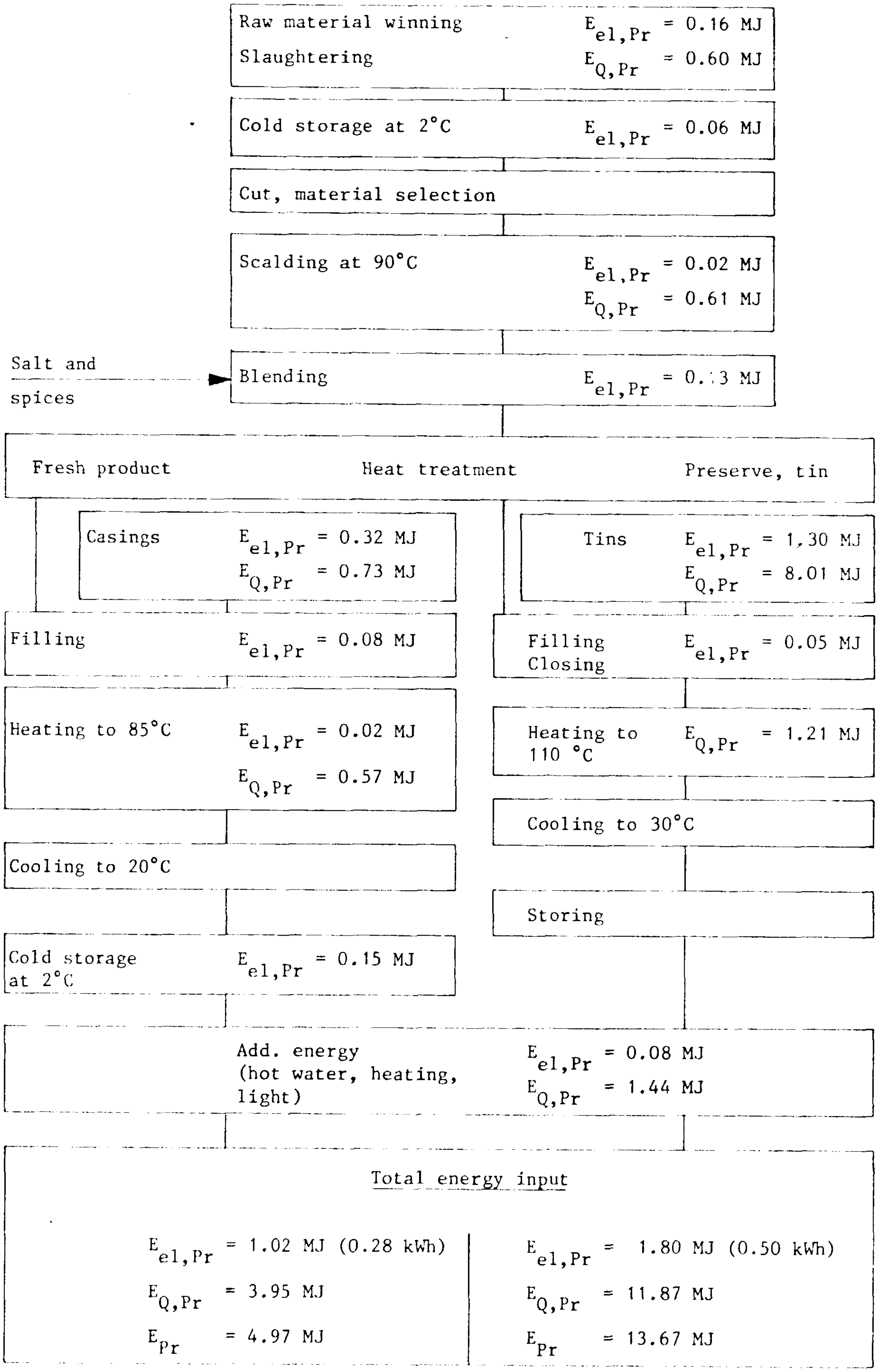


Fig. 4.1-2 Energy input per kg of cooked sausage /4.1-8/

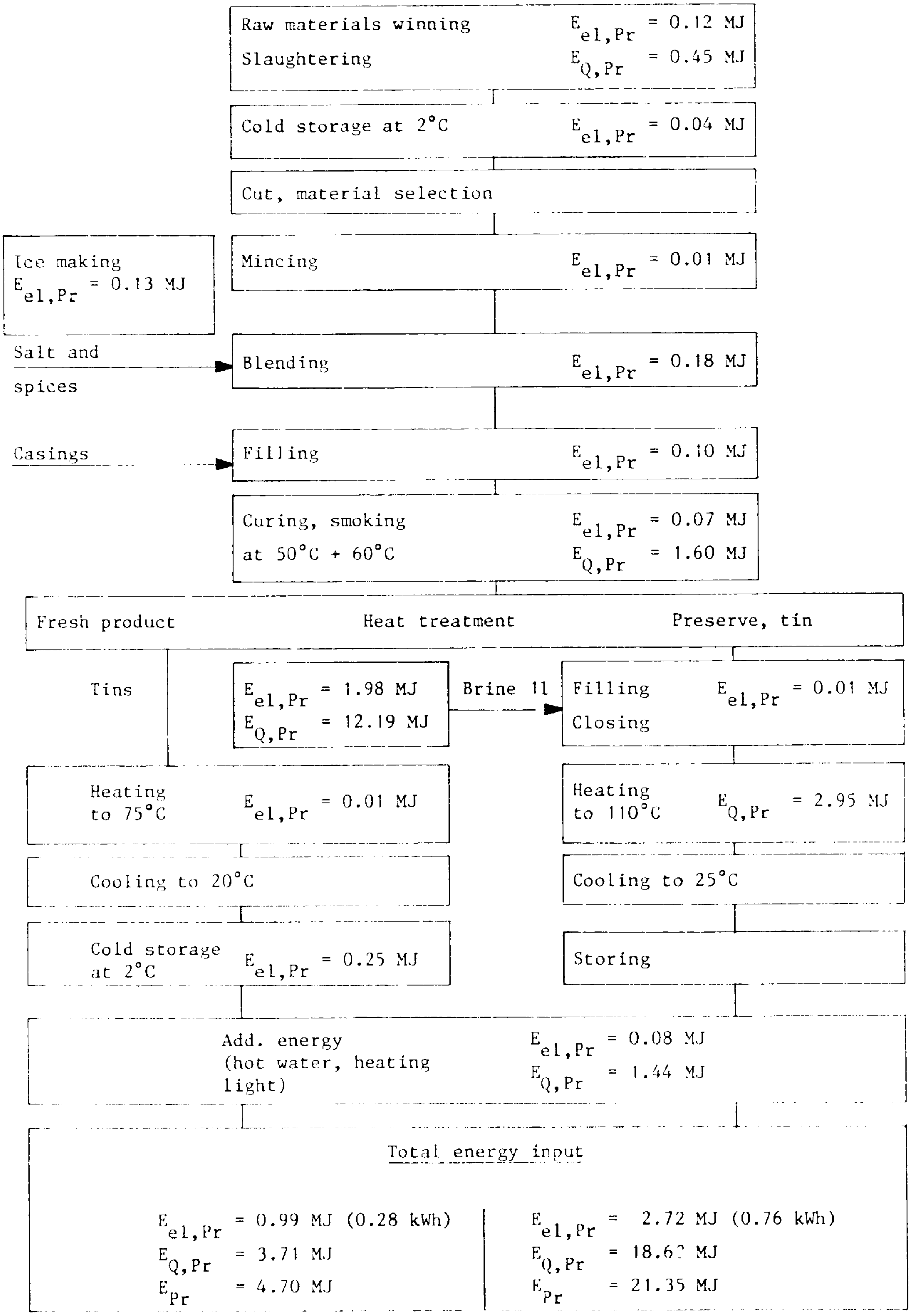


Fig. 4.1-3 Energy input per kg of boiling sausage /4.1-8/

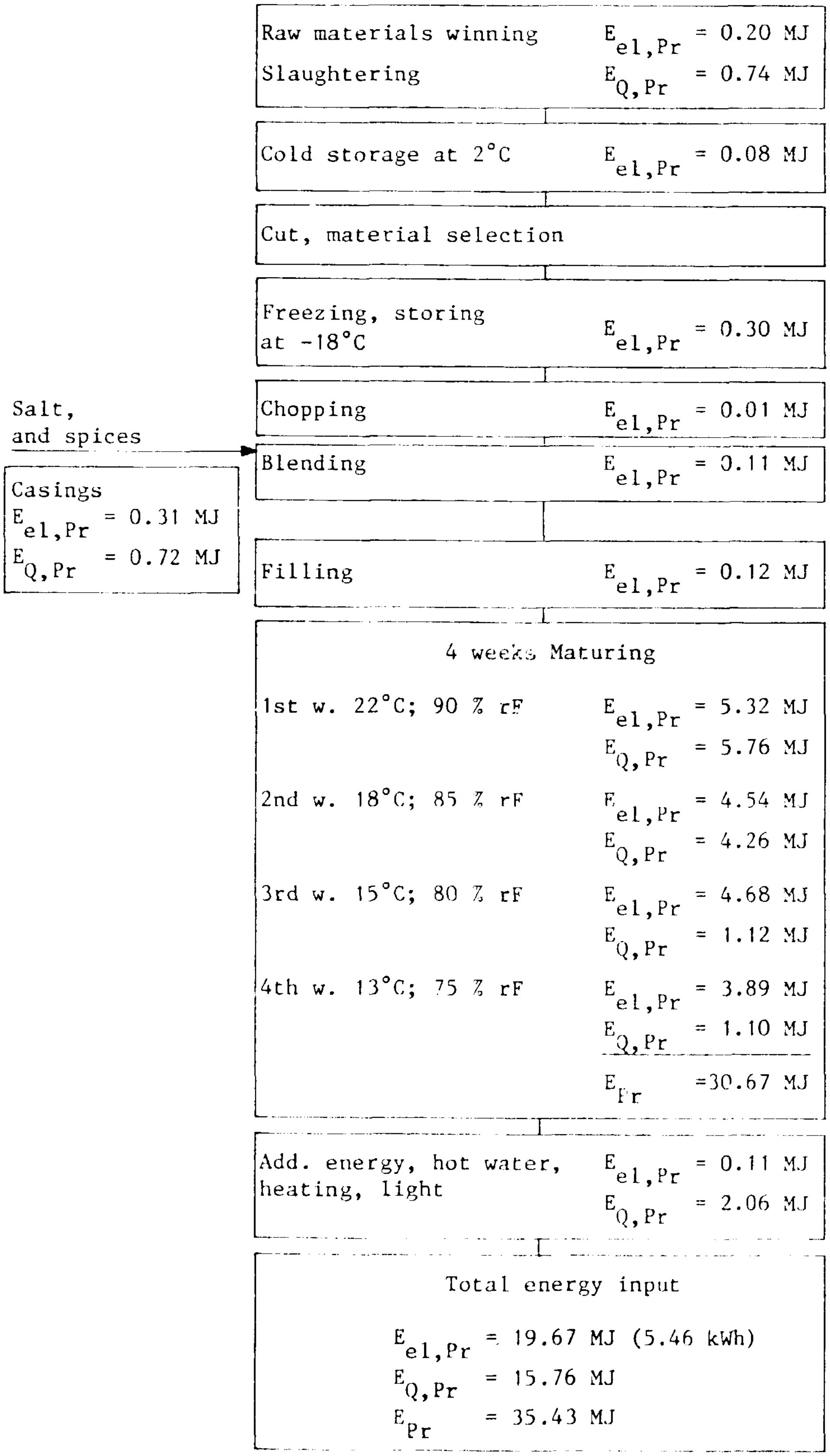
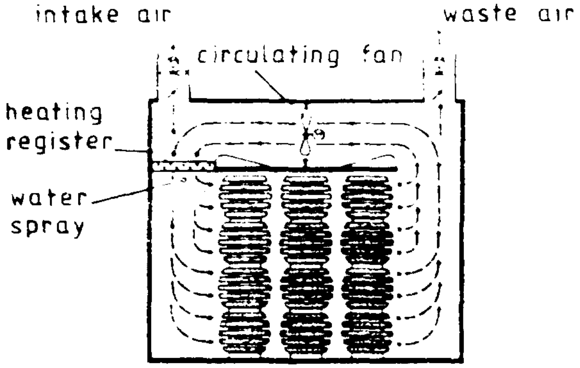


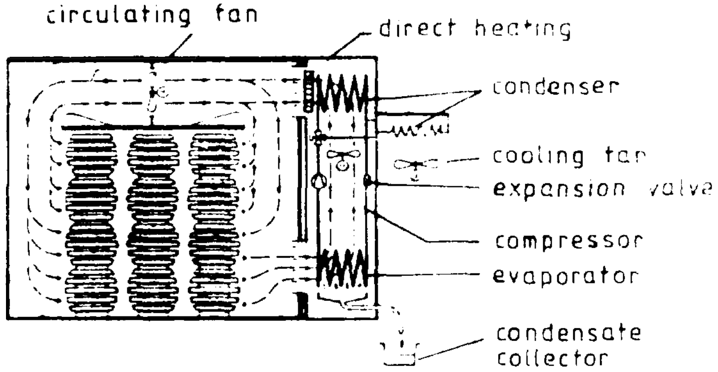
Fig. 4.1-4 Energy input per kg of raw sausage /4.1-8/

Convection timber drying

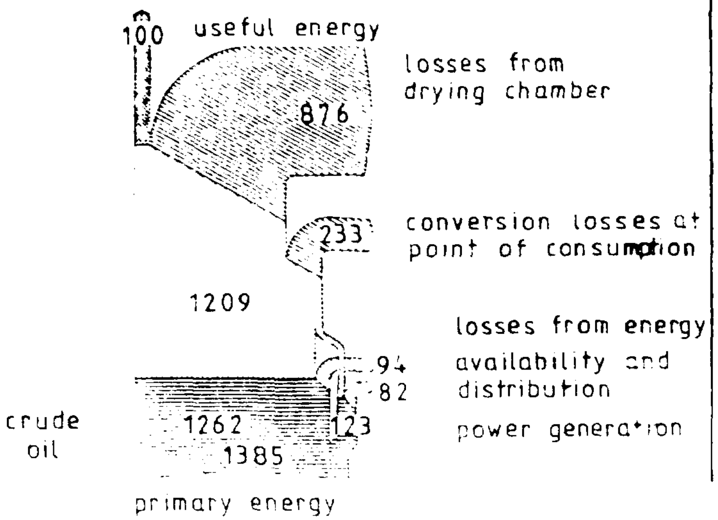
air-to-air heat pump timber drying

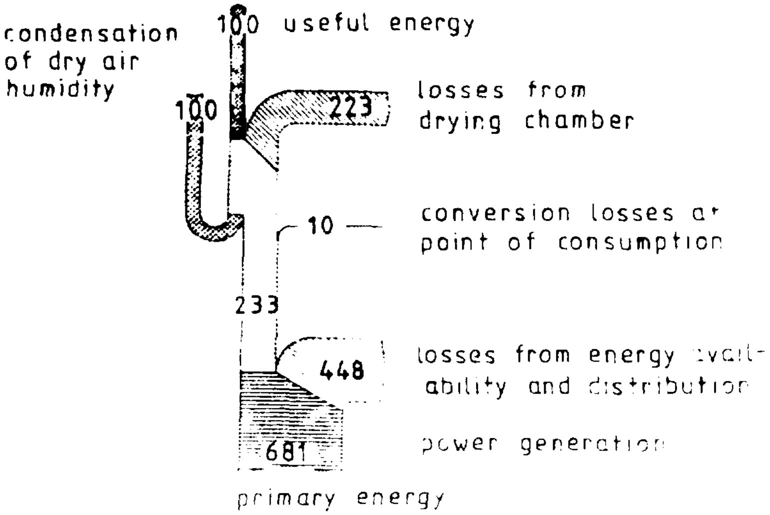
Basic Setup





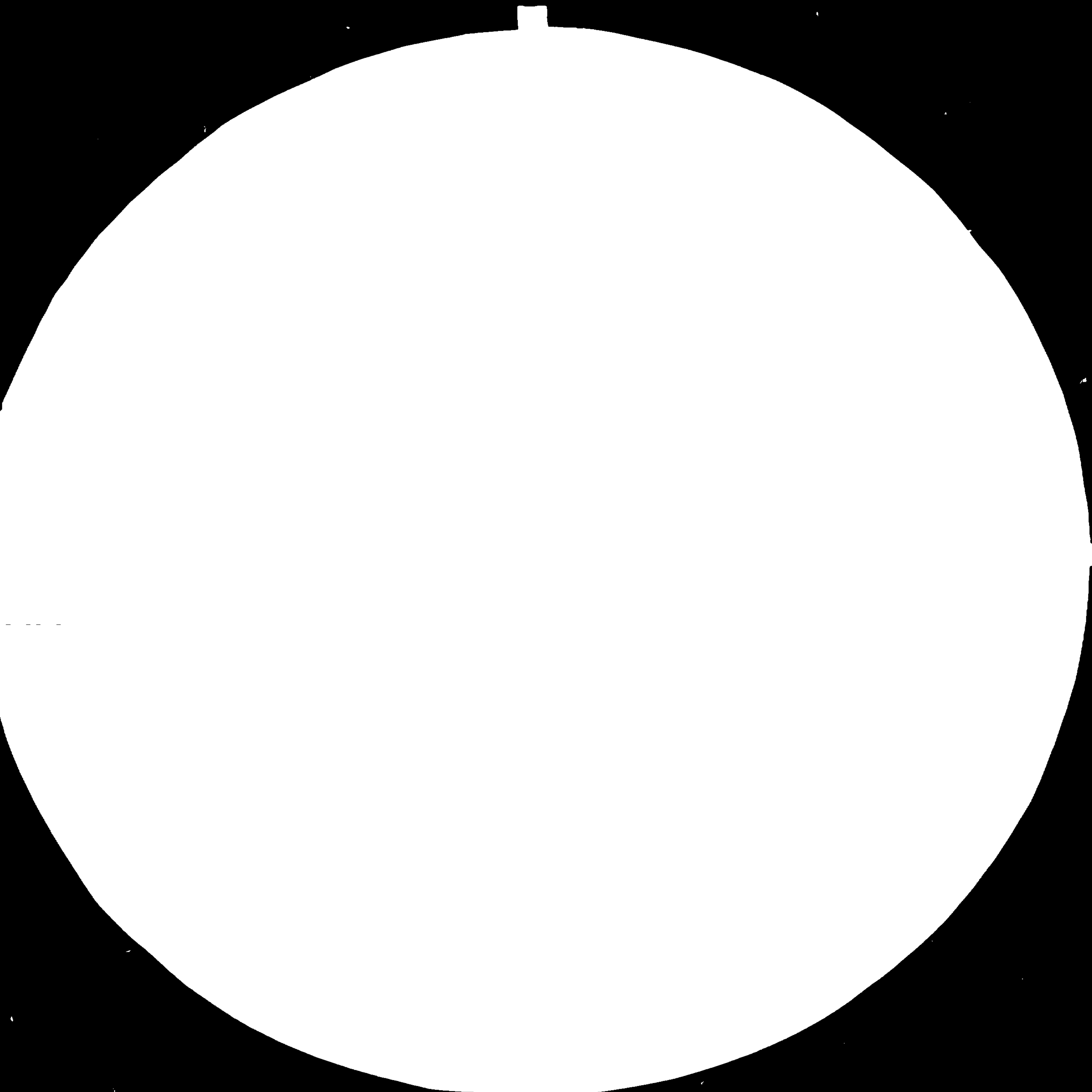
Energy flow diagrams for drying oak timber

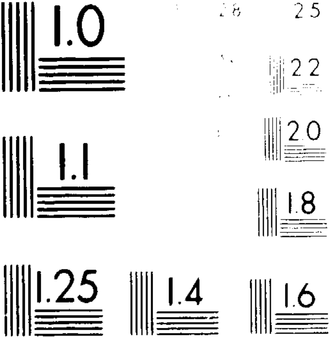




Comparison between condensation and convection drying

Fig. 5-1





MICROCOPY RESOLUTION TEST CHART

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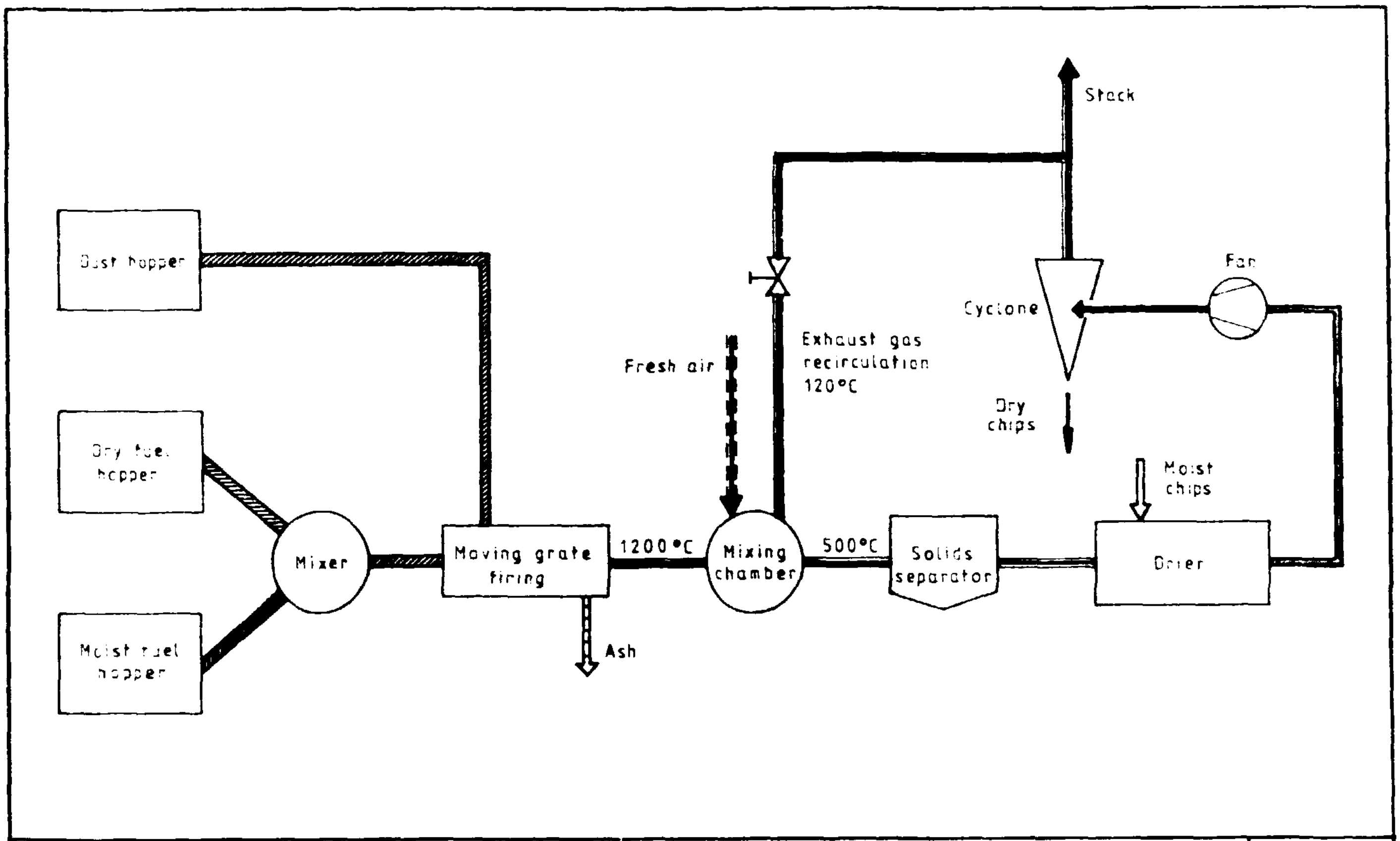
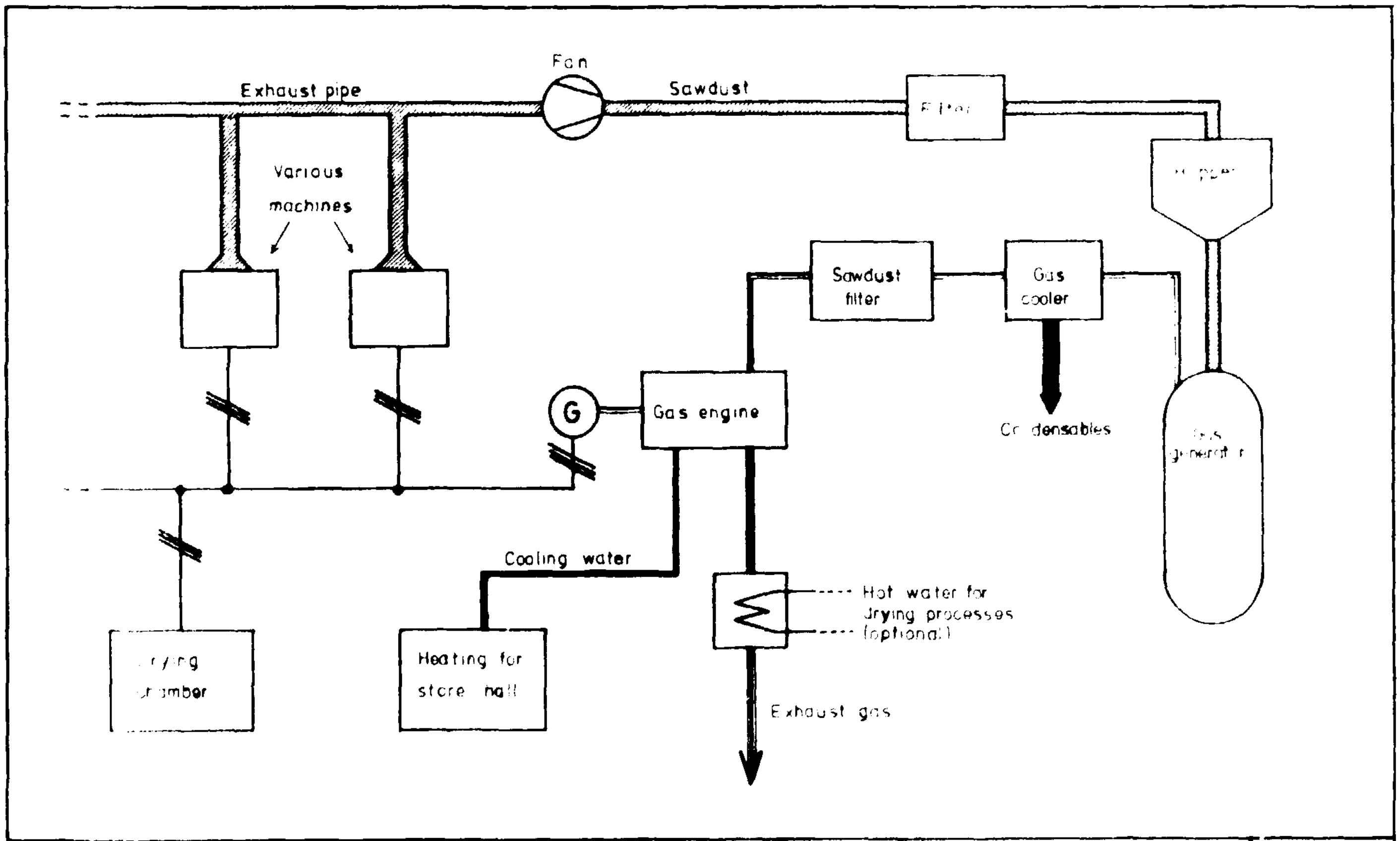


Solid-fuel chip drying plant



Wood gas power plant