



EUROPEAN COMMISSION

Integrated Pollution Prevention and Control (IPPC)

**Best Available Techniques Reference Document on the
Production of Iron and Steel**

December 2001

Executive summary

This Reference Document on best available techniques in the Iron and Steel Industry reflects an information exchange carried out according to Article 16(2) of Council Directive 96/61/EC. The document has to be seen in the light of the preface, which describes the objectives of the document and its use.

Scope

It covers the environmental aspects of iron and steel making in integrated steelworks (sinter plants, pelletisation plants, coke oven plants, blast furnaces and basic oxygen furnaces including continuous or ingot casting) and electric arc furnace steelmaking. Ferrous metal processing downstream to casting is not covered in this document.

Submitted information

The most important environmental issues of iron and steelmaking relate to emissions to air and to solid wastes/by-products. Wastewater emissions from coke oven plants, blast furnaces and basic oxygen furnaces are the most relevant emissions to water in this sector.

It is no surprise, therefore, that there is good information available on these aspects but only limited information is available about noise/vibration emissions and related measures to minimise them. The same is for soil pollution, health & safety and also for natural aspects. In addition, little information is available on the sampling methods, analysis methods, time intervals, computation methods and reference conditions used as a basis for the data submitted.

Structure of the document

The overall structure of this BREF is characterised by three main parts:

- General information on the sector
- Information on integrated iron and steelworks
- Information on electric arc furnace steelmaking

The general information includes statistical data about iron and steel production in the EU, the geographical distribution, economic and employment aspects together with rough assessment of the environmental significance of the sector. Because of the complexity of integrated steelworks an overview is given (chapter 3) before providing a full information set for the main production steps which are:

- sinter plants (chapter 4)
- pelletisation plants (chapter 5)
- coke oven plants (chapter 6)
- blast furnaces (chapter 7)
- basic oxygen steelmaking incl. casting (chapter 8)

A full information set means all the information for these production steps according to the General Outline for IPPC BAT Reference Documents. Such a “plant wise” compilation of information is made to assist the use of the document in practice.

Electric arc furnace steelmaking differs totally from integrated steelworks and is therefore presented in a separate chapter (chapter 9).

Finally, to complete the picture, information is submitted on new/alternative ironmaking techniques (chapter 10)

Chapter 11 contains the conclusions and recommendations.

General information

Iron and steel are important products that are widely used. The production of crude steel in the European Union stood at 155.3 million tonnes in 1999, equivalent to about 20% of world production.

In the EU about two thirds of crude steel are produced via the blast furnace route at 40 sites and one third is produced in 246 electric arc furnaces.

In 1995, around 330,000 people were employed in the iron and steel industry, with large numbers working in dependent industries such as construction, car manufacturing, mechanical engineering etc.

Production of iron and steel

The iron and steel industry is a highly material and energy intensive industry. More than half of the mass input becomes outputs in the form of off-gases and solid wastes/by-products. The most relevant emissions are those to air. Those from sinter plants dominate the overall emissions for most of the pollutants. Although big efforts have been made to reduce emissions, the contribution of the sector to the total emissions to air in the EU is considerable for a number of pollutants, especially for some heavy metals and PCDD/F. The rate of reuse and recycling of solid wastes/by-products has been increased dramatically in the past but considerable amounts are still disposed to landfills.

The information on the main production plants in integrated steelworks (see above) and for electric arc furnace steelmaking, begins with a concise description of applied processes and techniques in order to achieve a proper understanding of both the environmental problems and the further information.

The emission and consumption data characterise in detail the input and output mass streams structured according to the media of air, water and soil and also to energy and noise aspects (for sinter plants: table 4.1; for pelletisations plants: table 5.1; for coke oven plants: tables 6.2 and 6.3; for blast furnaces: table 7.1; for basic oxygen steelmaking and casting: table 8.2). All these data derive from existing installations and are very necessary for the evaluation of the described techniques to consider in the determination of BAT. The description of these techniques follows a certain structure (description of the technique, main achieved levels, applicability, cross-media effects, reference plants, operational data, driving force, economics, reference literature) and ends up in conclusions as to what is considered as BAT. These conclusions are based upon expert judgement in the TWG.

BAT for sinter plants (chapter 4)

Sinter, as a product of an agglomeration process of iron-containing materials, represents a major part of the burden of blast furnaces. The most relevant environmental issues are the off-gas emissions from the sinter strand, which contains a wide range of pollutants such as dust, heavy metals, SO₂, HCl, HF, PAHs and organochlorine compounds (such as PCB and PCDD/F). Thus most of the described techniques to consider in the determination of BAT refer to the reduction of emissions to air. The same applies to the conclusions; therefore the most important parameters are dust and PCDD/F.

For sinter plants, the following techniques or combination of techniques are considered as BAT.

1. Waste gas de-dusting by application of:
 - Advanced electrostatic precipitation (ESP) (moving electrode ESP, ESP pulse system, high voltage operation of ESP ...) *or*
 - electrostatic precipitation plus fabric filter *or*
 - pre-dedusting (e.g. ESP or cyclones) plus high pressure wet scrubbing system.

Using these techniques dust emission concentrations < 50 mg/Nm³ are achieved in normal operation. In case of application of a fabric filter, emissions of 10-20 mg/Nm³ are achieved.

2. Waste gas recirculation, if sinter quality and productivity are not significantly affected, by applying:
 - recirculation of part of the waste gas from the entire surface of the sinter strand,
or
 - sectional waste gas recirculation
3. Minimising of PCDD/F emissions, by means of:
 - Application of waste gas recirculation;
 - Treatment of waste gas from sinter strand;
 - use of fine wet scrubbing systems, values < 0.4 ng I-TEQ/Nm³ have been achieved.
 - Fabric filtration with addition of lignite coke powder also achieves low PCDD/F emissions (> 98 % reduction, $0.1 - 0.5$ ng I-TEQ/Nm³. – this range is based on a 6 hours random sample and steady state conditions).
4. Minimisation of heavy metal emissions
 - Use of fine wet scrubbing systems in order to remove water-soluble heavy metal chlorides, especially lead chloride(s) with an efficiency of > 90 % or a bag filter with lime addition;
 - Exclusion of dust from last ESP field from recycling to the sinter strand, dumping it on a secure landfill (watertight sealing, collection and treatment of leachate), possibly after water extraction with subsequent precipitation of heavy metals in order to minimise the quantity to dump.
5. Minimisation of solid waste
 - Recycling of by-products containing iron and carbon from the integrated works, taking into account the oil content of the single by-products (< 0.1 %).
 - For solid wastes generation, the following techniques are considered BAT in descending order of priority:
 - Minimising waste generation
 - Selective recycling back to the sinter process
 - Whenever internal reuse is hampered, external reuse should be aimed at
 - If all reuse is hampered, controlled disposal in combination with the minimisation principle is the only option.
6. Lowering the hydrocarbon content of the sinter feed and avoidance of anthracite as a fuel. Oil contents of the recycled by-products/residues < 0.1 % are achievable.
7. Recovery of sensible heat:

Sensible heat can be recovered from the sinter cooler waste gas and is feasible in some cases to recover it from the sinter grate waste gas. The application of waste gas recirculation can also be considered a form of sensible heat recovery.
8. Minimisation of SO₂ emissions by, for example:
 - Lowering the sulphur input (use of coke breeze with low sulphur content and minimisation of coke breeze consumption, use of iron ore with low sulphur content); with these measures emission concentrations < 500 mg SO₂/Nm³ can be achieved.
 - With wet waste gas desulphurisation, reduction of SO₂ emissions > 98 % and SO₂ emission concentrations < 100 mg SO₂/Nm³ are achievable.

Due to the high cost wet waste gas desulphurisation should only be required in circumstances where environmental quality standards are not likely to be met.
9. Minimisation of NO_x emissions by, for example:
 - waste gas recirculation
 - waste gas denitrification, applying
 - regenerative activated carbon process
 - selective catalytic reduction

Due to the high cost waste gas denitrification is not applied except in circumstances where environmental quality standards are not likely to be met.

10. Emissions to water (not cooling water)

These are only relevant when rinsing water is used or when wet waste gas treatment system is employed. In these cases, the effluent water to the receiving environment should be treated by heavy metal precipitation, neutralisation and sand filtration. TOC concentrations < 20 mg C/l and heavy metal concentrations < 0.1 mg/l (Cd, Cr, Cu, Hg, Ni, Pb, Zn) are achieved.

When the receiving water is fresh, attention has to be paid to salt content.

Cooling water can be recycled.

In principle the techniques mentioned in points 1 - 10 are applicable to both new and existing installations considering the preface

BAT for pelletisation plants (chapter 5)

Pelletisation is another process to agglomerate iron-containing materials. While sinter is practically always produced at the steelworks site for various reasons, pellets are mainly produced at the site of the mine or its shipping port. Therefore in the EU there is only one pelletisation plant as part of an integrated steelworks and four stand-alone plants. Also for these plants, emissions to air dominate the environmental issues. As a consequence, most of the described techniques to consider in the determination of BAT refer to emissions to air and the conclusions as well.

For pelletisation plants, the following techniques or combination of techniques are considered as BAT.

1. Efficient removal of particulate matter, SO₂, HCl and HF from the induration strand waste gas, by means of:
 - Scrubbing *or*
 - Semi-dry desulphurisation and subsequent de-dusting (e.g. gas suspension absorber (GSA)) *or any other device with the same efficiency.*

Achievable removal efficiency for these compounds are:

- Particulate matter: >95%; corresponding to achievable concentration of < 10 mg dust/Nm³
- SO₂: >80%; corresponding to achievable concentration of < 20 mg SO₂/Nm³
- HF: >95%; corresponding to achievable concentration of < 1 mg HF/Nm³
- HCl: >95%; corresponding to achievable concentration of < 1 mg HCl/Nm³

2. Emissions to water from scrubbers are minimised by means of water cycle closure, heavy metal precipitation, neutralisation and sand filtration.

3. Process-integrated NO_x abatement;

Plant design should be optimised for recovery of sensible heat and low-NO_x emissions from all firing sections (induration strand, where applicable and drying at the grinding mills).

In one plant, of the grate-kiln type and using manetite ore emissions < 150 g NO_x/t pellets are achieved. In other plants (existing or new, of the same or other type, using the same or other raw materials), solutions have to be tailor-made and the possible NO_x emission level might vary from site to site.

4. Minimisation of end-of-pipe NO_x emissions by means of end-of-pipe techniques:

Selective Catalytic Reduction or any other technique with a NO_x reduction efficiency of at least 80%.

Due to high cost waste gas denitrification should only be considered in circumstances where environmental quality standards are otherwise not likely to be met; to date there are no de-NO_x systems in operation at any commercial pelletisation plant.

5. Minimising solid waste/by-products
The following techniques are considered BAT in descending order of priority:
 - Minimising waste generation
 - Effective utilisation (recycling or reuse) of solid wastes/by-products
 - Controlled disposal of unavoidable wastes/by-products.
6. Recovery of sensible heat;
Most pelletisation plants already have a high rate of energy recovery. For further improvements, tailor-made solutions are usually necessary.

In principle the techniques listed in points 1 - 6 are applicable to both new and to existing installations considering the preface.

BAT for coke oven plants (chapter 6)

Coke is needed as the primary reducing agent in blast furnaces. Also for coke oven plants, emissions to air are most significant. However, many of these are fugitive emissions from various sources such as leakages from lids, oven doors and leveller doors, ascension pipes and emissions from certain operations like coal charging, coke pushing and coke quenching. In addition, fugitive emissions arise from the coke oven gas treatment plant. The main point source for emissions to air is the waste gas from the underfiring systems. Because of this special emission situation, detailed information is compiled in order to provide an adequate understanding. Consequently most of the techniques to consider in the determination of BAT refer to the minimisation of emissions to air. Emphasis has been placed on smooth and undisturbed operation as well as on maintenance of coke ovens, which appears to be essential.

Desulphurisation of coke oven gas is a measure of high priority to minimise SO₂ emissions, not only at coke oven plants themselves but also at other plants where the coke oven gas is used as a fuel.

Wastewater disposal is another major issue for coke oven plants. Detailed information provides a clear picture together with described techniques in order to minimise emissions to water.

The conclusions reflect the above mentioned issues. Therefore it has to be noted that coke dry quenching is not considered generally as BAT but only under certain circumstances.

For coke oven plants, the following techniques or combination of techniques are considered as BAT.

1. General:
 - Extensive maintenance of oven chambers, oven doors and frame seals, ascension pipes, charging holes and other equipment (systematic programme carried out by specially trained maintenance personnel);
 - Cleaning of doors, frame seals, charging holes and lids and ascension pipes after handling.
 - Maintaining a free gas flow in the coke ovens.
2. Charging:
 - Charging with charging cars.
From an integrated point of view, "smokeless" charging or sequential charging with double ascension pipes or jumper pipes are the preferred types, because all gases and particulate matter are treated as part of coke oven gas treatment. If however the gases are extracted and treated outside the coke oven, charging with land-based treatment of the extracted gases is the preferred method. Treatment should consist of efficient evacuation and subsequent combustion and fabric filtration. Emissions of particulate matter < 5 g/t coke are achievable.

3. Coking:
A combination of the following measures:
 - Smooth, undisturbed coke oven operation, avoiding strong temperature fluctuations;
 - Application of spring-loaded flexible-sealing doors or knife edged doors (in case of ovens $\leq 5\text{m}$ high and good maintenance) achieving:
 - $< 5\%$ visible emissions (frequency of any leaks compared to the total number of doors) from all doors in new plants *and*
 - $< 10\%$ visible emissions from all doors in existing plants.
 - Water-sealed ascension pipes, achieving $< 1\%$ visible emissions (frequency of any leaks compared to the total number of ascension pipes) from all pipes;
 - Luting charging holes with clay-suspension (or other suitable sealing material), achieving $< 1\%$ visible emissions (frequency of any leaks compared to the total number of holes) from all holes;
 - Levelling doors equipped with sealing package achieving $< 5\%$ visible emissions.
4. Firing:
 - Use of desulphurized COG
 - Prevention of leakage between oven chamber and heating chamber by means of regular coke oven operation *and*
 - Repair of leakage between oven chamber and heating chamber *and*
 - incorporation of low- NO_x techniques in the construction of new batteries, such as stage combustion (emissions in the order of $450 - 700 \text{ g/t coke}$ and $500\text{-}770 \text{ mg/Nm}^3$ respectively are achievable in new/modern plants).
 - Due to the high cost, flue gas denitrification (e.g. SCR) is not applied except in new plants under circumstances where environmental quality standards are not likely to be met.
5. Pushing:
 - Extraction with an (integrated) hood on coke transfer machine and land-based extraction gas treatment with fabric filter and usage of one point quenching car to achieve less than $5 \text{ g particulate matter/t coke}$ (stack emission).
6. Quenching:
 - Emission minimised wet quenching with less than $50 \text{ g particulate matter/t coke}$ (determined according VDI method). The use of process-water with significant organic load (like raw coke oven wastewater, wastewater with high content of hydrocarbons etc.) as quenching water is avoided.
 - Coke dry quenching (CDQ) with recovery of sensible heat and removal of dust from charging, handling and sieving operations by means of fabric filtration. With respect to present energy prices in the EU, “instrument/operational cost-environmental benefit”- consideration sets strong limitations on the applicability of CDQ. In addition a use of recovered energy must be available.
7. Coke oven gas desulphurisation:
 - Desulphurisation by absorption systems (H_2S content grid gas $500\text{-}1000 \text{ mg H}_2\text{S/Nm}^3$) *or*
 - Oxidative desulphurisation ($< 500 \text{ mg H}_2\text{S/Nm}^3$), provided that cross-media effects of toxic compounds are abated to a large extent.
8. Gas-tight operation of gas treatment plant:
All measures to enable virtually gas-tight operation of the gas treatment plant should be considered like:
 - Minimising the number of flanges by welding piping connections wherever possible;
 - Use of gas-tight pumps (e.g. magnetic pumps);
 - Avoiding emissions from pressure valves in storage tanks, by means of connection of the valve outlet to the coke oven gas collecting main (or by means of collecting the gases and subsequent combustion).

9. Wastewater pre-treatment:
 - Efficient ammonia stripping, using alkalis.
Stripping efficiency should be related to subsequent wastewater treatment. Stripper effluent NH_3 concentrations of 20 mg/l are achievable;
 - Tar removal.
10. Wastewater treatment:

Biological wastewater treatment with integrated nitrification/denitrification achieving:

 - COD removal: > 90%
 - Sulphide: < 0.1 mg/l
 - PAH (6 Borneff): < 0.05 mg/l
 - CN^- : < 0.1 mg/l
 - Phenols: < 0.5 mg/l
 - Sum of NH_4^+ , NO_3^- and NO_2^- : < 30 mgN/l
 - Suspended solids: < 40 mg/l

These concentrations are based on a specific wastewater flow of 0.4 m³/t coke.

In principle the techniques listed under points 1 - 10 are applicable to new as well as to existing installations considering the preface except low-NO_x techniques (only for new plants).

BAT for blast furnaces (chapter 7)

The blast furnace remains by far the most important process to produce pig iron from iron containing materials. Because of the high input of reducing agents (mainly coke and coal) this process consumes most of the overall energy input of an integrated steelworks.

Relevant emissions to all media occur and these are described in detail. Therefore the techniques described to consider in the determination of BAT cover all these aspects including minimisation of energy input. The subsequent conclusions are mainly concerned with the reduction of dust from the cast house, treatment of wastewater from blast furnace gas scrubbing, reuse of slag and dusts/sludges and finally the energy input minimisation and the reuse of blast furnace gas.

For blast furnaces, the following techniques or combination of techniques are considered as BAT.

1. Blast furnace gas recovery;
2. Direct injection of reducing agents;
e.g. a pulverised coal injection of 180 kg/t pig iron is already proven, but higher injection rates could be possible.
3. Energy recovery of top BF gas pressure where prerequisites are present;
4. Hot stoves
 - emission concentration of dust < 10 mg/Nm³ and of NO_x < 350 mg/Nm³ (related to an oxygen content of 3%) can be achieved
 - energy savings where design permits
5. Use of tar-free runner linings;
6. Blast furnace gas treatment with efficient de-dusting;
Coarse particulate matter is preferably removed by means of dry separation techniques (e.g. deflector) and should be reused. Subsequently fine particulate matter is removed by means of:
 - a scrubber *or*
 - a wet electrostatic precipitator *or*
 - any other technique achieving the same removal efficiency;

A residual particulate matter concentration of < 10 mg/Nm³ is possible.

7. Cast house de-dusting (tap-holes, runners, skimmers, torpedo ladle charging points); Emissions should be minimised by covering the runners and evacuation of the mentioned emission sources and purification by means of fabric filtration or electrostatic precipitation. Dust emission concentrations of 1-15 mg/Nm³ can be achieved. Regarding fugitive emissions 5-15 g dust/t pig iron can be achieved; thereby the capture efficiency of fumes is important.
Fume suppression using nitrogen (in specific circumstances, e.g. where the design of the casthouse allows and nitrogen is available).
8. Treatment of blast furnace gas scrubbing wastewater:
 - a. Reuse of scrubbing water as much as possible;
 - b. Coagulation/sedimentation of suspended solids (residual suspended solids < 20 mg/l can be achieved as an annual average, a single daily value up to 50 mg/l may occur);
 - c. Hydrocyclonage of sludge with subsequent reuse of the coarse fraction when grain size distribution allows reasonable separation.
9. Minimising slag treatment emissions and slag to landfill;
Slag treatment preferably by means of granulation where market conditions allow.
Condensation of fume if odour reduction is required.
Whenever pit slag is produced, forced cooling with water should be minimised or avoided where possible and where space restrictions allow.
10. Minimising solid waste/by-products.
For solid wastes, the following techniques are considered BAT in descending order of priority:
 - a. Minimising solid waste generation
 - b. Effective utilisation (recycling or reuse) of solid wastes/by-products; especially recycling of coarse dust from BFGas treatment and dust from cast house de-dusting, complete reuse of slag (e.g. in the cement industry or for road construction)
 - c. Controlled disposal of unavoidable wastes/by-products (fine fraction of sludge from BFGas treatment, part of the rubble)

In principle the techniques listed as points 1 - 10 are applicable to both new and existing installations considering the preface.

BAT for basic oxygen steelmaking and casting (chapter 8)

The objective of oxygen steelmaking is to oxidise the undesirable impurities still contained in the hot metal from blast furnaces. It includes the pre-treatment of hot metal, the oxidation process in the basic oxygen furnace, secondary metallurgical treatment and casting (continuous and/or ingot). The main environmental issues are emissions to air from various sources as described and various solid waste/by-products which are also described. In addition wastewater arises from wet de-dusting (when applied) and from continuous casting. Consequently the techniques to consider in the determination of BAT cover these aspects as well as the recovery of basic oxygen furnace gas. The conclusions are mainly concerned with minimisation of dust emissions from the different sources and measures to reuse/recycle solid waste/by-products, wastewater from wet de-dusting and the recovery of basic oxygen furnace gas.

For basic oxygen steelmaking and casting, the following techniques or combination of techniques are considered as BAT.

1. Particulate matter abatement from hot metal pre-treatment (including hot metal transfer processes, desulphurisation and deslagging), by means of:
 - Efficient evacuation;
 - Subsequent purification by means of fabric filtration or ESP.Emission concentrations of 5-15 mg/Nm³ are achievable with bag filters and 20-30 mg/Nm³ with ESP.

2. BOF gas recovery and primary de-dusting, applying:
 - Suppressed combustion *and*
 - Dry electrostatic precipitation (in new and existing situations) *or*
 - Scrubbing (in existing situations).

Collected BOF gas is cleaned and stored for subsequent use as a fuel. In some cases, it may not be economical or, with regard to appropriate energy management, not feasible to recover the BOF gas. In these cases, the BOF gas may be combusted with generation of steam. The kind of combustion (full combustion or suppressed combustion) depends on the local energy management.

Collected dusts and/or sludges should be recycled as much as possible. Note the usually high zinc content of the dust/sludge. Special attention should be paid to the emissions of particulate matter from the lance hole. This hole should be covered during oxygen blowing and, if necessary, inert gas injected into the lance hole to dissipate the particulate matter.
3. Secondary de-dusting, applying:
 - Efficient evacuation during charging and tapping with subsequent purification by means of fabric filtration or ESP or any other technique with the same removal efficiency. Capture efficiency of about 90% can be achieved. Residual dust content of 5-15 mg/Nm³ in case of bag filters and of 20-30 mg/Nm³ in case of ESP can be achieved. Note the usually high zinc content of the dust.
 - Efficient evacuation during hot metal handling (reladling operations), deslagging of hot metal and secondary metallurgy with subsequent purification by means of fabric filtration or any other technique with the same removal efficiency. For these operations emission factors below 5 g/t LS are achievable.

Fume suppression with inert gas during reladling of hot metal from torpedo ladle (or hot metal mixer) to charging ladle in order to minimise fume/dust generation.
4. Minimisation/abatement of emissions to water from primary wet de-dusting of BOF gas applying the following measures:
 - Dry BOF gas cleaning can be applied when space permits;
 - Recycling of scrubbing water as much as possible (e.g. by means of CO₂ injection in case of suppressed combustion systems);
 - Coagulation and sedimentation of suspended solids; 20 mg/l suspended solids can be achieved.
5. Abatement of emissions to water from direct cooling at the continuous casting machines by:
 - Recycling of process and cooling water as much as possible;
 - Coagulation and sedimentation of suspended solids;
 - Removal of oil using skimming tanks or any other effective device;
6. Minimisation of solid waste
For solid waste generation, the following techniques are considered BAT in descending order of priority:
 - Minimising waste generation
 - Effective utilisation (recycling or reuse) of solid wastes/by-products; especially recycling of BOF slag and coarse and fine dust from BOF gas treatment
 - Controlled disposal of unavoidable wastes

In principle the techniques according to items 1 - 6 are applicable to new as well as to existing installations (if there are no other indications) considering the preface.

BAT for electric steelmaking and casting (chapter 9)

The direct smelting of iron-containing materials, mainly scrap is usually performed in electric arc furnaces which need considerable amounts of electric energy and causes substantial emissions to air an solid wastes/by-products mainly filter dust and slags. The emissions to air from the furnace consist of a wide range of inorganic compounds (iron oxide dust and heavy metals) and organic compounds such as the important organochlorine compounds chlorobenzenes, PCB and PCDD/F. The techniques to consider in the determination of BAT reflect this and focus on these issues. In the conclusions, regarding emissions to air, dust and PCDD/F are the most relevant parameters. Scrap preheating is also considered as BAT just as reuse/recycling of slags and dusts.

For electric steelmaking and casting, the following techniques or combination of techniques are considered as BAT.

1. Dust collection efficiency
 - With a combination of direct off gas extraction (4th or 2nd hole) and hood systems
or
 - dog-house and hood systems *or*
 - total building evacuation98% and more collection efficiency of primary and secondary emissions from EAF are achievable.
2. Waste gas de-dusting by application of:
 - Well-designed fabric filter achieving less than 5 mg dust/Nm³ for new plants and less than 15 mg dust/Nm³ for existing plants, both determined as daily mean values.The minimisation of the dust content correlates with the minimisation of heavy metal emissions except for heavy metals present in the gas phase like mercury.
3. Minimising of organochlorine compounds, especially PCDD/F and PCB emissions, by means of:
 - appropriate post-combustion within the off gas duct system or in a separate post-combustion chamber with subsequent rapid quenching in order to avoid de novo synthesis *and/or*
 - injection of lignite powder into the duct before fabric filters.Emission concentrations of PCDD/F 0.1 - 0.5 ng I-TEQ/Nm³ are achievable.
4. Scrap preheating (in combination with 3.) in order to recover sensible heat from primary off gas
 - With scrap preheating of part of the scrap about 60 kWh/t can be saved, in case of preheating the total scrap amount up to 100 kWh/t liquid steel can be saved. The applicability of scrap preheating depends on the local circumstances and has to be proved on a plant by plant basis. When applying scrap preheating it has to be taken care of possibly increased emissions of organic pollutants.
5. Minimising solid waste/by-products
 - For solid wastes, the following techniques are considered BAT in descending order of priority:
 - Minimisation of waste generation
 - Waste minimisation by recycling of EAF slags and filter dusts; depending on local circumstances filter dust can be recycled to the electric arc furnace in order to achieve a zinc enrichment up to 30%. Filter dust with zinc contents of more than 20% can be used in the non-ferrous metal industry.
 - Filter dusts from the production of high alloyed steels can be treated to recover alloying metals.

- For solid wastes, which can not be avoided or recycled, the generated quantity should be minimised. If all minimisation/reuse is hampered, controlled disposal is the only option.
6. Emissions to water
- Closed loop water cooling system for the cooling of furnace devices.
 - Wastewater from continuous casting
 - Recycling of cooling water as much as possible
 - Precipitation/sedimentation of suspended solids
 - Removal of oil in shimming tanks or any other effective device.

In principle the techniques according to items 1 - 6 are applicable to new as well as to existing installations considering the preface.

Level of consensus

This BREF enjoys a high level of consensus. No split views had to be noted during TWG and IEF discussions. There is a broad agreement on the document.

Preface

1. Status of this document

Unless otherwise stated, references to “the Directive” in this document means the Council Directive 96/61/EC on integrated pollution prevention and control. This document forms part of a series presenting the results of an exchange of information between EU Member States and industries concerned on best available techniques (BAT), associated monitoring, and developments in them. It is published by the European Commission pursuant to Article 16(2) of the Directive, and must therefore be taken into account in accordance with Annex IV of the Directive when determining “best available techniques”.

2. Relevant legal obligations of the IPPC Directive and the definition of BAT

In order to help the reader understand the legal context in which this document has been drafted, some of the most relevant provisions of the IPPC Directive, including the definition of the term “best available techniques”, are described in this preface. This description is inevitably incomplete and is given for information only. It has no legal value and does not in any way alter or prejudice the actual provisions of the Directive.

The purpose of the Directive is to achieve integrated prevention and control of pollution arising from the activities listed in its Annex I, leading to a high level of protection of the environment as a whole. The legal basis of the Directive relates to environmental protection. Its implementation should also take account of other Community objectives such as the competitiveness of the Community’s industry thereby contributing to sustainable development.

More specifically, it provides for a permitting system for certain categories of industrial installations requiring both operators and regulators to take an integrated, overall look at the polluting and consuming potential of the installation. The overall aim of such an integrated approach must be to improve the management and control of industrial processes so as to ensure a high level of protection for the environment as a whole. Central to this approach is the general principle given in Article 3 that operators should take all appropriate preventative measures against pollution, in particular through the application of best available techniques enabling them to improve their environmental performance.

The term “best available techniques” is defined in Article 2(11) of the Directive as “the most effective and advanced stage in the development of activities and their methods of operation which indicate the practical suitability of particular techniques for providing in principle the basis for emission limit values designed to prevent and, where that is not practicable, generally to reduce emissions and the impact on the environment as a whole.” Article 2(11) goes on to clarify further this definition as follows:

- “techniques” includes both the technology used and the way in which the installation is designed, built, maintained, operated and decommissioned;
- “available” techniques are those developed on a scale which allows implementation in the relevant industrial sector, under economically and technically viable conditions, taking into consideration the costs and advantages, whether or not the techniques are used or produced inside the Member State in question, as long as they are reasonably accessible to the operator;
- “best” means most effective in achieving a high general level of protection of the environment as a whole.

Furthermore, Annex IV of the Directive contains a list of “considerations to be taken into account generally or in specific cases when determining best available techniques ... bearing in mind the likely costs and benefits of a measure and the principles of precaution and prevention”. These considerations include the information published by the Commission pursuant to Article 16(2).

Competent authorities responsible for issuing permits are required to take account of the general principles set out in Article 3 when determining the conditions of the permit. These conditions must include emission limit values, supplemented or replaced where appropriate by equivalent parameters or technical measures. According to Article 9(4) of the Directive, these emission limit values, equivalent parameters and technical measures must, without prejudice to compliance with environmental quality standards, be based on the best available techniques, without prescribing the use of any technique or specific technology, but taking into account the technical characteristics of the installation concerned, its geographical location and the local environmental conditions. In all circumstances, the conditions of the permit must include provisions on the minimisation of long-distance or transboundary pollution and must ensure a high level of protection for the environment as a whole.

Member States have the obligation, according to Article 11 of the Directive, to ensure that competent authorities follow or are informed of developments in best available techniques.

3. Objective of this Document

Article 16(2) of the Directive requires the Commission to organise “an exchange of information between Member States and the industries concerned on best available techniques, associated monitoring and developments in them”, and to publish the results of the exchange.

The purpose of the information exchange is given in recital 25 of the Directive, which states that “the development and exchange of information at Community level about best available techniques will help to redress the technological imbalances in the Community, will promote the world-wide dissemination of limit values and techniques used in the Community and will help the Member States in the efficient implementation of this Directive.”

The Commission (Environment DG) established an information exchange forum (IEF) to assist the work under Article 16(2) and a number of technical working groups have been established under the umbrella of the IEF. Both IEF and the technical working groups include representation from Member States and industry as required in Article 16(2).

The aim of this series of documents is to reflect accurately the exchange of information which has taken place as required by Article 16(2) and to provide reference information for the permitting authority to take into account when determining permit conditions. By providing relevant information concerning best available techniques, these documents should act as valuable tools to drive environmental performance.

4. Information Sources

This document represents a summary of information collected from a number of sources, including in particular the expertise of the groups established to assist the Commission in its work, and verified by the Commission services. All contributions are gratefully acknowledged.

5. How to understand and use this document

The information provided in this document is intended to be used as an input to the determination of BAT in specific cases. When determining BAT and setting BAT-based permit conditions, account should always be taken of the overall goal to achieve a high level of protection for the environment as a whole.

The rest of this section describes the type of information that is provided in each section of the document.

Chapters 1, 2 and 3 provide general information on the industrial sector and the first sections of Chapters 4 to 9 gives information on the industrial processes used within the sector. Current emission and consumption levels are then presented in the second sections of Chapter 4 to 9 reflecting the situation in existing installations at the time of writing.

The third sections of Chapters 4 to 9 describe in more detail the emission reduction and other techniques that are considered to be most relevant for determining BAT and BAT-based permit conditions. This information includes the consumption and emission levels considered achievable by using the technique, some idea of the costs and the cross-media issues associated with the technique, and the extent to which the technique is applicable to the range of installations requiring IPPC permits, for example new, existing, large or small installations. Techniques that are generally seen as obsolete are not included.

A conclusions section in each of Chapters 4 to 9 presents the techniques and the emission and consumption levels that are considered to be compatible with BAT in a general sense. The purpose is thus to provide general indications regarding the emission and consumption levels that can be considered as an appropriate reference point to assist in the determination of BAT-based permit conditions or for the establishment of general binding rules under Article 9(8). It should be stressed, however, that this document does not propose emission limit values. The determination of appropriate permit conditions will involve taking account of local, site-specific factors such as the technical characteristics of the installation concerned, its geographical location and the local environmental conditions. In the case of existing installations, the economic and technical viability of upgrading them also needs to be taken into account. Even the single objective of ensuring a high level of protection for the environment as a whole will often involve making trade-off judgements between different types of environmental impact, and these judgements will often be influenced by local considerations.

Although an attempt is made to address some of these issues, it is not possible for them to be considered fully in this document. The techniques and levels presented in the conclusion section on Best Available Techniques in each of chapters 4 to 9 will therefore not necessarily be appropriate for all installations. On the other hand, the obligation to ensure a high level of environmental protection including the minimisation of long-distance or transboundary pollution implies that permit conditions cannot be set on the basis of purely local considerations. It is therefore of the utmost importance that the information contained in this document is fully taken into account by permitting authorities.

Since the best available techniques change over time, this document will be reviewed and updated as appropriate. All comments and suggestions should be made to the European IPPC Bureau at the Institute for Prospective Technological Studies at the following address:

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Scope

This BREF covers the processes involved in the production of iron and steel in an integrated works as well as the production of steel in electric arc furnace steel works

The main operations covered by the descriptions are:

- loading, unloading and handling of bulk raw materials,
- blending and mixing of raw materials,
- coke production,
- sintering and pelletisation of iron ore,
- the production of molten iron by the blast furnace route, including slag processing
- the production and refining of steel using the basic oxygen process, including upstream ladle desulphurisation, downstream ladle metallurgy and slag processing
- the production of steel by electric arc furnaces, including downstream ladle metallurgy and slag processing
- continuous casting

Reheat or heat treatment furnaces, power plants, oxygen plants as well as all other steel related downstream processes like rolling, pickling, coating etc. are not included and will be treated by separate BREFs.

Furthermore there are environmental aspects which do not specifically relate to the primary iron and steel as well as to the electric arc furnace steel production such as:

- particulate matter emissions from storage and handling of raw materials, additives and auxiliaries as well as of intermediate products,
- occupational safety and hazard risk,
- cooling systems and
- emission monitoring.

In this document, these aspects are only briefly treated but will be covered by other reference documents.

1 GENERAL INFORMATION

1.1 Overall Steel production in Europe and Worldwide

Iron and steel have played an important role in the development of human civilisation over several millennia and have found uses in agriculture, construction, the generation and distribution of power, the manufacturing of machinery and equipment, in the household and in medicine.

Together with coal and cotton, iron and steel were the principal materials upon which the industrial revolution was based. Technical developments from the early eighteenth century onwards allowed dramatic increases in output, for example by replacing relatively scarce charcoal with hard coal/lignite and coke respectively and by the development of the puddling process for converting pig iron into steel. Steel production has grown exponentially in the second half of the twentieth century, rising to a world total of 757 million tonnes in 1995 (Figure 1.1).

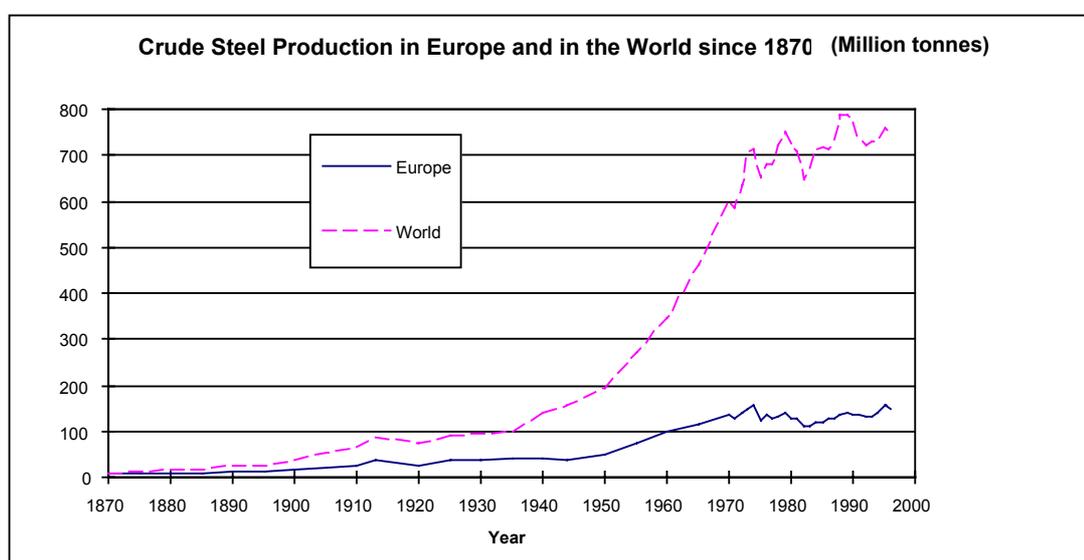


Figure 1.1 : Crude steel production in Europe and Worldwide since 1870 - [Stat. Stahl, 1997]

Figure 1.1 also shows how the European share of world steel production has steadily declined, falling to a level of 21% at the end of the period. Since the oil crisis of 1974-75 production has been virtually stagnant worldwide, with Europe being particularly affected.

EU production of crude steel grew at a rate of 1.2% per annum between 1985 and 1994 in terms of volume (see Figure 1.2) Growth was more rapid in the three years to 1989 (3.7% per annum). Subsequently production in the EU 12 declined from 140 million tonnes in 1992 and 1993, before recovering to reach 139 million tonnes in 1994 and 143 million tonnes in 1995. The entry of the three new Member States –Austria, Finland and Sweden– brought EU production of crude steel up to 156 million tonnes in 1995, as compared with Eastern European production of 112 million tonnes (Russia's share was 51 million tonnes), and production by Japan, the USA and China of 102, 94 and 93 million tonnes, respectively [EC Panorama, 1997]. In 1999, the production of crude steel in the EU was 155.3 million tonnes or 19.7% of world production, according to Eurofer and IISI Sources.

Figure 1.2 also shows how production of oxygen steel has remained fairly steady, whereas electric arc furnace steel production gradually increased. The latter's share of total steel production reached 34.4% in 1995. Nevertheless, the blast furnace - basic oxygen furnace route is predicted to remain the dominant means of steel production, at least for the next twenty years [Luengen, 1995].

The apparent steel consumption per capita in the EU in 1995 was 367 kg/capita (varying from 152 kg/capita in Ireland to 533 kg/capita in Italy) [Stat. Stahl, 1997].

Furthermore, Figure 1.2 shows the decline in the number of electric arc furnaces and oxygen converters after 1990, whilst the capacities of both remaining and new installations increased. The higher number for 1995 is a reflection of the entry of the three new Member States.

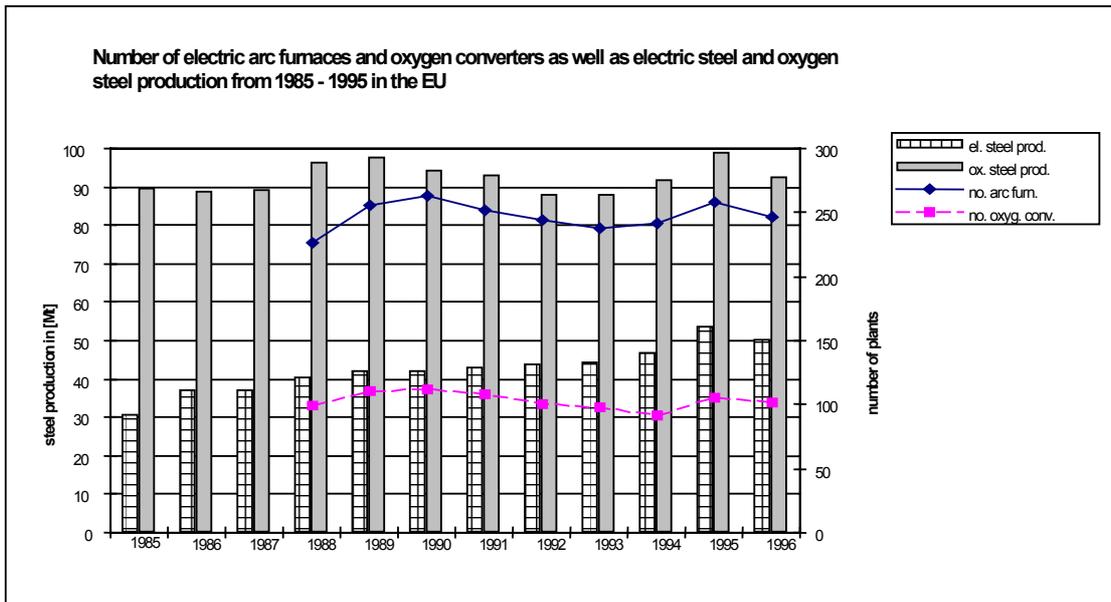


Figure 1.2 : Production of electric-arc furnace and oxygen steel in the EU from 1985 - 1995
-[Stat. Stahl, 1997]

1.2 Geographical distribution of steel production in the EU

The locations of integrated steelworks in the EU 15 are shown in Figure 1.3, in which the concentration of steelworks along the coal belt in Central Europe is clearly visible. Nevertheless, there are integrated steelworks located in most of the Member States. The number of electric arc furnaces is much higher. According to [Stat. Stahl, 1997] there are 246 electric arc furnaces in the EU 15. Their locations are not included in Figure 1.3.

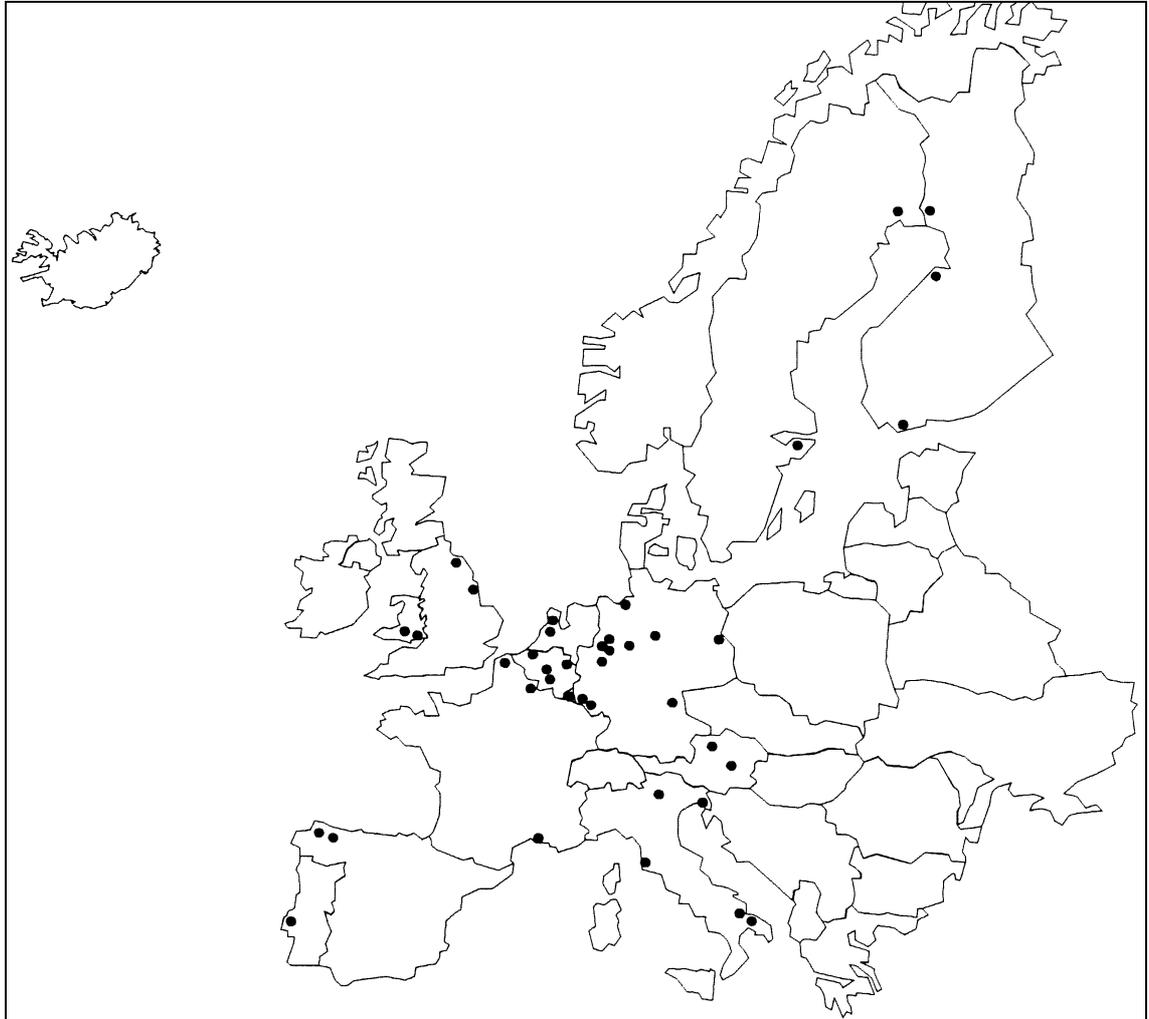


Figure 1.3 : Geographical distribution of Integrated Steelworks in the European Union

Figure 1.4 shows Germany to have the largest steel industry in Europe, producing 40 million tonnes of crude steel in 1996, i.e. 27% of EU 15 production. It is followed by Italy which accounts for 16% of EU 15 production. The six largest EU producers (Germany, Italy, France, the UK, Spain and Belgium) together accounted for 83% of EU production in 1996.

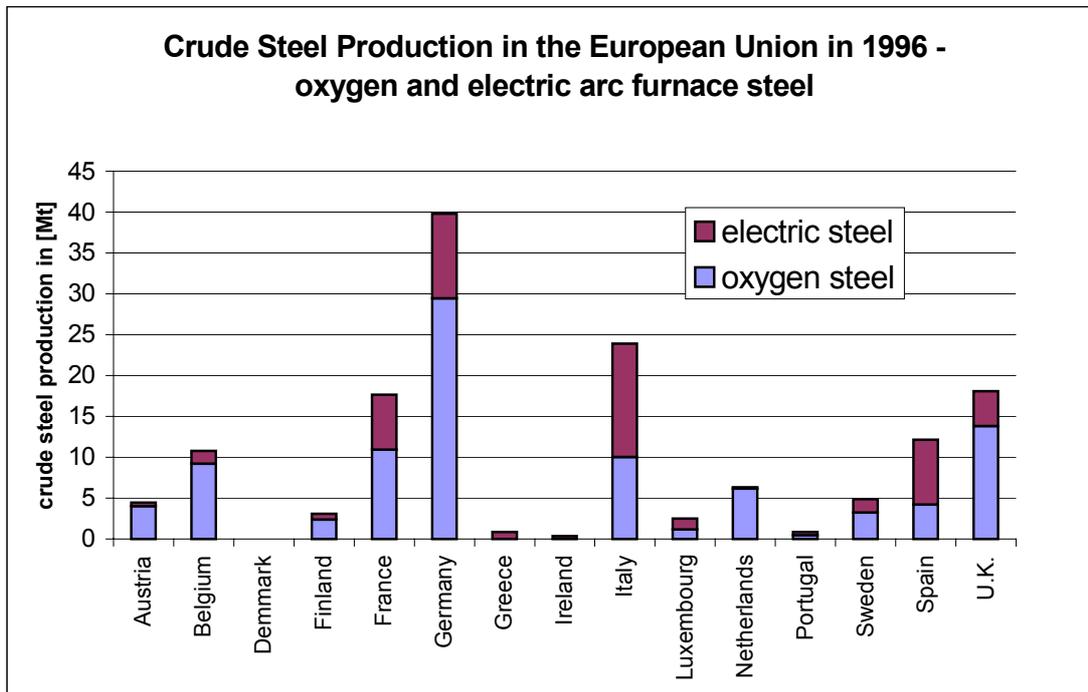


Figure 1.4 : Oxygen and electric arc furnace steel production in the EU Member States in 1996
- [Stat. Stahl, 1997]

In 1996 the average production capacity utilisation factor in the EU was 73.3% varying between 22.2% in Greece and 93.2% in the Netherlands [Stat. Stahl, 1997].

The number of plants in the EU 15 for the main processing steps in pig iron and steel production (coke plants, sinter plants, blast furnaces, basic-oxygen furnaces and electric-arc furnaces) are given in Figure 1.5.

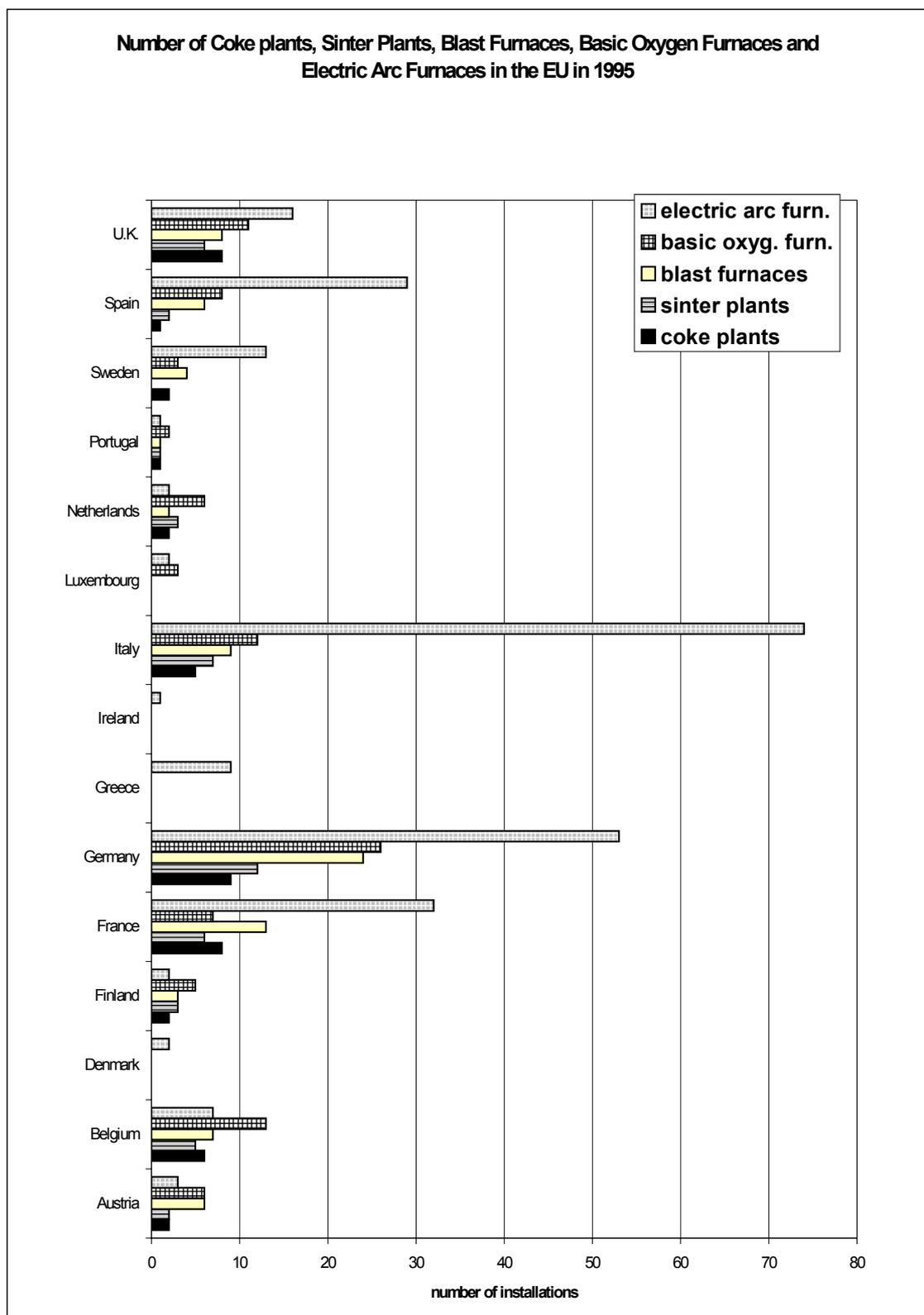


Figure 1.5 : Number of pig iron and steel making plants in the EU 15 - [Stat. Stahl, 1997; Stahl, 1996]

A coke oven plant normally consists of 2 – 4 coke oven batteries.

The distribution of plants by capacities and other characteristics in the EU 12 are compiled in Table 1.1, which also includes data for continuous casting (data only available for the EU 12)

Table 1A: Coke oven Batteries

Number of installations	Average age [years]	average age of last important modernisation [years]	Average number of ovens	Average dimensions of ovens			Production in 1993 [Mt]			
				Height [mm]	Width [mm]	Length [mm]				
106	27	17	40	4957	438	13590	34.1			
Annual Technical Capacity (ATC) [in kt/a] for the different sizes of coke oven batteries, the number of installations (No.) for the different size classes and the Total Annual Technical Capacity (TATC) of each size class [in Mt/a]										
ATC	<200	200-299	300-399	400-499	500-599	600-699	700-799	800-899	900-999	≥1000
No.	35	25	16	9	3	13	2	0	0	3
TATC	5.43	6.03	5.31	3.84	1.59	8.22	1.43	-	-	4.17

Table 1B: Sinter plants (strand sintering)

Number of Installations	average age	Average age of last important modernisation [years]	Average strand width [mm]	Average area of total strand [m ²]	Average annual technical capacity [Kt/a]		
47	23	16	3420	252	2521		
Size distribution of sinter strands according to their area of total strand [in m ²] (Area), the number of installations (No.) for the different size classes and the Total Annual Technical Capacity (TATC) of each size class [in Mt/a]							
Area	<100	100-199	200-299	300-399	400-499	500-599	≥ 600
No.	5	19	7	4	8	2	2
TATC	2.8	31.9	17.3	13.1	37.9	10.6	4.9

Table 1C: Blast furnaces

Number of Installations	average age [years]	Average age of last important Modernisation [years]	Average useful volume [m ³]	Average hearth diameter [m]	Average daily technical capacity [t/24hrs]				
81	28	7	1721	9.4	3931				
Size distribution of blast furnaces according to their hearth diameter [in m] (HD), the number of installations (No.) for the different size classes and the Total Daily Technical Capacity (TDTC) of each size class [in kt/24hrs]									
HD	4.0-5.99	6.0-6.99	7.0-7.99	8.0-8.99	9.0-9.99	10.0-10.99	11.0-11.99	12.0-12.99	≥ 13
No.	4	6	5	15	20	17	7	1	6
TDTC	3.0	8.6	9.0	38.5	67.6	83.6	41.8	7.0	59.3

Table 1D: Basic Oxygen furnaces (oxygen converters)

Number of installations	average age		average age of last important modernisation		Average capacity per heat		
	[years]		[years]		[t/heat]		
95	22		11		204		
Capacity per heat (C/H) and number of installations (No.) for the different size classes (data for the Total Annual Technical Capacity (TATC) are not available)							
C/H	50-99	100-149	150-199	200-249	250-299	300-349	350-399
No.	17	12	17	18	13	13	5

Table 1E: AC electric arc furnaces (DC arc furnaces, of which there are only 4, not included)

Number of Installations	average age	average age of last important modernisation	Average capacity per heat	Average rating of transformer	Average annual technical capacity					
	[years]	[years]	[t/heat]	[1000 kVA]	[kt/a]					
203	21	9	73	44	335					
Size distribution of electric arc furnaces according to their capacity per heat (C/H) [in t/heat], the number of installations (No.) for the different size classes and the Total Annual Technical Capacity (TATC) of each size class [in Mt/a]										
C/H	< 20	20-39	40-59	60-79	80-99	100-119	120-139	140-159	160-179	≥ 180
No.	19	27	28	44	33	18	18	9	4	3
TATC	0.3	2.6	5.3	15.6	15.1	9.3	9.6	7.2	1.8	1.4

Table 1F: Continuous casting for billets or blooms

Number of Installations	average age	average age of last important modernisation	Number of installations attached to an oxygen melting shop	Average annual technical capacity			
	[years]	[years]		[Kt/a]			
149	16	8	30	514			
Annual Technical Capacity (ATC) [in kt/a] for the different sizes of continuous casting for billets and blooms, the number of installations (No.) for the different size classes and the Total Annual Technical Capacity (TATC) of each size class [in Mt/a]							
ATC	<200	200-399	400-599	600-799	800-999	1000-1199	≥1200
No.	17	47	29	29	18	4	5
TATC	2.3	14.1	14.3	20.0	15.5	4.2	6.2

Table 1G: Continuous casting for slabs

Number of installations	average age [years]	average age of last important modernisation [years]	Number of installations attached to an electric melting shop	Average annual technical capacity [Kt/a]			
65	14	8	16	1399			
Annual Technical Capacity (ATC) [in kt/a] for the different sizes of continuous casting for slabs, the number of installations (No.) for the different size classes and the Total Annual Technical Capacity (TATC) of each size class [in Mt/a]							
ATC	<500	500-999	1000-1499	1500-1999	2000-2499	2500-2999	≥3000
No.	10	11	14	14	10	4	2
TATC	3.5	7.1	17.8	23.0	21.9	11.0	6.6

Table 1.1 : Number and characterisation of plants for pig iron and crude steel production (coke oven batteries, sinter plants, blast furnaces, basic oxygen furnaces, electric arc furnaces and continuous casting) in the EU 12 in 1993 - [Eurostat, 1993];

The number of EAF do not correspond with number mentioned under 1.2 because one refers to EU 12 and the other to EU 15.

The development of continuous casting had an extremely positive effect on the economics of the steel industry in the 1970s and 1980s. The 1990s, however, have not been marked by mass introduction of radical new technologies in the EU. Rather, the classical methods of production have been refined at various stages (blast furnaces, steelworks) and have been rendered much more efficient by improvements all along the production chain. These have allowed significant reductions in energy use and pollution, whilst increasing product quality.

A number of DC electric-arc furnaces have been built and put into operation since 1993 (three in Germany, two in Belgium and one in France).

1.3 Investments and employment in the EU iron and steel industry

Investment expenditure declined from 5.9 billion Ecu in 1991 to 2.7 billion Ecu in 1994 (EU 12), but rose again to 4.0 billion Ecu in 1996 (EU 15) [EC Panorama, 1997]. Of this, around 16% was spent on iron making (sinter plants, coke plants and blast furnaces), 13% on electric steelworks, 4% on oxygen steelworks, 7% on continuous casting 31% on rolling mills and the remainder on coating and miscellaneous [Eurofer 1, 1997].

In the EU, employment in the iron and steel industry declined steadily during the 1990s (Figure 1.6) as a result of static production levels and rationalisation in the industry. The result of this introduction of new techniques and working practices was an increase in productivity of 64% between 1985 and 1994. In 1995, around 330 thousand people were employed by the iron and steel industry in the EU 15 [EC Panorama, 1997], with large numbers working in dependent industries such as construction, car manufacturing, mechanical engineering etc.

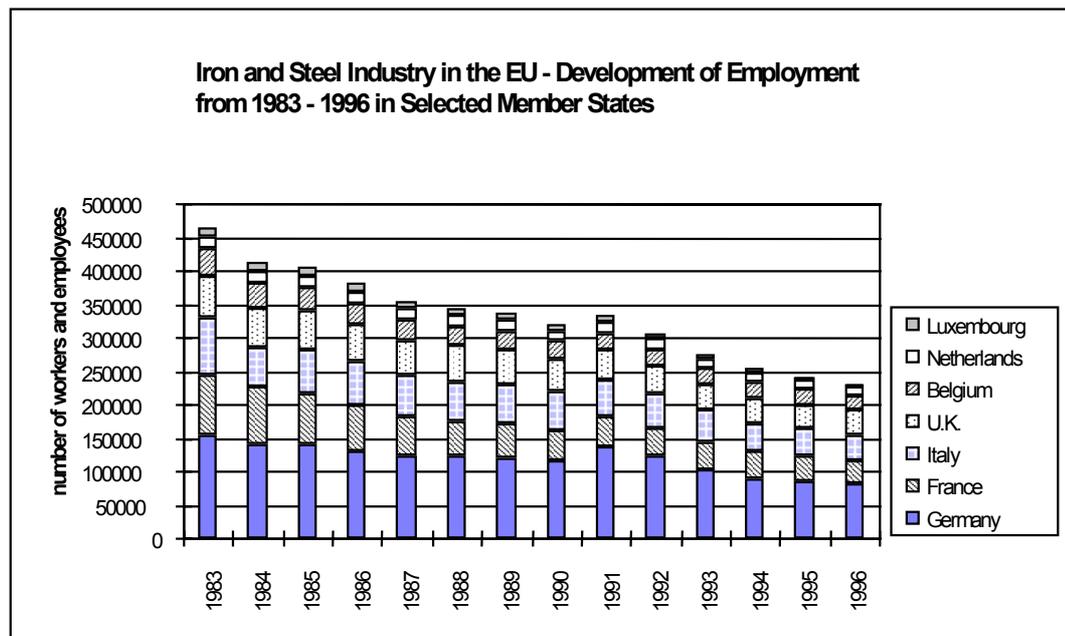


Figure 1.6 : Development of employment in the iron and steel industry in the EU 15 from 1983 - 1996 - [Stat. Stahl, 1997]

In order to unify the European market for coal, iron and steel and promote its development the European Coal and Steel Community (ECSC) was founded in 1951 [ECSC Treaty, 1951]. This was an important step in the evolution of the European iron and steel industry, but will end in 2002.

1.4 Economic situation

The globalisation of the world economy has had a profound effect on the steel industry and it will continue to do so. Despite, or perhaps because of, market stagnation, the industry is undergoing intensive structural change. This is characterised by the development of new concepts in steelworking (e.g. mini-electric steel mills, new concepts for electric arc furnaces, new casting techniques and direct or smelting reduction techniques). Highly competitive market conditions may accelerate this structural change and encouraging consolidation in the steel industry. This is evident from the growing number of alliances, co-operative ventures and take-overs.

1.5 Environmental relevance of the iron and steel industry

The iron and steel industry is highly intensive in both materials and energy. Figure 1.7 presents a simplified input/output scheme illustrating and numbering the main input mass streams together with the quantity of crude steel produced in the EU 15 in 1995. The figure illustrates the way in which more than half of the input ends up as off gases and solid co-products/residues.

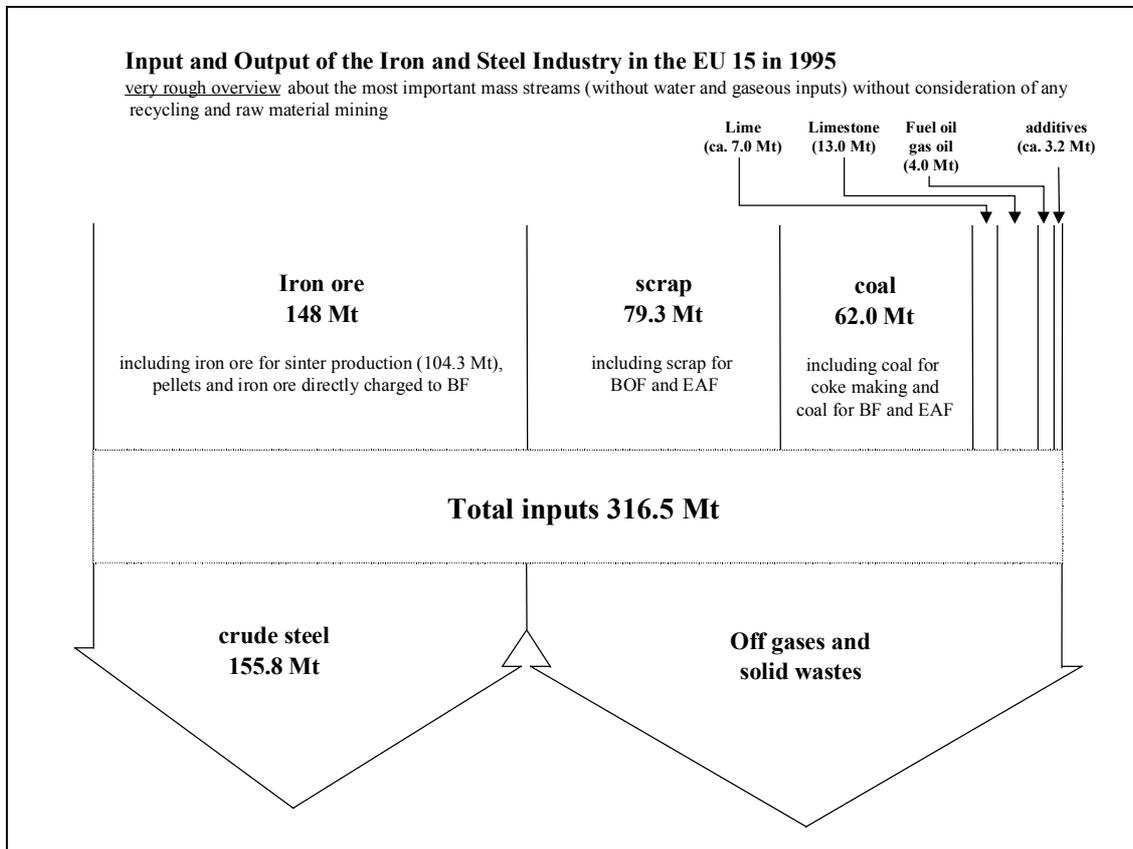


Figure 1.7 : Overview of the inputs and outputs of the iron and steel industry in the EU 15 in 1995 - based on [Stat. Stahl, 1997]

The overview given in Figure 1.1 is simplified and shows only the most important mass streams without considering recycling or the mining of raw materials.

Important subject for action in response to environmental concerns are generally considered to relate to controlling air emissions and managing solid waste.

Air pollution remains an important issue. In integrated steelworks, sinter plants dominate the overall emissions for most atmospheric pollutants, followed by coke-oven plants (Figure 1.8).

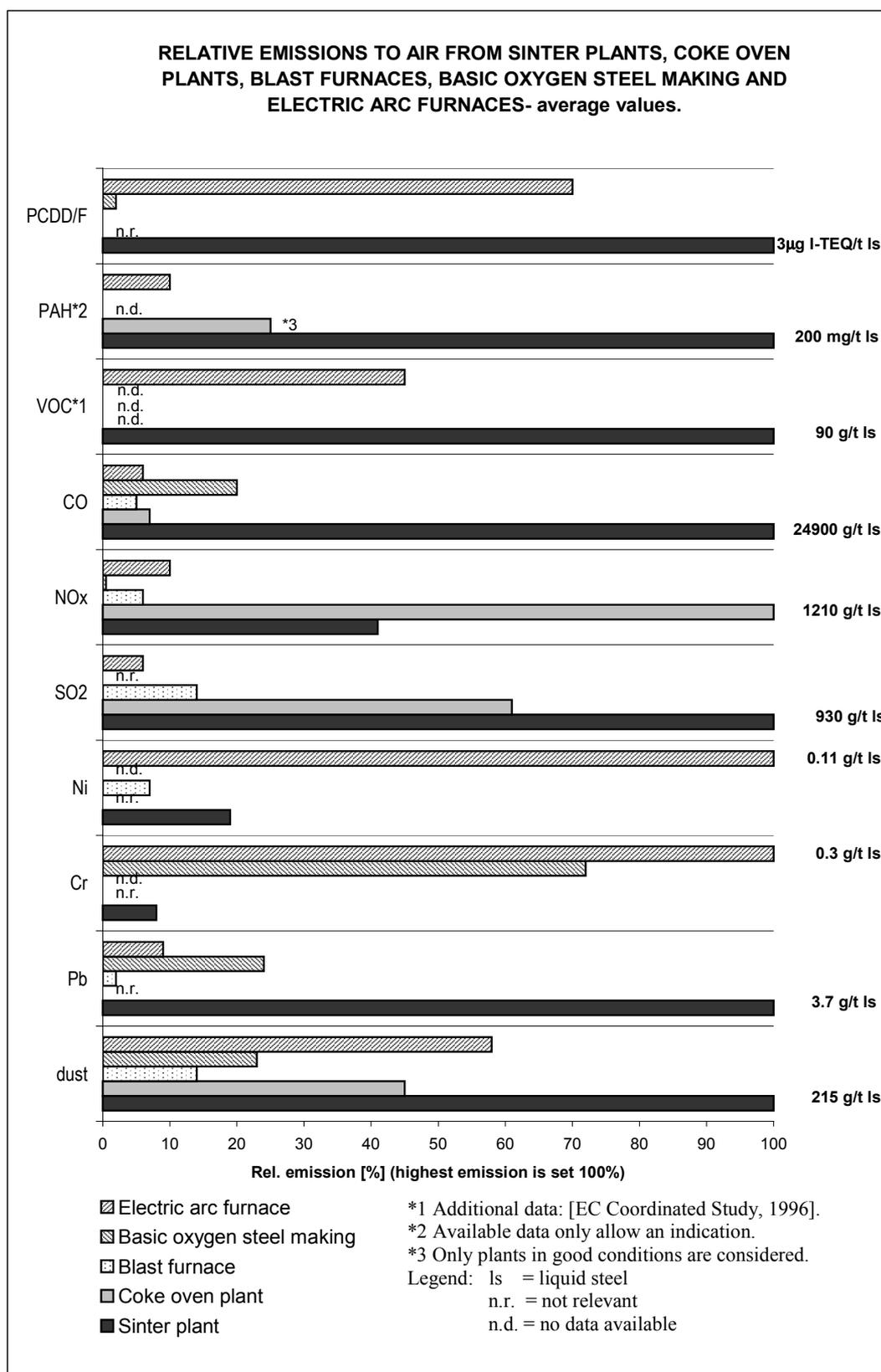


Figure 1.8 : Relative emissions to air of selected pollutants from sinter plants, coke oven plants, blast furnaces, basic-oxygen steelmaking and electric-arc furnaces

In the figure a relative value of 100% is assigned to the type of plant giving rise to the highest emission for each individual pollutant (annotated with the absolute specific quantity). The emissions of the others plant categories are presented relative to these plants with the highest emissions.

Blast furnaces, basic oxygen steelmaking, coke ovens as well as electric arc furnaces have considerable relative percentages of dust emissions. The gaseous emissions from electric-arc furnace steelmaking can be relevant for mercury and for the organic pollutants mentioned in Figure 1.8.

The first steps towards air pollution control were taken with dust collection and removal. In the eighties and nineties dust removal has become increasingly effective (especially secondary dedusting). This has reduced the directly related heavy metal emissions except in the case of those with high vapour pressure such as mercury. Efforts to minimise SO₂ and NO_x emissions have also been made. In addition the emissions of organohalogen compounds such as polychlorinated dibenzo-p-dioxins and -furans (PCDD/F), hexachlorobenzene (HCB) and polychlorinated biphenyls (PCB) together with polycyclic aromatic hydrocarbons (PAH) and monocyclic aromatic hydrocarbons, especially benzene, became increasingly important. The so-called diffuse emissions from plants and emissions from open yard storage also became subject to control.

The contribution of the iron and steel industry to the overall emissions to air in the EU is significant for heavy metals and PCDD/F, but less so in the cases of NO_x and SO₂ (Table 1.2).

Parameter	Year	Emissions in the EU 15 [t/a]	Percentage Contribution of the Iron and Steel Industry [%]
SO ₂	1994	12088000 ^{*1,2}	Ca. 1.5 ^{*6}
NO _x	1994	12435000 ^{*1,2}	ca. 1 ^{*6}
Cd ^{*3}	1990	200	19
Cr ^{*3}	1990	1170	55
Cu ^{*3}	1990	3040	5
Hg ^{*3}	1990	250	3
Ni ^{*3}	1990	4900	3
Pb ^{*3,4}	1996	12100	9 ^{*7,8}
Zn ^{*3}	1990	11100	35
PCDD/F ^{*5}	1995	5800 g I-TEQ	19

^{*1} power generation is not covered

^{*2} data source: [EEA, 1997]

^{*3} data source: [TNO Report, 1997]; numbers cover integrated steelworks (sinter plants, coke ovens, blast furnaces, basic oxygen steelmaking and electric arc furnace steelmaking)

^{*4} data corrected because of recent drastic reduction of vehicle lead emissions, based on [UN-ECE Lead, 1998]

^{*5} data from [LUA NRW, 1997]

^{*6} calculated from the emission factors given in chapter 3

^{*7} with the phaseout of lead in petrol the percentage will increase

^{*8} the crosscheck with the data given in Figure 1.8 and in chapter indicates, that the percentage is only 4%

Table 1.2 : Contribution of integrated steelworks (sinter plants, coke oven plants, blast furnaces, basic-oxygen steelworks) and of electric-arc furnace steel-making to the overall emissions of SO₂, NO_x, heavy metals and PCDD/F in the EU 15.

The quantities of solid residues, waste, and by-products are generally high (Table 1.3). Recycling and reuse has reached fairly high levels but varies greatly across the EU and would benefit from further optimization.

Solid wastes/residues/by-products	specific quantity [kg/t LS] ^{*1}	percent. disposed of on landfills [%] ^{*2}
sinter plants ^{*3}		
• dust	0.9-15	?
coke oven plants ^{*4}	-	-
blast furnaces		
• Cast house dust	?	?
• dust and sludge from BFGas purification	14	33
• slag	280	2
pig iron desulphurisation	9 - 18	41
basic oxygen steelmaking		
• coarse dusts and sludges from BOF gas treatment	3-12 ^{*8}	42
• Fine dusts and sludges from BOF gas treatment	9-15 ^{*8}	12
• converter slag	99	26
• slags from pig iron ladle, mixer, steel ladle and tundish	34	9
• slags from secondary metallurgy	11 ^{*5}	?
• dismantled refractories	6	76
electric arc furnace		
• slag		
• carbon steels	129	69
• low alloyed steels	109	59
• high alloyed and stainless steels	161	34
• dust from furnace and building evacuation	15 ^{*6}	63 ^{*6}
continuous casting	4 - 6	- ^{*7}

^{*1} used conversion factors (weighted average of all European basic oxygen steelworks): 940 kg pig iron/t LS

^{*2} the residual percentage is recycled within the works or used externally

^{*3} dusts from sinter strand, cooler and building air dedusting are normally fully recycled to the strand except some cases in which the dust from the last field of the electrostatic precipitator is disposed to landfill; in case of application a fine scrubber the sludge from wastewater treatment is also disposed to landfill (in Europe only very few cases)

^{*4} the by-products benzene, tar, naphthalene, sulphuric acid, elementary sulphur are fully recycled/reused

^{*5} average from three basic oxygen steelworks

^{*6} average for carbon, low alloyed and high alloyed/stainless steels

^{*7} mill scale normally is recycled to the sinter plant

^{*8} from [EUROFER BOF, 1997]

all data from [EC Study, 1996], unless stated otherwise. Average values shown.

Legend: LS = liquid steel; BF = blast furnace; BOF = basic oxygen furnace

Table 1.3 : Average specific quantity and average percentage of landfill disposal of solid residues/wastes/by-products from integrated steelworks and from electric-arc furnace steelmaking

Wastewater discharge from coke oven plants is of significant higher relevance than discharges from the water circuits at blast furnaces, basic-oxygen steelmaking, and continuous casting plants.

Noise emission, local soil pollution as well as groundwater pollution are other matters of concern for the iron and steel industry.

The energy consumption is considerable. The specific energy consumption for 1 t liquid steel, produced via the coke oven/sinter plant/blast furnace route is about 19.3 GJ (calculated according to [Lüngen, 1995; Peters, 1994] and Tables 4.1, 5.1, 6.2, 7.1 and 8.2). This amount is dominated by the coal input.

It also includes the consumption of electricity, which is about 1.6 GJ/t LS (the electricity consumption itself has been multiplied by factor three to make it comparable with primary energy).

The specific energy consumption for the production of electric arc furnaces steel is about 5.4 GJ/t LS (according to Table 10.1). The electricity consumption has also been multiplied by factor three.

2 STORAGE AND HANDLING OF RAW MATERIALS

Primary iron and steelmaking involves high input mass streams of raw materials such as ores, pellets, scrap, coal, lime, limestone (in some cases also heavy oil and plastics) and of additives and auxiliaries. These materials are usually transferred to the site in bulk carriers by road, rail or water transport. Both these and the intermediate products, such as coke and sinter, have to be stored on stockyards or silos and transported to the individual processing plants, usually by conveyor belt. Figure 2.1 shows a typical material flow diagram. Wind-borne dust from the stockyards and conveyor belts, including transfer points, can be a significant source of emissions. When material including leachable compounds and such as hydrocarbons from mill scale or scrap, is stored in unpaved stockyards attention also has to be paid to soil and ground water pollution and to run-off water.

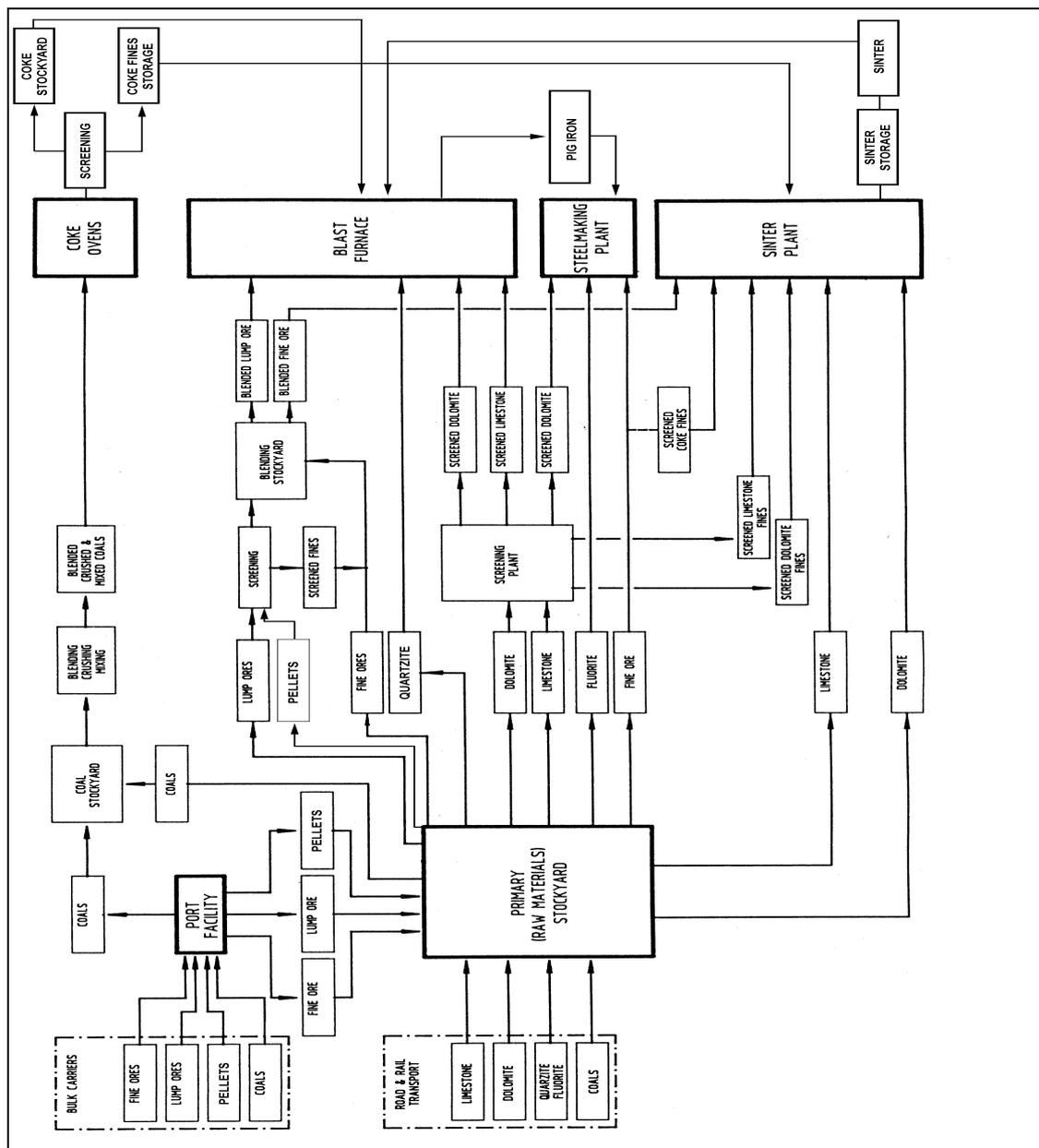


Figure 2.1 : Mass flow diagram of a typical material handling in an integrated steel works - [UK HMIP, 1993]

3 STEEL-MAKING - OVERVIEW

3.1 Steel-making process routes

Four routes are currently used for the production of steel: the classic blast furnace/basic-oxygen furnace route, direct melting of scrap (electric arc furnace), smelting reduction and direct reduction (Figure 3.1).

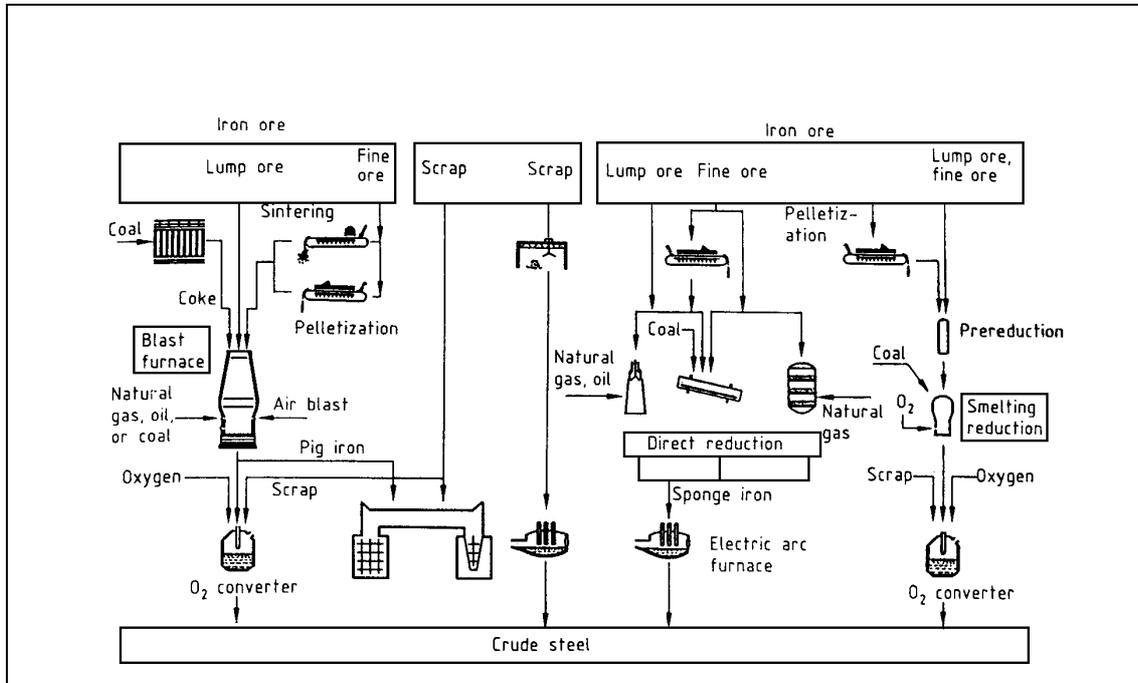


Figure 3.1 : Crude steel production methods - [Ullmann's, 1994]

At present (1998), EU (15) steel production is based on the blast furnace/ basic-oxygen route (approximately 65%) and the electric arc furnace (EAF) route (approximately 35%) (see Figure 1.2). The percentage of world crude steel production via smelting and direct reduction was about 4% in 1996 [Hille, 1997]. In Europe production of direct reduced iron (DRI) is limited to about 500000 t/a (Germany and Sweden) representing approximately 1.5% of world output. Consumption of DRI in EAF steel-making was reported to be 400000 t in the EU (15) in 1995, but interest in this material is increasing and new production technologies are emerging. In the EU (15), there are currently no smelting reduction units on a commercial scale. These routes are therefore not described here, but will be mentioned in the chapter "Alternative iron-making techniques".

3.2 Integrated steelworks

Of the four steel-making routes described in 3.1 the classic blast furnace/basic oxygen furnace route is by far the most complex, taking place in large industrial complexes known as integrated steelworks, covering areas up to several square kilometres (Figure 3.2). Integrated steelworks are characterised by networks of interdependent material and energy flows between the various production units, most of which are covered in this BREF (sinter plants, pelletisation plants, coke oven plants, blast furnaces and basic oxygen steel-making plants with subsequent casting). Before describing these individual types of plant in detail an overview of the interdependencies mentioned will be given.



Figure 3.2 : Aerial view of an integrated steelworks located near the coast

3.2.1 Process overview

The process routes of an integrated steelworks considered in this BREF are shown in Figure 3.3. The figure gives a schematic view of the main material inputs and outputs (emission mass streams) for each stage of the process route.

In an integrated steel works the blast furnace is the main operational unit where the primary reduction of oxide ores takes place leading to liquid iron, so-called pig iron. Modern high-performance blast furnaces require physical and metallurgical preparation of the burden. The two types of iron ore preparation plants are the sinter plants and the pellet plants. Pellets are nearly always made from one well-defined iron ore or concentrate (see Figure 2.1) at the mine and are transported in this form. In Europe there is only one integrated steelworks also operating a pelletisation plant. Sinter is generally produced at the ironworks from predesigned mixtures of fine ores, residues and additives.

The main reducing agents in a blast furnace are coke and powdered coal forming carbon monoxide and hydrogen which reduce the iron oxides. Coke and coal also partly act as a fuel.

Coke is produced from coal by means of dry distillation in a coke oven and has better physical and chemical characteristics than coal. In many cases, additional reducing agents/fuels are

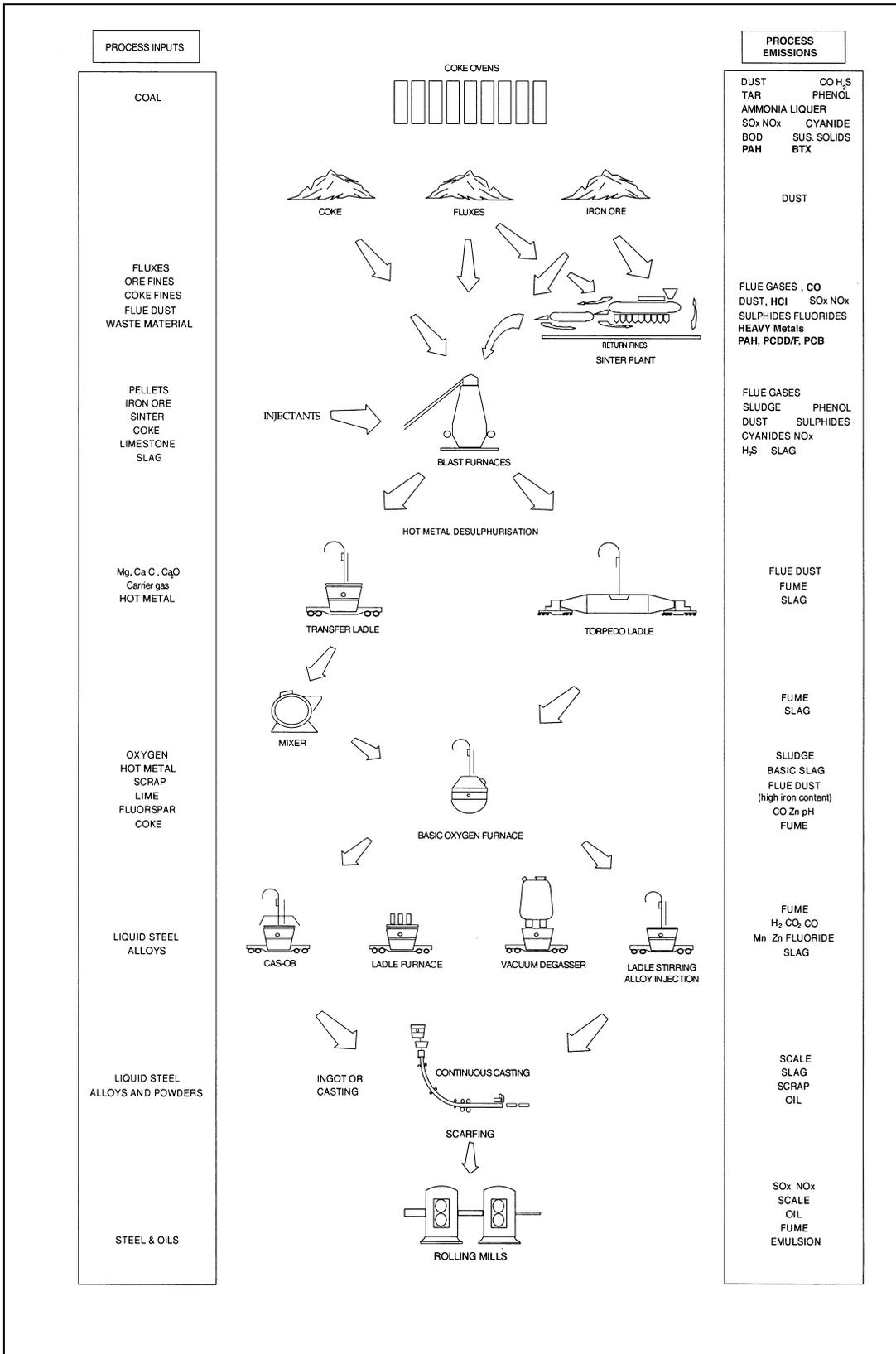


Figure 3.3 : Overview of the process route of an integrated steel works - [UK IPR 2/1, 1994] - (a pelletisation plant is not shown, as only one integrated steelworks in Europe has one)

supplied by injection of oil, natural gas and (in a few cases) plastics. A hot blast provides the necessary oxygen to form the carbon monoxide (CO), which is the basic reducing agent for the iron oxides.

The blast furnace is charged at the top with burden consists of alternate layers of coke and a mixtures of sinter and/or pellets, lump ore and fluxes. In the furnace the iron ore is increasingly reduced and liquid iron and slag are collected in the bottom of the furnace, from where they are tapped.

The slag from the blast furnace is granulated, pelletised, or tapped into slag pits. The slag granules or pellets are usually sold to cement manufacturing companies. Also, slag from pits can be used in road construction.

The liquid iron from the blast furnace (pig iron) is transported to a basic oxygen furnace, where the carbon content (approx. 4%) is lowered to less than 1%, thereby resulting in steel. Upstream ladle desulphurisation of the pig iron and downstream ladle metallurgy of the steel is generally applied in order to produce steel with of the required quality. On leaving the basic oxygen furnace the liquid steel is cast, either into ingots or by means of continuous casting. In some cases vacuum degassing is applied in order to further improve the quality of the steel.

Casting products, whether ingots, slabs, billets or blooms, are subsequently processed in rolling mills and product finishing lines in order to prepare them for market.

3.2.2 Interdependency of the different production processes/units in terms of energy, by-products/residues, air and water

The process overview in Figure 3.3 shows the various production units of an integrated steelworks. The individual units are connected both in terms of product flows and internal flows of residues (mill scale, filter dusts, sludges from scrubbing BFGas or BOF gas etc.), water (common treatment of various wastewater streams, cascade usage of cooling water, etc.) and energy (COG, BFGas, BOF gas, steam from BF top pressure turbines or basic oxygen furnaces etc.). These interdependencies have been installed in order both to minimize emissions and to optimize productivity and reduce costs.

3.2.2.1 Energy

Energy interdependency is the most complex of these interdependencies. Figure 3.4 illustrates an example of the input and output flows of each different kind of energy together with the internal energy flow of an integrated steelworks. The dominant energy inputs are coal and, if bought from an external supply, coke. Also electricity, natural gas, oil and (in a few cases) plastics represent the energy inputs. Coke oven gas (COG), blast furnace gas (BFGas) and basic oxygen furnace gas (BOF gas) are used for many purposes (heating coke oven batteries, provision of hot blast, ignition of the sinter feed, heating furnaces for hot rolling etc.). Steam from top pressure turbines of the blast furnaces or from basic oxygen furnaces is also used for various processes.

COG and BFGas are recovered and used at all integrated steelworks. However, this is not the case for BOF gas or for steam recovery using BF top pressure turbines. Steam recovery is dependent on top pressure of the blast furnace, on the operation condition of the BOF and the usability of BOF gas.

Figure 3.5 gives a detailed description of the quantitative distribution of different energy types in an integrated steelworks. All data in this figure relate to total energy input, including that from external electricity sources. Approximately 88% of the imported energy is ultimately

derived from coal, 83% of which is converted into coke. Blast furnaces consume about 60% of the overall energy demand of the steelworks, followed by rolling mills (25%), sinter plants (about 9%) and coke ovens (about 7%).

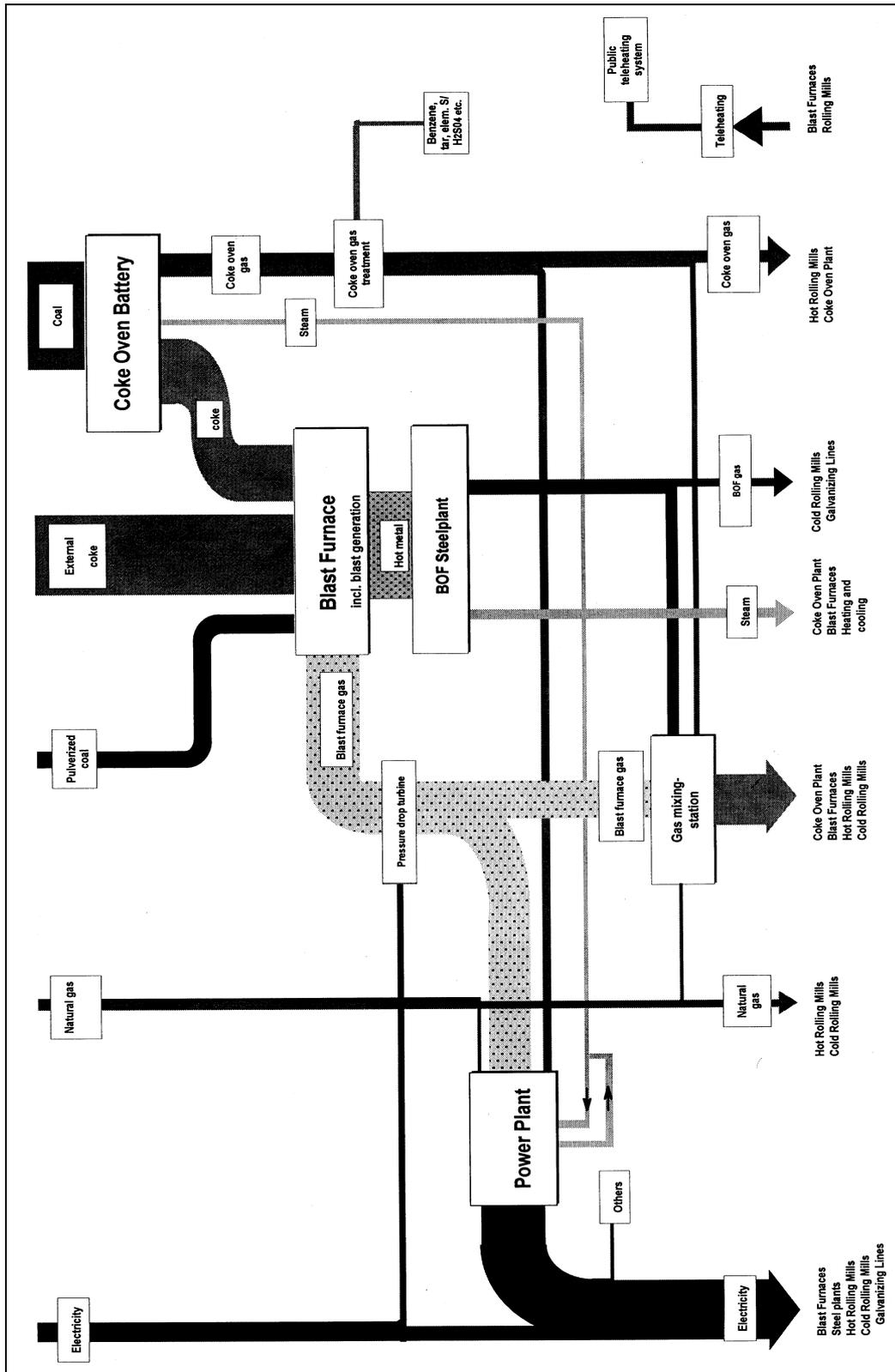


Figure 3.4 : Example of the input, output and internal flows in a modern integrated steelworks energy utilization system - [Joksch, 1998].

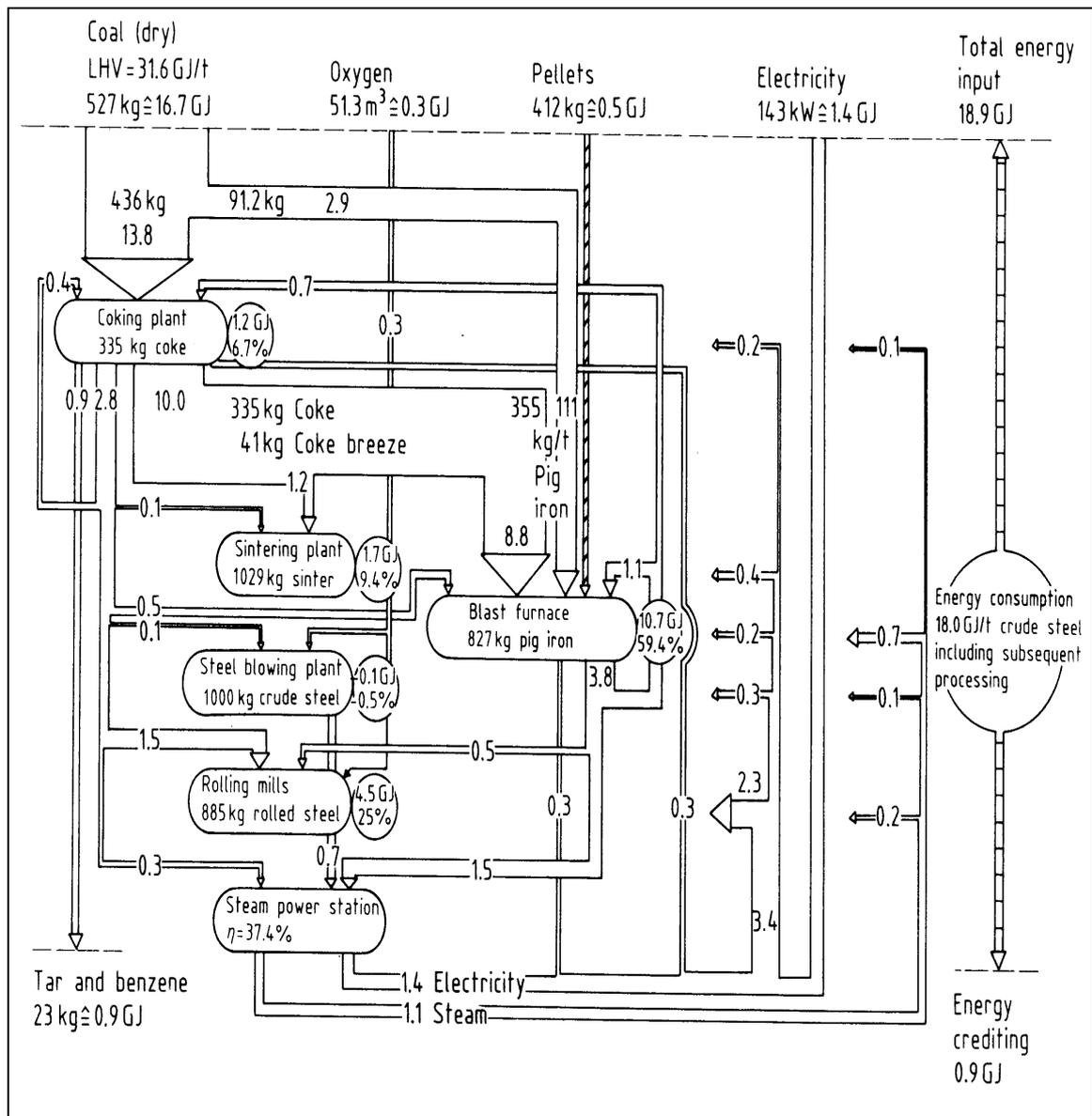


Figure 3.5 : Typical energy demand distribution in an integrated steelworks per tonne of crude steel - [Ullmann's, 1989];

This example confirms the numbers given in 1.5.

3.2.2.2 Solid residues/by-products

The management of residues in an integrated steelworks is characterized by advanced techniques for extracting value from the various types of slag and by recycling of most of the residues in the sinter plant, which can be considered the 'digester of an integrated steelworks'. Thus, beside the sintering process itself, the sinter plants plays this important role of recycling residues for which no adequate alternatives exist. Only small parts of the overall quantity of residues are landfilled. This often consists of fine dust from blast-furnace gas (BFgas) cleaning, rubble, fine dust from basic oxygen furnace (BOF) gas scrubbing (if wet cleaning is used) and , in some cases, the high alkali chlorides and heavy metal chlorides from the last field of electrostatic precipitators treating the off-gas from sinter strands. Figure 3.6 gives an typical example of the management of by-products and residues in an integrated steelworks.

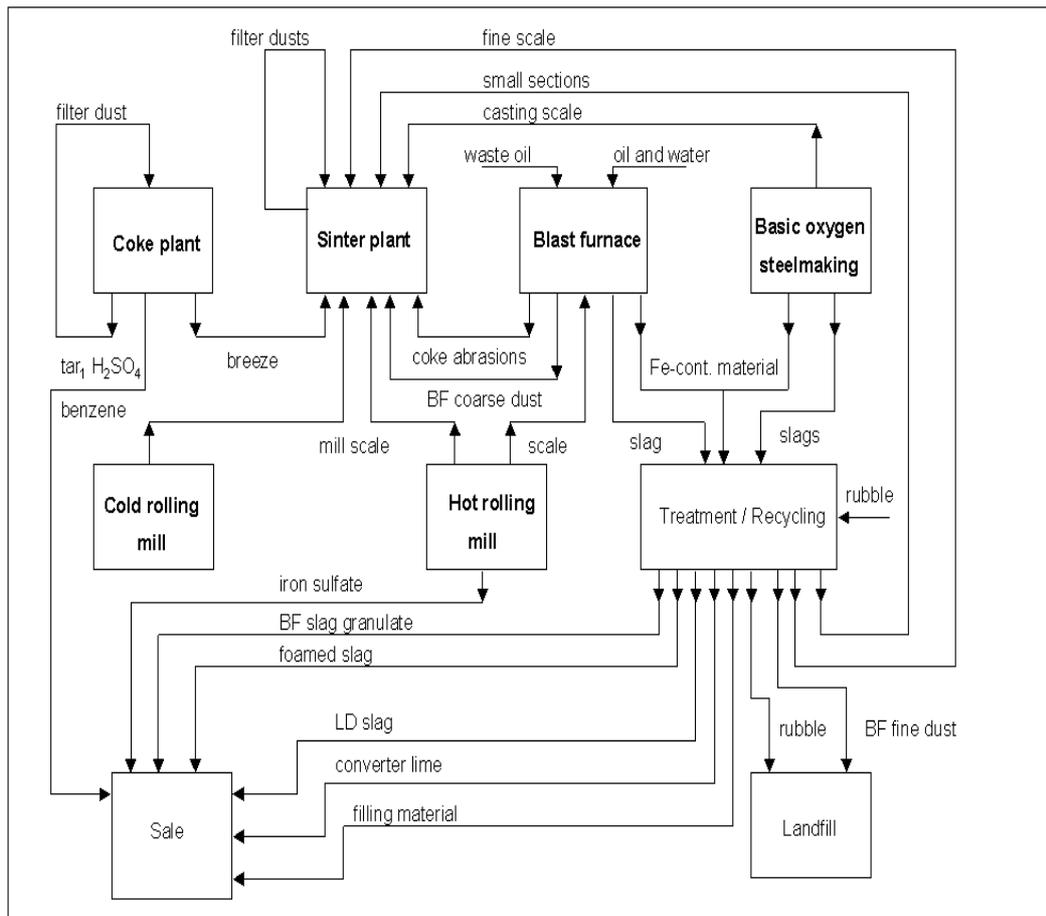


Figure 3.6 : Typical example for the management of residues and by-products in an integrated steelworks - based on [Bothe, 1993]

3.2.2.3 Water

The water management in an integrated steelworks primarily depends on the local conditions, above all on the availability of fresh water and on legal requirements. Legal restraints would tend to focus on minimizing discharges of cooling water and materially polluted wastewater but there are also cases where the authorities demand the avoidance of plumes from re-cooling towers, which prevents further cooling water recycling.

Figure 3.7 gives an example of water management with an indication of the water treatment of an integrated steelworks with surplus of intake water availability, thus explaining the presence of many once-through cooling systems, resulting in a specific water consumption of more than 100 m³/t steel. At sites with very low fresh water availability there is a need to save water as much as possible. In such cases the specific water consumption can be less than 10 m³/t steel, and sometimes less than 5 m³/t steel, in which case the interdependencies are much more intensive.

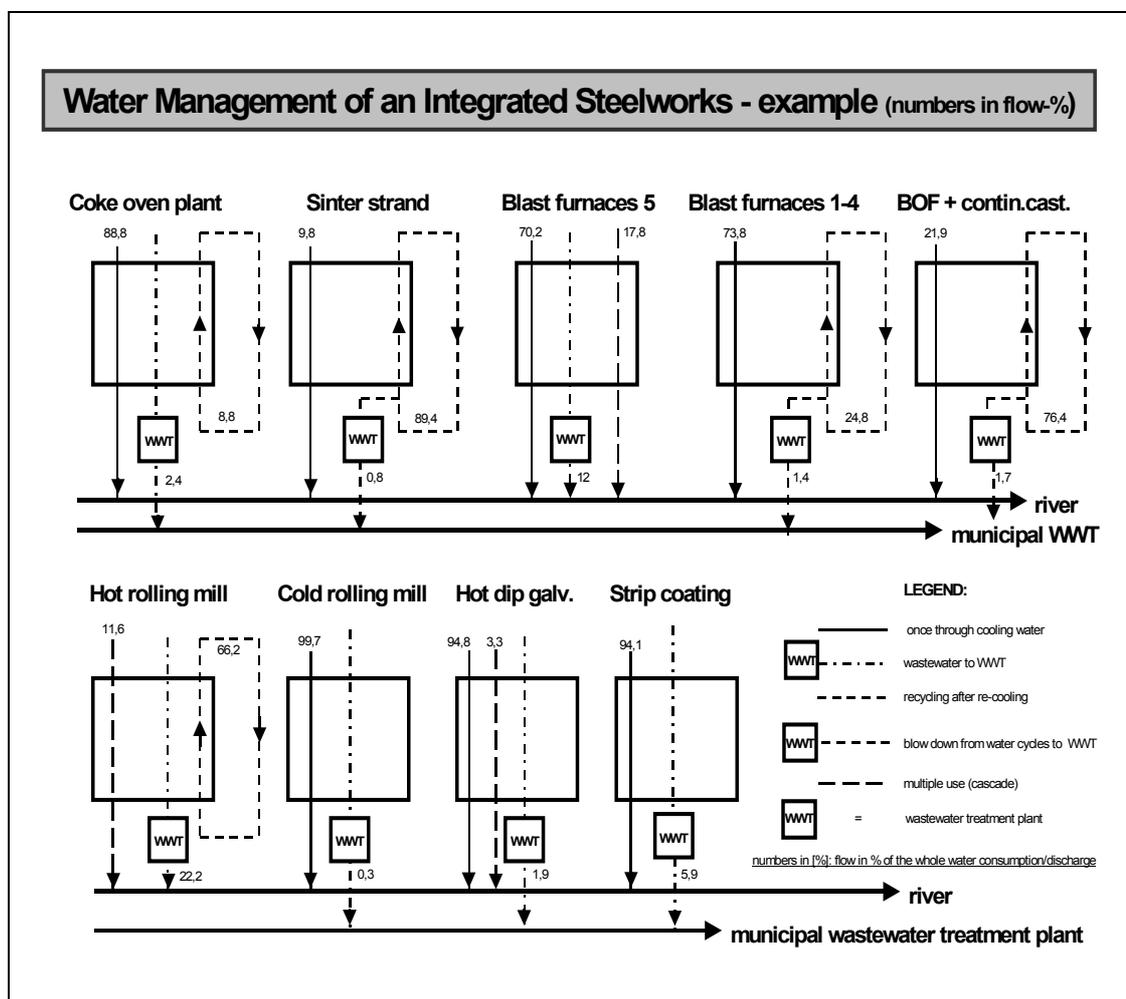


Figure 3.7 : Example for the water management of an integrated steelworks at a location with high surplus of water availability

4 SINTER PLANTS

4.1 Applied processes and techniques

4.1.1 Purpose of the sinter process

Modern high-performance blast furnaces achieve improved performance by prior physical and metallurgical preparation of the burden which improves permeability and reducibility. This preparation entails agglomerating the furnace charge either by sintering or pelletisation (see chapter 5). The charge consists of a mixture of fine ores, additives, iron-bearing recycled materials from downstream operations such as coarse dust and sludge from blast-furnace gas (BFgas) cleaning, mill scale, casting scale, etc. to which coke breeze is added enabling the ignition of it. In Europe, down draft sintering on continuous travelling grates is exclusively used (Figure 4.1 and Figure 4.2).



Figure 4.1 : Photograph of a sinter strand with the charging facility (drums or chutes) and the ignition canopy at the starting end

4.1.2 Blending and mixing of raw materials

Raw materials require blending prior to the sintering operation. This generally involves layering the materials on prepared areas in the precise quantities required by the sintering operation. Some flux material may also be added at this stage as can recycled materials from the downstream operations mentioned above. The ore beds are usually finished by covering with a layer of coarse material to prevent wind whipping. At the start of the sintering operation the ore blend is transferred from the beds to storage bunkers at the start of the sinter plant.

Other additives, such as lime, olivine, collected dust and mill scale, dusts (and to a much lower extent sludges) from gas cleaning in blast furnaces and recycled sinter (particles in the range of <math><5\text{ mm}</math>) from sinter screening (Figure 4.2), may be added to the ore blend at the mixing stage.

Coke breeze (small-grade coke with particle sizes of <math><5\text{ mm}</math>) is the most commonly used fuel for the sintering process. It is usually produced directly by an on-site coke oven plant and stored in hoppers for later use. Alternatively breeze may be obtained by crushing coke. In some cases anthracite is used as a fuel. Integrated steelworks with insufficient coke capacity to meet the needs of their sinter plant rely on external coke breeze suppliers.

The ore blend and the coke breeze are weighed on conveyer belts and loaded into a mixing drum. Here, they are blended completely and the mixture is dampened to enhance the formation of micro pellets, which improve the permeability of the sinter bed (Figure 4.2).

Where bunker blending and mixing are employed the emissions are abated by evacuation of particulate matter and subsequent purification of the collected gas.

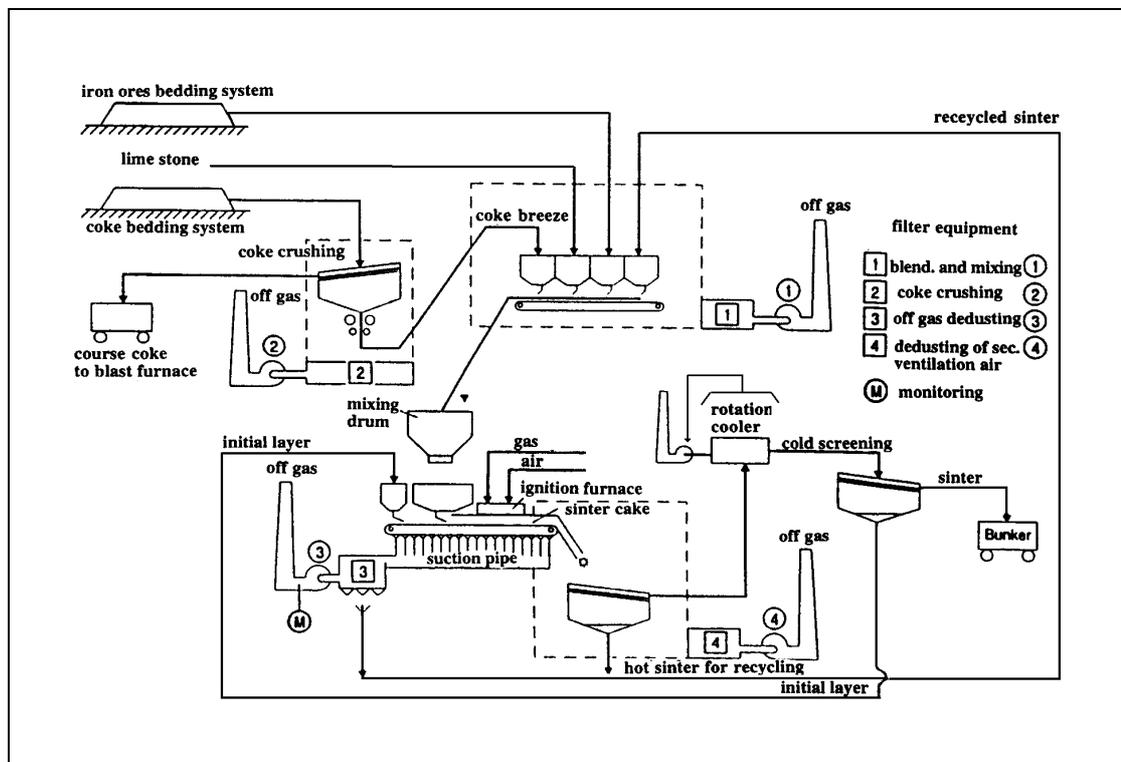


Figure 4.2 : Schematic diagram of a sinter plant showing the main emission points - [Theobald 1, 1995];

Other designs are also used; the figure is for plants using coke crushing and bunker blending; the stacks in the figure indicate emission sources; in reality there would be fewer stacks as more than one off gas stream may be combined in each stack

4.1.3 Sinter strand operation

The sinter plant essentially consists of a large travelling grate of heat resistant cast iron (Figure 4.1). The material to be sintered is placed on top of a 30-50 mm deep layer of recycled sinter. This bottom layer prevents the mixture from passing through the slots of the grate and protects the grate from direct heat of the burning mixture.

In modern sinter plants, the layer of materials to be sintered is approximately 400-600 mm deep but shallower beds are common in older plants. At the start of the grate a canopy of gas burners ignites the coke breeze in the mixture. In the down-draft process a powerful fan draws process air through the entire length of the sinter bed into distribution chambers located underneath the grate known as wind-boxes. The waste gas flow from a sinter plant varies from 350000 to 1600000 Nm³/hour, depending on the plant size and operating conditions [EC Sinter/BF, 1995]. Typically the specific waste gas flow is between 1500 and 2500 Nm³/t graded sinter [UBA Comments, 1997]. Most sinter plants with large suction areas (more than 250 m² and/or grate width of more than 3 m) have two off-gas collecting mains with separate fans and dedusting devices, which could be suitable for advanced emission reduction measures [Bothe, 1993].

As the sinter mixture proceeds along the grate, the combustion front is drawn downwards through the mixture. This creates sufficient heat (1300-1480°C) to sinter the fine particles together into porous clinker referred to as sinter.

A number of chemical and metallurgical reactions take place during the sintering process. These produce both the sinter itself, and also dust and gaseous emissions. The reactions overlap and influence each other, occurring as solid-state and heterogeneous reactions between the melt, solids and gaseous phases which are present in the sintering zone. The following processes and reactions take place in medium basicity sinter mixtures (1.5 - 1.7 which is very common, see Figure 4.9) [Matzke, 1987]:

- evaporation of moisture
- pre-warming and calcination of basic compounds, ignition of the coke breeze and reactions between carbon, pyrite, chloride- and fluoride-compounds and airborne oxygen
- decomposition of hydrates and cleavage of carbonates
- reaction between calcium oxide and hematite
- reaction between the silicate phase and calcium oxide and iron oxide phases to produce a silicate melt and increase the proportion of the molten phases
- formation of calcium-sulphur compounds and fluorine containing compounds together with alkali chlorides and metal chlorides
- reduction of iron oxides to metallic iron in the high temperature zone
- cavity and channel forming effects by coke combustion and moisture evaporation
- re-oxidation and re-crystallisation processes with shrinking, gearing and hardening effects during sinter cooling
- formation of cracks due to thermal strain during sinter cooling and defects in the sinter microstructure.

Figure 4.3 shows the temperature and reaction zones of a sinter layer 6 minutes after ignition. Emission profiles of various parameters (H₂O, O₂) (see Figure 4.5) and pollutants (CO₂, CO, SO₂, NO_x, alkali chlorides, heavy metal chlorides and PCDD/F) could be discussed (see Figure 4.5, Figure 4.8 and 4.2.2.1.2).

The number and variety of pollutants are present in the off-gas from the sinter strand, as is the case for most of the combustion processes. The gas contains particulate matter (heavy metals, mainly iron compounds but also other ones especially lead compounds), alkalichlorides, sulphur oxides, nitrogen oxides, hydrogen chloride, hydrogen fluoride, hydrocarbons, carbon monoxide and also significant trace amounts of PAH and aromatic organo-halogen compounds such as PCDD/F and PCB. As described in Figure 1.8 the gaseous emissions from the sinter plant dominate overall emissions (mass per time unit) from an integrated steelworks. Detailed information about formation pathways, emission quantities and sinter strand profile for individual pollutants are given in section 4.2.

The coke breeze is fully combusted before reaching the end of the grate and the last one or two wind-boxes are used to begin the cooling process. The cooler can be integrated into the sinter strand but it is the most common for it to be separate. As it is produced, sinter falls off the end

of the grate in the form of a cake, which is broken up on a crash deck and by a crusher. In many plants the sinter then goes through a hot screening process, in which fines measuring less than around 5 mm are separated and recycled to the feed mixture (Figure 4.2).

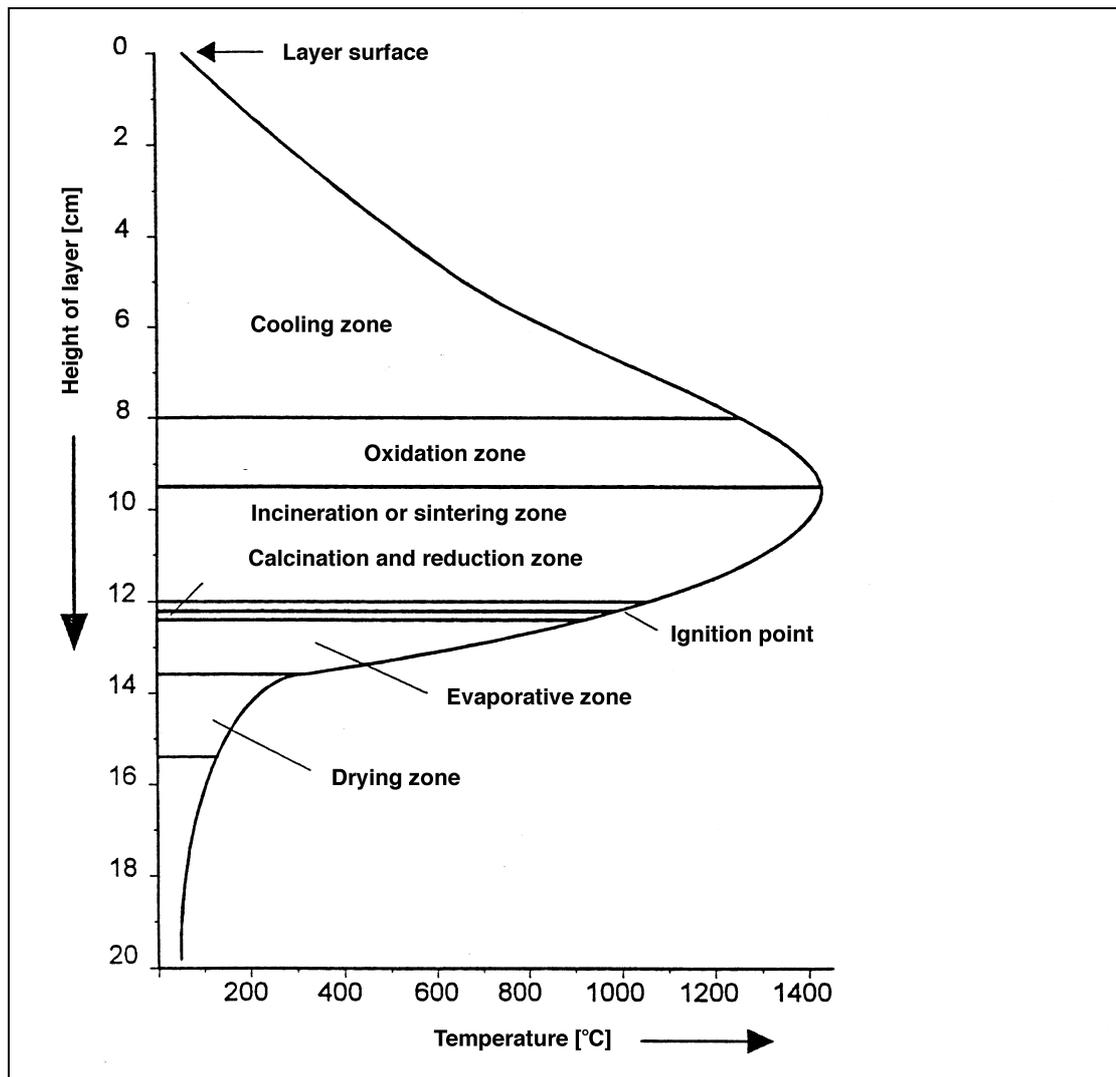


Figure 4.3 : Schematic diagram of temperature and reaction zones in a sintering process – based on [Dietrich, 1961]

4.1.4 Hot sinter screening and cooling

When the cooling is not integrated into the strand, the sintered material proceeds to a cooler after coming off the strand. The cooler is typically a rotating structure some 20-30 m in diameter in which the sinter is placed in a layer more than 1 m thick. The sinter is cooled by air, which is forced upwards through the layer. Sinter cooling gas flow is high and depends on the kind and age of the system used. Specific flow is between 1500 and 2500 Nm³/tonne sinter, which leads to flow rates of between 100000 and 1000000 Nm³/hour [EC Sinter/BF, 1995]. Sometimes, the sensible heat in the sinter cooling waste gas is used in a waste-heat boiler, in the sinter grate ignition hoods or to preheat the green feed. Other designs of cooler are also known to exist.

Cooled sinter is transferred to screens that separate the pieces to be used in the blast furnace (4-10 mm and 20-50 mm) from the pieces to be returned to the sinter process (0-5 mm as "return fines", 10-20 mm as "hearth layer").

4.2 Present consumption/emission levels

4.2.1 Mass stream overview and input/output data

Figure 4.4 shows an overview of the input and output mass streams of a sinter plant. This overview may be used for the collection of data from sinter plants.

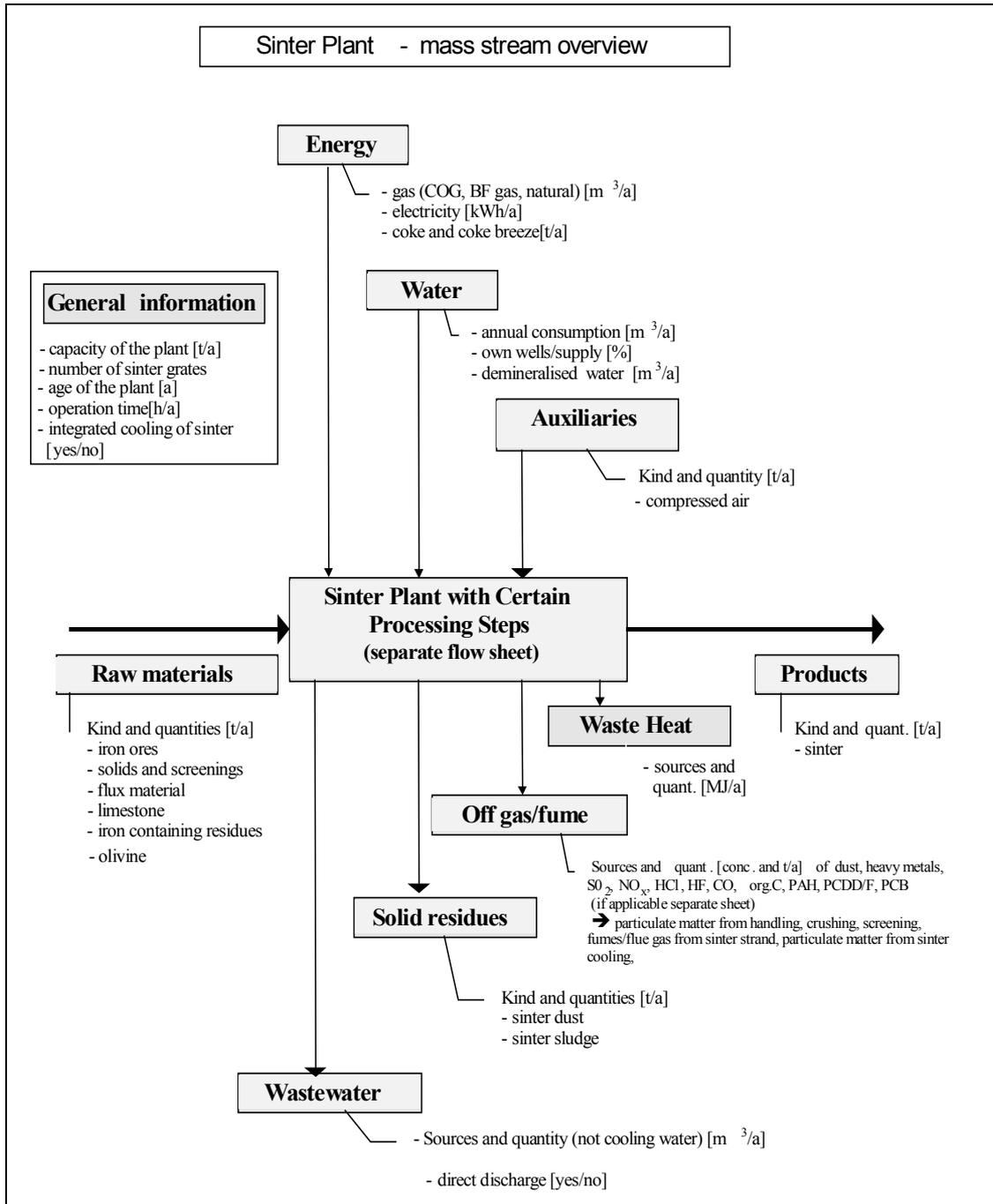


Figure 4.4 : Mass stream overview of a sinter plant

Using the mass stream overview both specific input factors and specific emission factors can be calculated. Table 4.1 shows this data for five sinter plants from five different EU Member States, thus providing a representative picture of modern plants.

The emission factors are stated per tonne of liquid steel in order to simplify addition of emissions from the various production steps.

Input			Output		
Raw materials			Product		
iron ore	kg/t sinter	680 - 850 ^{*1}	sinter	kg/t sinter	1000.0
other iron mat.	kg/t sinter	37 - 125	Gas. emissions		
lime	kg/t sinter	0.5 - 14	dust	g/t LS ^{*4}	170 - 280 ^{*5}
limestone	kg/t sinter	105 - 190	Cd	g/t LS	0.002 - 0.04 ^{*6}
additives	kg/t sinter	26 - 42	Cr	g/t LS	0.005 - 0.05 ^{*6}
BFG dust	kg/t sinter	11 - 27	Cu	g/t LS	0.007 - 0.16 ^{*6}
recycl. materials	kg/t sinter	42 - 113	Hg	mg/t LS	16-149 ^{*13}
intern. return sinter after screening	kg/t sinter	230 - 375	Mn	g/t LS	0.02 - 0.4 ^{*6}
			Ni	g/t LS	0.002 - 0.04 ^{*6}
			Pb	g/t LS	0.04 - 7 ^{*6}
			Tl	g/t LS	0.005 - 0.03 ^{*6}
Energy			V	g/t LS	0.005 - 0.02
COG/BF gas/nat. gas	MJ/t sinter	57 - 200 ^{*2}	Zn	g/t LS	0.002- 1.8 ^{*6}
coke	MJ/t sinter	1260 - 1380 ^{*3}	HCl	g/t LS	17 - 65
electricity	MJ/t sinter	96 - 114	HF	g/t LS	1.4 - 3.5
			NO _x	g/t LS	440 - 710
			SO ₂	g/t LS	900 - 1850
			CO	kg/t LS	13 - 43
Compressed air	Nm ³ /t sinter	1.2 - 3	CO ₂	kg/t LS	205 - 240 ^{*7}
			VOC ^{*8}	g/t LS	150 ^{*8}
			PAH ^{*9}	mg/t LS	115 - 915
			PCDD/F	µg I-TEQ/t LS	0.5 - 6.5
Water	m ³ /t sinter	0.01 - 0.35	PCB ^{*10}	mg/t LS	1 - 13
			Residues/ by-products		
			dusts ^{*11}	kg/t LS	0.9 - 15
			sludge ^{*12}	kg/t LS	0.3
Legend:			Wastewater ^{*12}	m ³ /t LS	0.06

^{*1} up to 1065 kg/t sinter in case of use of ore with lower iron content

^{*2} consumption depends on the efficiency of the ignition installation

^{*3} with 28650 kJ/kg coke breeze: 44 - 48 kg coke breeze/t sinter

^{*4} conversion factors used (weighted average of all European blast furnaces and basic oxygen steelworks):

1160 kg sinter/t pig iron; 940 kg pig iron/t LS

^{*5} values given for ESP; in case of cyclones: 560-740 g dust/t LS; in case of ESP + bag filter: 10-30 g dust/t LS (one plant in Europe) from the strand; in case of fine scrubber: <110 g dust/t LS (two plants in Europe)

^{*6} lower value in case of application of a quench and fine scrubber system after ESP or ESP with subsequent bag filter

^{*7} values up to 425 kg CO₂/t LS in case of usage of iron(II)carbonate containing iron ore decomposing to FeO and CO₂

^{*8} data only from one plant; VOC was measured continuously with a flame ionisation detector (11 measurement cycles)

^{*9} sum of EPA 16, calculated from Borneff 6 (EPA 16 = Borneff 6 x 4) with 2100 Nm³/t sinter

^{*10} sum of all PCB, calculated from (ΣPCB 28+52+101+153+138+180) x 5 (factor 5 according to [UN-ECE, 1997]) and with 2100 m³ off gas/t sinter; data available from only 2 plants.

^{*11} if part of the dust is put to landfill (dust from the last field of electrostatic precipitators)

^{*12} if a fine scrubber system is applied

^{*13} higher value when Hg is contained in iron ores in relevant amounts

Table 4.1 : Input/output-data from five sinter plants in four different EU Member States (Austria, Belgium, Germany and Netherlands)

the emission data represent the emissions to air after abatement from the whole sinter plant (off gas from the strand, from sinter cooling, from de-dusting of the buildings' atmosphere, etc)

information about the determination of the data like sampling methods, analysis methods, time intervals, computation methods and reference conditions not available;

Table 4.2 complements Table 4.1 by showing atmospheric emission factors (after abatement) for the main individual operations of a sinter plant. This table gives the SO₂, NO_x, CO and VOC content of the off-gas from plants distributed across the EU. It should be noted that large amounts of dust can be emitted from the discharge zone and from the cooling of sinter. This is due to missing or insufficient abatement techniques [EC Study, 1996].

	dust [g/t LS]	HF [g/t LS]	HCl [g/t LS]	SO ₂ [g/t LS]	NO _x [g/t LS]	CO [g/t LS]	CO ₂ [kg/t LS]	VOC [g C/t LS]	PAH [mg/t LS] ^{*2}	PCDD/F [µg/t LS] ^{*3}	PCB [mg/t LS]
Sinter plants											
- chrushing/blending	< 5 ^{*5}	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.
- sinter strand ^{*4} range(n°of data) x ± s	100 - 480 ^{*6} (23) 220±100	0.4-57(18)	23-95(9)	490-3000(22) 1100±600	75-1600(26) 550±310	7600-42500(24) 23500±7300	n/a	50-150 ^{*11} (7) 108±36	n/a	1 - 10 ^{*3}	n/a
- discharge zone ^{*4}	10 - 270 ^{*7}	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.
- sinter cooling ^{*4}	40 - 450 ^{*8}	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.
- building atmosphere	- ^{*9}	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.	n.r.
data from 5 sinter plants ^{*10}	see table 4.1 "Input/output of sinter plants"										

Legend: LS = liquid steel (crude steel); x±s = mean value and standard deviation (only calculated if enough data are available);
n.r. = not relevant; n/a = not available

^{*1} used conversion factors (weighted average of all European blast furnaces and basic oxygen steelworks): 1160 kg sinter/t pig iron; 940 kg pig iron/t LS; emissions factors for heavy metals are given in the table "Input/output of sinter plants"

^{*2} sum of EPA 16, calculated from Borneff 6 (EPA 16 = Borneff 6 x 4) with 2100 Nm³ off gas/t sinter

^{*3} in [µg I-TEQ/t LS], data from [LAI, 1995; Theobald 1, 1995; UN-ECE, 1997; LUA NRW, 1997]

^{*4} data from [EC Study, 1996]

^{*5} data from [EC Study, 1996], but without the 10% lowest and highest numbers (not reasonable) and without consideration of cyclones as abatement technique for which the emissions are significant higher (factor 2-4)

^{*6} calculated from [EC Sinter/BF, 1995] with 2100 m³ off gas/t sinter and verified with available data from sinter plants

^{*7} data only from one site; very often the off-gas is treated together with other streams like off-gas from dedusting of the building atmosphere and off-gas from sinter cooling

^{*8} calculated from [EC Sinter/BF, 1995] with 2000 m³ off gas /t sinter and verified with available data from sinter plants; the off-gas treatment by cyclones lead to emissions between 100 and 450 g/t LS, by bag filters significantly lower (40-110 g/t LS)

^{*9} the off-gas from dedusting of building atmosphere is very often treated together with the off-gases from other operations like sinter cooling and/or discharge zone/screening

^{*10} emission factors from 5 sinter plants in the EU 15 including all emission sources which are given in table 4.1 "Input/output of sinter plants"

^{*11} additional data from a German sinter plant (11 measurements) according to [Dropsch, 1997]

Table 4.2: Emission factors for emissions to air (after abatement) for individual sinter operations of sinter plants *1

4.2.2 Information on single emission mass streams

As already indicated the gaseous emissions from the sinter plant especially from the strand are of high environmental significance. When abatement techniques are applied to reduce emissions, cross-media effects can occur.

Detailed information is given below concerning both energy aspects and emissions to air, water, according to following structure:

4.2.2.1 Detailed information about emissions to air

- Particulate matter emissions from handling, crushing, screening and conveying of sinter feedstock and product;
- Waste gas emissions from the sinter strand;
- Particulate matter emissions from sinter cooling;

4.2.2.2 Information about emissions to water

- Waste water from waste gas treatment (optional);
- Cooling and rinsing water;

4.2.2.3 Information about solid waste

- Solid wastes from waste gas treatment (optional);

4.2.2.4 Information about energy aspects

- Energy demand.

4.2.2.1 Detailed information about emissions to air

4.2.2.1.1 Particulate matter emissions from handling, crushing, screening and conveying of sinter feedstock and product

When raw materials or sinter are handled, crushed, screened or conveyed, particulate matter emissions occur.

Particulate matter emissions from handling, crushing, screening and conveying can be reduced by means of proper suppression, or extraction and arrestment.

4.2.2.1.2 Waste gas emissions from the sinter strand

4.2.2.1.2.1 General

As mentioned in 4.1.3 the specific flow of waste gas is relatively high (1500 - 2500 m³/t sinter). Table 4.2 shows information concerning mixed waste gas from the strand as a whole. Although the composition of the waste gas extracted from individual wind-boxes varies considerably it shows certain characteristic features. Figure 4.5 shows the content of CO₂, CO, O₂ and H₂O in the waste gas along the sinter strand. These parameters also interfere with various reaction equilibria. Characteristic curves are also available for the temperature, and lead, SO₂, NO_x, PCDD/F content (see Figure 4.8 and 4.1.4) and also may exist for further pollutants. At present some parameters such as CO, CO₂, H₂O, O₂, NO_x and SO₂ can be calculated with appropriate models [Neuschütz, 1996].

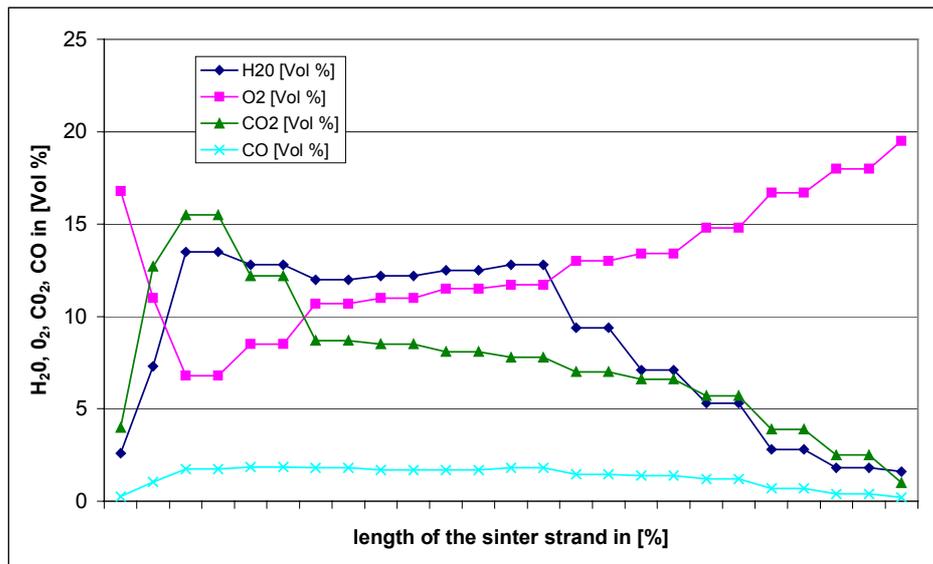


Figure 4.5 : Typical emission profile of CO_2 , CO , O_2 and H_2O in the waste gas (single windboxes) along the sinter strand - based on [Neuschütz, 1996]

4.2.2.1.2.2 Dust

Quantitative analysis of grain size distribution of the dust from a sinter strand before abatement shows two maxima; one for coarse dust (with a grain size about $100 \mu\text{m}$) and one for fine dust ($0.1\text{-}1 \mu\text{m}$) (Figure 4.6). This characteristic "two component dust mixture" can be explained by the existence of two dust forming processes.

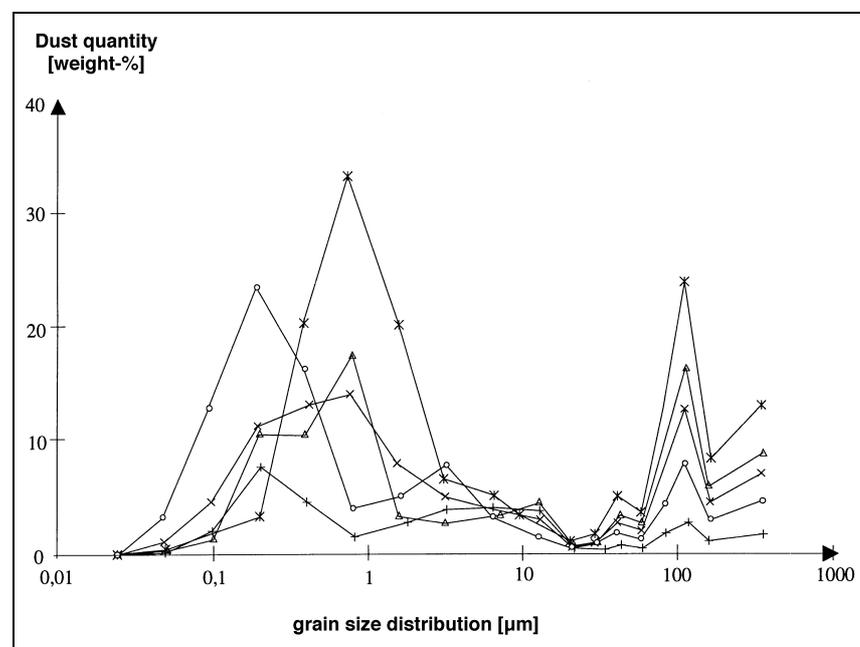


Figure 4.6 : Grain size and weight distribution of dust from various sinter strands - based on [Bothe, 1993]

The coarse dust results from the beginning of the strand and originates in the sinter feed and the lower layer, whereas the fine dust is formed in the sintering zone after complete water evaporation from the mixture. The composition of the coarse dust is related to that of the sinter feed and can be separated in electrostatic precipitators (ESPs) with a high efficiency. The fine dust, however, consists of alkali and lead chlorides formed during the sintering process itself (see 4.1.3). The alkali chlorides have high specific dust resistivity (between 10^{12} - 10^{13} Ω cm) (Figure 4.7) and thus form an insulating layer on the electrodes. This layer causes severe problems for dust removal as it reduces the efficiency of the precipitators [Bothe, 1993; Gebert, 1995]. According to [Bothe, 1993] alkali chlorides can only be removed with an average efficiency of about 60%. This fine dust means that even well designed and operated conventional electrostatic precipitators cannot normally achieve emitted dust concentrations of below 100 - 150 mg/m^3 .

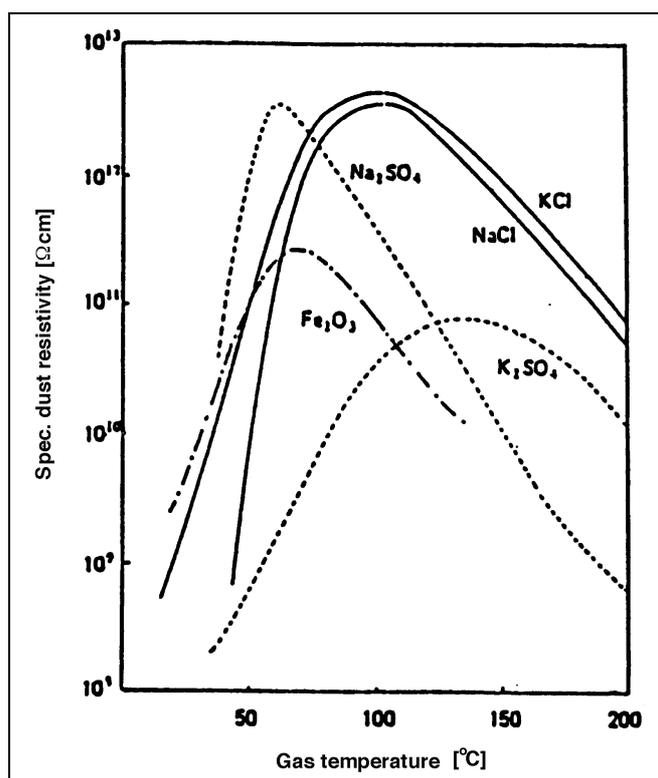


Figure 4.7: Specific dust resistivity of iron oxide and alkali chlorides and sulfates - [Reiche, 1990]

Lead chlorides behave similarly to alkali chlorides, hence the relatively high lead emissions from sinter strands (see Table 4.1).

The fine dust emitted contributes to ambient dust levels and having regard to Council Directive 96/62/EC [EC Air, 1996] under which limit values for particulate matter (PM 10) is envisaged, [EC LV, 1998], this may influence further discussions regarding dust emissions from sinter plants.

4.2.2.1.2.3 Heavy metals

Heavy metal emissions from sinter plants can be of high significance, especially for lead (see Figure 1.8).

Lead

During the sintering process lead reacts to PbO-PbCl_2 , PbCl_2 probably also to PbCl_4 . These lead compounds are relatively volatile and enter the gas phase [Matzke, 1987]. Their properties (volatility, no further increase of its grain size, the very fine PbCl_2 -crystals) give rise to relatively high concentrations in the raw waste gas (up to 70 mg Pb/Nm^3 , with $2100 \text{ Nm}^3/\text{t}$ sinter, 150 g Pb/t sinter). The removal efficiency of well-designed common ESP is not high for the very fine particles. Thus considerable emissions of lead in the g Pb/t sinter-range occur (see Table 4.1). From leaf deposits it is known that ambient air concentrations in the impact area of sinter plants can be significantly higher compared to the overall level, even if the emissions are minimized by application of high-efficient wet scrubbers after common ESP [UA-OÖ, 1998].

The most important factor in lead chloride formation is not the lead concentration in the sinter feed (normally $40 - 100 \text{ g Pb/t}$ sinter feed) but chloride concentration (normally $200 - 700 \text{ g Cl/t}$ sinter feed) [Matzke, 1987]. For the chloride content also ten times lower numbers are reported [BS PCDD/F, 1998] which may correlate with lower lead emissions.

The emission profile of lead compounds along the sinter strand shows that the volatilization of lead occurs in the first two thirds of the strand [Matzke, 1987].

Many of the sinter plants in Europe are operated using closed filter-dust cycles. This means that all precipitated filter dust from the ESP is recycled to the strand. Compared to partly open filter-dust cycles (the dust from the last field of the ESP, which mainly consists of alkali and metal chlorides, is put to landfill) closed cycle plants have higher alkali and metal chloride emissions [Matzke, 1987]. It has not been possible to obtain a precise quantification of this non-negligible cross-media effect.

Depending on the fluoride content of the sinter feed, lead fluorides may also be formed. These are even more volatile and also contribute to overall lead emissions.

Mercury

Mercury enters the gas phase directly during the sintering process. Emission levels depend on the mercury content of the sinter feed, but are normally very low. In case of iron ore which contains relevant amounts of mercury emissions are considerable. In such a case emissions of about $15-54 \text{ } \mu\text{g Hg/Nm}^3$ or $38-136 \text{ mg Hg/t}$ sinter are reported when well-designed and operated ESP plus fine wet scrubbing system are applied as abatement techniques [Linz, 1996]. The values have been determined in 1994. Such releases can lead to significant higher environmental concentrations (e.g. determined from deposits on tree leaves) in the impact area of a sinter plant [UA-OÖ, 1998]. However the concentration of mercury in ambient air as average values has been determined $0.18-0.19 \text{ ng Hg/m}^3$ which is far below prescribed standards.

Zinc

Zinc content in the feed is normally in the range $70 - 200 \text{ g Zn/t}$ sinter feed [Matzke, 1987]. At the high temperatures in the incineration/calcination/oxidation zone (see Figure 4.3) zinc evaporates but subsequently reacts to form zinc ferrite, which either remains in the sinter [Matzke, 1987] or may be removed using a well designed and operated ESP with high efficiency. The temperature during the sintering process, which is mainly influenced by the coke content of the mixture, may have a significant influence on zinc volatilization.

In the raw gas, zinc can be present at levels of up to 50 mg Zn/Nm^3 or 100 g Zn/t sinter [Bothe, 1993]. Treated waste gas can contain up to 1.65 g Zn/t sinter (see Table 4.1) which could be environmentally significant.

4.2.2.1.2.4 Alkali chlorides

The alkali content of the sinter feed normally varies from 600 - 1000 g K_2O /t sinter feed and 250 - 500 g Na_2O /t sinter feed [Matzke, 1987]. In some cases, depending on the ore quality, up to 3000 g K_2O /t sinter feed have been reported. The formation of alkali chlorides during the sintering process and their adverse impact on ESP removal efficiency has already been explained above (see "dust").

4.2.2.1.2.5 Sulphur oxides (SOx)

Sulphur oxides (mainly SO_2) in the waste gas originate from the combustion of sulphur compounds in the sinter feed. These sulphur compounds are primarily introduced through the coke breeze. The contribution from iron ore is normally about ten times smaller. Total sulphur input varies between 0.28-0.81 kg/t sinter [InfoMil, 1997]. The emission concentrations for SO_2 normally are 400 - 1000 mg SO_2 /Nm³ or 800 - 2000 g SO_2 /t sinter (Table 4.1) [Bothe, 1993]. These values represent average waste gas concentrations from the sinter strand as a whole. However, the emission profile along the sinter strand clearly varies, with considerable differences in SO_2 concentrations in the individual wind-boxes (Figure 4.8).

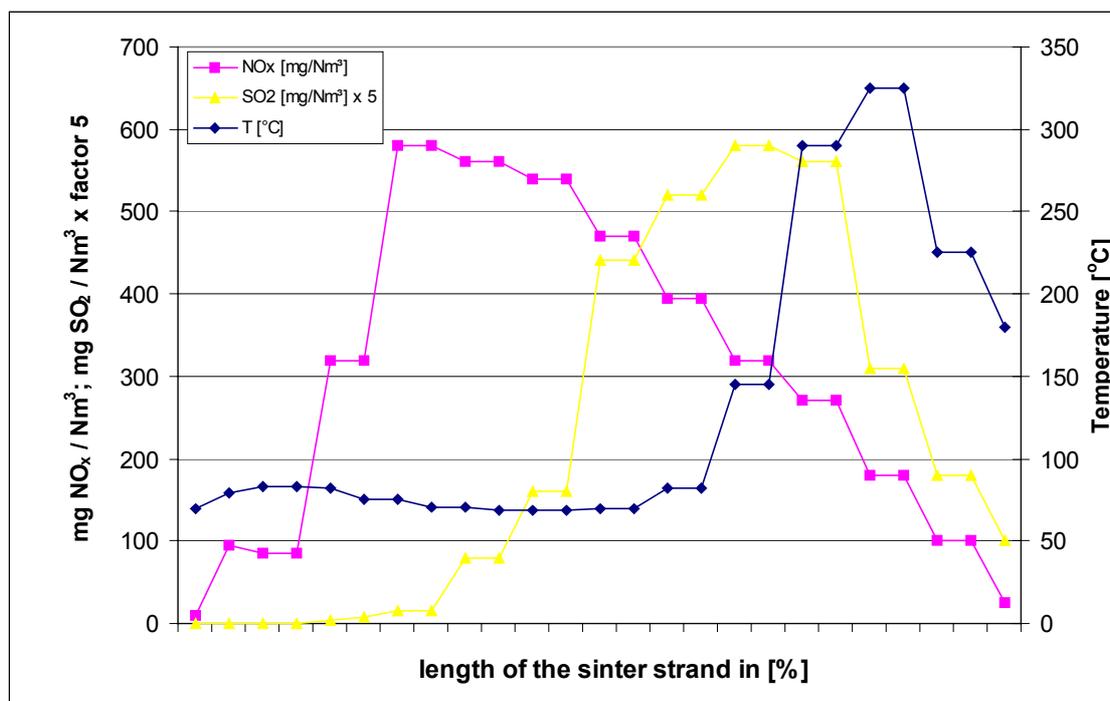


Figure 4.8: Typical emission profile of SO_2 and NO_x in the waste gas (individual wind-boxes) and the temperature curve along the sinter strand - based on [Neuschütz, 1996]

At the start of the strand the SO_2 content is low. Only when higher temperatures in the lower layers of the sinter bed are reached do emissions increase markedly. The highest concentrations occur at the end of the strand before sinter cooling starts. This emission profile may offer the possibility, if required, of treating only part of the waste gas.

The above mentioned SO_2 emissions are achieved by using raw materials with low sulphur content and by minimizing the fuel consumption. Whereas in the early eighties a coke breeze input of about 80 kg/t sinter was common, current consumption is 38 - 55 kg/t sinter [Lüngen, 1991]. Apart from the sulphur input with the raw materials, there are other factors which influence the quantity of SO_2 emitted. One of these is the degree to which sulphur is included in the sinter. This depends on the basicity of the sinter feed. At basicities up to 1, more than 90% of the sulphur volatilizes. This percentage decreases at basicities above 1.5 [Bothe, 1993]. At a basicity of 2, 80 - 90 % of the sulphur is released. In Germany the basicity of sinter has been increased to an average of 1.7 since the seventies in order to achieve a self fluxing burden and this is representative for Europe as a whole (Figure 4.9). This has contributed to decreasing SO_2 emissions, but only slightly. Moreover, the uptake of sulphur by high-basicity sinter is only possible when CaCO_3 is used as the alkali and not with MgCO_3 [Bothe, 1993].

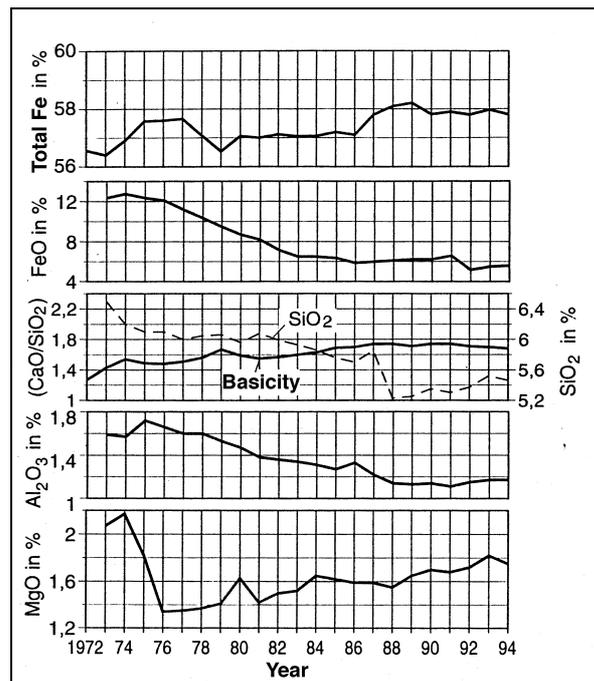


Figure 4.9 : Average composition of sinter in Germany - [Stahl, 1995]

The slight positive effect of CaCO_3 on SO_2 emissions is counteracted by its adverse effect resulting from the increased specific dust resistivity caused by higher basicity, leading to decreased dust-removal efficiency in the ESP (Figure 4.10).

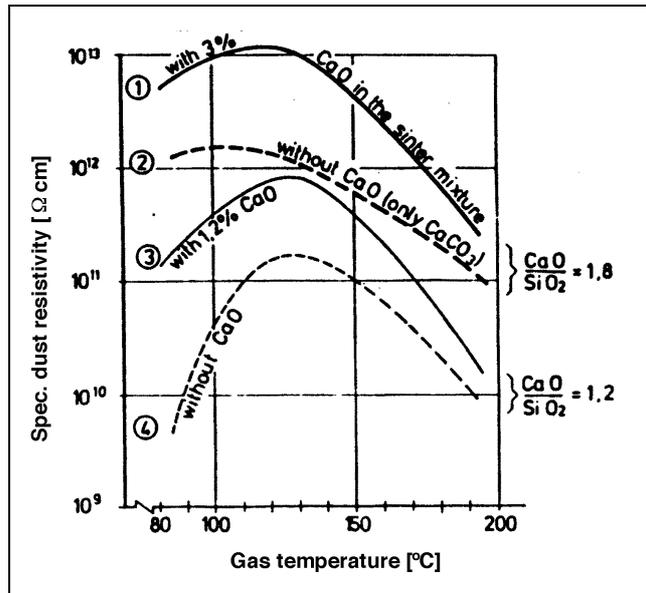


Figure 4.10 : Influence of sinter basicity (CaO/SiO_2) on spec. dust resistivity - [Bothe, 1993]

The grain size distribution of the coke breeze has a significant influence on SO_2 emissions. The use of coarser coke breeze (5mm – 6mm) instead of normal grain sizes (< 3mm) can reduce SO_2 emissions considerably. An example has been reported in which SO_2 emissions of about 800 mg SO_2/Nm^3 using fine grain breeze (1 mm) were reduced to 500 mg SO_2/Nm^3 after switching to coarser coke breeze (6 mm) [Beer, 1991].

4.2.2.1.2.6 Fluorides

Fluorides emissions primarily depend on the fluoride content of the ore and on the basicity of the sinter feed. Ores rich in phosphorous contains significant amounts of fluoride (1900 - 2400 ppm). In Europe such types of ore are especially explored and exported from Sweden (Kiruna D, Graengesberg) [Bothe, 1993]. The fluoride emissions strongly depend on the basicity of the sinter feed (Figure 4.11). The increase in basicity of sinter feed (see Figure 4.9) over the past two decades has lead to significantly lower fluoride emissions. As mentioned above, a disadvantage of sinter feed with higher basicity is the generation of dust with higher specific dust resistivity (see Figure 4.10). According to Table 4.1 the fluoride emissions are 1.3 - 3.2 g F/t sinter or (with 2100 Nm^3/t sinter) 0.6 - 1.5 mg F/ Nm^3 .

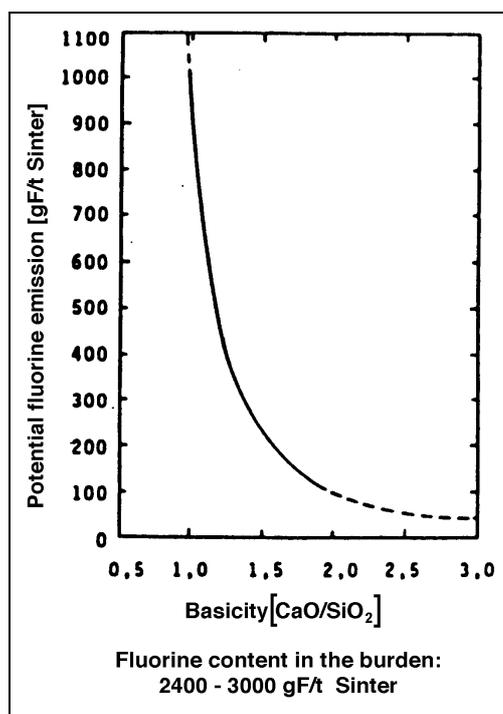


Figure 4.11: Relationship between sinter-feed basicity fluoride emissions - [Bothe, 1993]

4.2.2.1.2.7 Nitrogen oxides (NO_x)

The temperatures at the flame front in the sinter bed inherently cause NO_x formation. This NO_x can be formed in three ways: combustion of organic nitrogen compounds in the sinter feed ("fuel-NO_x"); the reaction of decomposing components with molecular nitrogen (N₂) in the combustion zone ("prompt-NO_x"); and the reaction of molecular oxygen (O₂) with molecular nitrogen (N₂) in the combustion air ("thermal-NO_x"). Fuel NO_x can be most important representing about 80% of the total, but also thermal NO_x can be dominating by 60-70% [InfoMil, 1997]. In Table 4.1 emission factors of 400 - 650 g NO_x/t sinter are mentioned, which means concentrations (with 2100 Nm³/t sinter) of 200 - 310 mg NO_x /Nm³. Emission concentrations of up to 700 mg NO_x /Nm³ are reported, which relate primarily to nitrogen content in the fuels [Bothe, 1993].

When the NO_x concentrations are determined at individual wind-boxes, it can be seen that the NO_x emissions are not evenly distributed along the sinter strand. However, compared to SO₂ the differences are not that significant (Figure 4.8).

4.2.2.1.2.8 Hydrocarbons

Hydrocarbon emissions mainly consist of products formed from pyrolysis and incomplete combustion of carbon-bearing raw materials. Recycled residues, such as mill scale (containing up to 10% oil [Gebert, 1995] from rolling processes are the major sources of hydrocarbon emissions from sinter strands [Gebert, 1995; Pütz, 1996]. Most integrated steelworks have introduced internal standards for the oil content of residues recycled to the sinter strand (some < 0.1%, some < 0.5% and some < 1%) in order to avoid operational problems with the ESP (scaling, glow fire) and also to minimise emissions.

Hydrocarbon emissions are a result of evaporation before the oxidation/incineration zone (see Figure 4.3) reaches the lower sinter bed layers. These emissions include methane, aliphatic compounds, phenols, olefins and aromatics. (Gebert, 1995; Bothe, 1993). Reported emissions concentrations, as determined with flame ionisation detector, are 49 - 109 mg C/Nm³ (11 measurements, $x \pm s$: 66 \pm 18 mg C/Nm³) [Dropsch, 1997] and 20 - 90 mg C/Nm³ (32 measurements, $x \pm s$: 51 \pm 21 mg C/Nm³) [BS PCDD/F, 1998]. These emissions directly depend on the input.

4.2.2.1.2.9 Polychlorinated dibenzo-p-dioxins and -furans (PCDD/F)

It has only recently become clear that sinter plants can be a significant source of PCDD/F emissions [Broeker, 1993; Lahl, 1994]. In one case, emission concentrations of up to 43 ng I-TEQ/Nm³ (replicate analysis) have been determined [Broeker, 1993]. Due to process optimisation, especially by input control, actual emission concentrations normally are in the range of 0.5 - 5 ng I-TEQ/Nm³ which is (with 2100 Nm³/ t sinter) 1 - 10 μ g I-TEQ/t sinter [LAI, 1995; Theobald 1, 1995; UN-ECE, 1997; LUA NRW, 1997]. These emission concentrations/factors refer to emissions after abatement, which is normally carried out using an electrostatic precipitator with three to four fields.

There is no exhaustive explanation for the emissions of PCDD/F (quantities and distribution of homologues and congeners, formation mechanisms, etc.). The following observations are considered important:

- with respect to PCDD/F, the electrostatic precipitators in primary dedusting of sinter plants achieve only low separation rates. This is documented by the fact that there is no detectable change in the PCDD/F profiles;
- the distribution of PCDD/F homologues and congeners (PCDD/F profile) is similar to so-called "thermal PCDD/F profile" [Pütz, 1996] with a higher percentage of tetra/penta-CDF and, when compared to PCDF, indistinct PCDD profiles with more pronounced scattering of the homologues;
- the waste gas of the sinter plants studied to date exhibits only minor scattering of the mass concentrations of the PCDD/F homologues groups;
- at 40 to 60%, 2,3,4,7,8-penta-CDF represents by far the largest proportion of toxicity equivalents (Figure 4.12).

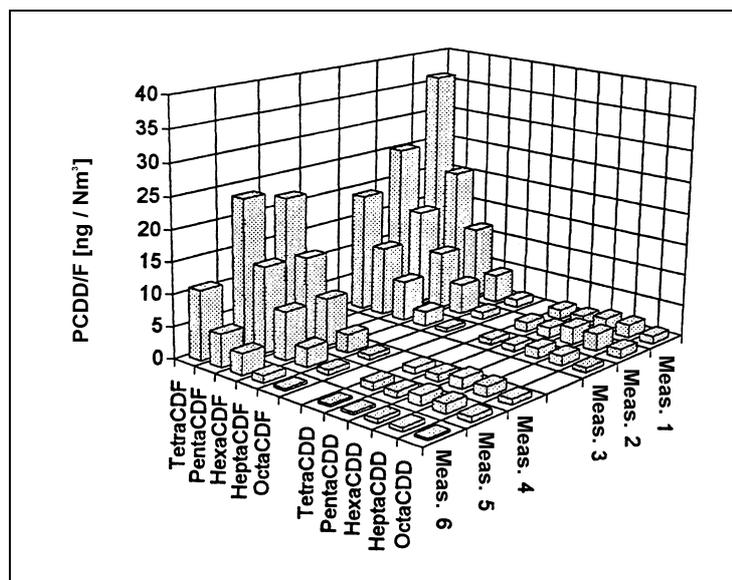


Figure 4.12 : Typical homologue groups profile of the raw waste gas of a sinter plant (before abatement) for 6 measurements - [Pütz, 1996]

Recent investigations have shown that for de novo synthesis there are two completely different precursors and reaction pathways leading to PCDD and PCDF. De novo synthesis of PCDD from carbon proceeds, at least partly, via the condensation of intermediate phenyl rings, e.g. phenols. In contrast, reactions involving single phenyl compounds play no role in the de novo synthesis of PCDF from carbon. In the case of PCDF the precursors are preformed biphenyl structures [Stieglitz, 1997].

PCDF dominate in the waste gas from sinter plants, suggesting that precursors with a phenyl ring structure are less important than the formation of PCDF from solid carbon structures and inorganic chloride. Carbon and chloride are present in the sinter feed so cannot be limiting factors for PCDD/F formation. This consideration does not directly support the reported positive correlation between volatile hydrocarbons in the sinter feed and the PCDD/F concentration in the waste gas [Gebert, 1995; Pütz, 1996]. But other sources indicate no obvious correlation between PCDD/F emissions and (volatile) hydrocarbon content in the sinter feed or in the waste gas (Figure 4.13).

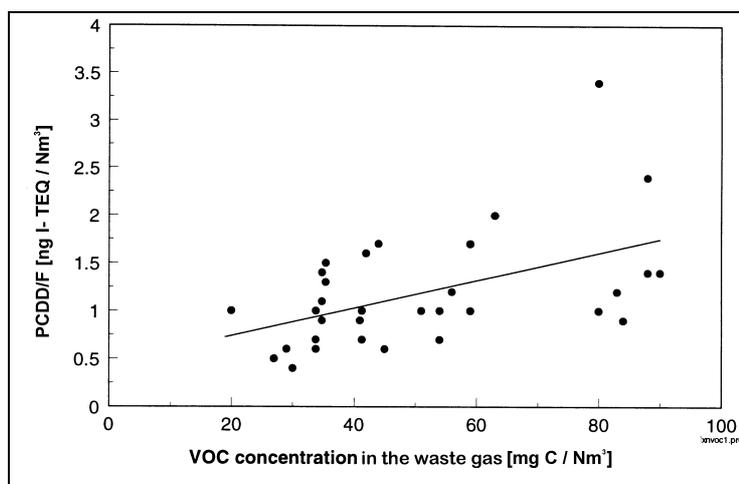


Figure 4.13 : No obvious correlation between PCDD/F concentration and VOC concentration (FID measurement) in the waste gas of sinter plants (correlation coefficient $r = 0.25$) - [BS PCDD/F, 1998]

As reported for other parameters (see Figure 4.5 and Figure 4.8) there is also an emission profile along the sinter strand for PCDD/F (Figure 4.14). After full evaporation of the moisture the temperature increases (see Figure 4.5) and seems then to correlate with PCDD/F emissions (Figure 4.14). One possible explanation of these observations is that PCDD/F are formed in upper regions of the sinter bed, and then condense on cooler burden beneath and subsequently undergo further reactions and re-volatilization as the combustion zone (see Figure 4.3) travels down through the bed.

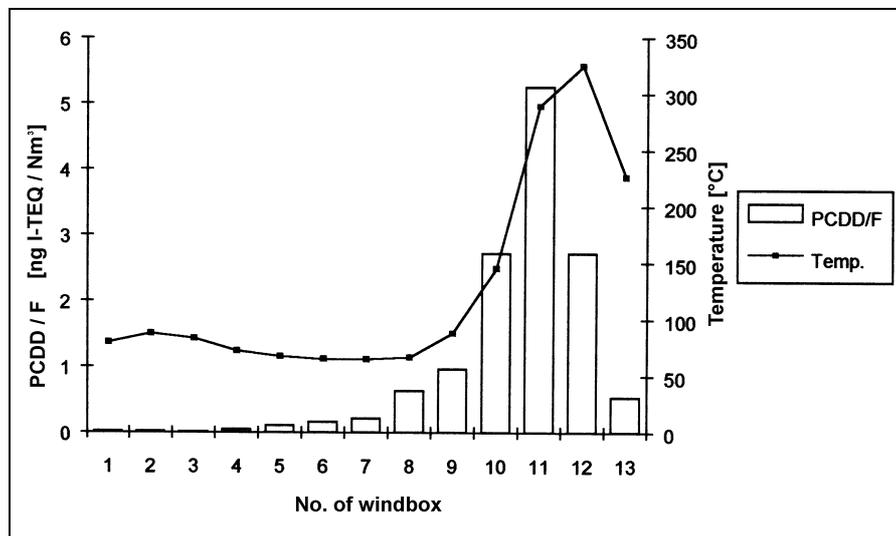


Figure 4.14 : PCDD/F and temperature profile in the waste gas along the sinter strand - [Pütz, 1996]

Another factor may be the Deacon equilibrium ($2 \text{HCl} + \frac{1}{2} \text{O}_2 \rightleftharpoons \text{H}_2\text{O} + \text{Cl}_2$), which shifts to the chlorine side with decreasing moisture content. The influence of this interaction remains unclear.

The distinct PCDD/F profile offers the possibility of treating the more concentrated parts of the waste gas.

4.2.2.1.2.10 Polychlorinated Biphenyls (PCB)

Although it was for a long time maintained that there is no de novo synthesis in thermal processes, it has been shown that in fact there is [Blaha 1995; Hagenmaier, 1996; Scholz 1997]. PCB can be formed under the same conditions as those described above for PCDD/F. However, during the formation pathway no scrambling of the phenyl rings takes place. In a further reaction the PCB produced may undergo a ring closure to yield PCDF, but not PCDD [Scholz, 1997]. Thus, as is the case for PCDF, no aromatic precursors are necessary for it to be formed.

PCB may also be present in the raw materials. PCB concentrations in coke breeze and iron ores are reported to be around 1 - 1.6 mg/t and limited calculations for one sinter plant indicated a potential input of 850 µg/t graded sinter [BS PCDD/F, 1998]. Clearly there is a possibility that PCB could be destroyed in the combustion zone (see Figure 4.3) but it must be remembered that the combustion process is not as intense as in waste incinerators, for example, and it is thought unlikely that all the PCB in the process air are destroyed in the combustion zone.

Moreover, PCB are relatively volatile and may be driven off as the burden is heated by the gaseous combustion products ahead of the combustion zone [BS PCDD/F, 1998].

Data concerning PCB emissions from sinter plants are only available from 2 plants (see Table 4.1). For these the emission factors are 1 - 12 mg Σ PCB/t sinter. This concentration level is relatively high and may be of environmental relevance.

4.2.2.1.2.11 Further organohalogen compounds

The presence of PCDD/F and PCB can be considered as indicators for the formation of organohalogen compounds such as chlorobenzenes, chlorophenols, chloronaphthalenes etc. [Stieglitz, 1997]. Thus these compounds have to be expected in the waste gas of sinter plants.

4.2.2.1.2.12 Polycyclic aromatic hydrocarbons (PAH)

As mentioned in 4.1.3 the reactions in the sinter bed are complex. The combustion process is not homogenous and is incomplete, resulting in significant quantities of polycyclic aromatic hydrocarbons (PAHs) being produced. As for PCB, the data about PAH emissions are available only for a few sinter plants (see Table 4.1). Emissions factors of 105 - 840 mg Σ EPA16/t sinter indicate the significance of PAH emissions.

4.2.2.1.3 Particulate matter emissions from sinter cooling

After falling off at the end of the sinter grate, product sinter is crushed, screened and cooled. In these operations there is the potential for emitting dust, the composition of which is basically that of the sinter itself. When the air used for cooling is returned to the process in order to use the sensible heat, the dust from the sinter screening and cooling is also returned to the process.

When the air is not recycled to the process particulate matter emissions are usually abated by means of an electrostatic precipitator. Often the waste gas from the cooler is treated together with waste gas extracted from the discharge zone. The precipitated dust can be recycled to the sinter process.

The dust abatement of the waste gas from sinter cooling and discharge zone is often missing although the dust emissions are considerable high (40 - 450 g/t sinter) [EC Study, 1996] which is at least in the same range as dust emissions from the sinter strand (see Table 4.2). However the composition of the dust is different from dust emitted from the sinter strand (especially no/very low emissions of alkali chlorides and organic micropollutants).

4.2.2.2 Information about emissions to water

4.2.2.2.1.1 Rinsing water

The type of processes in an iron and steel plant inherently cause dust deposition on the plant premises. In order to prevent run-off to surface water, this should be removed, preferably by dry techniques. A few plants, however, do use rinsing water cleaning techniques. The resulting waste water contains suspended solids (including heavy metals) and is usually treated before discharge. For example, in a sinter plant with a production of approximately 11000 tonnes of

sinter per day, the rinsing water flow is about 460 m³/day. This waste water is treated by means of sedimentation in the recirculation circuit and enhanced settling prior to discharge [InfoMil, 1997]. No data from other plants are available.

4.2.2.2.1.2 Cooling water

In the sinter plant, cooling water can be used for the cooling of the ignition hoods and the fans as well as for the sinter machines. In an integrated steelworks producing 4 Mt steel per year, the sinter plant cooling would require a water flow of approximately 600 m³ per hour [EC Haskoning, 1993]. The cooling water is normally completely recycled.

4.2.2.2.1.3 Waste water from waste gas treatment

Waste water from waste gas treatment will only be generated if a wet abatement system is applied. The water flow contains suspended solids (including heavy metals), organohalogen compounds such as PCDD/F and PCB, PAH, sulphur compounds, fluorides and chlorides. It is usually treated before discharge. The quantity and quality of such wastewater is described in more detail under “cross-media effects” in the description of the concerned technique (see EP.4).

4.2.2.3 Information about solid waste

Normally all solid wastes originating from the sinter plant (dusts from dedusting devices and sieving fractions) are recycled to the strand. However, there can be two exceptions: The first concerns sludge from wet waste gas treatment systems, which is put to landfill. At present, there are only two plants in Europe operating such a system. The second exception is filter dust from the last field of electrostatic precipitators (ESP). As mentioned in 4.2.2.1.2.3, most European sinter plants are operated with fully closed dust cycles [Bothe, 1993]. However, some plants exclude fine dust from the last field of the ESP. This dust mainly consists of alkali and metal chlorides. This so-called partly open filter dust cycle is carried out in order to improve the operation of ESP or (in one case) of the bag filter, or to reduce alkali and metal chloride emissions.

4.2.2.4 Information about energy aspects

The European Blast Furnace Committee survey of the operational data for sinter plants (1996) reveals that sinter plants use 1125 – 1920 MJ/t sinter of thermal energy (solid fuels including flue dust and ignition fuel), with an average consumption of 1480 MJ/t sinter. These are 39.5 - 67 kg coke breeze equivalents/t sinter, with an average of 52 kg coke breeze equivalents/t sinter. Total electrical consumption is in the ranges 68 - 176 MJ/t sinter, with an average of 105 MJ/t sinter. There is only a slight difference in fuel consumption between low basicity sinter (< 1.7 CaO/SiO₂) and higher basicity sinter (≥ 1.7 CaO/SiO₂).

Table 4.1 shows data from five sinter plants to which these figures correspond. Coke is the dominant sinter plant energy input (about 85%), with electricity and gas (COG and/or BFGas and/or natural gas) supplying the remainder in equal shares. The main energy outputs are via waste gas, water evaporation, the reaction energy required and the sinter itself. Sinter cooling is often combined with the recovery of sensible heat.

4.2.2.5 Information about noise emissions

The following noise sources are dominating in sintering plants:

- Sinter waste gas fans
- Sinter cooling fans

4.3 Techniques to consider in the determination of BAT

This section looks at both process-integrated and end-of-pipe techniques for environmental protection and energy saving at sinter plants. A description of each technique, the main emission levels achieved, its applicability, emission monitoring, cross-media effects, reference plants, operational data, economics and driving force for implementation are given in those cases where relevant information is available and pertinent.

4.3.1 Process-integrated techniques

The following process-integrated techniques are known to be use at sinter plants:

- PI.1 Process optimisation for minimisation of PCDD/F emissions
- PI.2 Recycling of iron-containing waste into the sinter plant
- PI.3 Lowering the content of volatile hydrocarbons in the sinter feed
- PI.4 Lowering the sulphur content of the sinter feed
- PI.5 Heat recovery from sintering and sinter cooling
- PI.6 Top-layer-sintering
- PI.7 Waste gas recirculation e.g. Emission Optimised Sintering (EOS)
- PI.8 Sectional waste gas recirculation

4.3.2 End-of-pipe techniques

The following end-of-pipe techniques are known to be in use at sinter plants:

- EP.1 Electrostatic precipitator (ESP)
- EP.2 Fabric filter system
- EP.3 Cyclone
- EP.4 Fine wet scrubber, e.g. Airfine
- EP.5 Desulphurisation
- EP.6 Regenerative activated carbon (RAC)
- EP.7 Selective catalytic reduction (SCR)

4.3.1 Process-integrated techniques

PI.1 Process optimisation for minimisation of PCDD/F emissions

Description: Extensive research into formation of polychlorinated dibenzo-p-dioxins and –furans (PCDD/F) in the sintering process [BS PCDD/F, 1998] has shown that PCDD/F are formed within the sinter bed itself, probably just ahead of the flame front as the hot gases are drawn through the bed (see 4.2.2.1.2.9). It has also been shown that disruptions to flame front propagation, i.e. non-steady state operations, result in higher PCDD/F emissions. The solution therefore has been to operate the sintering process in as consistent a manner as possible in terms of strand speed, bed composition (particularly consistent blending of revert materials minimisation of chloride input), bed height, the use of additions such as burnt lime and the control of millscale oil content to a consistent level of < 1% and keeping the strand, ductwork and ESP air tight to minimise, as far as possible, the amount of air ingress in the operation. This added advantages in terms of operational performance (productivity, sinter quality) improvements.

Main achieved emission levels: A mean of 1.0 ng I-TEQ/Nm³ has been achieved from a total of 41 samples at four sites in the UK. The typical range is 0.5-1.5 ng I-TEQ/Nm³, although the majority of samples lie close to 1 ng I-TEQ/Nm³. The samples were obtained using US EPA Method 23. PCDD/F analysis is carried out in accredited trace organic laboratories. Detailed results are shown in Table 4.3. However at other plants in other EU Member States performing the same or very similar operation conditions such low values can not be achieved. In Germany usually 2-3 ng I-TEQ/Nm³ are achieved. From one plant values between 5 to 6 ng I-TEQ/Nm³ are reported.

No specific measure could be identified that enables to achieve the relatively low PCDD/F emission levels, rather it seems to be a combination of a number of measures mentioned above.

Applicability: Can be applied to existing and new plants.

Cross media effects: Energy usage is minimised by consistency of operations. There are no negative cross media effects.

Reference plants: British Steel Scunthorpe, British Steel Teesside, British Steel Port Talbot, British Steel Llanwern (2 plants).

There are further sinter plants in the EU which are operated in a similar manner. Partly operators can confirm these low emission levels (0.5-1.5 ng I-TEQ/Nm³) but others achieve significantly higher concentrations up to 5-6 ng I-TEQ/Nm³.

Operational Data: Process optimisation can result in improved operational performance in terms of consistently high productivity and consistent sinter quality.

Economics: There are no installation cost and there are operational benefits from consistency of operations.

Reference literature: [BS PCDD/F, 1998]

British Steel Teesside (Redcar)		British Steel Scunthorpe		British Steel Port Talbot		British Steel Llanwern			
						B Strand		C Strand	
Sampling date	PCDD/F [ng I-TEQ/Nm ³]	Sampling date	PCDD/F [ng I-TEQ/Nm ³]	Sampling date	PCDD/F [ng I-TEQ/Nm ³]	Sampling date	PCDD/F [ng I-TEQ/Nm ³]	Sampling date	PCDD/F [ng I-TEQ/Nm ³]
08/03/95	1.0	20/02/95	0.6	24/02/95	1.6	09/04/97	1.6	11/04/95	1.0
09/03/95	1.7	20/02/95	0.7	24/02/95	0.9	10/04/97	1.3	11/04/95	0.4
26/04/95	0.7	20/02/95	1.0	24/02/95	0.6	11/04/97	1.1	12/04/95	0.6
26/04/95	0.9	06/07/95	1.1	19/04/95	1.0	11/04/97	1.0	12/04/95	0.5
27/04/95	0.9	06/07/95	1.4	19/04/95	0.7			09/06/95	1.4
27/04/95	1.2	06/07/95	1.1	20/04/95	1.0				
17/12/96	1.0	06/07/95	0.9	20/04/95	1.2				
17/12/96	1.0	17/05/96	1.5						
20/07/98	0.6	17/05/96	1.3						
21/07/98	0.6	18/05/96	1.3						
21/07/98	1.5	30/06/97	1.5						
		04/08/98	1.2						
		04/08/98	0.3						
		04/08/98	0.8						
Range: 0.6 – 1.7		Range: 0.3 – 1.5		Range: 0.6 – 1.6		Range: 1.0 – 1.6		Range: 0.4 – 1.4	
Mean: 1.0 (n = 11)		Mean: 1.1 (n = 14)		Mean: 1.0 (n = 7)		Mean: 1.25 (n = 4)		Mean: 0.8 (n = 5)	

Table 4.3: PCDD/F emissions from five sinter plants after process optimisation (in order to minimise PCDD/F emissions)

PI.2 Recycling of iron-containing materials into the sinter plant

Description: An integrated steel plant generates by-products, mainly consisting of iron scale from the rolling mills and a wide variety of dusts and sludges from waste gas treatment devices. Whenever these dusts, sludges and mill scale have a high enough iron or carbon content (normally >50%), they can be considered for use as a raw material in the sinter plant. Materials with a high lime content, such as steel slag may also be accepted, reducing the lime input. Currently, nearly all sinter plants in the world recycle dusts, sludges and mill-scale. In most plants these account for 10-20% of the sinter feed. At least one plant uses 100% dusts, sludges, slags and additives.

Main environmental benefits achieved: The amount of raw material saved is equal to the amount of sludges, dusts and mill scale used. Furthermore, dumping of these by-products is avoided. In this respect, the sinter plant has an important function in an integrated iron and steel plant.

Applicability: Can be applied at new and existing plants

Cross-media effects: Some by-products have a significant oil content, which may lead to higher emissions of hydrocarbons and (possibly) PCDD/F. Furthermore, undesired components (alkalis, chlorides) may accumulate as a result of recycling and cause increased waste gas emissions. To comply with existing emission limits, it might be necessary to install improved abatement equipment (e.g. fabric filter or improved/advanced ESP techniques). This would have a strong influence on dust generation and on economics (UBA Comments , 1997).

Reference plants: (Nearly) all sinter plants around the world recycle sludges, dusts and mill scale. Worth mentioning is the sinter plant of Warren Consolidated Industries, Youngstown Sinter Company, Warren, Ohio, USA. In 1992, this plant used only iron containing dusts/sludges and slags as an iron source for its sinter plant.

Operational data: Trouble-free operation is possible.

Economics: Saves raw material costs and avoids costs for dumping or waste processing.

Driving Force for Implementation: In many cases there are limited possibilities to dispose residues/by-products on landfills. In addition considerable cost (taxes for landfilling corresponding with stringent standards) as well as legal requirements are the most important driving forces for the implementation of measures to optimise recycling.

Reference literature: [InfoMil, 1997; Rentz, 1996]

PI.3 Lowering the content of volatile hydrocarbons in the sinter feed

Description: The input of hydrocarbons can be minimised, especially by reduction of the oil input and by avoidance of anthracite as well. Oil enters the sinter feed mainly by addition of mill scale. The oil content of mill scales can vary significantly, depending on their origin. Sometimes, oil content up to 10% is found [Gebert, 1995].

Low oil content in the recycled dusts and mill scale is preferred for several reasons (avoidance of fires and scaling in the ESP or fabric filter). High oil content also could give rise to higher PCDD/F emissions (see 4.2.2.1.2.8). Lower oil contents lead to lower VOC emissions.

Most of the oil hydrocarbons volatilise from the sinter mixture at temperatures in the 100 to 800°C range and are emitted from the sinter plant via the waste gas.

Two "techniques" can be applied to minimise oil input via dusts and mill scale:

1. Limiting input of oil by selecting only dusts and mill scale with low oil content. The use of 'good housekeeping' techniques in the rolling mills can result in a substantial reduction in the contaminant oil content of mill scale.
2. De-oiling of mill scale. Generally two methods may be developed/applied:
 - a. By heating the mill scale to approximately 800°C, the oil hydrocarbons are volatilised and "clean" mill scale is yielded. The volatilised hydrocarbons can be combusted.
 - b. Extracting oil from the mill scale using a solvent.

Neither of these treatment techniques are used commercially in the EU steel industry at present.

In the EU coke breeze is usually used as the fuel for the sintering process. However some plants still use a mixture of coke breeze and anthracite resulting in significantly higher emissions of hydrocarbons because of degrassing effects. This can be avoided by exclusive use of coke breeze.

Applicability: Can be applied both at new and existing plants.

Main emission level achieved: Non-methane hydrocarbon concentrations of < 20 mg/Nm³ can be achieved. The figure can be significantly higher if precautionary measures are not taken to reduce the oil content of the sinter feed materials and/or anthracite is also used as a fuel.

Cross-media effects: When some of the iron-containing wastes are rejected as suitable sinter plant feedstock, they enter the waste stream and so are either processed in another way or disposed of. Thus, emissions may in effect be transferred to another site.

Whenever mill scale is de-oiled, energy is consumed as it is heated. The volatilised hydrocarbons should be combusted properly (rule of thumb: temp. >850°C for > 2 seconds at >6% O₂).

Reference plants: Many plants regulate oil input in the sinter plant via the iron-containing wastes, especially those with an ESP or with a fabric filter. In most of the EU sinter plants anthracite is not used as a fuel.

In Mülheim-Ruhr, Thyssen Altwert Umweltservice GmbH operates a demonstration rotary kiln process for de-oiling of mill scale with a capacity of 15000 t/a. Several German steelworks (e.g. Thyssen Stahl AG), have run pilot tests with different processes (UBA Comments, 1997).

Operational data: At Inland Steel, Indiana Harbor Works, East Chicago, USA the maximum input of oil is limited to 0.3 kg/t sinter. De-oiling is applied whenever a higher hydrocarbon content is expected. However, it is not known whether this system is still operated. The sinter plant is equipped with a fabric filter.

A mill scale de-oiling facility has been installed at Nippon Kokan, Keihin Works and at Nippon Steel Corporation, Wakamatsu/Yawata Works, Japan. [InfoMil, 1997] reports that the NSC, Yawata Works does not operate this de-oiling facility because the oil content in the raw materials is regulated to avoid fire in the ESP.

At Kobe Steel, Kakogawa Works, Japan, the hydrocarbon concentration in the ESP precipitate is kept below 1% by reduced input of mill scale in the sinter plant (max. 3%).

Economics: no data available regarding de-oiling; the use of coke/anthracite mixtures is cheaper than the only use of coke, but precise figures are not available.

Reference literature: [Gebert, 1995; UBA Comments, 1997; InfoMil, 1997]

PI.4 Lowering the sulphur content of the sinter feed

Description: Sulphur compounds mainly enter the sintering process from the ores and coke breeze (see 4.2.2.1.2.5), with the ores contributing the much smaller percentage. Part of the sulphur remains in the sinter product (in the order of 13 – 25%) depending on the sinter basicity and the grain size distribution (see 4.2.2.1.2.5). Consequently the use of coke breeze and iron ores with lower sulphur content ($\leq 0.8\%$ S) directly correlate with lower SO₂ emissions. However, minimisation of the specific coke breeze consumption is also very important. During the last 15 years consumption at many sinter plants in the EU has been reduced from about 80 kg/t sinter to 38 – 55 kg/t sinter (see Table 4.1 and [Bothe, 1993]).

In addition the usage of coarser coke breeze (6 mm) can lead to significant lower SO₂ emissions compared to fine grain breeze (1 mm). Reductions from about 800 mg SO₂/Nm³ to about 500 mg SO₂/Nm³ have been reported [Beer, 1991].

Main emission levels achieved: Table 4.1 and Table 4.2 show emission factors below 1 kg SO₂/t sinter or (with 2100 Nm³/t sinter) emission concentrations below 500 mg/Nm³ can be achieved.

Applicability: Minimising sulphur inputs into the sinter feed (use of low sulphur coke breeze and iron ore and minimising specific coke breeze consumption) can be applied both at new and existing plants. However, it should be noted that the availability of low sulphur breeze and ore might be a constraint.

Cross-media effects: No cross media effects are known but the retention of additional sulphur in the sinter could have an adverse effect on the operation of the blast furnace when a low sulphur iron is required.

Reference plants:

- 1) Sinter plant, Nippon Steel Corporation, Yawata/Wakamatsu Works: using low-sulphur coke breeze
- 2) Sinter plant 4, Sumitomo Metal Industries, Wakayama Works
- 3) Sinter plant of Sidmar, B-Gent

Operational data: No specific problems are known. The plants operate smoothly.

Economics: Not available

Reference literature: [Bothe , 1993]

PI.5 Heat recovery from sintering and sinter cooling

Description: Two kinds of potentially reusable waste energy are discharged from the sinter plants: the sensible heat from the main exhaust gas from the sintering machines, and the sensible heat of the cooling air from the sinter cooler.

The sensible heat from the exhaust gases in the stack may be used by means of a heat exchanger. The energy savings are unspecified. Waste gas recirculation is a special case of heat recovery. The sensible heat is transferred directly back to the sinter bed by the hot recirculated gases. This is currently the only practical method of recovering heat from the waste gases.

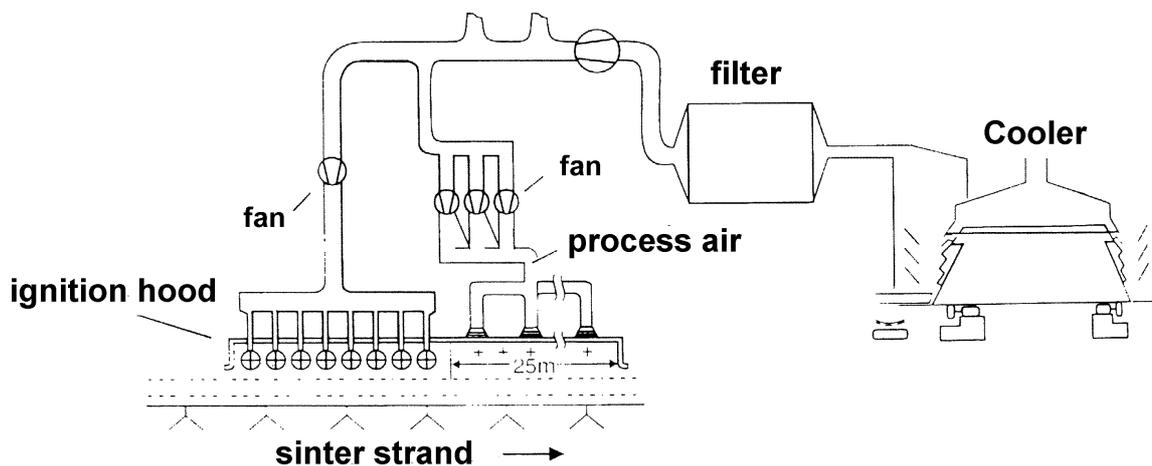


Figure 4.15 : Heat recovery from cooling air from the sinter cooler – [Beer, 1991]

The sensible heat in the hot air from the sinter cooler is one or more of the following ways:

- Steam generation in a waste heat boiler;
- preheating combustion air in the ignition hood; e.g. Figure 4.15
- preheating the green feed.

The amount of waste heat recovered can be influenced by the design of the sinter plant and the heat recovery system.

Some examples of heat recovery are given below:

- *Sinter cooler waste heat recovery with conventional as well as EOS-sintering*

The sensible heat of the sinter cooling hot air gas is used for producing steam in a waste heat boiler and for preheating the combustion air in the ignition hoods.

Main energy savings achieved: Reported energy recovery amounts to 18% of the total energy input for the waste heat boiler and 2.2% of total energy input for recirculation to the ignition hoods [InfoMil, 1997].

- *Sinter cooler and waste gas heat recovery with sectional waste gas recirculation*

At Sumitomo Heavy Industries Kokura No. 3 sinter plant, sectional waste gas recirculation is applied. Before recirculation, the waste gases are led through a waste heat boiler. The gases from the sinter cooler are also led through a waste heat boiler.

Main energy savings achieved: Energy recovery reported at this plant by means of this system is 23.1% of the energy input; 120 kg steam/t sinter was produced at a temperature of 273°C and a pressure of 9 bar [SHI, 1987].

- *"Strand cooling" and waste heat recovery with partial waste gas recirculation*

At the Sumitomo Heavy Industries Wakayama No. 4 sinter plant, the sinter cooler is integrated into the sinter strand ("strand cooling"). At this plant, waste gases from both the sintering and the cooling zone on the grate are led through waste heat boilers and subsequently recirculated to the strand.

Main energy savings achieved: Recovered heat amounts to 30% of the input heat. Approximately 120 kg steam/t sinter is produced at a pressure of 25 bar and a temperature of 375°C. The plant is designed for a sinter production of 10000 t/day and has a sinter area of 360 m² [SHI, 1987] but productivity could be adversely affected if the system were applied to an existing plant operating at or near maximum output.

Applicability: Waste heat recovery from stack or sinter cooling can be applied both at new and existing plants. It is recognised, however that investment costs are lower for a new plant incorporating heat recovery systems from the planning stage and at some existing plants the existing configuration may make costs prohibitive.

Cross-media effects: Reduces energy consumption and in some cases also emissions of dust, due to the pre-installed coarse dust separators.

Economics: Capital costs will be site specific, but application of waste heat recovery reduces operational costs.

Reference plants: Heat recovery from sinter cooling is applied frequently (e.g. at Hoogovens IJmuiden, NL-IJmuiden, Thyssen Stahl in Germany and reportedly 64% of the Japanese sinter plants) [OECD, 1988; Arimitsu, 1995].

In 1995, 43% of the Japanese sinter plants applied waste heat recovery from the stack [Arimitsu, 1995] and three strands at British Steel recover heat from the sinter cooler hot air.

Operational data: No specific problems are known.

Reference literature: [Arimitsu, 1995; InfoMil, 1997; OECD, 1988; SHI, 1987]

PI.6 Top-layer sintering

Description: In PI.2 the recycling of iron-containing materials to the sinter strand is described. With respect to recycled materials containing hydrocarbons, especially oil, in PI.3 techniques are described to lower the oil/hydrocarbon input. There is another possibility to recycle such materials with varying oil contents of up to 3%. It is called top-layer-sintering and is claimed to be much cheaper compared to de-oiling techniques. Top-layer-sintering means that a certain mixture of oil/hydrocarbons containing by-products/residues conditioned to approximately 7% water content and then deposited with a feeding drum on the main sinter layer. A second ignition hood with an energy output of 25 to 35% of the energy output of the main burner is used to ignite this second layer. For achieving a high sinter quality from the second sinter layer of oil-containing by-products/residues, it is important that a balanced selection of energy carriers is present within this layer in order to satisfy the enthalpy requirements for water and oil evaporation, for the cracking of the inherent organic components as well as for the complete sintering of this layer. In addition to this careful proportion of the sinter feed components, the exact positioning and timing of ignition of the second sintering layer is of critical importance.

Achieved environmental benefit: The combustion of hydrocarbons (mainly deriving from the oil content of recycled materials) within the sinter layers is optimised in order to protect the ESP (prevention of fires) and to avoid blue haze (which represents incomplete combusted organic compounds). In addition the emissions of PCDD/F can be reduced; pot grate tests showed a reduction of 60-65% [VAI, 1996] but data from sinter strand is not available.

Applicability: Limited applicability to existing plants because of missing space for additional storage facilities and for a second feed system as well. New plants would normally go for techniques minimising the hydrocarbon/oil input of the recycled by-products/residues. Flexibility in the selection of raw materials is very limited.

Cross-media effects: Additional fuel for the ignition of the second layer has to be provided.

Reference plants: Sinter plant of Voest-Alpine Stahl Donawitz GmbH, A-Leoben-Donawitz

Operational data: The sinter strand of Voest-Alpine Stahl Donawitz GmbH has a suction area of 120 m² and a productivity of 35 t/m². 24h. The productivity did not change after introduction of top-layer-sintering in January 1995. The capacity for recycling ferrous containing residues/by-products/materials is 18 t/h.

Economics: Not available.

Reference literature: [VAI, 1996]

PI.7 Emission Optimised Sintering (EOS)

Description: In 1992 results were published showing that recycling of part of the waste gas from the sinter strand can significantly reduce the quantity of waste gas for end-of-pipe-treatment, limit emissions of pollutants at source and decrease solid fuel consumption [Gudenau, 1992]. An initial demonstration/commercial application in May 1994 at Hoogovens IJmuiden in the Netherlands fully confirmed the potential of this approach. A strand with a suction area of 132 m² was completely covered with a sealed hood for waste gas in accordance with the EOS (Emission Optimised Sintering) Lurgi process (Figure 4.16).



Figure 4.16 : Covered sinter strand according to the EOS process - [Panne, 1997]

The concept is to recycle a part of the mixed waste gas from the whole strand back to the entire surface of the strand. The recycling rate of the sintering waste gas is of the order of 40-45 %, corresponding to a 14-15 % oxygen concentration in the wet gas/air mixture in the hood and resulting in a 45-50 % decrease in waste gas flow emitted to the atmosphere. The waste gas is de-dusted in a cyclone before recycling. Under these conditions, strand productivity remains unchanged and coke breeze consumption is reduced by 10-15 % compared with conventional practice. The sinter quality, defined as reduction disintegration, appears constant, FeO in sinter is 1.5 % higher, reducibility increases, cold strength decreases slightly and the mean diameter remains approximately 17 mm [Panne, 1997]. The use of "EOS sinter" in the blast furnace does not show any adverse effects but it should be noted that it is only accounts for about 50 % of the charge because of the high pellet percentage used at Hoogovens, NL-IJmuiden blast furnaces. Elsewhere the levels of sinter used in the blast furnace burden can be much higher (up to 95%).

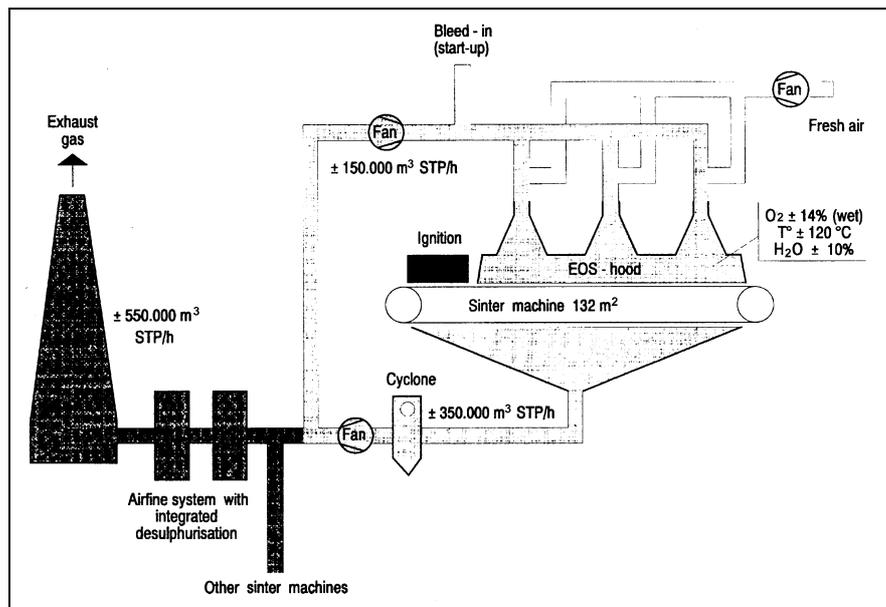


Figure 4.17 : Schematic diagram of the Emission Optimised Sintering (EOS) process - [Kersting, 1997]

Main emission levels achieved: EOS was primarily developed to reduce waste gas flow and thus the mass emissions of particulate matter and PCDD/F with the benefit that additional abatement equipment to further treat the waste gas before discharge to the atmosphere would have to treat lower volumes, with implied capital and operational cost savings. The results from this particular commercial plant for high basicity sinter ($\geq 1.7 \text{ CaO/SiO}_2$) indicate that successful implementation of EOS is possible (Table 4.4). The results for low basicity sinter are similar.

Component	Unit	Conventional sintering		Emission optimised sintering	
		July 1994	October 1994	July 1994	October 1994
Total gas flow	Nm ³ /h	394000	372000	328000	328000
Recirculation flow	Nm ³ /h	0	0	153000	120000
Emitted waste gas flow	Nm ³ /h	394000	372000	175000	208000
Temperature	°C	164	114	155	149
Percentage of moisture	%	10	11	16	19
Acid dewpoint	°C	46±5	n.d.	71±5	n.d.
Particulate matter *1	g/t sinter	500	n.d.	170	n.d.
O ₂	%	15	15	11.5	12.1
CO ₂	%	7.5	7	11.7	11.2
CO	%	1	1.2	1	1
SO ₂	g/t sinter	1430	890	840	680
NO _x	g/t sinter	630	570	300	410
C _x H _y	g/t sinter	200	145	95	83
PCDD/F	µg I-TEQ/ t sinter	2	n.d.	0.6	n.d.

*1 at this time the particulate emissions have been abated by cyclones only

Table 4.4 : Comparison of characteristics and emission values of Conventional Sintering and Emission Optimised Sintering (EOS) at a sinter strand of a Hoogovens IJmuiden with a suction area of 132 m² and a production of 4700 t high basicity sinter/d - [Panne, 1997]

The emission reductions (by mass) resulting from EOS of the commercial plant described are:

Parameter	Emission reduction* (mass %)	
	Low basicity sinter	high basicity sinter
Waste gas flow	40	40 - 50
Particulate matter	50	60
Carbon monoxide (CO)	45	50
SO ₂	?	15 - 20
NO _x	35	30 - 45
Hydrocarbons (C _x H _y)	n/a	50
PCDD/F	70	65

- emission reduction is expressed as a mass flow reduction of the component. This does not necessarily imply a reduction in concentration
- ? = no consistent data available; n/a = data not available

Table 4.5 : Reported emission reduction (by mass) of Emission Optimised Sintering (EOS) - [Panne, 1997]

EOS allows coke breeze consumption to be reduced from 60 kg/t sinter to 48 kg/t sinter [Panne, 1997]. This significant saving in thermal energy in the Dutch plant described is not necessarily transferable to other plants since these plants may already be operating at low thermal energy inputs from the solid fuels as a result of the application of process optimisation measures. Both the data in Table 4.1 and data from the European Blast Furnace Committee survey of the operational data for sinter plants (1996), clearly demonstrate that many European plants already operate at significantly lower levels than those quoted by [Panne, 1997].

The application of EOS requires the installation of extra suction fans. This will result in an additional installed electric capacity of 200 to 400 kW, amounting to an energy consumption increase of 0.003-0.008 GJ/t sinter, which is low compared to the overall potential saving.

Where carbonate iron ores are used the concentration of CO₂ in the waste gas is increased so that the application of EOS may strongly inhibit the sinter process. The CO₂ emissions in plants which do not use carbonate ores average from 190 to 220 kg/t sinter (Table 4.1) whereas in plants which do use carbonate ores the average can be up to twice as high.

Applicability: EOS can be applied at both new and existing plants although it is recognised that investment costs are lower in the case of new plant incorporating the system from the planning stage and at some existing plants costs may be considerably high due to plant arrangement.

Cross-media effects: EOS reduces both emissions to air and the energy consumption of the sintering process. The additional suction fans increases electricity consumption, but this increase is negligible compared to the savings in coke breeze. Special attention must be paid to carbon monoxide (CO) in the recirculated waste gas in order to prevent carbon monoxide poisoning of employees.

Reference Plant: Hoogovens IJmuiden, NL-IJmuiden. All sinter production at this plant is currently performed with EOS. Note should be taken of the Wierton plant in the USA which has had a similar arrangement for many years.

Economics: The investment required to implement EOS at the sinter plant at Hoogovens IJmuiden, with a total conventional waste gas flow of approximately 1.2 MNm³/h from three sinter strands, was 17 million Ecu₁₉₉₆. Operational costs were decreased compared to conventional sintering due to reduced coke breeze input. Operational savings are estimated at 2.5 million Ecu₁₉₉₆/a. This figure is based on a reduced coke breeze consumption of 6 kg/t sinter, at a price of 100 Ecu₁₉₉₆/t coke breeze and sinter production of 4.2 Mt/a [Goverde, 1995]. These figures may be exceptional and may not be possible to achieve at many other EU sinter plants, particularly those already operating at significantly lower solid fuel inputs than those quoted for Hoogovens (see above). It is also stressed that the estimated average price of the coke breeze used at other EU sinter plants is about 60 Ecu₁₉₉₆/t, rendering any possible coke breeze savings less economically attractive.

However, the smaller volume of waste gases emitted from the sinter strand following the application of waste gas recirculation means the cost of end-of-pipe abatement will be reduced.

Driving force for implementation: The most important driving forces for implementation have been stringent requirements and emission standards laid down by authorities.

Operational data: EOS first came into operation in May 1994 on sinter strand 31 of the Hoogovens IJmuiden sinter plant. Now, EOS is operated at all three sinter strands at the plant. To date, availability of the system has been >95%. In case of a failure, the system is automatically switched to conventional sintering.

Reference literature: [Panne, 1997; Goverde, 1995]

PI.8 Sectional waste gas recirculation

Description: The concept of selective recycling technology is based on local suction of the sintering waste gas under the strand and its local recycling above the sinter bed. This selective suction and recycling is the main difference between this and the EOS process. Figure 4.18 shows a schematic diagram of such a sectional waste gas recirculation implemented at a sinter plant in Japan.

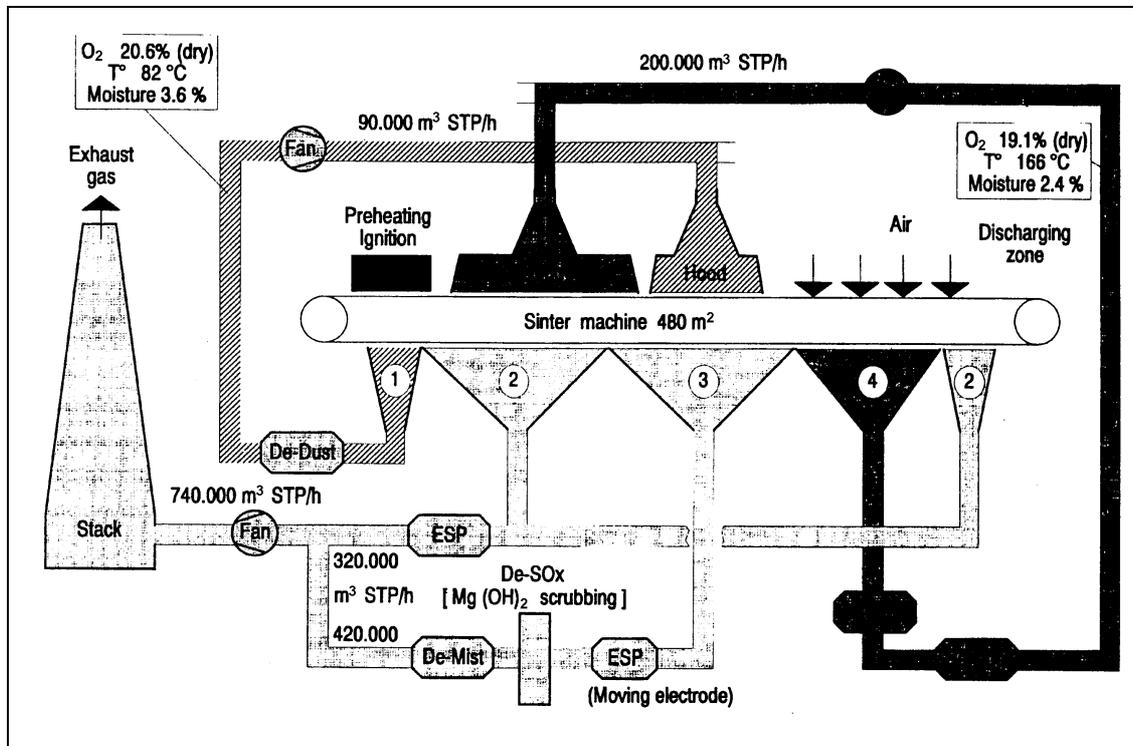


Figure 4.18 : Schematic diagram of selective waste gas recirculation (Nippon Steel Corporation – Yawata Works – Tobata No. 3 Plant) - [Kersting 1997]

In this case the 480 m²-sintering surface is divided into 4 different zones:

- zone 1: gas corresponding to the preheating section of the raw mix to be recycled in the middle of the strand (high O₂, low H₂O, low temperature) (see Figure 4.5 and Figure 4.8 and Table 4.6),
- zone 2: gas low in SO₂, to be discharged at the stack after de-dusting (low O₂, high H₂O, low temperature) (see Figure 4.5 and Figure 4.8 and Table 4.6),
- zone 3: gas rich in SO₂, to be discharged at the stack after de-dusting and desulphurisation (scrubbing with a magnesium hydroxide solution, in the present case) (low O₂, high H₂O, low temperature) (see Figure 4.5 and Figure 4.8 and Table 4.6),
- zone 4: gas rich in SO₂, corresponding to the hot section around the flame front, to be recycled in the first half of the strand just after the ignition zone (high O₂, low H₂O, very high temperature) (see Figure 4.5 and Figure 4.8 and Table 4.6).

In this procedure the O₂ concentration of the recycled waste gas remains high (19%) and the moisture low (3.6%). A recycling rate of 25% is achieved without negative impact on sinter quality (the RDI remains practically constant and the SI – Shatter Index – increases by 0.5%). A solid fuel saving 6% is also reported.

There are two advantages to this system compared to conventional sintering:

1. The unused oxygen in the waste gas can be used effectively by recirculation.
2. The waste gas from the different sections can be treated separately depending on the composition of the gas. Thus, investments and operational costs of waste gas treatment facilities can be significantly reduced compared to conventional sintering even in comparison with the EOS system.

Waste gas flow section	Waste gas composition					Waste gas treatment
	Flow [kNm ³ /h]	Temp. [°C]	O ₂ [%vol.]	H ₂ O [%vol.]	SO ₂ [mg/Nm ³]	
Windbox 1 - 3	62	82	20.6	3.6	0	Recirculated to sinter strand
Windbox 4 - 13 + 32	290	99	11.4	13.2	21	To stack after ESP (electrostatic precipitation)
Windbox 14 - 25	382	125	14.0	13.0	1000	To stack after ESP and desulphurisation
Windbox 26 – 31	142	166	19.1	2.4	900	Recirculated to sinter strand
Stack	672	95	12.9	13.0	15	Emission to air

Table 4.6 : Characteristics of waste gas flows, using sectional waste gas recirculation at Tobata No. 3 sinter plant, NSC's Yawata works – based on [Sakuragi, 1994]

Main emission levels achieved: The following improvements have been achieved in terms of pollution abatement: a substantial decrease in the waste gas released into the atmosphere (about 28%), in dust emission (about 56%, note that this includes the effect of revamping the electrostatic precipitator, which is now equipped with moving electrodes) and in the SO₂ discharge (about 63%, including the end-of-pipe-desulphurization of the gas issuing from zone 3). A slight decrease in the NO_x emission (about 3%) is also reported. Table 4.7 compares emissions before and after application of sectional waste gas recirculation.

Characteristic/ Component	Unit	Conventional (with desulphurization plant)	Sectional waste gas recirculation	Improvement
Waste gas flow	Nm ³ /h	925000	665000	28%
Particulate matter*	mg/Nm ³	50	30**	56% by mass
SO _x ***	mg/Nm ³	26	14	63% by mass
NO _x	mg/Nm ³	408	559	3% by mass
Net energy consumption	GJ/t sinter	1.662	1.570	6%****

* waste gas treated by electrostatic precipitation

** reduction in dust emissions partially achieved by repair of ESP

*** part of waste gas treated in desulphurisation unit

**** this reduction in net energy consumption has to be considered in relation to the relative productivity and quality requirements in Japan and the EU sinter plants

Table 4.7 : Comparison of final waste gas composition before and after rebuilt to sectional waste gas recirculation. Tobata No. 3 sinter plant, NSC's Yawata works – based on [Sakuragi, 1994]

Applicability: Sectional waste gas recirculation can be applied at both new and existing plants although it is recognised that investment costs are lower for a new plant incorporating the system from the planning stage and at some existing plants costs may be considerably high due to plant arrangement.

Cross-media effects: Installing additional fans results in more electricity being consumed. However, this increased energy demand is negligible compared to the decreased coke breeze consumption.

Reference plant: There are five plants in Japan which apply this technique. The data given here refer to Tobata no. 3 sinter plant, Yawata Works, Nippon Steel Corporation.

Operational data: A system using sectional waste gas recirculation was installed in the Tobata No.3 sinter plant at Nippon Steel Corporation's Yawata Works in October 1992. The waste gas is divided in four sections, each of which is treated separately. The system operates smoothly and waste gas recycling does not affect the sinter quality [Sakuragi, 1994]. However, this latter point has to be considered in relation to the lower productivity of the quoted Japanese plant in comparison with many sinter plants in the EU.

Economics:

Current European investment costs for the gas recycling system are quoted as 8 to 10 Million Ecu₁₉₉₇, excluding the de-NO_x, de-SO_x and other abatement equipment.

Operational costs were reported confidential. Nevertheless, a 6% reduced coke consumption was reported.

Reference literature: [InfoMil, 1997; Sakuragi, 1994]

4.3.2 End-of-pipe-techniques

EP.1 Electrostatic precipitator (ESP)

Description: The most commonly used abatement devices for treating the large volumes of waste gases in sinter plants in the EU are dry, electrostatic precipitators with three or four fields arranged in series. These work by generating an electrostatic field across the path of the particulate matter in the air stream. The particles become negatively charged and migrate towards positively charged collection plates. In dry electrostatic precipitators, the collected material is removed by "rappers" which periodically strike or vibrate the collection plates, dislodging the material and allowing it to fall into collection hoppers.

In wet electrostatic precipitators the collected material is removed by a constant flow of water, which is collected and subsequently treated.

In order to obtain good separation, the specific resistivity of the particulates has to be in the range 10^4 - 10^9 Ω m. Usually, most particulates in the waste gas from the sintering process are within this range but compounds with significant higher specific resistivity, such as alkali chlorides, heavy metal chlorides and calcium oxides (see Figure 4.7 and Figure 4.10) also occur and are difficult to remove with high efficiency.

Other factors that influence the efficiency are: waste gas flow rate; strength of the electric field; particulate loading rate; SO_3 concentration; moisture content; and, shape and area of the electrodes.

Performance improvements have been made to ESP's by the use of higher or variably pulsed voltages and rapid reaction voltage and current controls [Hodges, 1995]. Operations have been further enhanced through the introduction of systems to improve the rapping forces to 200 Gs, high energy pulse superimposition and refurbishment with increased plate spacing. Conditioning with SO_3 and/or water vapour may also increase de-dusting efficiency. As a disadvantage HCl emissions can increase.

Three newer types of ESP with good reported performance are highlighted below, although currently these techniques have been installed in only a few commercial plants to date.

- a) Moving Electrode Electrostatic Precipitator (MEEP): In the MEEP several groups of electrode plates move on caterpillar tracks. They are continuously cleaned by rotating brushes. Thus, the strongly adhesive dust is easily removed from the plates and the insulating effect of the dust layer is avoided (see 4.2.2.1.2.2) [InfoMil, 1997; Bothe, 1993].
- b) Use of energy pulse superimposition: The pulse system provides a voltage consisting of negative narrow pulses superimposed on a filtered voltage of negative polarity. These high voltage pulses have a width of 140 μ s and can be repeated at a frequency up to 200 pulses per second. The peak voltage is higher with pulse energisation, providing a better particle charging and current distribution in the precipitator. One of the most important characteristics of pulse energisation is its capability to cope with high resistivity dust (see 4.2.2.1.2.2) [Kim, 1998].
- c) Electrostatic Space Cleaner Super (ESCS). The ESCS is operated with a higher voltage (70-200 kV). This is possible by a larger distance between the electrode plates [InfoMil, 1997].

Main emission levels achieved: ESP reduces particulate matter concentrations with an efficiency of >95%. In some cases efficiency of 99% is achievable. Operational data for sinter plants are in the range of 20 to 160 mg/Nm³. Emission values for MEEP and ESCS may achieve < 40 mg/Nm³. ESP with energy pulse superimposition main achieve 20 – 30 mg/Nm³.

Applicability: Electrostatic precipitators can be installed at both new and existing plants. The MEEP can be installed as a last field of an existing electrostatic precipitator, or as a separate unit in its own housing [Bothe, 1993], but the arrangement and possibility of either type of installation will be site specific.

Cross-media effects: A solid waste flow is generated. In some cases, this waste flow can be recycled into the sintering process. Whenever the concentration of heavy metals and/or alkali compounds is too high, reuse may be hampered.

Furthermore, energy consumption is increased. For a sinter plant with a waste gas flow of approximately 1 MNm³/h, the energy consumption is 300 to 400 kW. At sinter production of 4 Mt/a, this amounts to 0.002 to 0.003 GJ/t sinter (or 0.1 to 0.15% of the total sinter energy consumption).

Reference plants: Nearly all sinter plants in Europe apply electrostatic precipitators. [UBA Comments, 1997] reports that all German sinter plants now apply dry ESP.

MEEP has been installed at two sinter plants in Japan and at two sinter plants in Germany, one at Krupp Hoesch Stahl, Dortmund and one at EKO Stahl, Eisenhüttenstadt.

Energy pulse superimposition has been installed at four sinter plants at Kwangyang Works, Posco in South-Korea and at two strands at Thyssen Krupp Stahl, D-Duisburg.

ESCS has been installed in the sinter plant of Nippon Steel Corporation, Wakamatsu/Yawata Works, Japan.

Operational data and economics: Electrostatic precipitation is the most common particulate matter abatement device for cleaning sinter plant waste gas and ESPs can usually be applied without problems. Attention needs to be paid to the amount of hydrocarbons in the waste gas in order to avoid the risk of fire. Operators control the mill scale input to avoid hydrocarbons in the waste gas.

Table 4.8 gives some operational and economic data concerning electrostatic precipitators in use at sinter plants.

Additional economics:

DHV (1996) reports the following costs for ESP:

Investment: 5 to 7.5 Ecu₁₉₉₆/(Nm³/h);

Operational: 0.05 to 0.08 Ecu₁₉₉₆/1000 Nm³ treated.

For a sinter plant with a capacity of 4 Mt/a, a waste gas flow of 1 Million Nm³/h and 8640 operational hours per year, this would correspond to the following costs:

Investment: 5 to 7.5 million Ecu₁₉₉₆;

Operational: 0.11 to 0.16 Ecu₁₉₉₆/t sinter.

For a single MEEP filter added to the existing ESP to treat 500000 Nm³/h a cost of about 1.1 Million Ecu₁₉₉₇ has been quoted.

The decisive cost factor is the waste gas flow.

Driving Force for Implementation: The main driving force for the implementation of the techniques described has been stringent emission standards or other legal requirements.

Reference literature: [Gebert, 1995; InfoMil, 1997; Kim, 1998]

	Unit	Nippon Steel Corp. Wakamatsu/Yawata	Krupp Hoesch Stahl Dortmund	Kobe Steel Ltd. Kakogawa works	Sumitomo Metal Ind. Wakayama
Sinter production	(t/h)				
Design		1000	625	560	n/a
actual		600	n/a	375	185
Total waste gas flow	(10 ⁶ Nm ³ /h)				
Design		2	1.05	1	n/a
Actual		1	n/a	0.7	0.4
Sinter basicity	(CaO/SiO ₂)	1.92	n/a	1.8	2.2
Electrostatic precipitator					
Type		ESCS*	MEEP	n/a	dry ESP
Number		2 parallel	3 parallel	3 parallel	1
flow per ESP	(10 ⁶ Nm ³ /h)	0.5	2*0.3; 1*0.45	0.25-0.33	0.4
Particulate matter inlet concentration					
Design	(mg/Nm ³)	1000	n/a	1000	n/a
Actual	(mg/Nm ³)	n/a	700	1000	1100
Particulate matter outlet concentration					
Design	(mg/Nm ³)	50	n/a	50	n/a
Actual	(mg/Nm ³)	20-37	25	30-50	40
ESP efficiency					
Design	(%)	95	n/a	95	n/a
Actual	(%)	n/a	96	95-97	96
Conditioning ESP and/or pre-treatment sinter feed		ESP non-conditioned; Oil content raw materials regulated (no de-oiling)	ESP non-conditioned No special measures to prevent fire-hazard	ESP non-conditioned; Max. HC* in ESP-precipitate: 1%; achieved by max. input mill scale of 3% of feed	non-conditioned
By-product (dust)					
Amount	(kg/t sinter)	1-2	n/a	1-2	2
Processing		recycled to sinter process	recycled to sinter process	n/a	n/a
Energy demand	(GJ/t sinter)	0.00036	n/a	n/a	n/a
Investment	(1 x 10 ⁶ Ecu ₁₉₉₆)	n/a	9 in 1995	n/a	4.2 in 1975
Operational costs	(Ecu ₁₉₉₆ /t sinter)	0.0018-0.0024 in 1986 (only maintenance)	n/a	n/a	0.11 in 1986 (maint. and deprec. excluded)
n/a	not available				
*	ESCS	Electrostatic Space Cleaner Super			
**	MEEP	Moving Electrode Electrostatic Precipitator			

Table 4.8 : Operational and economical data for MEEP and ESCS electrostatic precipitators operated at sinter plants to treat sinter plant waste gas – [InfoMil, 1997]

EP.2 Fabric filter

Description: A fabric filter is highly efficient at reducing particulate matter emissions in a waste gas stream. Fabric filters enhanced with additives may also reduce emissions of PCDD/F, hydrochloric acid (HCl), hydrofluoric acid (HF) and to a lesser extent, sulphur dioxide (SO₂). In particular, emissions of PCDD/F may be reduced significantly.

The filters, often tubular, are placed on supports inside an airtight shell called a "baghouse", a term often used synonymously with "fabric filter". The air stream enters from below and particulate matter collects on the fabric as it passes through the bag. A filter cake builds up until the pressure drop reaches a set point. At this time, the filter comes off-line for cleaning by one of three mechanisms - air reversal, shakers or pulse jet. Several types of filtering material can be used, each of which have their specific qualities.

Application of fabric filters to treat sinter plant waste gas is often frustrated by high temperature, abrasiveness, large gas volumes, the additional pressure drop and "stickiness". Experience at the only EU plant to use fabric filters on sinter plant waste gas, following coarse de-dusting using an ESP (a prerequisite of employing fabric filters at sinter plants), is that the large gas volumes and the additional pressure drop do not frustrate the operation of the filter. In addition neither the abrasive nature of the dusts, nor the stickiness of the particles caused problems and the problem of high temperature can be largely resolved through the judicious choice of the filter bag material.

The presence of fine particulates (especially alkaline chlorides, heavy metal chlorides and calcium oxides (see 4.2.2.1.2.2)) and the relatively high concentrations of organic compounds may lead to form a dust cake on the bags which stays strongly cohesive and forms a relatively impermeable layer.

It is reported [Weiss, 1998] that the problem of fine particulate and high concentration of organic compounds can be resolved through the use of the basic addition of slaked lime which builds up a precoat layer preventing the formation of an impermeable layer and consequently the blinding of the filter material. In addition, the high concentration of hydrocarbons can cause filter fire [EC LECES, 1991]. [Gebert, 1995] stated that the concentration of hydrocarbons in the waste gas before the fabric filter should be limited to 20-30 mg/Nm³ in order to avoid blinding of the filter material. The experience at Bremen [Weiss, 1998] has shown that this limitation is not necessary when dosing with slaked lime and concentrations of hydrocarbons as high as 200 mg/Nm³ have been measured in the waste gas without detriment to the operation of the filter.

Figure 4.19 shows the layout of a bag filter which has been installed downstream to an existing ESP with two fields in order to minimise the emissions of dust and heavy metals.

Dosage with the slaked lime in this operation has been used to circumvent the problems previously associated with the application of bag filters to treat sinter plant waste gas.

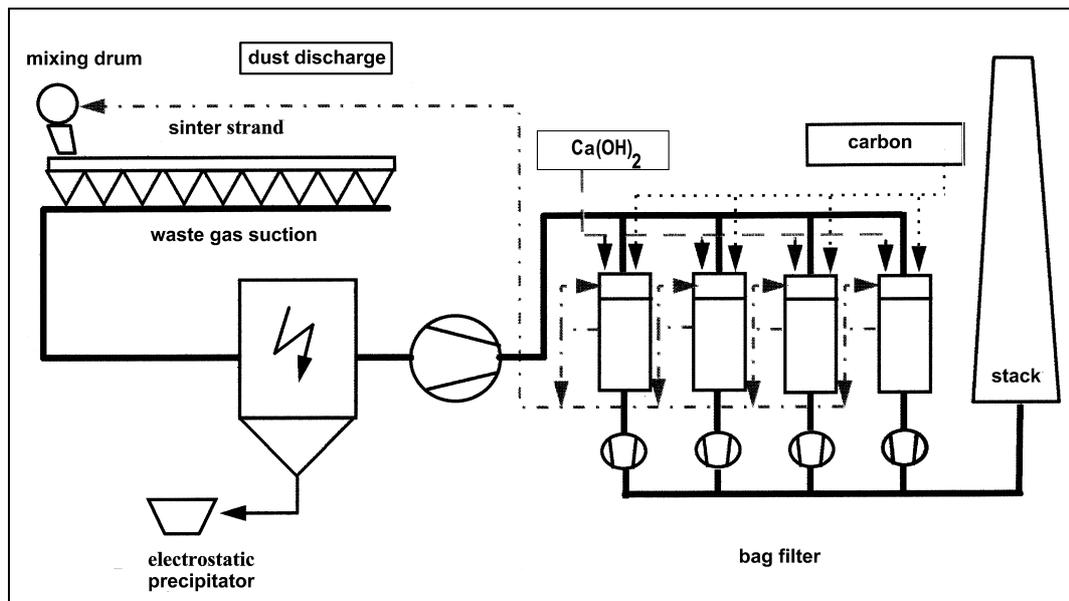


Figure 4.19: Layout of a bag filter after an ESP for the advanced treatment of waste gas from a sinter strand – [Weiss, 1996]

Additionally lignite coke powder is added to the waste gas before the bag filter to reduce PCDD/F (and mercury) (Figure 4.20).

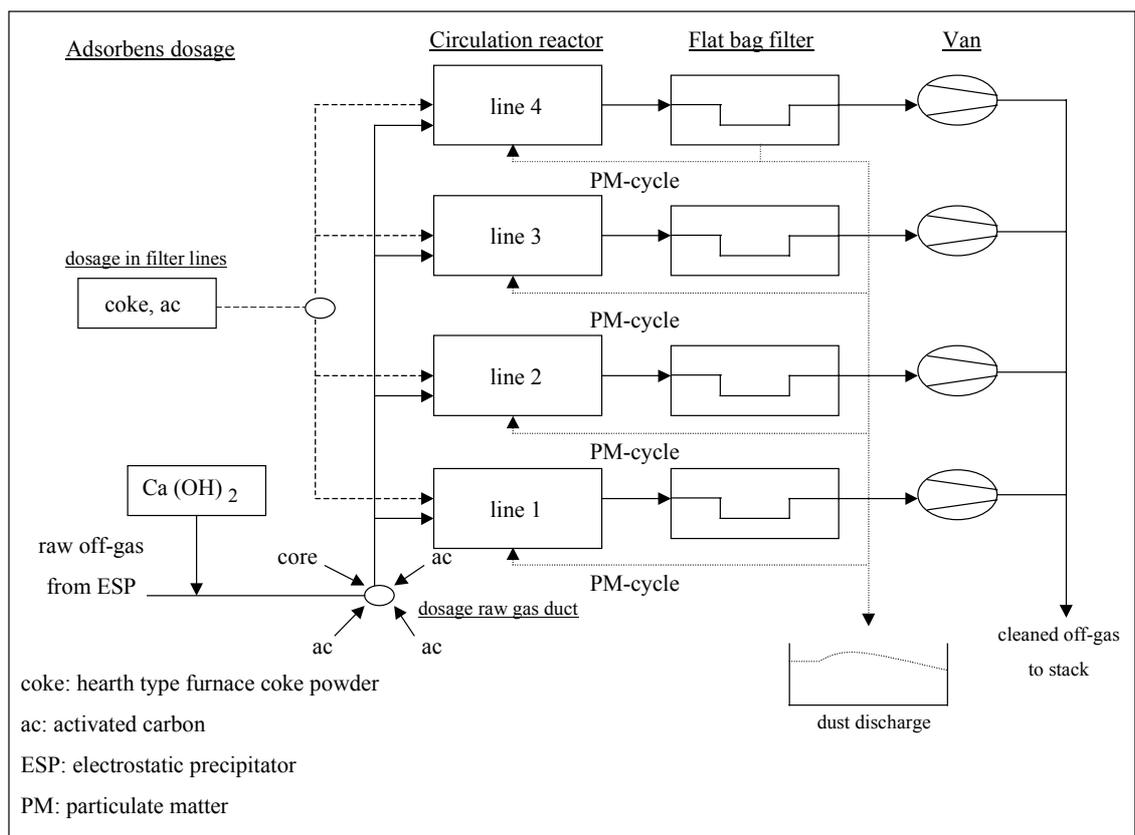


Figure 4.20 : Dosage of lignite coke powder and lime into the waste gas from a sinter plant before a bag filter – [Weiss, 1996]

As well as lignite coke, powder lime is dosed in order to form a precoat layer protecting the bag fabric filter and to lower the auto-ignition point. As a side effect also HF and HCl emissions are reduced. The separated dust and adsorbents are fully recycled to the sinter strand. It is noted that the dust from the ESP is landfilled because of the relatively high alkali chloride content, which increases the residual dust content in the treated waste gas.

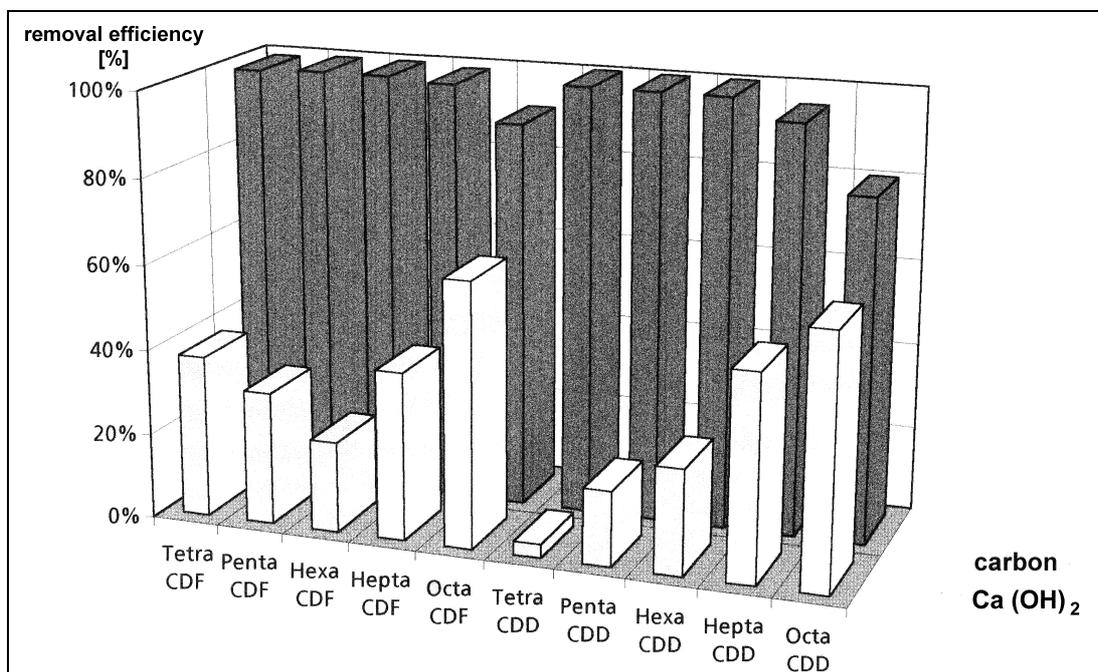


Figure 4.21 : PCDD/F removal efficiency of a bag filter with dosage of lignite coke powder – [Weiss, 1996]

Because of the relatively high inlet PCDD/F concentrations (after ESP) these can be 5-10 ng I-TEQ/Nm³ the residual concentration is between 0.2 to 1 ng I-TEQ/Nm³, although in most cases it is below 0.5 ng I-TEQ/Nm³. This residual content is in proportion to the fine dust concentration achieved.

Applicability: A fabric filter can be applied both at new and existing plants. However, it should be noted that its applicability is site-specific and depends on the characteristics of the waste gas and the particulate matter.

Cross-media effects: Fabric filters generate a solid waste flow (0.5 to 1 kg/t sinter), which is preferably recycled into the sintering process. In some cases the captured dust contains unwanted compounds (e.g. zinc, lead, alkalis), hampering its reuse. In this case, some or all of the solid waste has to be discarded.

Use of a fabric filter also increases electricity consumption (from 0.2 to 2.0 kWh/1000 Nm³). This would correspond to 0.002 to 0.015 GJ/t sinter or < 1% of total sinter energy consumption for a plant producing 4 Mt sinter per year, with a waste gas flow of 1 MNm³ and 8640 operating hours per year.

If carbon is used to achieve further reduction of PCDD/F emissions then special attention has to be paid to the risk of filter fire and operation temperatures close to or under the dew-point of the waste gas. Spark detection equipment may be installed and when necessary, the filter is bypassed. Fabric filters have to be bypassed during start-up.

Reference plants: There are at least three successful fabric filter devices in operation treating sinter plant waste gases. A further fabric filter device is currently under construction.

- 1) Stahlwerke Bremen GmbH, Bremen, Germany; the above described system was installed in 1993. The continuous dosage of lime and lignite coke powder has been in operation since 1996.
- 2) Inland Steel, East Chicago, USA.
- 3) Warren Consolidated Industries (WCI), Youngstown Sinter Company, Warren, Ohio, USA.
- 4) US Steel, Gary Works, Gary, Indiana, USA: fabric filter device currently under construction.

Operational data and economics:

Table 4.9 contains an overview of the operational data and economics of the fabric filters. More specific information is discussed here:

Additional economics:

Following costs are reported [InfoMil, 1997]:

Investment: 5 to 15 Ecu₁₉₉₆/(Nm³/h);

Operational: 0.25 to 1.5 Ecu₁₉₉₆/1000 Nm³ treated.

For a sinter plant with a capacity of 4 Mt/a, a waste gas flow of 1 Million Nm³/h and 8640 operational hours per year, this would correspond to the following costs:

Investment: 5 to 15 million Ecu₁₉₉₆;

Operational: 0.54 to 3.24 Ecu₁₉₉₆/t sinter.

Decisive cost factors are pressure drop, the waste gas flow, fabric material and filter loading

Driving force for implementation: In case of Stahlwerke Bremen the authorities required measures to minimise the dust, heavy metals and PCDD/F emissions. To meet the prescribed emission limit values Stahlwerke Bremen chose and implemented the described technique.

Reference literature: [InfoMil, 1997; Gebert, 1995; Weiss, 1996; EC LECES, 1991]

	Unit	Stahlwerke Bremen GmbH, Bremen, Germany	Inland Steel* East Chicago, USA	WCI-Youngstown Sinter Company**, Warren, Ohio, USA
Sinter capacity	(t sinter/h)			
design		260	167	125
actual		230	125	100
Waste gas flow	(1 x10 ⁶ Nm ³ /h)			
design		0.5	0.82	n/a
actual		0.4-0.45	0.75	0.42
Pre-inst. de-dusting		dry ESP (two fields)	none	none
Particulate matter				
inlet	(mg/Nm ³)	ca. 300	n/a	n/a
outlet	(mg/Nm ³)	< 5	12-16	20
efficiency	(%)	>99	n/a	>99%
Dioxin removal			n/a	n/a
inlet	(ng I-TEQ/Nm ³)	5-10		
outlet	(ng I-TEQ/Nm ³)	0.02-1.0		
efficiency	(%)	98-99.6		
HCl removal			n/a	n/a
inlet	(mg/Nm ³)	26.6		
outlet	(mg/Nm ³)	11.4		
efficiency	(%)	57		
HF removal			n/a	n/a
inlet	(mg/Nm ³)	5.9		
outlet	(mg/Nm ³)	0.2		
efficiency	(%)	>95		
Heavy metal removal		Inlet	Emission	
efficiency		(after ESP)	(after fabric filter)	n/a
	mg/Nm ³	Al: 0.2	0.04	
		As: 0.009	0.0003	
		Cd: 0.076	0.001	
		Cr: 0.01	0.003	
		Cu: 0.93	0.003	
		Hg: 0.013	0.0013	
		Ni: 0.01	0.006	
		Pb: 13.4	0.02	
		Zn: 0.41	0.12	
Additives	(mg/Nm ³)	Slaked lime (Ca(OH) ₂): ca. 450 Lignite coke powder: ca. 100	none	none
Waste product				
generation	(kg/t sinter)	Ca. 1	n/a	n/a
reuse		recycled to sinter plant	to BOF briquetter	n/a
Type of fabric		Nomex, Rastex	Dacron	non-woven Nomex
Type of cleaning		pulse jet	reverse air flow	pulse jet
Pressure drop	mm watercolumn	150	75-180	75-100
Investments	(1 x 10 ⁶ Ecu ₁₉₉₆)	Fabric filter: 7.3 (in 1993) Additive dosage device: 1.1	8 (in 1987)	5 (in 1991)
Operational costs	(Ecu ₁₉₉₆ /t sinter)	Energy: 0.11 (in 1996) Other: strongly depending on type and operation time of fabric	not separated from other operational costs	n/a

n/a not available

* at Inland Steel, East Chicago, USA a fabric filter is operated since 1978 (PWS, 1987). The input of oily mill-scale is limited to < 3 kg/t sinter in order to reduce fire hazard in the fabric filter. A de-oiling facility was operated for over one year to eliminate a large stock pile of oily scale. The process used solvent to reduce the oil content of the scale from 1-2% to 0.1%.

** at the sinter plant of WCI, Youngstown Sinter Company, Warren, Ohio, USA a fabric filter was commissioned in 1991. Some problems have occurred with fire in the filter bags.

Table 4.9 : Operational data and economics of fabric filters at sinter plants – based on [InfoMil, 1997; Weiss, 1996]

EP.3 Cyclone

Description: Cyclones separate out particulate matter by centrifugal action. As they rely on inertia, cyclones are only efficient abatement devices when the particulate matter is relatively coarse. A multicyclone applies the same principle by means of a parallel series of cyclones, thus obtaining greater efficiency. Cyclones are sometimes used in sinter plants as an intermediate gas cleaning device in order to protect equipment (e.g. ducts and fans) from the abrasive effect of the particulate matter present in the waste gas.

Main emission level achieved: For particles $> 10 \mu\text{m}$ [InfoMil, 1997] reports a removal efficiency of 90-95% achieved by use of a multicyclone. However, because of the relatively small size of particulate matter in sinter plant waste gas, 60 to 80% removal efficiency is assumed here. Thus, outlet concentrations from sinter plants are between 300 and 600 mg/Nm^3 , depending on the inlet concentration and the particle size distribution.

Applicability: Applicable both at new and at existing plants

Cross-media effects: A slight pressure drop (0.5 kPa) increases the energy consumption of the waste gas suction pumps by approximately 200 kW for a sinter plant with a waste gas flow of 1 MNm^3/h and a production of 4 Mt sinter per year. This amounts to 0.001 GJ/t sinter, or less than 0.1% of the sinter plant energy consumption. Furthermore, a solid waste flow is generated, which is recycled into the sinter process completely.

Reference plant: Many sinter plants in the world use cyclones as a coarse particulate matter abatement device. For example: Hoogovens IJmuiden, NL-IJmuiden; Wakamatsu/Yawata Works, Nippon Steel Corporation, Japan.

Operational data: Operates smoothly under abrasive and moist conditions, reducing particulate matter concentrations at sinter plants with an efficiency of approximately 60 to 80%, depending on the specific weight of the dust. At Hoogovens IJmuiden, an outlet concentration of 300 mg/Nm^3 was achieved with a multicyclone.

Economics: Investments are estimated at 500 to 750 Ecu_{1996} per 1000 Nm^3/h [InfoMil, 1997]. For a sinter plant with a waste gas flow of 1 MNm^3/h , this amounts to 0.5 to 0.75 million Ecu_{1996} . Operational costs depend on the pressure drop and thus the energy costs. A figure of 0.007 to 0.015 Ecu_{1996} per 1000 Nm^3 treated is mentioned. For the sinter plant mentioned, this would amount to 60000 to 127000 Ecu_{1996} per year, corresponding to 0.02 to 0.04 $\text{Ecu}_{1996}/\text{t}$ sinter.

Reference literature: [InfoMil, 1997]

EP.4 Fine wet scrubber, e.g AIRFINE

Description: In a scrubber, the particulate matter in the waste gas is abated by using a liquid to arrest the particulates. The contaminated liquid is removed and (usually) recycled after treatment.

Due to the high concentration of hydrocarbons and the relatively fine particulate matter in sinter plant waste gas, traditional scrubbers (e.g. venturi-scrubbers, spray column scrubbers) are usually unable to significantly reduce particulate matter concentrations. Traditional scrubbers are not applied at sinter plants in Europe.

Recently, a new type of scrubber has been developed: the high performance scrubber, manufactured under the name AIRFINE.

The main components of the gas cleaning system (Figure 4.22) include:

- An electrostatic precipitator (ESP) for the removal of coarse dust;
- A system for waste gas cooling and moisture saturation;
- A fine scrubber system for fine dust separation and simultaneous gas cleaning;
- A water treatment facility for by-product separation and recovery.

The heart of this process is the fine scrubber system, where dual flow nozzles eject water and compressed air as high pressurised mist jets into the cooled waste gas stream.

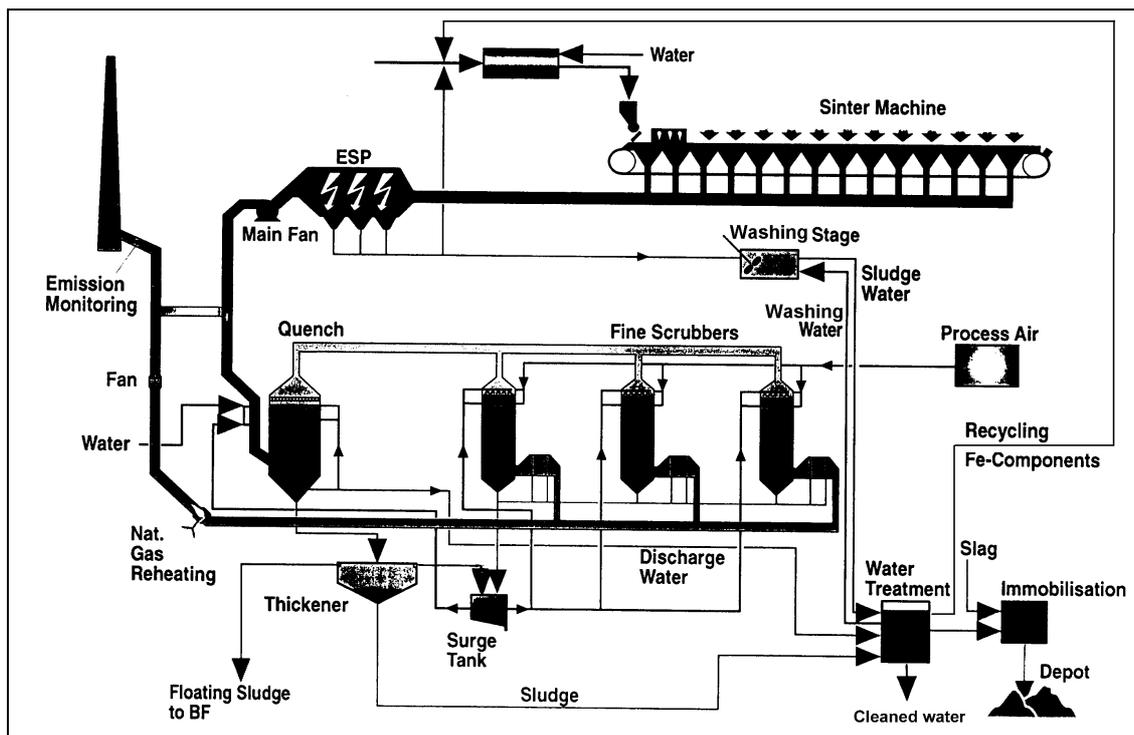


Figure 4.22 : Treatment of the waste gas from the sinter plant at Voest-Alpine Stahl AG, A-Linz with the fine scrubber system

The AIRFINE scrubber allows simultaneous removal of the finest dust particles (including alkali and heavy-metal chlorides – see 4.2.2.1.2.2) and noxious waste gas components. The latter (PCDD/F, heavy metals, PAH) are mainly associated with the fine dust. Compared with dry abatement systems this system can also remove water soluble compounds, such as alkali chlorides (see 4.2.2.1.2.2) and heavy metal chlorides (see 4.2.2.1.2.3). In case of addition of alkalines to the scrubbing water also acidic components like HF, HCl and SO₂ can be removed

significantly. The aqueous solution from the scrubber containing alkali and heavy metal salts is consequently treated by precipitation/flocculation (Figure 4.23). The solids are deactivated with slag followed by disposal to secure landfill. The over-flow is neutralised and passed through several gravel beds before discharge to the municipal sewage system.

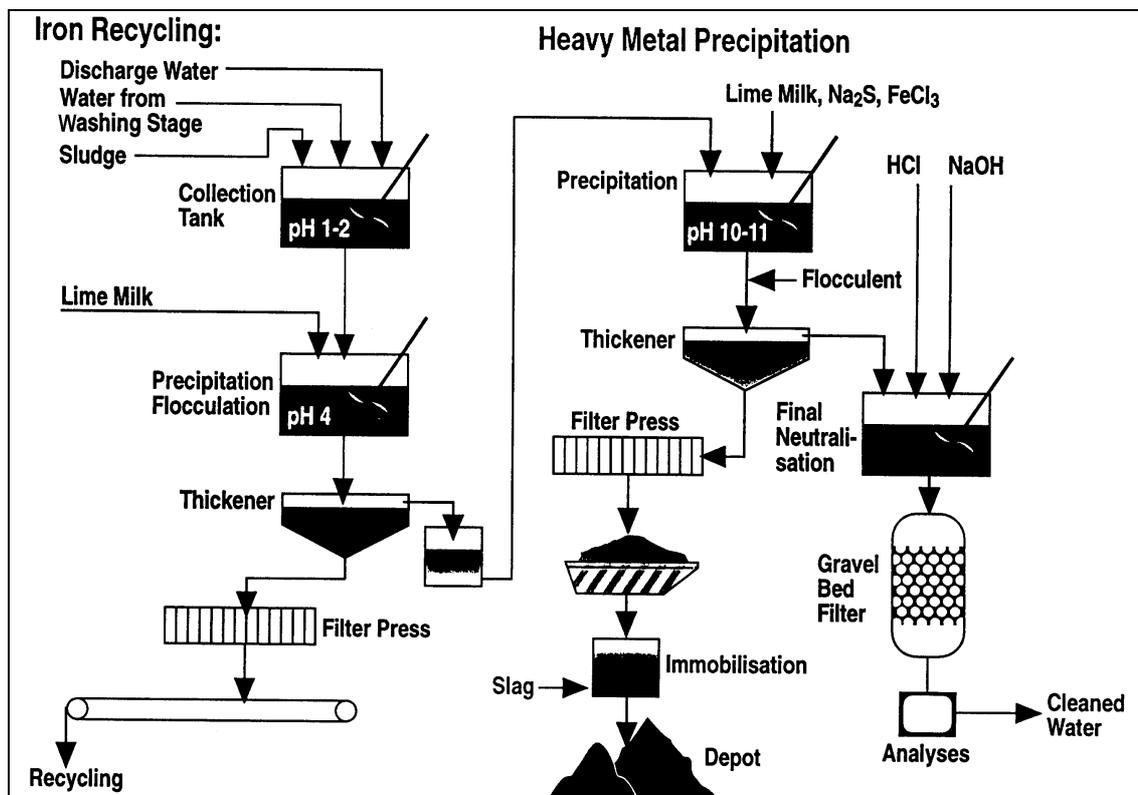


Figure 4.23 : Treatment of the washing water from the fine scrubber and of the water from ESP filter dust extraction at Voest-Alpine Stahl AG, A-Linz

The filter dust from ESP is extracted with recycled water (Figure 4.22) in order to remove the water soluble alkali and heavy metal salts. The extracted filter dust is fully recycled to the sinter strand.

Main emission levels achieved:

One of the two commercial plants is operated at Voest-Alpine Stahl AG, A-Linz, has been in operation since 1993 and the second one at Hoogovens IJmuiden, NL-IJmuiden since spring 1998. A third one is planned to be erected at BHP-ISD, Whyalla Australia. The achieved emission concentrations and emission factors are shown in Table 4.10. The emissions of particulate matter are $< 50 \text{ mg/Nm}^3$, corresponding to a particulate matter removal efficiency of about 95%. The actual removal efficiency is related to the energy input (compressed air for atomisation). Currently, electricity consumption is 39 MJ/t sinter and 79 MJ/t for re-heating the waste gas before discharge. Note that coarse dust separation is achieved by a pre-installed ESP with an outlet concentration of approximately $150\text{-}250 \text{ mg/Nm}^3$. The reported pressure drop is 8-15 mbar.

The technique is also efficient at removing PCDD/F. Under normal operating conditions, $0.4 \text{ ng I-TEQ/Nm}^3$ is guaranteed and achievable, corresponding to approximately 95% efficiency for an inlet of 8 ng I-TEQ/Nm^3 . No additives (activated carbon) are added to achieve this value. Furthermore, 80 to 95% of HCl and HF are removed. SO_2 emissions can be removed with high efficiency if facilities with injection of additives are installed (see EP.5 and point 8 of BAT conclusion – Minimisation of SO_2 emissions). Heavy metal concentrations are also reduced efficiently ($> 90\%$) as a result of their water solubility. The technique gives levels of efficiency

significantly higher than those achieved with ESP (see Table 4.1). Also non-polar pollutants such as PAH are reduced because of the efficient removal of fine dust.

Component	Achieved emission concentration [mg/Nm ³]	Achieved emission factor with 2200 Nm ³ /t [g/ t sinter]	Removal efficiency [%]	Measuring method
Particulate matter	48	110	95.2	VDI 2066 B13
As	< 0.001	< 0.002	87	VDI 3868
Cd	0.003	0.0067	92	VDI 3868
Co	< 0.001	< 0.002	95	VDI 3868
Cr	0.002	0.0045	92	VDI 3868
Cu	0.02	0.044	96	VDI 3868
Hg	0.01	0.02	96	VDI 3868
Mn	0.01	0.02	93	VDI 3868
Ni	< 0.001	< 0.002	95	VDI 3868
Pb	0.05	0.1	96	VDI 3868
Sb	< 0.001	< 0.002	87	VDI 3868
Se	0.001	0.002	90	VDI 3868
Tl	0.002	0.0045	93	VDI 3868
V	< 0.001	< 0.002	87	VDI 3868
Zn	0.001	0.002	94	VDI 3868
HCl	26.5	60	96	VDI 3480 B11
HF	0.6	1.3	95	VDI 2470 B11
SO ₂	370	820	10	Infrared
VOC (FID-measurement)	11	25	50-60	Instruction UBA EM-K1
PAH (EPA 16)	50 µg/Nm ³	110 mg/t	n/a	EPA-Method
PCDD/F	0.4 ng I-TEQ/Nm ³	1.0 µg I-TEQ/t	94	VDI 3499 B11

n/a = not available

Table 4.10 : Achieved emission concentrations and emission factors of the AIRFINE system at Voest-Alpine Stahl AG, A-Linz

The waste-water emission concentrations and emission factors achieved by fine scrubber and ESP filter dust extraction after treatment (precipitation/flocculation) are given in Table 4.11.

Component	Achieved concentration [mg/l]	Achieved emission factor [mg/t sinter]	Removal efficiency [%]	Measuring method
Treated waste water flow	14.2 m ³ /h	0.064 m ³ /t sinter	-	induction
pH	8.8 – 9.3			DIN 384
Suspended solids	8	0.5	99.6	DIN 38409-H2
Al	0.28	0.018	98	DIN 38406
As	0.001	0.00006	99	DIN 38406
Cd	0.002	0.00013	99	DIN 38406
Cr	0.009	0.0006	95.5	DIN 38406
Cu	0.062	0.004	99	DIN 38406
Fe	0.23	0.014	99.9	DIN 38406
Hg	0.001	0.00009	99.8	DIN 38406
Ni	0.048	0.003	99	DIN 38406
Pb	0.006	0.0004	99.9	DIN 38406
Zn	0.026	0.0016	99	DIN 38406
Chloride	4770	310	-	DIN 38405-D1
Sulphate	2420	160	20	DIN 38405-D5
Fluoride	6.7	0.43	81-90	DIN 38405-D4
Sulfide	0.06	0.004	-	DIN 38405-D26
CN-volatile	0.02	0.0013	-	DIN 38405-D13
Ammonium-N	110	7.1	-	DIN 38406-E5
Nitrate-N	17	1.1	-	DIN 38405-D9
Nitrite-N	0.64	0.04	-	DIN 38405-D19
TOC	17	1.1	-	DIN 38409-H3

Table 4.11 : Achieved emission concentrations and emission factors of the treated waste water from waste gas scrubbing and ESP filter dust extraction at Voest-Alpine Stahl AG, A-Linz (mean values)

Applicability: Applicable both new and existing plants. However, it should be noted that the applicability is site-specific.

Cross-media effects: A contaminated waste water flow is generated, which must be treated. The sludge from waste water treatment has to be disposed of in secure landfill.

At Hoogovens IJmuiden, NL-IJmuiden the flow of cooling water is 800 m³/h and of wastewater 50 m³/h.

The fine scrubber system has a relatively large energy demand. This is for the scrubber itself and for the re-heating of the treated waste gas before discharge.

Reference plant: Sinter plant of Voest Alpine Stahl Linz AG, A-Linz in operation since 1993; sinter plant of Hoogovens IJmuiden, NL-IJmuiden in operation since May 1998; the latter plant also includes desulphurisation.

Economics:

Voest-Alpine Stahl AG, A-Linz:

Investments: 39.2 million Ecu₁₉₉₆ (total investment, including installation, engineering etc. These relatively high costs result from the fact that the first plant was a pilot installation.)

Operational data: The costs of energy consumables are also important. Furthermore, one worker is required per shift for wastewater treatment and maintenance (1.5% of investment). The operating costs, excluding energy costs, are 280000 Ecu₁₉₉₇ per annum.

Hoogovens IJmuiden, NL-IJmuiden:

Investments: ca. 40 million Ecu₁₉₉₇, which also includes infrastructure investments.

Operational data: In operation since May 1998; the waste gas from three sinter strands (in total 550000 m³/h) is treated.

Actual costs will differ widely dependent upon the plant and the local requirements for waste water treatment services.

Operational data: At the sinter plant of Voest Alpine Stahl AG, A-Linz a high performance scrubber was commissioned in August 1993 under the name AIRFINE. The scrubber is installed after a coarse dust separator (ESP) and treats 600000 Nm³/h at a sinter production of 270 t/h. Today the system offers high reliability and efficiency.

The energy demand is approximately 2 MW for air compression and scrubber pumps. The additional extra electrical energy demand equates to about 39 MJ/t sinter, representing an increase of about 35%. Furthermore, 650 Nm³/h (79 MJ/t sinter) of natural gas is used for re-heating the gases after the scrubber. The re-heating depends on local regulations and is not specific for this technique.

The iron containing sludge is recycled into the sinter plant. The heavy metal contaminated sludge is conditioned and subsequently disposed of (0.5 t dry matter per day).

The cooling water flow (for the process air compressor station) is 200-300 m³/h, depending on operating conditions.

Reference literature: [Gebert, 1995]

EP.5 Wet Desulphurisation

Description: There are wet and dry desulphurisation processes but only a wet system is considered here. After cooling of the waste gases, SO₂ is absorbed in a spray tower with a solution containing calcium (Ca) or magnesium (Mg). This forms gypsum (CaSO₄) or magnesium sulphate (MgSO₄), which is removed from the column as a slurry. Several reaction agents may be used:

- Steel slag (Steel Slag Desulphurisation (SSD)-process). Steel slag, which contains 30-40% CaO is pulverised, mixed with water and added as a Ca(OH)₂-containing slurry;
- Slaked lime (Ca(OH)₂);
- CAL. Calcium chloride (CaCl₂) and slaked lime (Ca(OH)₂);
- Slaked lime (Ca(OH)₂) and chalk (CaCO₃);
- Magnesium hydroxide (Mg(OH)₂).

The gypsum is de-watered. The quality of the gypsum strongly depends on the efficiency of the pre-installed particulate matter abatement device. In some countries the gypsum can be sold to the cement manufacturing industry.

Water is separated from the gypsum slurry and most of it is recirculated. Because of chloride build-up, there will be a scrubbing water bleed. Thus, both a solid (gypsum) and waste water are generated.

When NH₄OH is used as a reaction agent, an NH₄HSO₃-solution is generated. This solution is processed by wet oxidation in the gas treatment plant of the coke oven. Ammonium sulphate ((NH₄)₂SO₄) is produced.

Wet desulphurisation can be also achieved with fine scrubber systems (see EP.4).

Main emission levels achieved: Desulphurisation efficiencies of >90% can be easily achieved. Efficiencies of 95-99% have been reported. Furthermore, HCl, HF and particulate matter are scrubbed from the waste gas. The system does not remove NO_x [InfoMil, 1997].

Applicability: Can be applied both at new and existing plants although the requirements of space may be of significance.

Cross-media effects: A slurry containing gypsum is generated. The disposal of the gypsum can present considerable problems. In countries where there is no market for gypsum the de-watered material has to be disposed of as an active waste, requiring further stabilisation before dumping and thereby attracting higher disposal costs. Even where there is a market for gypsum the contamination of the product with particulate matter makes it less attractive and it may have to be dumped. Furthermore, additional wastewater treatment is necessary when the unit removes particulate matter as well.

In one case, the gypsum is separated in a contaminated part (10%), which is disposed, and a "clean" part (90%), which is sold. This is the case at Thyssen Stahl AG, Duisburg [InfoMil, 1997]. The design of the desulphurisation unit determines the possibility to separation of gypsum flow.

Most of the scrubbing water can be recirculated, especially when a particulate matter abatement device is operated prior to the desulphurisation unit. A bleed is usually needed to avoid the build up of chloride. Bleeds in the range of 5 to 20 m³/h are reported. However, in some cases no bleed occurs, and water is supplemented.

Electricity consumption is estimated at 6.1-7.2 MJ/t sinter. Furthermore, the cleaned waste gas has to be reheated before emission in order to achieve a sufficient plume rise. In some cases, coke oven gas is used for this purpose and sometimes the CO present in the sinter plant waste gas is catalytically combusted.

Slaked lime consumption depends on the SO₂ inlet concentration, the amount of waste gas treated and the desired efficiency. Reported consumption is 3-4 kg slaked lime (Ca(OH)₂) per m³ SO₂ at Wakamatsu Works, NSC, Japan.

Reference plants: In Table 4.12 a number of sinter plants is given, which apply wet desulphurisation. It should be mentioned that the Mitsubishi-process operated at Thyssen Stahl AG is the only desulphurisation plant treating sinter plant waste gas in Europe. The Thyssen plant treats a selected portion of the waste gas flow (the part with a high SO₂ concentration (see Figure 4.8) and is only operated when local air quality standards are breached. It is reported that this plant has not been operated in the last two years (1997 and 1998) making this method of SO₂ reduction very expensive.

Company	Works	Plant no.	Amount of gas treated (design) (1 x 10 ⁶ Nm ³ /h)	Reaction agent
Nippon Steel Corporation	Wakamatsu/Yawata Sakai		1	slaked lime
			1	SSD (steel slag)
Nippon Kokan KK	Fukuyama Keihin		0.76	NH ₄ OH
			1.2-1.3	NH ₄ OH
Kawasaki Steel Corporation	Chiba	sinter plant no.4	0.62-0.75	slaked lime
		Chiba	0.32	slaked lime
		Mizushima	0.75; 0.75; 0.9	slaked lime
Kobe Steel Ltd.	Kakogawa Kobe		1	CAL*
			0.35	CAL*
Sumitomo Metal Industries	Wakayama Kashima Kokura	sinter plant no.5	0.4	Slaked lime and chalk
		2 sinter plants	0.88; 0.2	Slaked lime and chalk
			0.32	Magnesium hydroxide
Thyssen Stahl AG	Duisburg	sinter plant no. 4	0.32	Slaked lime and chalk

* CAL: Calcium chloride and slaked lime

Table 4.12 : Wet desulphurisation units at sinter plants – [InfoMil, 1997]

Operational data and economics: Table 4.13 gives an overview of operational and economic data from five desulphurisation units. All these units have been installed to meet environmental regulations set by the local authorities. No specific operational difficulties have been mentioned.

The desulphurisation efficiency figures quoted in the table do not take plant down time into account and so relate to 24 hour-a-day operation. Thus, in practice, the overall efficiency figures would be considerably lower. As noted above, the sale of the resulting gypsum may not be possible in some European countries, thus leaving a disposal problem.

Additional economic data:

[InfoMil, 1997] reports the following costs for wet desulphurisation equipment:

Investments: 50-80 Ecu₁₉₉₆/(Nm³/h);

Operational: 0.5 to 1.0 Ecu₁₉₉₆ per kg SO₂ removed.

For a sinter plant with a capacity of 4 Mt/a, a waste gas flow of 1 MNm³/h, 8640 operational hours per year and a untreated SO₂ emission of 1200 g/t sinter and 90% desulphurisation efficiency, this would correspond to the following costs:

Investment: 50 to 80 million Ecu₁₉₉₆;

Operational: 0.5 to 1.1 Ecu₁₉₉₆/t sinter.

Because the investments are relatively high (greater than the cost of a new sinter plant) and depend strongly on the waste gas flow, it may be beneficial to restrict desulphurisation to the waste gas sections with the highest sulphur concentration (see Figure 4.8) and concentrate efforts to lowering the sulphur input to the process (see PI. 4).

Reference literature: [InfoMil, 1997]

	Unit	Thyssen Stahl AG Duisburg	Kawasaki Steel Corp. Chiba	Nippon Kokan KK Keihin	Nippon Steel Corp. Wakamatsu/Yawata	Sumitomo Metal Ind. Wakayama
Sinter production	(t/h)					
design		400	290	500	1000	n/a
actual		375	180	n/a	600	185
Total waste gas flow	(10 ⁶ Nm ³ /h)					
design		0.32	0.62-0.75	1.2-1.3	2	n/a
actual		0.24	0.38	n/a	1	0.4
Sinter basicity	(CaO/SiO ₂)	1.75 - 2.1	1.8	1.8	1.92	2.2
Desulphurisation capacity	(10 ⁶ Nm ³ /h)					
design		0.32	0.62-0.75	1.2-1.3	1 (partial deSO _x)	n/a (partial deSO _x)
actual		0.1 (partial deSO _x)	0.38	n/a	0.45-0.51 (partial deSO _x)	0.2 (partial deSO _x)
Pre-installed de-dusting						
Type		3-field dry ESP	dry ESP	dry ESP	cyclone	dry ESP
clean gas	(mg/Nm ³)	130-140	70-80	50	n/a	40
Reaction agent		slaked lime and chalk	slaked lime	NH ₄ OH	slaked lime	slaked lime and chalk (50%:50%)
SO ₂ -inlet concentration	(mg/Nm ³)					
design		1500	880-1170	n/a	1140	1830
actual		n/a	370	560-840	513-684	730-940
SO ₂ -outlet concentration	(mg/Nm ³)					
design		<150	30	n/a	< 110	370
actual		<30	9	3-9	6-29	n/a
Desulphurisation efficiency	(%)					
design		>90	96-98	n/a	90	80
actual		>98	98	>99	95-98	n/a
By-product						
type		gypsum	gypsum	ammonium sulphate	gypsum	gypsum
amount	(kg/t sinter)	0.9	2.3	n/a	1.0-1.34	4.5
processing		90% sold; 10% disposed	sold	sold	sold	mainly disposed
Energy demand	(GJ/t sinter)	n/a	n/a	n/a	0.0061-0.0072 (electricity) (< 0.3% of tot. energy cons.)	n/a
Investment	(10 ⁶ Ecu ₁₉₉₆)	13 (in 1980)	26 (in 1976)	n/a	n/a	19.4 (in 1976)
Operational costs	(Ecu ₁₉₉₆ /t sinter)	0.6 (in 1986)	2.1 (in 1986); without depreciation	n/a	0.016 (in 1986); only maintenance	0.62 (in 1986); without maintenance without depreciation

n/a not available

Table 4.13 : Operational data and economics of sinter plant wet desulphurisation units – [InfoMil, 1997]

EP.6 Regenerative activated carbon (RAC)

Description: Dry desulphurisation techniques are based on adsorption of SO₂ on activated carbon. When the SO₂ laden activated carbon is regenerated, the system is called Regenerated Activated Carbon (RAC). In this case, a high quality, expensive activated carbon type may be used and sulphuric acid (H₂SO₄) is yielded as a by-product. The bed is regenerated either with water or thermally. This technique is applied at municipal waste incinerators, refineries, power plants and one sinter plant.

In some cases, lignite-based activated carbon is used. In this case the SO₂-laden activated carbon is usually incinerated under controlled conditions. This technique is usually only applied for "fine-tuning" down-stream of an existing desulphurisation unit.

The RAC-process allows the removal of several components from the waste gas: SO₂, HCl, HF, mercury (Hg) and (optional) NO_x. The system can be developed as a 1 stage or a 2 stage process. In the 1 stage process, the waste gases are led through a bed of activated carbon. The pollutants are adsorbed by the activated carbon. NO_x removal only occurs when ammonia (NH₃) is injected into the gas stream before the catalyst bed.

In the 2 stage process, the waste gases are led through two beds with activated carbon. Ammonia can be injected before the bed to reduce NO_x emissions.

Main emission levels achieved: Highly efficient desulphurisation is possible (> 95%). The denitrification efficiency can be as high as 80-90%, depending on operating temperature, NH₃-addition and design. These efficiency figures do not take account of plant down time and are for 24 hour-a-day operation. Thus actual efficiency figures would be considerably lower.

Applicability: The RAC-process can be applied as an end-of-pipe technique both at new and existing plants. The process is usually installed for simultaneous removal of several components from the off-gas (e.g. SO₂, HF, HCl, NO_x). The layout of the plant and space requirements are important factors in consideration of this technique.

Cross-media effects: When the RAC-process is applied, the total energy consumption per t sinter increases and a small water flow is generated. In an integrated iron and steel plant, the water flow can be treated in the existing waste water treatment plant if a suitable plant exists or is conveniently situated. If not, additional costs would be incurred for a suitable waste-water treatment. Sulphuric acid is produced as a by-product.

No solid waste is generated, because the activated carbon is regenerated and partially combusted. Electricity consumption is 1200 kW or 8.6 MJ/t sinter (around 0.4% of total sinter energy consumption).

When the non-regenerative process is applied, the total energy consumption per t sinter increases and a contaminated solid waste is yielded, which should be properly treated.

Reference plant: The RAC process has been in operation at Sinter plant No.3, Nagoya Works, Nippon Steel Corporation in Japan, since August 1987.

Operational data: The sinter plant has a capacity of 12000 t sinter per day, with a waste gas flow of 900000 Nm³/h. Sinter basicity was 1.72-2.1 in 1991. The RAC process was designed as two parallel 1 stage absorption towers, each with a capacity of 450000 Nm³/h. Prior to desulphurisation, particulate matter is removed in a cyclone and subsequently an ESP (clean gas concentration: 20-30 mg/Nm³). SO₂ inlet concentrations were 360 mg/Nm³ and the reported desulphurisation efficiency was 97%. This would lead to an outlet concentration of 11 mg/Nm³. Particulate matter outlet concentrations are 15-20 mg/Nm³ (particle size 2-4 µm; 60% carbon). Denitrification is only applied in one of the two absorption towers, by injection of NH₃. NO_x removal efficiency is low at 15% overall (or 30% on the absorption tower with NH₃ injection), depending on NH₃ injection.

The SO₂ laden activated carbon is regenerated by thermal treatment at 380-430°C. The carbon is indirectly heated by a coke oven gas fired heater. Nitrogen (N₂) is admixed (500 m³/h). The water saturated SO₂ rich (15%) gas is led to the sulphuric acid plant of the coke oven gas treatment. The regenerated carbon is screened and fed back to the adsorption beds. The small size particles are used as sinter plant feedstock. These arrangements are specific to the plant considered and should not be considered as generally applicable at other plants. Activated carbon consumption amounts to 150 kg/h. Steam consumption (40 bar) is 600 kg/h and water consumption is 0.08 m³/h.

Special attention has to be paid to the inlet temperature of the waste gas (< 140°C), and to the particulate matter inlet concentration (max. 50 mg/Nm³).

Economics:

Investments (excluding sulphuric acid plant and waste water treatment plant) in 1987 were approximately 21 million Ecu₁₉₉₆. In 1991 Voest Alpine Stahl AG, A-Linz were quoted as being approximately 73 million Ecu₁₉₉₇ for a RAC plant.

Operating costs (excluding maintenance and depreciation) were 0.75 Ecu₁₉₉₆ per t sinter in 1991.

Maintenance costs are estimated at 0.17 Ecu₁₉₉₆ per t sinter (750000 Ecu₁₉₉₆ per year).

Note that these data are for a RAC-installation for the combined removal of SO₂ and NO_x.

Reference literature: [InfoMil, 1997]

EP.7 Selective catalytic reduction (SCR)

Description: In this process, NO_x in the waste gas is catalytically reduced by ammonia (NH_3) or urea to N_2 and H_2O . Vanadium pentoxide (V_2O_5) or tungsten oxide (WO_3) on a titanium oxide (TiO_2) carrier are often used as a catalyst. Other possible catalysts are iron oxide and platinum. Optimal operating temperatures are in the range of 300 to 400°C range.

The SCR can be operated as a high-dust system, a low-dust system and as a clean-gas system; each with its own characteristics. Until now only clean gas systems have been operated at sinter plants.

Special attention should be paid to de-activation of the catalyst, the accumulation of explosive ammonium nitrate (NH_4NO_3), ammonia slip and the formation of corrosive SO_3 .

Usually, the off-gases have to be reheated before entering the SCR device in order to achieve the required operation temperature.

Applicability: Can be applied as an end-of-pipe technique at both new and existing plants. At sinter plants, only "clean gas systems", after de-dusting and desulphurisation, have been commissioned. It is essential that the gas is very clean, $< 40 \text{ mg dust/Nm}^3$, and that it has a minimum temperature of about 300°C. This requires an energy input.

Main emission levels achieved: A NO_x reduction efficiency of more than 90% can be achieved at sinter plants, depending on catalyst type, operating temperature and NH_3 -addition.

Cross media effects: Because SCR is a dry technique, there are no emissions to water. The only waste generated is de-activated catalyst, which can be reprocessed by the manufacturer. This technique implies storage and use of ammonia, but not necessarily as liquid ammonia; urea or ammonia solutions may also be used.

Overall energy consumption increases as the off-gases have to be reheated before coming in contact with the catalyst. This reduces the possibilities for recovery of sensible heat from the sinter process. Furthermore, the SCR-device consumes electrical energy.

Reference plants:

1. Sinter Plant, Kawasaki Steel Corporation, Chiba Works, Japan
2. Sinter Plant, Nippon Kokan, Keihin Works, Japan.
3. Sinter Plants 3 & 4, China Steel, Taiwan

Operational data:

1. Sinter Plant, Kawasaki Steel Corporation, Chiba Works, Japan
An SCR was installed at the Kawasaki Steel Corporation, Chiba Works sinter plant in 1975 to reduce NO_x emissions. The sinter plant has a capacity of 7000 t sinter/day and a waste gas flow of 620000 to 750000 Nm^3/h . The NO_x emissions were reduced from 450 mg/Nm^3 to 37 mg/Nm^3 on a platinum (Pt) catalyst, thus achieving a reduction efficiency of $>90\%$. The SCR is operated as a "clean gas system", which means the SCR is installed after de-dusting and desulphurisation.

The SCR can be operated at 260°C or at 400°C. At 260°C, the catalyst has to be revitalised several times a year by heat treatment. At this temperature, the 90% efficiency is only obtained by adding NH_3 at 1.2 times above the stoichiometric ratio. In this situation, NH_3 slip occurs. After 3 years of operation at 260°C, a CO converter was installed upstream of the SCR device. In the converter, CO in the waste gas is catalytically combusted and heat is generated. The CO converter increases the gas temperature from approximately 400°C to 480°C. NO_x removal of approximately 90% efficiency has been achieved with addition of NH_3 at a stoichiometric ratio of 0.9. No NH_3 slip occurs. The pressure drop over the catalyst bed (1 m) is 140 mm (water column).

2. Sinter Plant, Nippon Kokan, Keihin Works, Japan.

At the sinter plant of Nippon Kokan, Keihin Works, an SCR was installed in 1979. This system is operated as a clean gas system after de-dusting and desulphurisation. The sinter plant has a capacity of 12000 t sinter per day and a waste gas flow of 1.2 to 1.3 million Nm³/h.

The SCR uses lump ore (limonite and gasite ores from Asia) as a catalyst. Used catalyst is screened and used as feedstock in the blast furnace or the sinter plant.

The gases enter the SCR after being preheated at a temperature of 340°C with a NO_x concentration of 410 mg/Nm³. The clean gas concentration is 100-120 mg/Nm³ equivalent to a SCR efficiency of 70-75%.

The pressure drop over the SCR device is 60-70 mm (water column). The heat exchanger has a pressure drop of 80 mm (water column).

NH₃ consumption is 120 g/t sinter. Electricity consumption is 0.014 GJ/t sinter, or <1% of total sinter energy consumption.

The SCR operates without any reported problems.

Economic data:

1. Sinter plant, Kawasaki Steel Corporation, Chiba Works, Japan

Investments: 27.3 x10⁶ Ecu₁₉₉₆ in 1975

Maintenance costs: 0.08 Ecu₁₉₉₆/t sinter in 1992 (200000 Ecu₁₉₉₆/a)

Operational costs: 0.40 Ecu₁₉₉₆/t sinter in 1992 (1000000 Ecu₁₉₉₆/a); including preheating costs

2. Sinter plant, Nippon Kokan, Keihin Works, Japan

Investments: 50 x 10⁶ Ecu₁₉₉₆ in 1979

Maintenance costs: 0.57 Ecu₁₉₉₆/t sinter in 1992 (900000 Ecu₁₉₉₆/a)

Operational costs: 0.75 Ecu₁₉₉₆/t sinter in 1992

[InfoMil, 1997] reports the following costs for SCR:

Investment: 25-45 Ecu₁₉₉₆/(Nm³/h);

Operational: 0.7-0.9 Ecu₁₉₉₆/1000 Nm³ treated.

For a sinter plant with a capacity of 4 Mt/a, a waste gas flow of 1 MNm³/h and 8640 operational hours per year, this would correspond to the following costs:

Investment: 25 to 45 million Ecu₁₉₉₆;

Operational: 1.5 to 2.0 Ecu₁₉₉₆/t sinter.

In 1992 an estimated cost of about 30 million Ecu₁₉₉₇ was quoted for an SCR unit including re-heating system at Hoogovens IJmuiden, NL-IJmuiden.

Decisive cost factors are the use of catalyst, the ammonia consumption and (if applicable) costs for preheating the waste gas.

No sinter plants in Europe currently operates a de-NO_x system of this type.

Reference literature: [InfoMil, 1997]

4.4 Conclusions

In understanding this chapter and its contents, the attention of the reader is drawn back to the preface of this document and in particular the fifth section of the preface: “How to understand and use this document”. The techniques and associated emission and/or consumption levels, or ranges of levels, presented in this chapter have been assessed through an iterative process involving the following steps:

- identification of the key environmental issues for the sector; for sinter plants these are dust, heavy metals, NO_x, SO_x, dioxins and energy efficiency (heat);
- examination of the techniques most relevant to address those key issues;
- identification of the best environmental performance levels, on the basis of the available data in the European Union and world-wide;
- examination of the conditions under which these performance levels were achieved; such as costs, cross-media effects, main driving forces involved in implementation of this techniques;
- selection of the best available techniques (BAT) and the associated emission and/or consumption levels for this sector in a general sense all according to Article 2(11) and Annex IV of the Directive.

Expert judgement by the European IPPC Bureau and the relevant Technical Working Group (TWG) has played a key role in each of these steps and in the way in which the information is presented here.

On the basis of this assessment, techniques, and as far as possible emission and consumption levels associated with the use of BAT, are presented in this chapter that are considered to be appropriate to the sector as a whole and in many cases reflect current performance of some installations within the sector. Where emission or consumption levels “associated with best available techniques” are presented, this is to be understood as meaning that those levels represent the environmental performance that could be anticipated as a result of the application, in this sector, of the techniques described, bearing in mind the balance of costs and advantages inherent within the definition of BAT. However, they are neither emission nor consumption limit values and should not be understood as such. In some cases it may be technically possible to achieve better emission or consumption levels but due to the costs involved or cross media considerations, they are not considered to be appropriate as BAT for the sector as a whole. However, such levels may be considered to be justified in more specific cases where there are special driving forces.

The emission and consumption levels associated with the use of BAT have to be seen together with any specified reference conditions (e.g. averaging periods).

The concept of “levels associated with BAT” described above is to be distinguished from the term “achievable level” used elsewhere in this document. Where a level is described as “achievable” using a particular technique or combination of techniques, this should be understood to mean that the level may be expected to be achieved over a substantial period of time in a well maintained and operated installation or process using those techniques.

Where available, data concerning costs have been given together with the description of the techniques presented in the previous chapter. These give a rough indication about the magnitude of costs involved. However, the actual cost of applying a technique will depend strongly on the specific situation regarding, for example, taxes, fees, and the technical characteristics of the installation concerned. It is not possible to evaluate such site-specific factors fully in this document. In the absence of data concerning costs, conclusions on economic viability of techniques are drawn from observations on existing installations.

It is intended that the general "BAT" in this chapter could be used to judge the current performance of an existing installation or to judge a proposal for a new installation and thereby assist in the determination of appropriate "BAT" based conditions for that installation. It is foreseen that new installations could be designed to perform at or even better than the general "BAT" levels presented here. It is also considered that many existing installations could reasonably be expected, over time, to move towards the general "BAT" levels or do better.

While the BREFs do not set legally binding standards, they are meant to give information for the guidance of industry, Member States and the public on achievable emission and consumption levels when using specified techniques. The appropriate limit values for any specific case will need to be determined taking into account the objectives of the IPPC Directive and the local considerations.

For sinter plants, the following techniques or combination of techniques are considered as BAT. The order of priority and the selection of the techniques will differ according to local circumstances. Any other technique or combination of techniques achieving the same or better performance or efficiency can also be considered; such techniques may be under development, an emerging technique or a technique which is already available but not mentioned or described in this document.

1. Waste gas de-dusting by application of:
 - Advanced electrostatic precipitation (ESP) (moving electrode ESP, ESP pulse system, high voltage operation of ESP ...) *or*
 - electrostatic precipitation plus fabric filter *or*
 - pre-dedusting (e.g. ESP or cyclones) plus high pressure wet scrubbing system.Using these techniques dust emission concentrations $< 50 \text{ mg/Nm}^3$ are achieved in normal operation. In case of application of a fabric filter, emissions of $10\text{-}20 \text{ mg/Nm}^3$ are achieved.
2. Waste gas recirculation, if sinter quality and productivity are not significantly affected, by applying:
 - recirculation of part of the waste gas from the entire surface of the sinter strand, *or*
 - sectional waste gas recirculation
3. Minimising of PCDD/F emissions, by means of:
 - Application of waste gas recirculation;
 - Treatment of waste gas from sinter strand;
 - use of fine wet scrubbing systems, values $< 0.4 \text{ ng I-TEQ/Nm}^3$ have been achieved.
 - Fabric filtration with addition of lignite coke powder also achieves low PCDD/F emissions ($> 98 \%$ reduction, $0.1 - 0.5 \text{ ng I-TEQ/Nm}^3$. – this range is based on a 6 hours random sample and steady state conditions).
4. Minimisation of heavy metal emissions
 - Use of fine wet scrubbing systems in order to remove water-soluble heavy metal chlorides, especially lead chloride(s) with an efficiency of $> 90\%$ or a bag filter with lime addition;
 - Exclusion of dust from last ESP field from recycling to the sinter strand, dumping it on a secure landfill (watertight sealing, collection and treatment of leachate), possibly after water extraction with subsequent precipitation of heavy metals in order to minimise the quantity to dump.

5. Minimisation of solid waste
 - Recycling of by-products containing iron and carbon from the integrated works, taking into account the oil content of the single by-products (< 0.1 %).
 - For solid wastes generation, the following techniques are considered BAT in descending order of priority:
 - Minimising waste generation
 - Selective recycling back to the sinter process
 - Whenever internal reuse is hampered, external reuse should be aimed at
 - If all reuse is hampered, controlled disposal in combination with the minimisation principle is the only option.
6. Lowering the hydrocarbon content of the sinter feed and avoidance of anthracite as a fuel. Oil contents of the recycled by-products/residues < 0.1% are achievable.
7. Recovery of sensible heat:

Sensible heat can be recovered from the sinter cooler waste gas and it is feasible in some cases to recover it from the sinter grate waste gas. The application of waste gas recirculation can also be considered a form of sensible heat recovery.
8. Minimisation of SO₂ emissions by, for example:
 - Lowering the sulphur input (use of coke breeze with low sulphur content and minimisation of coke breeze consumption, use of iron ore with low sulphur content); with these measures emission concentrations < 500 mg SO₂/Nm³ can be achieved.
 - With wet waste gas desulphurisation, reduction of SO₂ emissions > 98% and SO₂ emission concentrations < 100 mg SO₂/Nm³ are achievable. Due to the high cost wet waste gas desulphurisation should only be required in circumstances where environmental quality standards are not likely to be met.
9. Minimisation of NO_x emissions by, for example:
 - waste gas recirculation
 - waste gas denitrification, applying
 - regenerative activated carbon process
 - selective catalytic reduction

Due to the high cost waste gas denitrification is not applied except in circumstances where environmental quality standards are not likely to be met.
10. Emissions to water (not cooling water)

These are only relevant when rinsing water is used or when wet waste gas treatment system is employed. In these cases, the effluent water to the receiving environment should be treated by heavy metal precipitation, neutralisation and sand filtration. TOC concentrations < 20 mg C/l and heavy metal concentrations < 0.1 mg/l (Cd, Cr, Cu, Hg, Ni, Pb, Zn) are achieved.

When the receiving water is fresh, attention has to be paid to salt content. Cooling water can be recycled.

In principle the techniques mentioned in points 1 - 10 are applicable to both new and existing installations considering the preface.

4.5 Emerging techniques and future developments

In this section, a number of techniques are mentioned which are not yet applied at an industrial scale or which although applied at an industrial scale are not being used in the iron and steel industry. Most of the mentioned techniques are end-of-pipe techniques.

Furthermore, it should be noted that new iron-making techniques may in future strongly reduce the need for sinter plants. These new iron-making techniques are described in chapter 11.

4.5.1 PCDD/F removal

In order to reduce emissions of PCDD/F and other organic micro-pollutants end-of-pipe measures like injection of lignite coke powder with a subsequent bag filter (EP. 2) or fine scrubber with subsequent wastewater treatment (EP. 4) have been described. Another possibility is the application of an adsorption technique with, if necessary subsequent catalytic oxidation of these pollutants. The operators of sinter plants in Germany and Sidmar N.V., B-Gent founded in co-operation with the Verein Deutscher Eisenhüttenleute (VDEh) an association in order to develop techniques to lower PCDD/F emissions from sinter plants. Recently Sollac (USINOR, France) joined this group of steel companies. This association has the goal to develop a process which is secure in operation, needs no water and can be applied to existing plants. On the basis of preliminary experiments and cost calculations the association erected a demonstration plant at Thyssen Krupp Stahl AG, D-Duisburg. It consists of an adsorption stage (entrained flow reactor), an existing ESP and a subsequent oxidative acting catalyst (Figure 4.24). The device to inject lignite coke powder (see EP.2) as an adsorbent into the crude gas main is installed about 27 m upstream of the ESP. A controller allows dosing of the adsorbent between 0 to 300 mg/m³. PCDD/F are essentially present in a gaseous state and are adsorbed by the lignite coke powder. They are then precipitated in the ESP and are recycled to the sinter strand. The remaining PCDD/F are destroyed, reducing them to < 0.1 ng I-TEQ/Nm³, in a honeycomb catalyst located between the fan and stack but values of <0.1 ng I-TEQ/Nm³ after catalytic oxidation have not been confirmed in practice yet. Preliminary tests, without catalytic oxidation, indicated the appropriateness of the process.

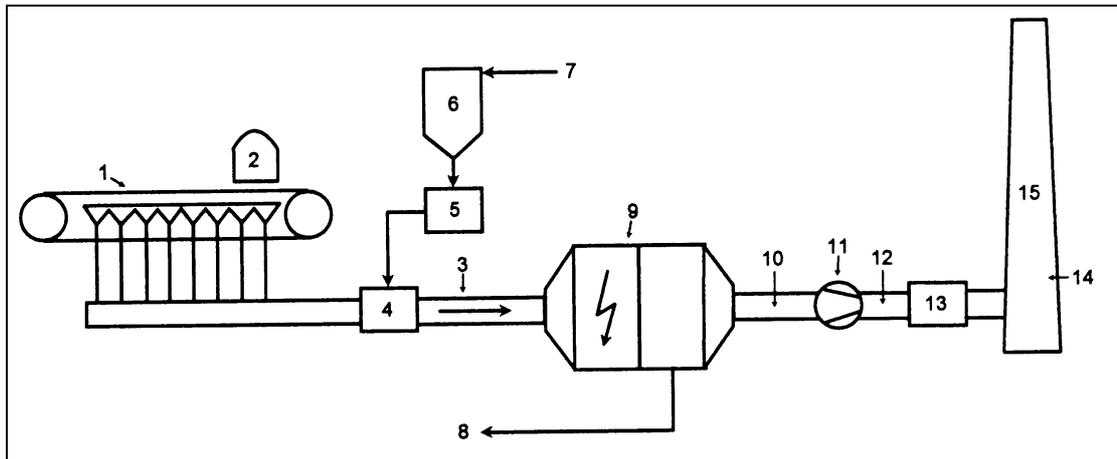


Figure 4.24 : Block diagram of a waste gas treatment system with adsorption stage and catalytic converter - [Kersting, 1997; Philipp, 1988]

- | | |
|---------------------------------------|---|
| 1 sinter strand | 9 electrostatic precipitator |
| 2 ignition furnace | 10 measuring station for dust concentrations and for PCDD/F |
| 3 waste gas main with reactor | 11 fan |
| 4 injection pipes | 12 measuring station for off gas composition |
| 5 dose station with control equipment | 13 oxidation catalytic converter |
| 6 feed tank for adsorbents | 14 measuring station for PCDD/F |
| 7 filling device | 15 stack |
| 8 dust recycling | |

It should be noted that the two systems -injection of lignite coke powder and catalytic oxidation- were tested separately and that not both systems are necessarily needed to achieve values <0.1 ng I-TEQ/Nm³.

5 PELLETISATION PLANTS

As mentioned in 3.2.1, pelletisation and sintering of iron ore are complementary process routes for the preparation of iron oxide raw materials for primary iron and steel making. Each has its own specific advantages and drawbacks. These are highly influenced by local conditions such as the availability and type of raw materials. For various reasons, sinter is practically always produced at the steel works side: it allows solid wastes to be recycled; coke breeze is available at steel works for use as a fuel; sinter is prone to degradation during transport and handling. Pellets are formed from the raw materials – fine ore and additives of < 0.05 mm - into 9-16 mm spheres using very high temperatures and this is mainly carried on at the site of the mine or its shipping port. In the EU 15 there is only one integrated steel works which includes a pelletisation plant (in the Netherlands). Sweden has four stand-alone pelletisation plants.

Pellet production in the five EU plants mentioned above was 15.1 Mt in 1996. In 1995 total pellet consumption in the EU 15 was about 35 Mt whereas sinter consumption was three times higher.

5.1 Applied processes and techniques

Pellets are small crystallized balls of iron ore with a size of 9-16 mm. Figure 5.1 shows the balling drum, which is part of a pelletisation plant.



Figure 5.1 : Balling drum as part of a pelletisation plant

The pelletisation process consists of grinding and drying or de-watering, balling and induration followed by screening and handling (Figure 5.2).

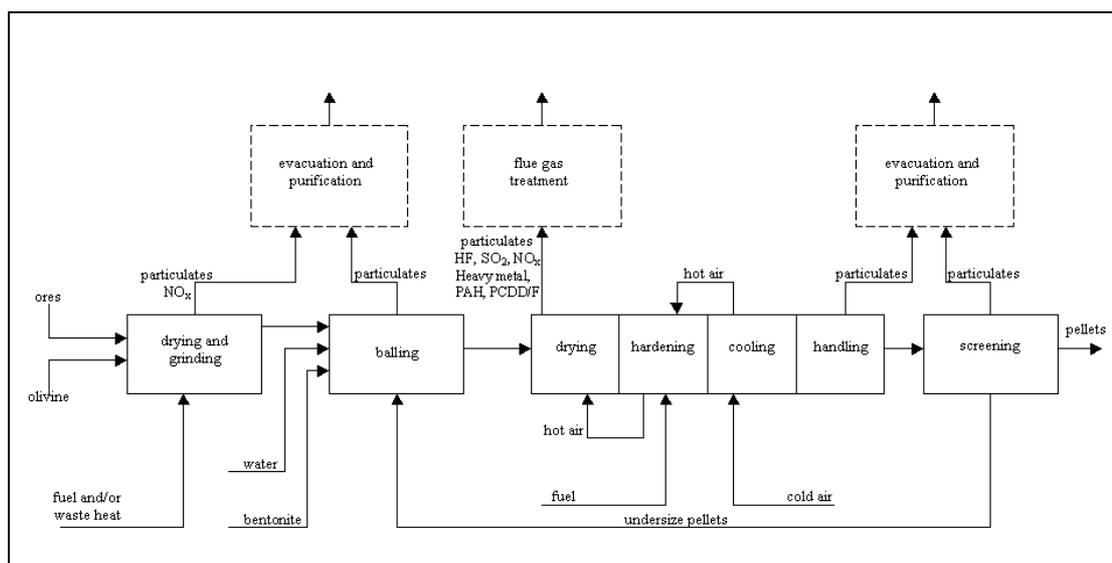


Figure 5.2 : Schematic of a pelletisation plant – [InfoMil, 1997]

5.1.1 Grinding and drying/de-watering

Before being fed into the pelletisation plant the ore is upgraded by several sorting and beneficiation steps with intermediate crushing and grinding. The properties of the ore make magnetic separation the predominant means of concentrating. At the Swedish plants, grinding and concentration are wet processes. In the Dutch plant grinding is carried out at relatively high temperatures (approx. 100°C). In the wet process additives (olivine, dolomite and/or limestone depending on the end product) are ground and then added to the ore slurry, typically at a level of 3 to 3.5%, before de-watering. In the other process after hot grinding the material is re-wetted in paddle-type mixers and combined with additives. In both cases the moisture content is adjusted to 8–9%.

5.1.2 Green ball preparation

De-watered or re-wetted pellet feed is mixed with additives and then processed in the (green) ball preparation plant. This is typically equipped with 4 to 6 balling circuits consisting of a feed bin, balling drum, roller screens and conveyors for circulating the materials. The balling drum is inclined 6 to 8° to the horizontal plane. To obtain a well defined green ball size, typically in the range 9 to 16 mm, under- and oversized fractions are screened off and recirculated.

5.1.3 Induration

Induration, which means thermal treatment, consisting of drying, heating and cooling. It can be carried out in two different systems; in 'straight grate' or 'grate kiln' systems. During thermal treatment magnetite is almost completely oxidised to hematite. This contributes to the large amounts of heat needed to operate the process.

5.1.3.1 The straight grate process

The straight grate process consists of a travelling grate divided into a number of different sections (Figure 5.3).

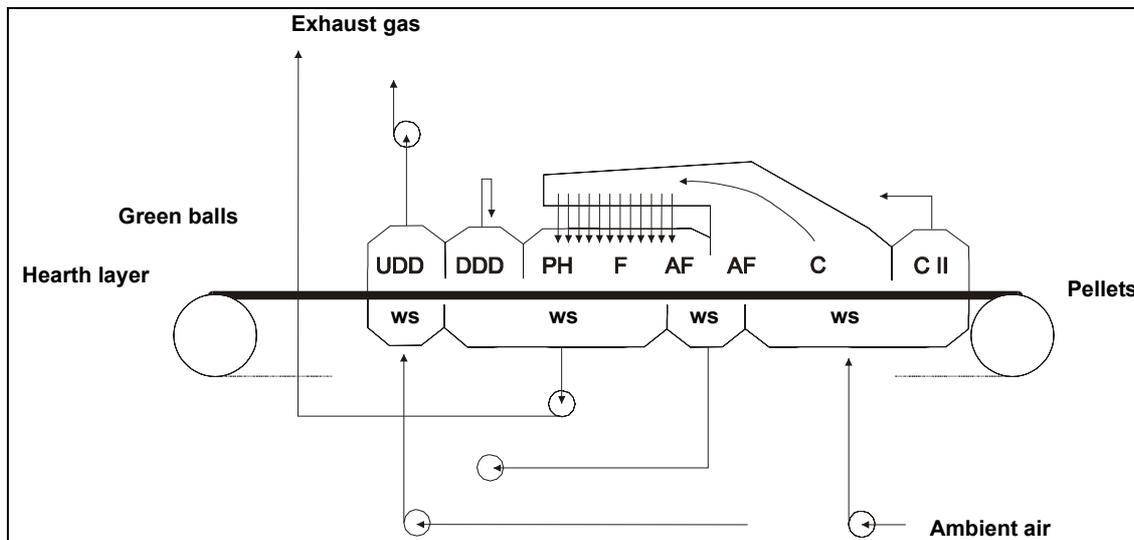


Figure 5.3 : Schematic of the straight grate process

UDD	up-draught drying with hot air from C II
DDD	down-draught drying with hot air from AF
PH	preheating with recuperated hot air from C
F	firing zone oil and hot air from C
AF	after firing sections with hot air from C
C	cooling section using cool (ambient) air
C II	second cooling section using cool (ambient) air

Before the green balls are fed onto the grate bars, the bars are covered with a 5 to 10 cm thick hearth layer of fired pellets. The green balls are then charged on top of the hearth layer to form a gas permeable bed with a total depth of 40-55 cm. The green balls must be heated to approximately 1250 °C during oxidation and sintering to obtain pellets with high strength. This can be achieved by means of a row of burners on each side of the travelling grate, usually fired with oil.

At the end of the induration strand a fraction of the pellets are recycled for use as the hearth layer.

5.1.3.2 The grate kiln process

The grate kiln system consists of three main parts: a travelling grate, a rotary kiln, and a separate annular cooler.

The grate consists of an endless chain of grate plates which are connected with wind-boxes in a gas tight manner. The green balls are fed directly onto the grate plates to create a bed about 20-25 cm deep. After pre-treatment on the grate, the pellets are charged via a chute to the rotary kiln. The kiln has a single oil- or coal-fired burner at the discharge end, and operates at a temperature of approximately 1250 °C. The annular cooler has a grate bottom which permits cool air to penetrate the pellet layer. Walls divide the cooler into sections so as to provide hot air to the earlier process steps.

The overall process can be seen from Figure 5.4 .

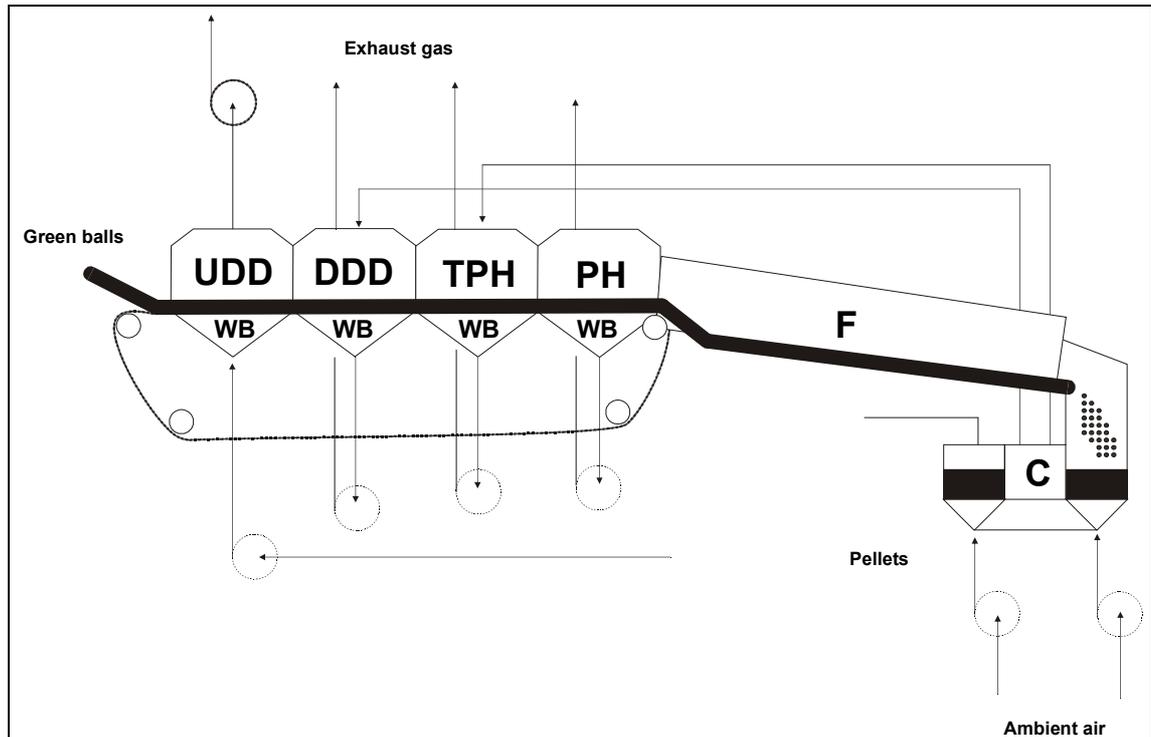


Figure 5.4 : Schematic of the grate kiln process

UDD	up-draught drying with hot air from latter part of the cooler
DDD	down-draught drying with hot air from an intermediate part of the cooler
TPH	tempered preheat with hot air from an intermediate part of the cooler
PH	preheating with hot waste gas from the rotary kiln
F	firing zone using hot air from the first part of the cooler
C	cooling zones using cool (ambient) air

When indurating magnetite, oxidation to hematite and sintering takes place in the TPH, PH, F and C sections of the process (see Figure 5.4).

5.1.4 Screening and handling

At the end of the induration strand, the pellets are collected and screened. Undersize or broken pellets can be recycled. Significant particulate matter emissions may occur.

For the stand-alone plants in Sweden, the indurated pellets are kept in covered product bins before charging to open railway wagons for transportation to the harbours at N-Narvik and S-Luleå. In connection with ship loading 2-3% of the material is screened off as undersize, which is sold as sinter feed. As for all blast furnace material (coke, sinter, pellets and lump ore) a final screening takes place at the blast furnace site.

5.2 Present consumption/emission levels

5.2.1 Mass stream overview and input/output-data

Figure 5.5 gives an overview of the input and output mass streams of a pelletisation plant. This overview may be used for the collection of data from single pelletisation plants.

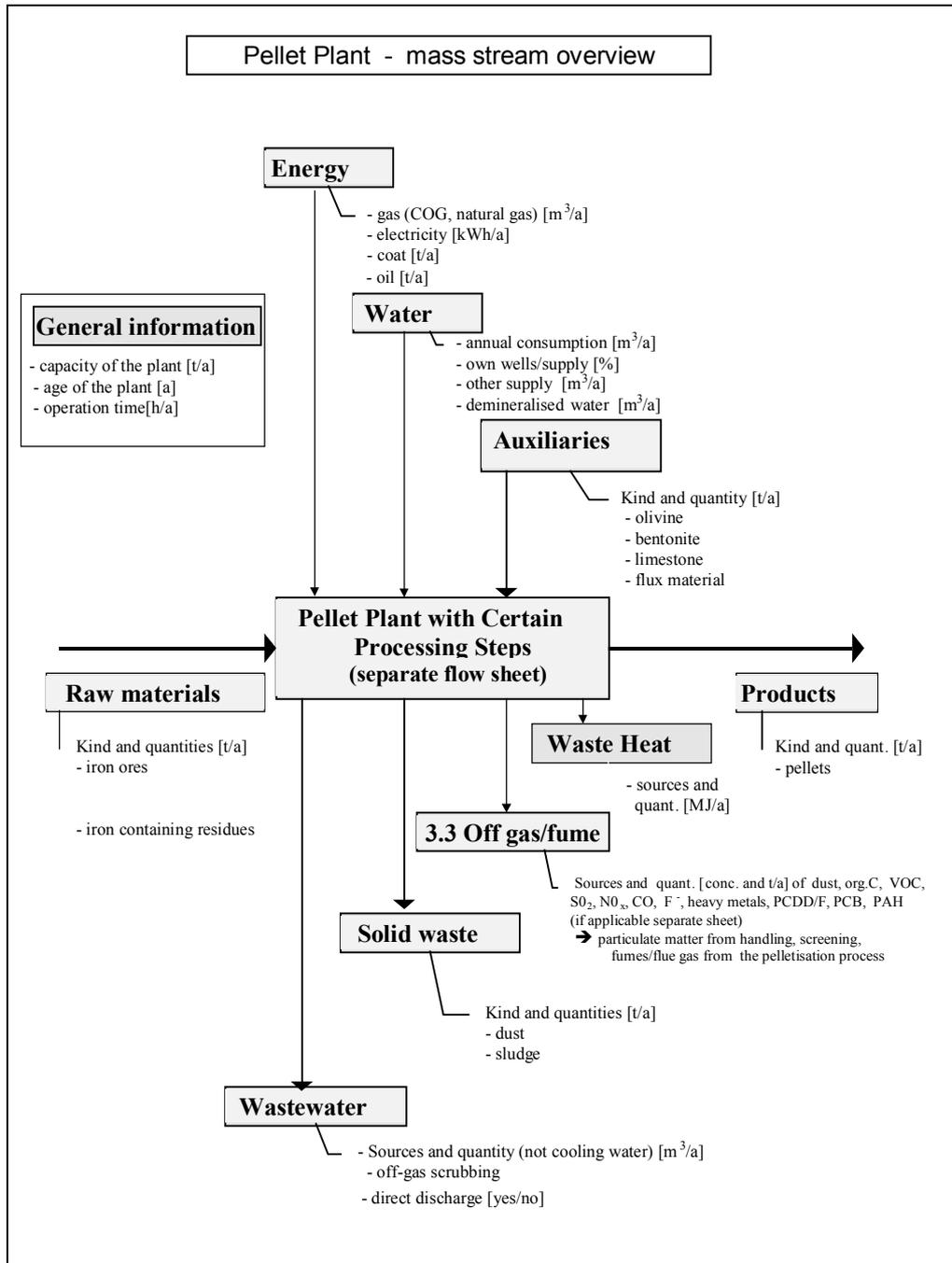


Figure 5.5 : Mass stream overview of a pelletisation plant.

Subsequently, both specific input factors and emission factors can be calculated. Values of these factors at the five pelletisation plants in the EU are shown in Table 5.1 The emission factors are not referred to 1 t liquid steel like for sinter plants, coke oven plants and blast furnaces because the necessary conversion factors are not available. The specific waste gas flow is about 1940 to 2400 Nm³/t pellets.

Input			Output		
Raw materials			Product		
iron ore	kg/t Pel	935 - 1120	pellets	kg/t Pel	1000.00
bentonite	kg/t Pel	5.1 - 7.2			
olivine	kg/t Pel	31 - 35.8	Emissions		
Limestone ^{*1}	kg/t Pel	0 - 3	dust	g/t Pel	20 - 130
dolomite ^{*2}	kg/t Pel	31	Cd	mg/t Pel	0.02 - 0.4
			Cr	mg/t Pel	1 - 4.4
			Cu	mg/t Pel	1.7 - 7.5
			Hg	mg/t Pel	< 0.1 - 0.4
			Mn	mg/t Pel	8 - 38
Energy			Ni	mg/t Pel	5 - 25
COG ^{*3}	MJ/t Pel	398.7	Pb	mg/t Pel	3 - 130
natural gas ^{*3}	MJ/t Pel	209.0	Tl	mg/t Pel	n/a
coke ^{*3}	MJ/t Pel	283.0	V	mg/t Pel	21 - 150
Coal ^{*4}	MJ/t Pel	213 - 269	Zn	mg/t Pel	2.4 - 110
oil ^{*4}	MJ/t Pel	38 - 171	HF ^{*5}	g/t Pel	0.8 - 39
electricity	MJ/t Pel	51 - 128	HCl ^{*5}	g/t Pel	2 - 48
			SO _x ^{*6}	g/t Pel	18 - 250
			NO _x	g/t Pel	120 - 510
Water			CO	g/t Pel	< 10 ^{*4} - 410
			CO ₂	kg/t Pel	15.6 - 31.8
			VOC ^{*7}	g/t Pel	< 5 ^{*4} - 40 ^{*3}
Compressed air			PAH ^{*8}	mg/t Pel	0.19
			PCDD/F ^{*4}	µg I-TEQ/ t Pel	0.0057
			Residues/ by-products		
			Dusts	kg/t Pel	-

Legend: Pel = Pellets; n/a = not available

*1 in case of pellets production for direct reduction

*2 in case of pellets production for blast furnaces

*3 in case of the pelletisation plant is part of an integrated steelworks

*4 in case of stand-alone pelletisation plants in Sweden (magnetite ores)

*5 lower value if techniques for removal of acidic waste gas components are applied

*6 lower value if desulphurization techniques are applied

*7 measurement technique is not known

*8 information whether it is Borneff 6 or EPA 16 or benzo(a)pyrene is not available

*9 sum of EPA 16, calculated from Borneff 6 (EPA 16 = Borneff 6 x 4) with 2100 nm³/t sinter

Table 5.1 : Input/output-data from the five pelletisation plant in the EU 15;

data from 1996 to 1998

the emission data represent the emissions after abatement;

information about the determination of the data like sampling methods, analysis methods, time intervals, computation methods and reference conditions is not available

5.2.2 Information about single emission mass streams

The pelletisation plant is a source of primarily particulate matter and gaseous emissions to air. When abatement techniques are applied to reduce emissions, cross media effects occur. Generally, the following aspects are relevant:

1. Particulate matter emissions from grinding
2. NO_x emissions from induration and drying;
3. Particulate matter and gaseous emissions from the induration strand;
4. SO₂ emissions from induration;
5. HCl and HF emissions from induration
6. Waste water from waste gas treatment (optional);
7. Solid waste from waste gas treatment (optional);
8. Particulate matter emissions from screening and handling;
9. Energy demand.

5.2.2.1 Particulate matter emissions from grinding

Process discharge gas after the air classifier step contains large quantities of particulate matter. This particulate matter consists mainly of iron (Fe) and reflects the composition of the raw material. Emissions can be abated by means of electrostatic precipitation.

5.2.2.2 NO_x emissions from induration and drying

Emitted NO_x are formed during combustion via two mechanisms. The oxidation of nitrogen compounds in the hydrocarbon fuel generates “fuel-NO_x”. The other, and most important mechanism, is the formation of NO_x in the pelletisation process as a result of the high temperatures involved. These cause air-borne nitrogen and oxygen to dissociate and react to form “thermal-NO_x“. The data shown in Table 5.1 reflect the significant differences in NO_x formation.

5.2.2.3 Particulate matter and gaseous emissions from the induration strand

Emissions of gas and particulate matter originate from the firing zone of the induration strand. They are continuous and have to be removed by efficient ESP, bag filter or scrubbing.

5.2.2.4 SO₂ emissions from induration

Sulphur dioxide (SO₂) is formed during the induration process. Emission levels depend on the sulphur content in the ore, additives and the fuel used. Table 5.1 shows that SO₂ emissions are about ten times higher when abatement techniques are not applied.

5.2.2.5 HCl and HF emissions

The pelletisation plant is also a source of hydrofluoric acid (HF) and hydrochloric acid (HCl). These components are formed during induration from fluorine and chlorine containing minerals (apatite) which are present in the ore. Table 5.1 indicates that emissions of HF and HCl are about ten times higher without application of abatement techniques.

5.2.2.6 Wastewater

Whenever a scrubber is used to remove contaminants, a wastewater flow is generated. This wastewater needs a large bleed, due to the presence of HF.

In addition wastewater is discharged from wet rinsing of the plant and equipment. In the plant at Hoogovens IJmuiden, NL-IJmuiden the flow is about 0.04 m³/t pellets. Cooling water may

derive from the grinding and drying section as well as from the induration strand. For the mentioned Dutch plant, the specific flow is 0.16 m³/t pellets and 0.05 m³/t pellets respectively.

5.2.2.7 Solid waste

The pelletisation plant itself is not a primary source of solid waste. However, the sorting and beneficiation of raw materials is a source of solid waste. In addition, solid waste is generated whenever emissions of particulate matter are abated.

5.2.2.8 Energy demand

Table 5.1 shows significant higher energy consumption of the pelletisation plant being part of an integrated steelworks compared to stand-alone plants in Sweden. The main reason is the higher contribution of magnetite oxidation heat in the Swedish stand-alone plants, which use ores from predominately magnetite deposits.

5.3 Techniques to consider in the determination of BAT

This section describes both process-integrated and end-of-pipe techniques for environmental protection and energy saving at pelletisation plants. For each technique a description, the main achieved emission levels, applicability, cross-media effects, reference plants and literature are given. Where possible, this information is complemented with economic and operational data. The list is very short due to the fact that there are few plants in Europe (4 in Sweden and 1 in the Netherlands) and that little information is available about application of environmental techniques at pelletisation plants in general.

Process integrated measures

- PI.1 Recovery of sensible heat from induration strand
- PI.2 Process integrated NO_x abatement

End-of-pipe techniques

- EP.1 Electrostatic precipitation at the grinding mills (dry grinding)
- EP.2 Removal of particulate matter from the drying and induration zone
- EP.3 Gas suspension absorber

PI.1 Recovery of sensible heat from induration strand

Description: The pelletisation plant is designed in such a way that the sensible heat in gas flows in the induration strand is reused efficiently. For example, the hot air from the primary cooling section is used as secondary combustion air in the firing section. In turn, the heat from the firing section is used in the drying section of the induration strand. Heat from the secondary cooling section is also used in the drying section.

At a pelletisation plant which is part of an integrated steelworks, the cooling section generates more sensible heat than can be used in the induration strand. Formerly, this heat was not used, but since the mid 1980s, this sensible heat has been used in the drying chambers of the drying and grinding unit. The hot air is transported through an insulated pipe-line called a "BRAMA-leiding".

The "BRAMA-leiding" transports approximately 150000 m³/h hot air (250°C) from the cooling section of the induration strand to the mills in the drying and grinding section. In the drying section, hot air (600-800°C) is used to dry the concentrates and fines before grinding. By using the hot air from the cooling section, considerably less firing is needed in the drying chamber.

Main emission levels achieved: Gross energy consumption of the abovementioned pelletisation plant is approximately 1.8 GJ/t pellet (compared to stand-alone plants this consumption is significantly higher – see Table 5.1). About 0.7 GJ/t pellet is supplied by means of heat recuperation, whereas approximately 1.1 GJ/t pellet is introduced by means of fuel (see also table 3.2). The "BRAMA-leiding" is included in this calculation and accounts for an energy recovery of approximately 67.5 MJ/t pellet (ca. 4% of gross energy consumption).

Applicability: Recovery of sensible heat is a process-integrated part of pelletisation plants. New plants can be expected to have a more efficient design than existing ones. The "BRAMA-leiding" can be applied at existing plants with a comparable design and sufficient sensible heat supply.

Cross-media effects: The use of hot air from the cooling zone in the firing zone of the induration strand may give rise to higher flame temperatures in the burner(s) and thus to higher NO_x emissions.

Reference plant: Pelletisation plant, Hoogovens IJmuiden, NL-IJmuiden

Operational data: Not available

Economics: Heat recuperation in the induration strand is an integrated part of the plant design and does not require an additional investment. The "BRAMA-leiding" was commissioned in 1984. The estimated investment was 5 Million Ecu₁₉₈₄. Savings in energy costs total 2.8 million Ecu per year.

Reference literature: [InfoMil, 1997]

PI.2 Process-integrated NO_x abatement

Description: The high temperatures in the firing zone of the induration strand generate NO_x. At a pelletisation plant forming part of an integrated steelworks, it was estimated that some 50 to 75% of the total NO_x is formed in the burners ("thermal NO_x"), and some 25 to 50% in the pellet bed. The main factors for thermal NO_x formation are the high temperatures (1300-1400°C) and the high availability of oxygen in the burner zone.

NO_x that is formed in the pellet bed mainly consists of "prompt-NO_x" and "fuel-NO_x", which are formed by reactions between hydrocarbons in the coke breeze, nitrogen in the coke breeze and oxygen and nitrogen in the air. It is not possible to distinguish between the various forms of NO_x after they have formed, there being no difference in chemical composition.

The most important measures that can be taken to reduce overall NO_x emissions from the pelletisation plant aim at reducing formation of thermal NO_x. This can be achieved by lowering the (peak) temperature in the burners and reduction of the oxygen excess in the combustion air.

Main achieved emission level: At the KK3 pelletisation plant of the company LKAB, S-Kiruna, NO_x emissions from the firing zone are as low as 140 g/t pellet (see Table 5.1) or (with 2400 Nm³/t pellets) 58 mg/Nm³. This plant was commissioned in January 1996 and no end-of-pipe techniques are used to reduce NO_x emissions. According to [InfoMil, 1997] the low NO_x emissions were achieved by a combination of low nitrogen content in the fuel (coal and oil) and limiting the oxygen excess. The plant uses two large burners in the induration strand.

The situation at the Hoogovens IJmuiden pelletisation plant is somewhat different. The plant was commissioned in 1970 and uses 56 small burners (of which 44 are operated simultaneously). NO_x emissions from the induration strand are approx. 510-970 g/t pellet or (with 1940 Nm³/t pellets) 260-500 mg/Nm³; the lower value relates to the figure in Table 5.1. Process-integrated measures at this plant are complicated and were tailor made for it.

Applicability: When building a new plant, process-integrated measures to reduce generation of NO_x can be considered at the design stage. In existing plants, process-integrated measures are much harder to achieve and have to be customised to the particular plant.

Cross-media effects: No cross-media effects occur.

Reference plant: KK3 pelletisation plant, LKAB, S-Kiruna.

Operational data: Not available

Economics: Part of plant design in new plants. No data available for existing plants.

Reference literature: [InfoMil, 1997]

EP.1 Electrostatic precipitation at the grinding mills (dry grinding)

Description: After grinding the particles are separated by means of air classification. Coarse particles are recycled to the grinding mills and fine particles are transported to the balling drum, where the green pellets are made. The air used for the air classification contains large amounts of particulate matter. This is basically raw material and has to be captured. The characteristics of the particulate matter are such that emissions are usually abated effectively using electrostatic precipitation. The precipitate can be fed directly to the balling drum as it has the same composition as the raw material (iron ore (62%) and coke breeze (8.5%)) [InfoMil,1997].

Main emission levels achieved: The ESP operates more or less as an integrated part of the air-classification system. Emissions can be $< 50 \text{ mg/Nm}^3$. Modern design ESP's may achieve lower emissions and hence recover raw materials as well.

Applicability: ESP are applicable for existing as well as for new plants.

Cross-media effects: The precipitate from the ESP is used as raw material in the balling mill. The operation of the ESP consumes energy. At a waste gas flow of $300000 \text{ Nm}^3/\text{h}$ and a production of 4 Mt/a, the energy consumption is approximately 0.001 GJ/t pellet.

Reference plant: Pelletisation plant, Hoogovens IJmuiden, NL-IJmuiden.

Operational data: Not available

Economics: No reference data are available. ESP is integrated part of the plant design; part of the raw material from the air classification step is recovered by the ESP. However, it can be calculated that a dry ESP for treatment of a waste gas flow of $300000 \text{ Nm}^3/\text{h}$ will require an investment of approximately 2 million Ecu₁₉₉₆. Operational costs can be calculated as 0.03 to 0.05 Ecu₁₉₉₆/t pellet for a pelletisation plant with an annual production of 4 Mt and a drying mill waste gas flow of $300000 \text{ Nm}^3/\text{h}$.

Reference literature: [InfoMil, 1997]

EP.2 Removal of particulate matter from the drying and induration zone

Description: In pelletisation plants several techniques are used for dust (particulate matter) removal. The main techniques used are mechanical collectors (for coarse particles), multicyclones, wet scrubbers, fabric filters and electrostatic precipitators (ESP). ESP and fabric filters provide high cleaning efficiency but can operate only over a limited range of temperature and moisture conditions. Wet scrubbers are therefore also used in pelletisation plants to remove dust. Scrubbers can be highly efficient dust removers. Scrubbing water is collected, and treated by sedimentation (clarifier). The solid matter is recycled back into the process. Wet scrubbers also have an advantage over dry de-dusting systems that a wider range of pollutants can be captured such as acidic compounds (i.e. the easily water-soluble HCl and some HF), fumes and aerosols.

Main emission levels achieved: Dust removal efficiency for the techniques listed above is normally >95%, and in some cases >99%. Dust emission concentrations of 20 mg/Nm³ or lower are achieved. For example at the latest Swedish plant, KK3 in Kiruna, the dust concentrations in the emitted waste gas from the drying and induration zone are below 10 mg/Nm³ and the total dust emission from the plant as a whole is below 100 g/t pellets (see Table 5.1).

Applicability: The described end-of-pipe techniques described can be applied both at new and existing plants.

Cross-media effects: Normally the collected dust is recycled to the pelletisation process. In case of scrubbers some surplus water has to be discharged after appropriate treatment.

Reference plant: KK3 pelletisation plant, LKAB, S-Kiruna

Operational data: No specific problems are known.

Economics: Not available

Reference literature: [InfoMil, 1997]

EP.3 Gas suspension absorber (GSA)

Description: The Gas Suspension Absorber (GSA) is a semi-dry process. A wet lime slurry is sprayed in the waste gas in a fluidized bed reactor. The water from the lime slurry evaporates and the slaked lime reacts with the impurities (HF, HCl and SO₂). A high concentration of particulate matter in the waste gas enhances the performance of the GSA as it increases the reaction surface, particulate matter is coated with the slaked lime solution.

After the fluidized bed reactor, the waste gases are led through cyclones. Part of the dry product (reacted lime and particulate matter) is removed here and recirculated into the reactor, in order to maintain a high concentration of particulate matter in the reactor. Final cleaning of the waste gas is performed using electrostatic precipitation.

The rate of injection of slaked lime is proportional to the SO₂ concentration in the waste gas.

Main emission levels achieved: The performance data of GSA at a pelletisation plant are summarised in Table 5.2. These data were provided by the Swedish company LKAB, which operates a GSA to treat the waste gas from the induration strand at their KK3 pelletisation plant. Waste gas from the drying zone of the induration strand is led through venturi scrubbers to remove particulate matter before emission to air.

The values in Table 5.1 show the low overall emissions from this plant.

Component	Before GSA (mg/Nm ³)	After GSA (mg/Nm ³)	Removal efficiency (%)
Sulphur oxides (SO ₂)	300	9.6	96.8
Hydrofluoric acid (HF)	78	<0.1	99.9
Hydrochloric acid (HCl)	133	0.6	99.5
Particulate matter	1400	2	99.9

Table 5.2 : Performance data of the Gas Suspension Absorber (GSA) at KK3 pelletisation plant of LKAB, S-Kiruna in 1995 – [InfoMil, 1997]

The following data can be given for this plant:

The plant was commissioned in January 1995 and has an annual production capacity of 4.5 Mt. Actual production in 1995 was 2.8 Mt pellet. The plant is equipped with two burners: one burner in the firing zone of the induration strand and one burner to increase the temperature of the off gas from the second cooling section, which is used as preheating air in the first preheating zone.

Applicability: GSA is an end-of-pipe measure that can be applied both at new and existing plants.

Cross-media effects: The GSA is a semi-dry device, with an electrostatic precipitator (ESP) as the last cleaning stage. The dry precipitate from the ESP contains particulate matter, CaSO₃, CaCl₂, and CaF₂ and some unreacted lime (CaO). The dry precipitate is deposited.

Slaked lime consumption is 264 kg/h (design) and water consumption is 10.6 m³/h (design). Furthermore, pressurised air (7 bar) is used: 690 kg/h (design). Energy consumption is about 0.64 MJ/t pellet (design).

Reference plant: KK3 pelletisation plant, LKAB, S-Kiruna

Operational data: No specific problems are known.

Economics: Not available

Reference literature: [InfoMil, 1997]

5.4 Conclusions

In understanding this chapter and its contents, the attention of the reader is drawn back to the preface of this document and in particular the fifth section of the preface: “How to understand and use this document”. The techniques and associated emission and/or consumption levels, or ranges of levels, presented in this chapter have been assessed through an iterative process involving the following steps:

- identification of the key environmental issues for the sector; for pelletization plants these are dust, SO_x, NO_x, HCl and HF;
- examination of the techniques most relevant to address those key issues;
- identification of the best environmental performance levels, on the basis of the available data in the European Union and world-wide;
- examination of the conditions under which these performance levels were achieved; such as costs, cross-media effects, main driving forces involved in implementation of this techniques;
- selection of the best available techniques (BAT) and the associated emission and/or consumption levels for this sector in a general sense all according to Article 2(11) and Annex IV of the Directive.

Expert judgement by the European IPPC Bureau and the relevant Technical Working Group (TWG) has played a key role in each of these steps and in the way in which the information is presented here.

On the basis of this assessment, techniques, and as far as possible emission and consumption levels associated with the use of BAT, are presented in this chapter that are considered to be appropriate to the sector as a whole and in many cases reflect current performance of some installations within the sector. Where emission or consumption levels “associated with best available techniques” are presented, this is to be understood as meaning that those levels represent the environmental performance that could be anticipated as a result of the application, in this sector, of the techniques described, bearing in mind the balance of costs and advantages inherent within the definition of BAT. However, they are neither emission nor consumption limit values and should not be understood as such. In some cases it may be technically possible to achieve better emission or consumption levels but due to the costs involved or cross media considerations, they are not considered to be appropriate as BAT for the sector as a whole. However, such levels may be considered to be justified in more specific cases where there are special driving forces.

The emission and consumption levels associated with the use of BAT have to be seen together with any specified reference conditions (e.g. averaging periods).

The concept of “levels associated with BAT” described above is to be distinguished from the term “achievable level” used elsewhere in this document. Where a level is described as “achievable” using a particular technique or combination of techniques, this should be understood to mean that the level may be expected to be achieved over a substantial period of time in a well maintained and operated installation or process using those techniques.

Where available, data concerning costs have been given together with the description of the techniques presented in the previous chapter. These give a rough indication about the magnitude of costs involved. However, the actual cost of applying a technique will depend strongly on the specific situation regarding, for example, taxes, fees, and the technical characteristics of the installation concerned. It is not possible to evaluate such site-specific factors fully in this document. In the absence of data concerning costs, conclusions on economic viability of techniques are drawn from observations on existing installations.

It is intended that the general "BAT" in this chapter could be used to judge the current performance of an existing installation or to judge a proposal for a new installation and thereby assist in the determination of appropriate "BAT" based conditions for that installation. It is foreseen that new installations could be designed to perform at or even better than the general "BAT" levels presented here. It is also considered that many existing installations could reasonably be expected, over time, to move towards the general "BAT" levels or do better.

While the BREFs do not set legally binding standards, they are meant to give information for the guidance of industry, Member States and the public on achievable emission and consumption levels when using specified techniques. The appropriate limit values for any specific case will need to be determined taking into account the objectives of the IPPC Directive and the local considerations.

For pelletisation plants, the following techniques or combination of techniques are considered as BAT. The order of priority and the selection of the techniques will differ depending upon the local circumstances. Any other technique or combination of techniques achieving the same or better performance or efficiency can also be considered; such techniques may be under development or an emerging technique or already available but not mentioned/described in this document.

1. Efficient removal of particulate matter, SO₂, HCl and HF from the induration strand waste gas, by means of:
 - Scrubbing *or*
 - Semi-dry desulphurisation and subsequent de-dusting (e.g. gas suspension absorber (GSA)) *or any other device with the same efficiency.*
 Achievable removal efficiency for these compounds are:
 - Particulate matter: >95%; corresponding to achievable concentration of < 10 mg dust/Nm³
 - SO₂: >80%; corresponding to achievable concentration of < 20 mg SO₂/Nm³
 - HF: >95%; corresponding to achievable concentration of < 1 mg HF/Nm³
 - HCl: >95%; corresponding to achievable concentration of < 1 mg HCl/Nm³
2. Emissions to water from scrubbers are minimised by means of water cycle closure, heavy metal precipitation, neutralisation and sand filtration.
3. Process-integrated NO_x abatement;

Plant design should be optimised for recovery of sensible heat and low-NO_x emissions from all firing sections (induration strand and, where applicable drying at the grinding mills).

In one plant, of the grate-kiln type and using magnetite ore, emissions < 150 g NO_x/t pellets are achieved. In other plants (existing or new, of the same or other type, using the same or other raw materials), solutions have to be tailor-made and the possible NO_x emission level might vary from site to site.
4. Minimisation of end-of-pipe NO_x emissions by means of end-of-pipe techniques:

Selective Catalytic Reduction or any other technique with a NO_x reduction efficiency of at least 80%.

Due to high cost waste gas denitrification should only be considered in circumstances where environmental quality standards are otherwise not likely to be met; to date there are no de-NO_x systems in operation at any commercial pelletisation plant.

5. Minimising solid waste/by-products
The following techniques are considered BAT in descending order of priority:
 - Minimising waste generation
 - Effective utilisation (recycling or reuse) of solid wastes/by-products
 - Controlled disposal of unavoidable wastes/by-products.
6. Recovery of sensible heat;
Most pelletisation plants already have a high rate of energy recovery. For further improvements, tailor-made solutions are usually necessary.

In principle the techniques listed in points 1 - 6 are applicable to both new and to existing installations considering the preface.

5.5 Emerging techniques

5.5.1 Process-integrated NO_x abatement at the induration strand

Description: PI.2 already gives an introduction to the formation of NO_x at the pelletisation plant and the possible measures. However, not many of these measures have actually been implemented at pelletisation plants. A number of measures can be identified as emerging techniques. Whether a certain technique is applicable at a certain site depends on a great many factors and cannot be discussed here.

The measures below aim to reduce "thermal NO_x" formation (see PI.2) by lowering the (peak) temperatures in the burner flame(s) or by means of limiting the availability of molecular oxygen (O₂) or molecular nitrogen (N₂). The following measures have been identified:

1. *Water injection into the induration strand burners*
This measure reduces peak flame temperatures. However, energy efficiency decreases as the quantity of water injected increases.
2. *Use of exhaust gases from other processes as combustion air*
This measure aims to reduce availability of oxygen (O₂) in the burners, thus achieving a lower generation of NO_x. The used exhaust air can be obtained from within the pelletisation plant or from elsewhere (e.g. from hot stoves, the sinter plant, coke oven heating in cases where the pelletisation plant is part of an integrated steelworks). This measure may have a significant influence on plant operation.
3. *Indirect water injection in the cooling section*
The generated steam might reduce NO_x formation in the burners, but this is not certain. Furthermore, this measure affects the drying and cooling processes in the induration strand.

Reference literature: [InfoMil, 1997]

5.5.2 Cold bonded pellets/briquettes

Description: Cold bonding has recently been implemented by steelworks in Japan, USA and Sweden. This process achieves ore hardening by non-thermal means using a binder. Cold bonding processes are designed to complement existing sinter and pelletisation processes by agglomerating both coarse and fine ores and recycled material. Up to 15% (but typically 3-5%) of cold bonded pellets/briquettes can be subsequently used in the blast furnace charge.

A number of processes are used for cold-bonding. One process, developed by the Nippon Steel Corporation (Nagoya Works), combines dry dust, iron ore, wet dust, sludge and coke dust with a binder in a ball mill. After 10 days of airing in a stock yard, the pellets are tough enough for use in the blast furnace.

Other processes, such as the NKK-Corac process (NKK Niigata Works), employ a continuous curing process with slag as the binder. This gives a reduced curing time of approximately 10 hours.

Main achieved emission level: Cold bonding techniques do not emit SO_x and NO_x. Cold bonded pellets/briquettes cannot completely replace sinter or pellet agglomerates. This technique could be used for iron ore fines and for recycling material usually sent to the sinter plant.

Status: Commercially applied in Japan, USA and Sweden. However, cold-bonded pellets/briquettes can not fully replace self-fluxed sinter or pellet.

Reference: [EC Sinter/BF, 1995]

5.5.3 Other possible techniques

Although only a few techniques have been applied as end-of-pipe techniques at pelletisation plants, several other commercially available techniques can be expected to be applicable without serious technical problems.

With regard to NO_x emissions, the following available techniques might be applicable at pelletisation plants:

- Selective Catalytic reduction (SCR);
- Shell-denox;
- Degussa H₂O₂;
- Regenerated Activated Carbon (RAC);

With regard to emissions of SO₂, the following commercially available techniques might be applicable at pelletisation plants:

Wet desulphurisation:	MgO regeneration process Double-Alkali process NaOH process KHI system CFB process AIRFINE scrubbing process
Semi-dry desulphurisation:	Fläkt/Niro Fläkt/Drypac Fläkt CDAS Walther
Dry desulphurisation:	Dry alkali injection Regenerated Activated Carbon (RAC)
Desulphurisation with scrubbing liquid	Wellman-Lord process Fläkt-boliden process Degussa H ₂ O ₂ process Solinox

6 COKE OVEN PLANTS

Coal pyrolysis means the heating of coal in an oxidation free atmosphere to produce gases, liquids and a solid residue (char or coke). Coal pyrolysis at high temperature is called carbonisation. In this process the temperature of the flue gases is normally 1150 – 1350 °C indirectly heating the coal up to 1000 – 1100 °C for 14 – 24 hours. This produces blast furnace and foundry cokes. Coke is the primary reducing agent in blast furnaces and can not be wholly replaced by other fuels such as coal. Coke functions both as a support material and as a matrix through which gas circulates in the stock column.

Only certain coals, for example coking or bituminous coals, with the right plastic properties, can be converted to coke and, as with ores, several types may be blended to improve blast furnace productivity, extend coke battery life, etc.

6.1 Applied processes and techniques

By the 1940s the basic design of modern coke ovens had been developed. The ovens were about 12 m long, 4 m high and 0.5 m wide, and were equipped with doors on both sides. The air supply was preheated by the hot exit gas; waste heat recovery enabled higher temperatures and increased coking rates. Since the 1940s, the process has been mechanised and the constructing materials have been improved without significant design modifications. Current assemblies may contain up to 60 ovens as large as 14 m long and 6 m high. Because of heat transfer considerations widths have remained 0.3 – 0.6 m. Each oven in the battery holds up to 30 tonnes of coal. Figure 6.1 shows such a battery. Some recently constructed coke oven plants have increased dimensions yet further. For instance, the chambers of the new coke oven plant “Kaiserstuhl”, which went into operation end of 1992, are 18 m long, 7.6 m high and 0.61 m wide holding approx. 65 t coal [Schönmuth, 1994]



Figure 6.1 : Photograph of a coke oven battery showing the chambers, the coal tower and the coke oven gas collecting main

Developments in recent years have also been particularly aimed at minimising emissions from the processes and at improving working conditions for operators.

The coke making process can be subdivided into:

- coal handling,
- battery operation (coal charging, heating/firing, coking, coke pushing, coke quenching),
- coke handling and preparation,
- collection and treatment of coke oven gas (COG) with recovery of by-products.

In order to make the description of waste-water treatment clearer the water flows in the coking process are also described here

Figure 6.2 shows a simplified scheme of the whole sequence of operations and processes required to produce coke (showing emission sources). The main steps mentioned above are described in more detail below.

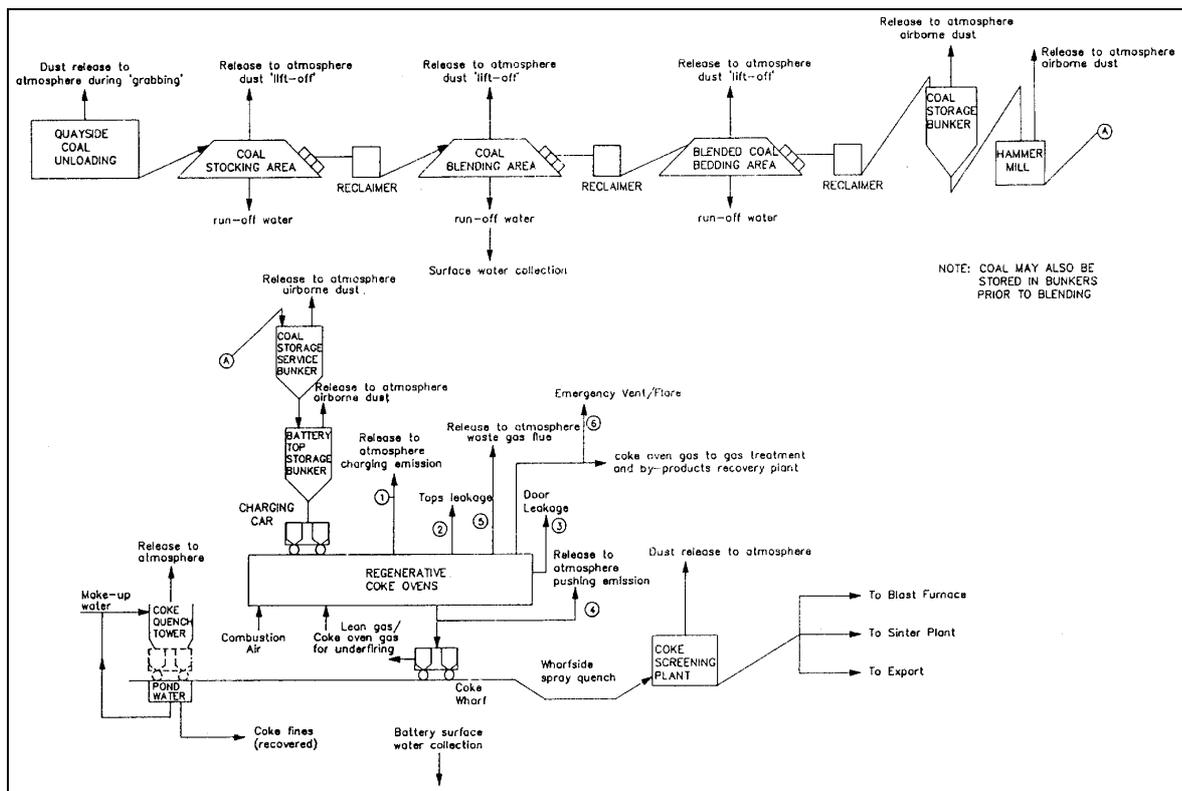


Figure 6.2 : Typical flow diagram of a coke oven plant showing emission sources – [UK Coke, 1995]

6.1.1 Coal handling

Coal handling comprises following steps:

- discharge of coal; the coal is discharged from ships or trains onto a transportation system or for storage. Usually large cranes with grabs are used. Wind may cause coal dust emissions.
- coal storage; coke oven plants are normally associated with large coal stocking areas. Wind may cause coal dust emissions. Attention has to be paid to proper treatment (sedimentation) of run-off water.
- coal transport; coal transport by conveyor, possible transfer points outside buildings and transportation by road has to be taken into consideration.

- coal preparation; coal preparation comprises bed blending, bunker blending and crushing which may lead to dust emissions. During blending recycled substances such as tar may be added which may lead to emissions of volatile compounds.
- Charging of the coal tower (coal dust emissions may occur)
- Charging of the charging car (coal dust emissions may occur)

6.1.2 Coke oven battery operations

The operation of a coke oven battery comprises:

- coal charging
- heating/firing of the chambers
- coking
- coke pushing
- coke quenching

These operations dominate the emissions from a coke oven plant. They are therefore described in more detail here, together with information about emission sources. Figure 6.3 shows the layout of the battery and shows the main emission sources.

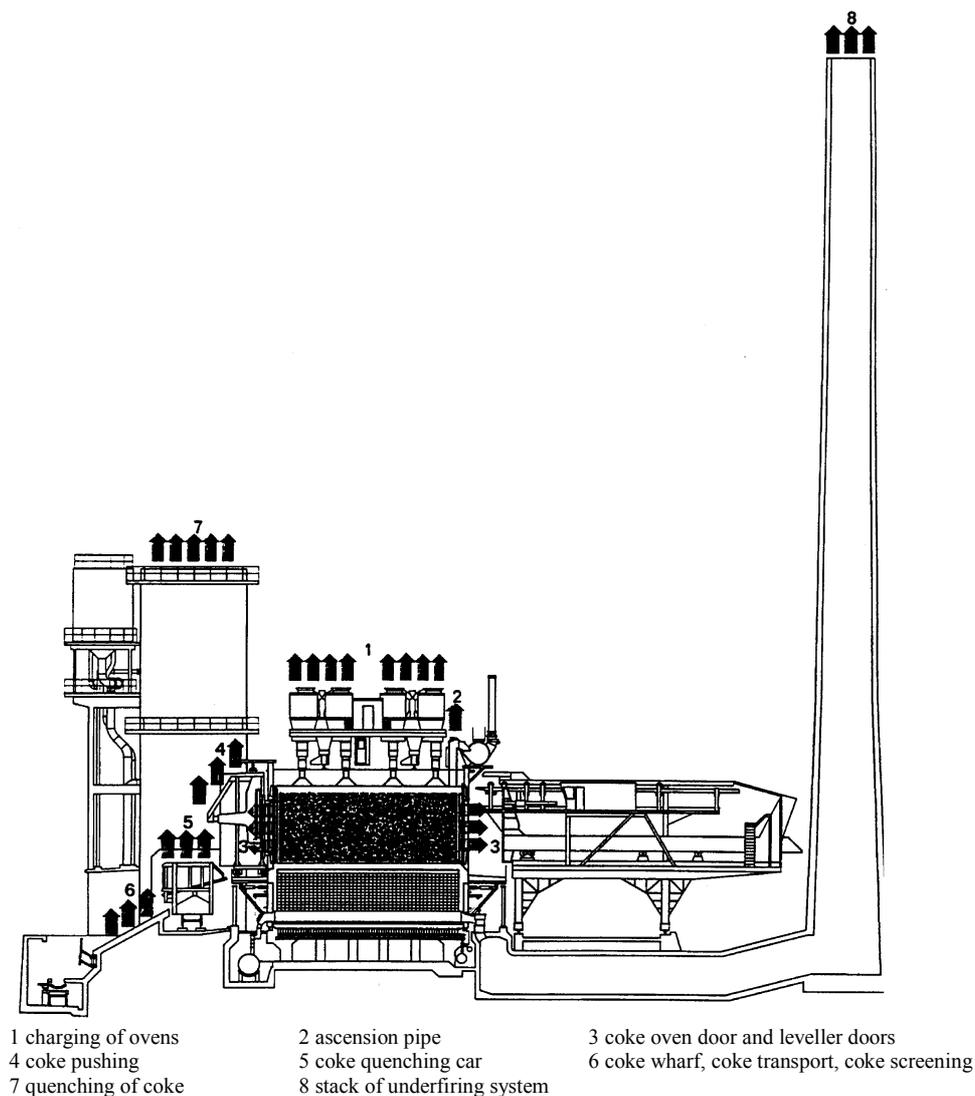


Figure 6.3 : Schematic diagram of a coke oven battery showing the main emission sources

6.1.2.1 Coal charging

There are a number of techniques for charging coke ovens with pulverised coal (70 – 85 % is < 3 mm) through the charging holes. The most common technique is gravity charging by charging cars (Figure 6.4). This can be simultaneous, sequential or stage charging by speed controlled horizontal screw feeders or turntables. Other systems are also possible. Regardless of the system the flow of the coal must be kept under control. General measures are given for all these systems. The aim of these measures is to achieve “smokeless” charging (charging with reduced emissions).

Pipeline charging or stamp charging are rarely applied.

The concept and geometry of the charging holes and systems is a very important element in the reduction of charging emissions.

Within the process a distinction can be made between:

- Emissions during the charging process itself.
- Evacuation and cleaning of the charging gases.
- Leveller door emissions during levelling the coal with the leveller bar.
- Fugitive emissions from material spilled on the oven deck.

One or a combination of several techniques may be used depending on plant configuration.

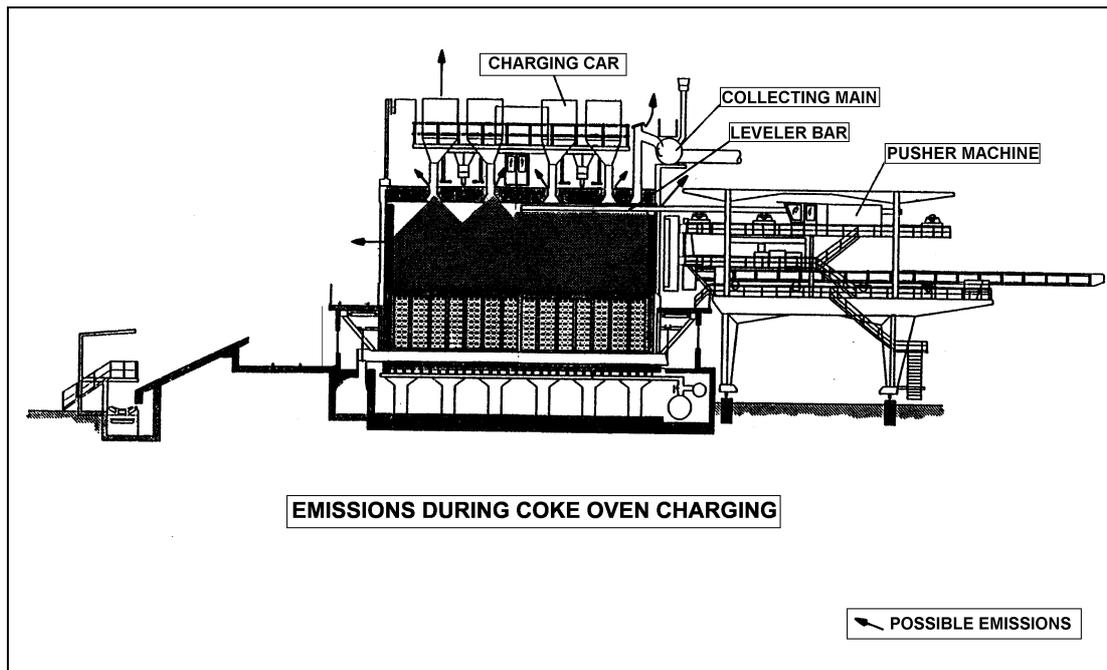


Figure 6.4 : Diagram of coal charging of a coke oven chamber using a charging car showing the emission points (indicated by arrows)

6.1.2.2 Heating/firing of the chambers

The individual coke oven chambers are separated by heating walls. These consist of a certain number of heating flues with nozzles for fuel supply and with one or more air inlet boxes, depending on the height of the coke oven wall. The average nozzle-brick temperature, characterising heating flue operation, is usually set between 1150 and 1350°C. Usually, cleaned

coke oven gas is used as a fuel, but other gases such as (enriched) blast furnace gas can be used as well.

In order to improve energy efficiency, regenerators are located right under the ovens, exchanging heat from flue gases with combustion air or blast furnace gas. Figure 6.5 shows a schematic diagram of a coke oven heating system together with the emission points. If the heating walls are not completely gas-tight because of cracks (which is very often the case), coke oven gas will reach the flue gas and will be emitted with it via the stack.

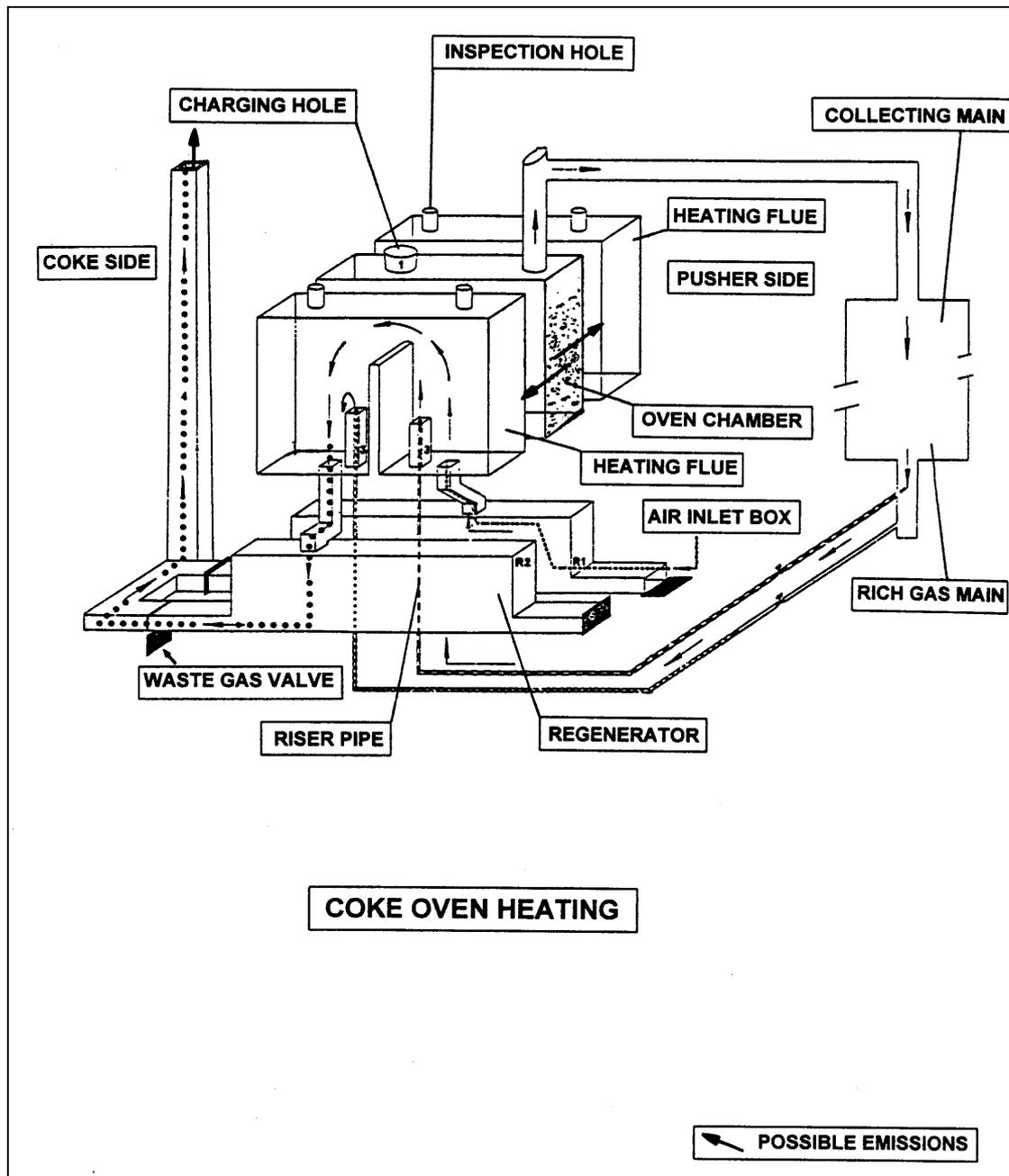


Figure 6.5 : Schematic diagram of a coke oven heating system showing the emission points (marked by arrows);

The shown design is for a single stage combustion whereas today most modern plants have multi-stage combustion; COG used as a fuel is normally desulphurised before.

6.1.2.3 Coking

The carbonisation process starts immediately after coal charging. The gas and moisture driven off account for about 8 - 11% of the charged coal. This crude coke oven gas (COG) is exhausted via ascension pipes into the collecting main. The high calorific content of this gas means that after purification (see COG treatment) it can be used as a fuel (e.g. for battery heating). The coal is heated by the heating/firing system described above and remains in the coke oven until the centre of the coal has reached a temperature of 1000 – 1100 °C.

Depending on oven width and heating conditions the coking process takes around 14 –24 hours to complete. Figure 6.6 shows the main emission sources during coking. These may be door emissions, emissions from charging holes and ascension pipes and also, in case of wall cracks, emissions of COG via the heating gases.

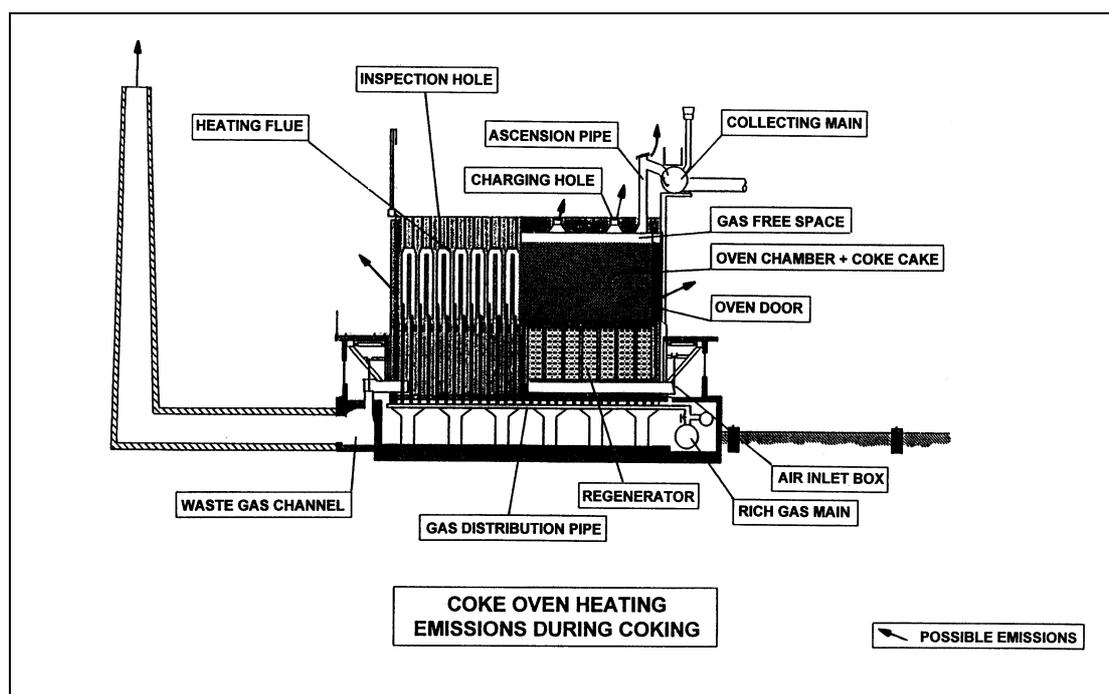


Figure 6.6 : Schematic diagram of a coke oven chamber showing possible emission points during coking (indicated by arrows)

1000 kg of coal yields between 750 to 800 kg coke and approximately 325 m³ COG corresponding to approximately 187 kg COG. However it should be noted that coke yield and COG production and composition depend to a large extent on coal composition and coking time.

6.1.2.4 Coke pushing and quenching

Fully carbonised coke is pushed out of the oven into a container by the ram of the pusher machine (Figure 6.7). Contact with atmospheric oxygen causes the coke to start burning instantaneously. The container used is generally a coke quenching car which transports the hot coke to a quenching tower. Here the coke is quenched directly by large volumes of water. The water fraction that does not evaporate can be collected and used in the next batch, thus preventing wastewater emissions.

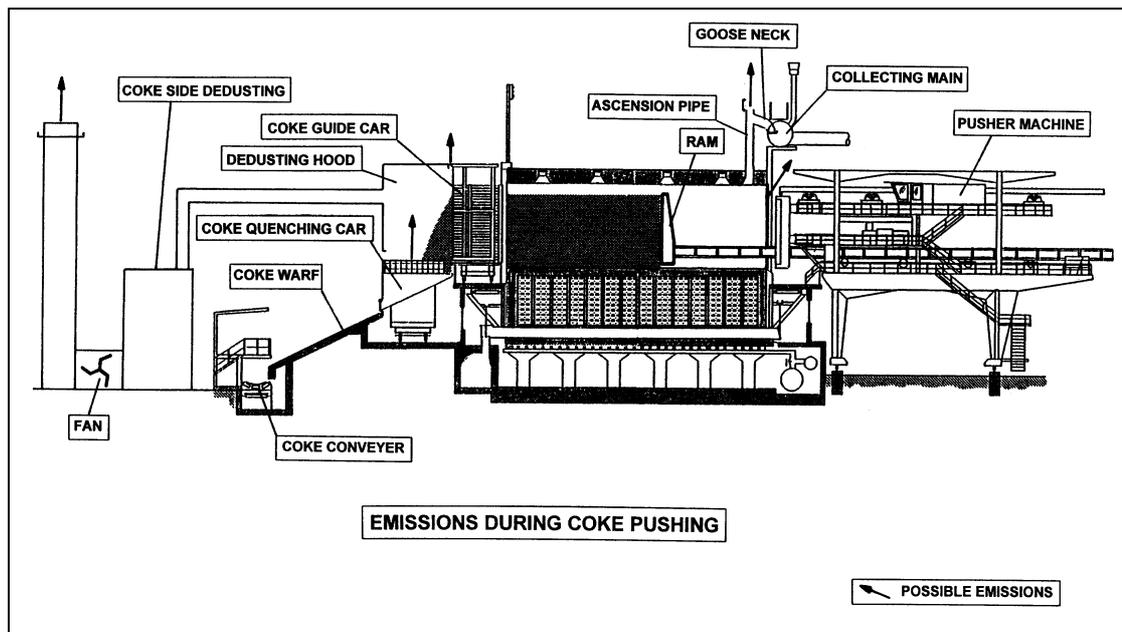


Figure 6.7 : Pushing of carbonised coke from the coke oven into a quenching car. Emission points are shown by arrows.

In an alternative system, known as dry quenching, the quenching car takes the hot coke to a vertical quenching chamber. Inert quenching gas circulates around the chamber, which is isolated from the atmosphere, thus preventing combustion whilst cooling the coke. The gas is cooled by a heat exchanger in order to recover thermal energy.

6.1.2.5 Coke handling and screening

After quenching, the coke is stored in stock piles from which it is transported by conveyors (with transfer points), road, rail or any combination of these. Finally the coke is crushed and screened. The smaller fraction (< 20 mm) is usually set aside for the sinter process, the larger fraction (20-70 mm) is used in the blast furnace.

Coke handling and screening give rise to dust emissions.

6.1.3 Collection and treatment of coke oven gas (COG) with recovery of by-products

Raw COG has a relatively high calorific content due to the presence of hydrogen, methane, carbon monoxide and hydrocarbons. Furthermore, the raw coke oven gas contains valuable products such as tar, light oil (mainly consisting of BTX (benzene, toluene and xylenes)), sulphur and ammonia. Table 6.1. shows the composition of raw coke oven gas. For several reasons the coke oven gas must be treated before use as a fuel.

Raw gas yield [m ³ /h/t coal]	Raw gas density [kg/Nm ³]	H ₂ [vol.%]	CH ₄ [vol.%]	C _x H _y [vol.%]	CO [vol.%]	H ₂ S [vol.%]	BTX [g/Nm ³]	PAH [mg/Nm ³]	NH ₃ [g/Nm ³]	CO ₂ [vol.%]
12 - 25	0.53 - 0.62	39 - 65	32 - 42	3.0 - 8.5	4.0 - 6.5	3 - 4	23 - 30	n/a	6 - 8	2 - 3

Legend: n/a = not available

Table 6.1 : Raw coke oven gas composition – after [InfoMil, 1997]

Coke oven gas composition depends on coking time and coal composition. The data given refer to water- and ash-free coal. The H₂S content of the example given is relatively high. Other plants are typically in the range 3.5- 4.5 g H₂S/Nm³.

Tar and naphthalene in the raw gas may clog the piping and equipment and should be removed first. For each tonne of coke produced, approximately 35 to 45 kg of tar may be recovered. Several products can be recovered from the tar, such as pitch, anthracene oil, wash oil, naphthalene oil, carbolic oil (phenol) and light oil.

Sulphur compounds and ammonia cause corrosion of piping and equipment and the sulphur compounds cause emissions of SO₂ when the coke oven gas is used as a fuel. For each tonne of coke produced, approximately 3 kg of ammonia and 2.5 kg of H₂S are generated.

In some cases light oil, and especially BTX is recovered from the raw coke oven gas as a valuable by-product. Up to 15 kg of light oil may be recovered per tonne of coke produced. This oil contains benzene, toluene, xylenes, non-aromatics, homologous aromatics, phenol, pyridine bases and other organic compounds such as polycyclic aromatic hydrocarbons (PAH).

Figure 6.8 shows a typical COG treatment plant. The main steps in the process are described below.

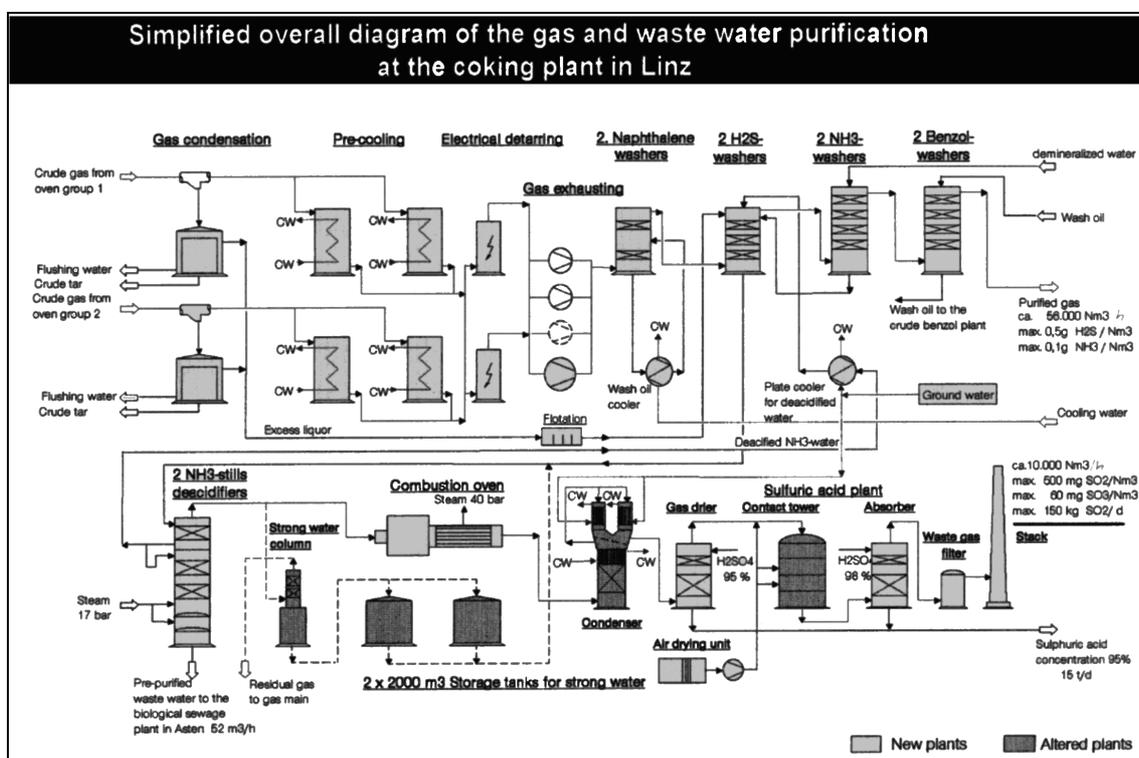


Figure 6.8 : Typical scheme of a COG treatment plant with recovery of by-products – [UK Coke, 1995]

6.1.3.1 Gas cooling

Hot crude oven gas enters the ascension pipes at a temperature of approximately 800°C. In the goose neck it is directly cooled by an ammonia liquor spray to a steam saturation temperature of around 80°C. This requires 2-4 m³ of ammonia liquor for each tonne of coal carbonised.

The liquid phase, i.e. the condensate, is fed to the tar/water separator, whilst the gas phase is led to the primary coolers. It used to be common for primary coolers to be operated as open systems. Nowadays, indirect cooling with closed cooling systems is more frequent. When sufficient cooling water of the right temperature is the gas can be cooled to below 20°C, given a moderate ambient temperature. Under these conditions most of the higher boiling point compounds and the water from the steam fraction of the gas will condense.

Droplets and particles are precipitated in the down-stream electrostatic tar precipitators before the gas is drawn into the washing facilities by exhausters (suction fans). Occasionally electrostatic tar precipitators are installed down-stream of the exhausters. The precipitate from the electrostatic tar precipitator is also led to the tar/water separator.

The suction fans cause compression of the gas, and even if the fans are such that this is only slight, the attendant temperature increase cannot be tolerated in view of the down-stream processing conditions. This makes it essential to use so called final coolers.

Final coolers may be indirect or direct, in which case the cooling water of which is used to absorb the impurities from the COG. Consequently at the end of the cooling cycle, during return-flow cooling using natural draft or fan coolers, emissions are inevitably generated. Closed systems are therefore usually preferred for final cooling, though open cycles are still operated at some plants.

6.1.3.2 Tar recovery from the coke oven gas

Most of the water and the high boiling point hydrocarbons are condensed during coke oven gas cooling. The condensate from the pipes and the electrostatic tar precipitator is led to the tar/water separator, where the tar is recovered. The water phase is separated off as so called "coal water" and led through the ammonia stripper/still prior to further treatment.

Sometimes scrapers are installed to remove tar from the condensate. These chunks are usually fed back to the coal feed.

6.1.3.3 Desulphurisation of coke oven gas

Coke oven gas contains hydrogen sulphide (H₂S) and various organosulphur compounds (carbon disulphide (CS₂), carbon oxisulphide (COS), mercaptans etc.). All desulphurisation techniques currently in use are highly efficient at removing H₂S. They are less efficient at removing organosulphur compounds. Commercial coke oven gas desulphurisation processes can be divided into two categories:

1. processes using wet oxidation to produce elemental sulphur (S⁰);
2. processes which absorb and strip H₂S for subsequent conversion into sulphuric acid (H₂SO₄) or elemental sulphur (S⁰).

All wet oxidation processes utilise a reduction-oxidation catalyst to facilitate the wet oxidation of hydrogen sulphide to elemental sulphur (S⁰) or sulphate. All these processes are characterised by very efficient removal of hydrogen sulphide (as low as 2 mg/Nm³), but have the

disadvantage of producing highly contaminated wastewater and/or air, which make elaborate treatment facilities a necessary part of the process (UN-ECE, 1990).

Absorption/stripping processes are characterised by generally lower H₂S removal (0.5-1 g/Nm³ after cleaning), but, since air is not included in the regenerating system and no toxic catalysts are used, emissions to air and water of process related chemicals are minimised or eliminated. The processes can be operated to produce sulphuric acid (sulphuric acid plants), or a very high-purity elemental sulphur (Claus plants).

A common process combination is NH₃/H₂S circuit scrubbing in the low pressure stage and potassium carbonate scrubbing (vacuum carbonate process) in the high pressure stage, combined with a BTX washer either at low or at enhanced pressure. Potassium scrubbing at both the pressure stages, combined with a BTX washer is also common.

6.1.3.4 Recovery of ammonia from the coke oven gas

The ammonia formed during coking appears in both the coke oven gas and the condensate (weak liquor) from the gas. Typically 20-30% of the ammonia is found in the weak liquor.

Three techniques are applied commercially to remove ammonia from the coke oven gas:

- The NH₃/H₂S scrubbing circuit. In this process, the ammonia is scrubbed from the coke oven gas in an ammonia scrubber using water or dilute liquor wash as a scrubbing liquid. The effluent from the ammonia scrubber is used as a scrubbing liquor in the H₂S scrubber. The effluent from the H₂S scrubber contains H₂S and NH₃ and is led to the ammonia stripper and the still. This process is also known as the Carl Still, Diamex or Ammoniumsulfide Kreislaufwäscher (ASK) process.
- Direct recovery as ammonium sulphate ((NH₄)₂SO₄). Two processes can be used: The Otto-type absorber and the Wilputte low differential controlled crystallisation process. In both systems the COG is sprayed with a dilute sulphuric acid solution and ammonium sulphate is yielded;
- Direct recovery as anhydrous ammonia (NH₃). The recovery of ammonia from the gas as anhydrous ammonia has been developed by the United States Steel Corporation under the name USS PHOSAM. In this process, ammonia is scrubbed from the coke oven gas by counter-current contact with an ammonia-lean phosphate solution (phosphoric acid). Ammonia removal efficiency is 98-99%.(UN-ECE, 1990)

6.1.3.5 Recovery of light oil from coke oven gas

The gas leaving the ammonia absorbers contains light oil, a clear yellow-brown oil with a specific gravity of 0.88. It is a mixture of the products of COG with boiling points mostly between 0 and 200°C, containing well over a hundred constituents. Most of these are present in such low concentrations that their recovery is seldom practicable. The light oil is usually referred to as BTX. The principal usable constituents are benzene (60-80%), toluene (6-17%), xylene (1-7%) and solvent naphtha (0.5-3%). Three main methods are used for the recovery of light oil (UN-ECE, 1990):

- Refrigeration and compression to temperatures below -70°C and pressures of 10 bar.
- Absorption by solid absorbents, in which the light oil is removed from the gas by passing the latter through a bed of activated carbon and recovering the light oil from the carbon by heating with indirect or direct steam.
- Absorption by solvents, consisting of washing the COG with a petroleum wash oil, a coal tar fraction or other absorbent, followed by steam distillation of the enriched absorbent to recover the light oil.

The ammonia liquor storage tank provides water for the goose neck spray equipment. The ascension pipe lids are sealed. The surplus water in the ammonia liquor storage tank is led to the ammonia stripper/still.

It should be noted that usually all water flows, except for water from closed cooling systems and wet oxidative desulphurisation systems, are eventually drained from the ammonia still and led to a wastewater treatment plant.

High concentrations of NH_3 are present in the ammonia still. There are several reasons for decreasing the ammonia concentration before discharging the water to a wastewater treatment plant or to the environment:

- the ammonia can be recovered as a valuable energy source (in a sulphuric acid plant) or as a valuable by-product (as ammonium sulphate or anhydrous ammonia).
- free ammonia is highly toxic for aquatic ecosystems (including biological wastewater treatment plants);
- ammonia has a very high specific oxygen demand (it requires 4.5 times its own weight of oxygen for oxidation to nitrates). Thus, there is a risk for oxygen depletion of the wastewater treatment plant or the recipient water.

This has led to ammonia strippers being installed in virtually all coke oven plants. This device strips H_2S and NH_3 from the liquid by steam and alkaline additives. The vapours are subsequently led to the crude gas or to the $\text{NH}_3/\text{H}_2\text{S}$ scrubbing circuit (to improve H_2S scrubbing efficiency) or to a sulphuric acid plant, where NH_3 and H_2S are incinerated together. Sometimes the NH_3 is removed from these vapours in saturators, producing ammonium sulphate.

The most commonly used alkali is caustic soda (NaOH). Formerly, slaked lime (CaOH_2) was often used.

Some possible water flows were not indicated in Figure 6.9:

1. Wastewater from BTX recovery is led to the tar/water separator.
2. Phenol (concentration > 3 g/l) may be recovered from the coal water by a solvent extraction process, before the coal water is led to the ammonia liquor storage tank.
3. Wastewater from oxidative desulphurisation processes is usually discharged separately after pre-treatment.
4. Chemical water from the (optional) sulphuric acid plant is usually led to the still.
5. Chemical water from the (optional) Claus-process will usually not be condensed but discharged to the atmosphere via a stack [EC Coke, 1996]. An alternative is to inject this water into the raw gas before treatment.
6. Cooling water. Indirect gas cooling water is recirculated and will not influence the wastewater quantity. In the case of direct gas cooling, the cooling water has to be considered to be a washing liquor and it is eventually drained via the still.

6.2 Present consumption/emission levels

6.2.1 Mass stream overview and input/output data

Figure 6.10 shows an overview of the input and output mass streams of a coke oven treatment plant. This overview might be used for the collection of data from single coke oven plants.

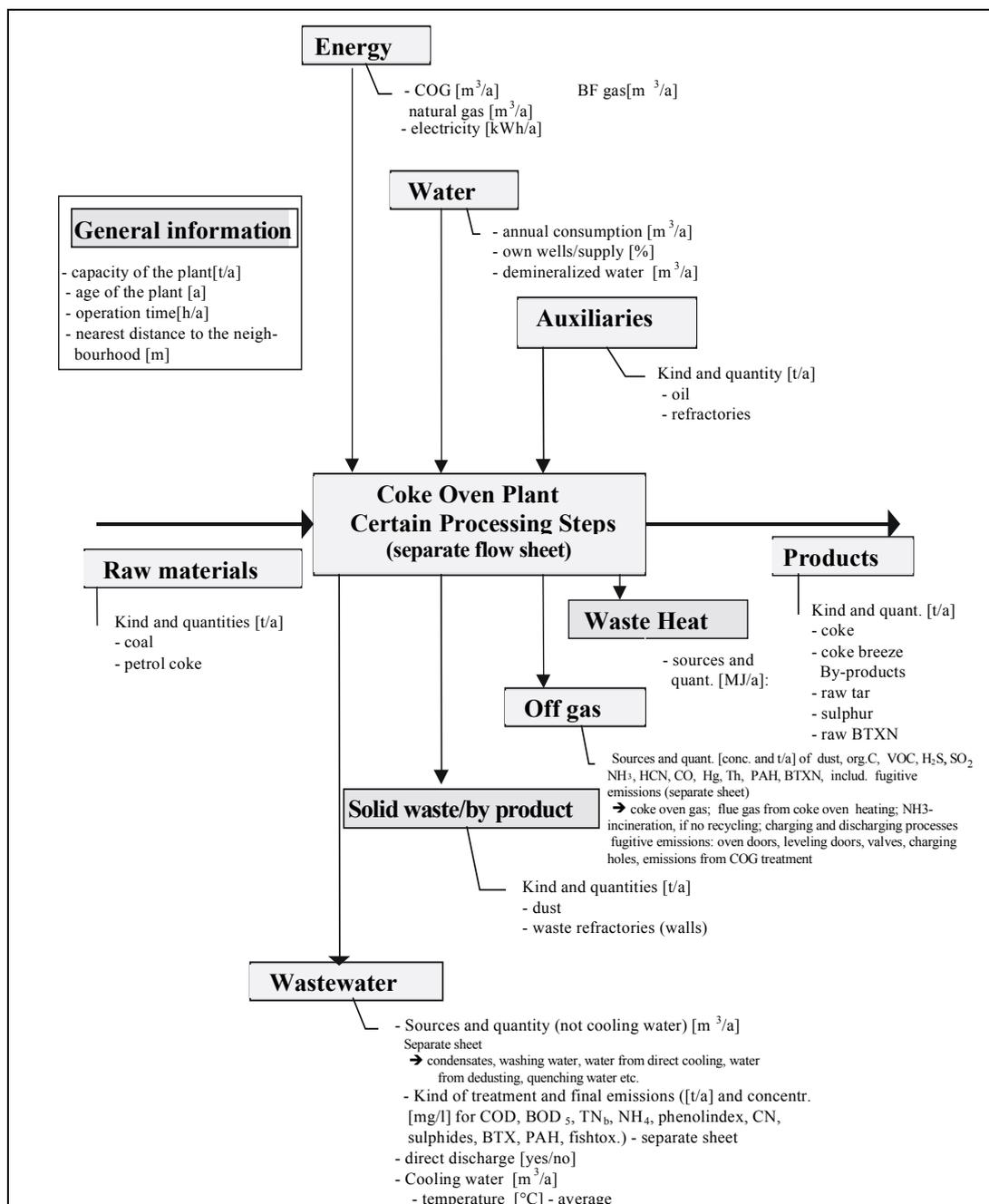


Figure 6.10 : Mass stream overview of a coke oven plant

Subsequently specific input factors as well as specific emission factors can be calculated. Factors of this kind are presented in Table 6.2. The data given refer to 11 coke oven plants in four EU member states.

The emission factors refer to 1 tonne liquid steel in order to make it easier to sum emission factors from various production units.

Input			Output		
Raw materials			Products		
Coal (dry)	kg/t coke	1250 - 1350	Coke (dry)	kg/t	1000.0
			COG ^{*3}	MJ/t LS	2500 - 3200
			steam ^{*3}	MJ/t LS	3 - 90/500 ^{*4}
Energy					
BFgas + COG	MJ/t coke	3200 – 3900	Gas. Emissions		
Electricity	MJ/t coke	20 – 170 ^{*1}	Dust	g/t LS ^{*3}	17 - 75 ^{*1}
			SO _x	g/t LS	27 - 950 ^{*5}
			NO _x	g/t LS	230 - 600 ^{*1}
			NH ₃	g/t LS	0.8 - 3.4
			H ₂ SO ₄	g/t LS	0.7 ^{*6}
Steam	MJ/t coke	60 - 300 ^{*2}	HCN	g/t LS	0.02 - 0.4
			H ₂ S	g/t LS	4 - 20 ^{*1}
			CO	g/t LS	130 - 1500 ^{*1}
Compressed air	Nm ³ /t coke	7 – 15	CO ₂	kg/t LS	175- 200
			CH ₄	g/t LS	27 ^{*7}
			VOC ^{*8}	g/t LS	4 - 8
Process water (≠ cooling water)	m ³ /t coke	0.8 - 10	Benzene	g/t LS	0.3 - 15 ^{*9}
			PAH ^{*10}	mg/t LS	170 - 500 ^{*11}
			Residues/ by-products		
			Benzene	kg/t coke	8 - 15
			H ₂ SO ₄ ^{*6}	kg/t coke	4 - 9
			Tar	kg/t coke	25 - 46
			(NH ₄) ₂ SO ₄ as SO ₄ ²⁻ ^{*12}	kg/t coke	1.7 - 3.4
			Sulphur ^{*13}	kg/t coke	1.5 - 2.3
			Wastewater	m ³ /t coke	0.3 - 0.4

Legend: LS = liquid steel (crude steel); VOC = volatile organic carbon; PAH = polycyclic aromatic hydrocarbons

^{*1} high value for old plants (older than 20 years)

^{*2} old plants (older than 20 years) can have a steam consumption of 1200 MJ/t coke

^{*3} used conversion factors (weighted average of all European blast furnaces and basic oxygen steelworks): 358 kg coke/t pig iron; 940 kg pig iron/t LS;

^{*4} high values in case of coke dry quenching (with heat recovery in form of steam); value of 90 MJ/tLS from two plants which are in operation since 14 and 15 years respectively; value 500 MJ/tLS from a plant which is in operation since about 7 years (see PI.7).

^{*5} high value in case of non-desulphurized COG; the specific SO₂ emissions are 27 – 300 g/t LS in case of desulphurization (the higher value of this range indicate insufficient desulphurization)

^{*6} in case of SO₂ absorption to H₂SO₄ as final by-product

^{*7} only data from one plant available

^{*8} VOC without methane

^{*9} low value for one modern plant (about 5 years old)

^{*10} PAH as EPA 16 (sum of following 16 PAH:

(Σ (FLU+PYR+TRI+CPP+BaA+CHR+BNT+BeP+BbF+BkF+BaP+DbahaA+BghiP+INP+ANT+COR))

^{*11} only data from two plant available

^{*12} in case of SO₂ absorption with ammonia liquor

^{*13} in case of SO₂ absorption and sulphur recovery as elemental sulphur

Table 6.2 : Input/output-data from eleven coke oven plants in four different EU Member States;

Detailed data from other plants concerned are not available; data from 1996;

the emission data represent the emissions after abatement, they do not comprise emissions from coal and coke handling because of missing information but these emissions are of less significance;

information about the determination of the data such as sampling methods, analysis methods, time intervals, computation methods and reference conditions is not available

Table 6.3 completes the information given in Table 6.2 with emission factors to air (after abatement) for the main coke oven plant operations.

Operation		PM [g/t LS]	CH ₄ [g/t LS]	ali. HC [gC/t LS]	benzene [mg/t LS]	BaP [mg/t LS]	PAH ^{*2} [mg/t LS]	CO [g/t LS]	SO ₂ [g/t LS]	H ₂ S [g/t LS]	NH ₃ [g/t LS]	NO _x [g/t LS]
Charging	“old plants” ^{*3}	1-1.5	17		34	3.5		7-13				
	[Eisenhut, 1990]				34-570	0.5-5 ^{*5}	5-74 ^{*5}					
	[EC coke, 1993]	0.1-3.5	0.02-25	0.003-10	2-400	0.007-1.5		0.02-24	0.003-3		<0.1	
	Kaiserstuhl ‘96 ^{*4}	0.3			3	0.01		0.3				
Carbonisation Doors	“old plants” ^{*3}	4			335	19						
	[Eisenhut, 1990]				2700-4700 ^{*6} 70-100 ^{*7}	19 ^{*6} 0.1-5 ^{*7}	3-1250 ^{*5}					
	[EC coke, 1993]	0.1-2	1.5-25	0.3-7	70-4700	1.5-15		0.5-10	0.05-0.5	0.006-0.3	0.03-0.5	0.01-0.15
	Kaiserstuhl ‘96 ^{*4}	0.3			30	0.3						
Lids	“old plants” ^{*3}				270	3						
	[Eisenhut, 1990]				270-9000 ^{*8}	1-90 ^{*5}	40-1100 ^{*5}					
	[EC coke, 1993]	0.06-0.3	1.5-8.5	0.5-3	270-2700	3-5		0.5-3	0.05-0.3	0.003-	0.03-0.1	0.01-0.05
	Kaiserstuhl ‘96 ^{*4}				0.03	0.05						
Ascens. Pipes	“old plants” ^{*3}											
	[Eisenhut, 1990]				3-600 ^{*9}	0.1-11 ^{*9}						
	[EC coke, 1993] ^{*9}	< 0.07	0.1-1	0.03-0.3	3-33	0.1-1		0.001-0.1	0.003-0.03	<0.003	<0.003	
	Kaiserstuhl ‘96 ^{*4}							2	5			

Legend LS = liquid steel (crude steel); PM = particulate matter; ali. HC = aliphatic hydrocarbons; BaP = benzo(a)pyrene; PAH = polycyclic aromatic hydrocarbons
:

^{*1} used conversion factors (weighted average of all European blast furnaces and basic oxygen steelworks): 358 kg coke/t pig iron; 940 kg pig iron/t LS;

^{*2} PAH as EPA 16 (sum of following 16 PAH (Σ (FLU+PYR+TRI+CPP+BaA+CHR+BNT+BeP+BbF+BkF+BaP+DBahaA+BghiP+INP+ANT+COR))

^{*3} data from [Hein et al., 1996] originating from old plants, which have been replaced by the modern plant “Kaiserstuhl”, D-Dortmund (see ^{*4})

^{*4} data from [Hein et al., 1996]

^{*5} higher values are for old plants in bad condition, the lower ones for new plants in good condition

^{*6} rigid hammer knife sealing

^{*7} flexible door sealing (e.g. membranes)

^{*8} high values, if lids are not carefully and/or appropriately sealed

^{*9} high values for metal/metal sealing, low values for water sealing

Table 6.3 : Emission factors for emissions to air for coke oven plants^{*1}

continued (Table 6.3 : emission factors for emissions to air for coke oven plants)

Operation		PM [g/t LS]	CH ₄ [g/t LS]	ali. HC [gC/t LS]	Benzene [mg/t LS]	BaP [mg/t LS]	PAH ^{*2} [mg/t LS]	CO [g/t LS]	SO ₂ [g/t LS]	H ₂ S [g/t LS]	NH ₃ [g/t LS]	NO _x [g/t LS]
Pushing	“old plants” ^{*3}	135-200 ^{*10}				17						
	[Eisenhut, 1990]	<2 ^{*11}			170 ^{*12} 7-25 ^{*13}	0.001-17 ^{*14}						
	[EC coke, 1993]	>70	0.3-2					2-12	2-8			
	Kaiserstuhl ‘96 ^{*4}	0.3				<0.001						
Quenching	“old plants” ^{*3}	20-40						30-50		22	1.5	
	[Eisenhut, 1990]				15-1000 ^{*15}	0.003-3 ^{*5}	0.2-33					
	[EC coke, 1993]	up to 45						100-470		17-27	1-3	
	Kaiserstuhl ‘96 ^{*4}	5										
Sieving/screening		2 ^{*16}										
By-product plant					30-8000 ^{*17}	0.02-0.03	0.5-1.5					
Underfiring ^{*18}								27-100 ^{*19} 100-500 ^{*20} up to 1250 ^{*21}				80-600

Legend: LS = liquid steel (crude steel); PM = particulate matter; ali. HC = aliphatic hydrocarbons; BaP = benzo(a)pyrene; PAH = polycyclic aromatic hydrocarbons

*1 conversion factors used (weighted average of all European blast furnaces and basic oxygen steelworks): 358 kg coke/t pig iron; 940 kg pig iron/t LS;

*2 PAH as EPA 16 (sum of following 16 PAH (Σ (FLU+PYR+TRI+CPP+BaA+CHR+BNT+BeP+BbF+BkF+BaP+DBaA+BghiP+INP+ANT+COR)))

*3 data from [Hein et al., 1996] originating from old plants, which have been replaced by the modern plant “Kaiserstuhl”, D-Dortmund (see *4)

*4 data from [Hein et al., 1996]

*5 higher values are for old plants in bad conditions, the lower ones for new plants in good conditions

*10 data for unabated emissions

*12 with cokeside car

*14 the low value is achieved by bag filters, the high one occurs without de-dusting device

*16 data from [US PM-10, 1994]

*18 data from 10 coke oven plants from 1997

*20 in case of use of desulphurized coke oven gas, but non-gastight chamber walls leading to higher emissions

*11 according to the German TA Luft [TA Luft, 1986] 5 g/t coke or 1.7 gPM/t LS must be achieved

*13 with hood car

*15 in case of incomplete coking

*17 the high value is for plants without gas balancing pipe system

*19 in case of use of desulphurized coke oven gas (in many cases in mixture with blast furnace gas)

*21 in case of usage of undesulphurized coke oven gas (in many cases in mixture with blast furnace gas)

6.2.2 Information on emissions to air

It should be noted that coke oven plants have a relatively large number of emission sources (see description in 6.1). The emissions from many of these vary considerably with time (e.g. semi-continuous emissions from doors, lids, ascension pipes and discontinuous emissions from pushing and quenching). Moreover, these emissions are difficult to quantify. Comparison of emission factors between plants also needs to take into account plant specific parameters. For instance specific door emissions vary widely depending upon the type of doors, the size of ovens and the quality of maintenance. Maintenance can be a determining factor. Indeed, one can well find examples of good results with traditional (knife edged) doors on well-maintained small ovens and bad results with modern flexible sealing doors on badly maintained large ovens. This should be borne in mind when assessing the information on individual emission sources to air given in Table 6.2 and Table 6.3. The wide range of values for emission factors can primarily be explained by the maintenance programme, the degree of care taken in its performance (enabling smooth and stable operation conditions) and the abatement techniques applied.

6.2.3 Information on emissions to water

More detailed information on the emissions to water and on energy demand is given below.

6.2.3.1 Continuous emissions to water

6.2.3.1.1 Quantities

A wide variety of processes and variants are used for the down-stream processing of coke oven gas. The amount of wastewater yielded by gas treatment plants depends on the water demand, for instance steam, fresh-water addition to the washing liquids, diluting water etc. Roughly speaking the quantity of wastewater is between 1.5 and 3 times the quantity of water resulting from the coking process.

A substantial proportion of water to be drained from the coking process is the coal moisture from the coke oven. The average moisture content of coal is about 8 - 11%, i.e. 0.08 – 0.11 m³/t coal. Thus a representative coke yield of 780 kg coke/t coal produces a water flow of about 0.1 to 0.13 m³/t coke.

Thermal decomposition of coal generates additional water. Depending on the rank of coal 3 to 5% by weight of so-called "chemical water" is produced which means 0.03 – 0.05 m³/t coal corresponding to 0.04 – 0.06 m³/t coke.

If steam injection is used for aspiration (suction of charging gases) condensation of this steam will result in further water being produced in the primary coolers. The condensate is then also part of the water from the coke oven plant.

The water leaves the coke oven plant via the ascension pipes and is subsequently condensed, together with the tar from the crude coke oven gas. The mixture of tar and water is led to the tar/water separator, where the so called "coal water" is separated off [EC Coke, 1996]. This water is led to the ammonia liquor storage tank and eventually to the ammonia stripper/still to remove the ammonia.

6.2.3.1.2 Wastewater from the coke oven plant

After ammonia removal by distillation (see 'still effluent' in Figure 6.9) the wastewater effluent contains various organic (such as phenols) and inorganic compounds (such as residual ammonia and cyanides). These compounds have a negative impact on the recipient water when directly discharged. The still effluent is therefore usually treated in a wastewater treatment plant before discharge.

Data concerning the composition of the wastewater before treatment show it to vary considerably from one plant to another, both in terms of concentration and load per tonne of coke produced. The main reason for the wide variations lies in the gas treatment systems, the type of coking processes (top charging or stamp charging) and the type of coal at the individual coke oven plants. Table 6.4 shows the influent characteristics for the most important design parameters of a biological treatment plant.

Parameter	Concentration [mg/l]		Emission factor [g/t coke]
	Mean value	Standard deviation	
Chemical oxygen demand (COD)	2250 - 4450	310 - 590	430 - 1700
Ammonia (NH ₃ /NH ₄ ⁺)	25 - 85	15 - 105	5 - 30

Table 6.4 : Influent characteristics of wastewater from four coke oven plants in Belgium, Germany, France and the Netherlands – [Löhr, 1996];

The data shown represent ranges of mean values and standard deviations for wastewater after ammonia stripping but before dilution and treatment (if applied).

The COD as a sum parameter comprises a big variety of different organic chemical compounds such as phenol (1000 – 2000 mg/l) organic nitrogen compounds (Kjeldahl-N: 250 – 500 mg/l) and polycyclic aromatic hydrocarbons (PAH) (up to 30 mg/l) [InfoMil,1997]. In addition to ammonia other inorganic nitrogen containing compounds are also present, such as thiocyanides (150 – 350 mg/l) [InfoMil, 1997].

6.2.3.1.3 Wastewater from wet oxidation desulphurisation processes

Wastewater from wet oxidative desulphurisation processes is usually treated separately owing to the presence of compounds that have a detrimental effect on the biological wastewater treatment plant.

Table 6.5 shows the wastewater composition of two wet oxidative desulphurisation processes. The application of one of the other wet oxidative techniques may lead to water emissions of arsenic compounds (Thylox process), 1,4-naphthoquinone-2-sulphonic acid (Takahax), picric acid and thiocyanides (Fumaks/rhodacs process).

Component	Stretford desulphurisation (g/l)	Perox desulphurisation (g/l)
NH ₃ (free)	0.1	7.5 - 24
NH ₃ (fixed)	?	48 - 61
Na ₂ CO ₃	5.7 - 65	-
CO ₂	?	13 - 34
SCN ⁻	80 - 300	61 - 73
Cl ⁻	?	1.5
H ₂ S	?	0.04 - 0.2
S ₂ O ₃ ²⁻	50	35 - 127
SO ₄ ²⁻	12.2	7 - 23
Vanadate (VO ₃ ⁻)	1.2	-
Tartrate (V ₄ O ₉ ²⁻)	1	-
Anthraquinodisulphonate (ADS)	1.9	-
Hydroquinone	-	0.3 - 0.6
pH	?	8.7

Table 6.5 : Composition of wastewater from two wet oxidative desulphurisation processes
- [EC Coke, 1996]

6.2.3.1.4 Cooling water

Normally most cooling water is used for indirect cooling in coke oven gas (COG) treatment. This cooling water is usually recirculated around the circuit. Fresh water consumption to replace evaporation and drainage losses is of around 6 – 10 m³/h (i.e. 0.06 m³/t coke).

Direct cooling of COG normally uses ammonia liquor in a closed circuit. The circuit flow is about 6 m³/t coke.

6.2.3.2 Discontinuous emissions to water

6.2.3.2.1 Wet coke quenching

Discontinuous emissions to water can in some cases be generated by wet coke quenching operations. However, when quenching is operated correctly, the excess water is collected and used for the next batch. The use of this excess water in other processes is also possible. This eliminates emissions to water.

6.2.4 Energy demand

Table 6.6 contains information on energy input of a coke oven plant without consideration COG treatment. The data in Table 6.2 confirm these values. The table also gives figures for the energy output, thus showing the considerable energy losses (approx. 3 GJ/t coke). The COG produced by coke oven plants means they play an important role in energy supply and management in integrated steelworks (see Figure 3.3 and Figure 3.4).

Energy Input		Energy Output	
Energy carrier	GJ/t coke	Energy carrier	GJ/t coke
Coal	40.19	Coke	27.05
Coke oven firing	3.01	Coke oven gas	8.08
Chemical reactions	0.32	Energy loss	3.33
		Additional products (S ⁰ , tar etc.)	2.56
		Screening waste of coke	1.92
		Coke dusts	0.26
Total	43.52	Total	43.20

Table 6.6 : Energy balance of a coke oven plant (not considering COG treatment) – based on [UN-ECE, 1990]; the assumed coke yield is 780 kg coke/t coal

6.2.5 Soil pollution

In the coke oven gas treatment plant, tar and other organic compounds (e.g. BTX) are recovered from the coke oven gas. Spillage or leakage of these compounds may cause a soil pollution hazard, depending on local soil conditions. Furthermore, spillage or leakage of coal water may also cause a soil pollution hazard.

In this document soil pollution is treated as a local aspect. Therefore, only a brief description of possible measures will be given here:

- minimising the number of flanges and appendages;
- pipes and flanges must be inspectable as much as possible (e.g. placed above ground or in mantle pipes), unless the age and design of the plant may make this impossible;
- storage and transport of substances which are potentially a soil pollution hazard should be done in such a way that precautions are taken to avoid spillage to soil occurring

6.3 Techniques to consider in the determination of BAT

This section includes both process-integrated and end-of-pipe techniques for environmental protection and energy saving at coke oven plants. A description of each technique is given, together with the main emission levels achieved, its applicability, the monitoring of emissions, cross-media effects, reference plants, operational data, economics and the driving force for implementation (where these data are available and pertinent).

Process integrated measures

The following process-integrated techniques are known to be used at coke ovens:

- PI.1 Smooth and undisturbed operation of the coke oven plant
- PI.2 Maintenance of coke ovens
- PI.3 Improvement of oven door and frame seals
- PI.4 Cleaning of oven door and frame seals
- PI.5 Maintaining free gas flow in the coke oven
- PI.6 Emission reduction during coke oven firing
- PI.7 Coke dry quenching (CDQ)
- PI.8 Larger coke oven chambers
- PI.9 Non-recovery coking

End-of-pipe techniques

The following end-of-pipe techniques are known to be in use at coke oven plants:

- EP.1 Minimising oven charging emissions
- EP.2 Sealing of ascension pipes and charging holes
- EP.3 Minimising leakage between coke oven chamber and heating chamber
- EP.4 De-dusting of coke oven pushing
- EP.5 Emissions minimised wet quenching
- EP.6 De-NO_x of waste gas from coke oven firing
- EP.7 Coke oven gas desulphurisation
- EP.8 Removing tar (and PAH) from the coal water
- EP.9 Ammonia stripper
- EP.10 Gas-tight operation of the gas treatment plant
- EP.11 Wastewater treatment plant

PI.1 Smooth and undisturbed operation of the coke oven plant

Description: Together with the maintenance regime (see PI.2) and cleaning operations (see PI.4), smooth and undisturbed operation of the coke oven is one of the most important process-integrated measures. A failure to do so leads to severe temperature fluctuations and increased chances on coke jam during pushing. This has an adverse effect on the refractory and on the coke oven itself and may lead to increased leakage and an increase in abnormal operation conditions.

A prerequisite for smooth and undisturbed operation is good reliability of the oven machinery and installations. This also leads to higher productivity.

Another prerequisite is an optimised coal pre-treatment – the optimal operation of a coke oven plant requires a coal burden, that is as homogenous as possible. A modern coal pre-treatment plant consists of two mixing hoppers, a crushing/screening plant, transporting equipment, dedusting equipment and possibly additional processes like a coal drying or addition of coal additives. Only by this process, the good operation of coke oven plants as described in this chapter is achieved.

By controlling the temperature of heating surface (measuring device at the ram bar) it is possible to determine the temperature distribution in the chamber which allows conclusions on the efficiency of the heating system.

Based on these results actions to repair or to optimise it can be taken.

Applicability: Can be applied both at new and existing plants.

Main achieved emission level: A significant part of the coke oven plant emissions is caused by leakage through cracks between the heating chamber and the oven chamber and as a result of deformed doors, door frames, buck-stays etc. These emissions can be avoided to a large extent by a more smooth and undisturbed operation of the coke oven plant. Furthermore, this measure may significantly increase the duration of the life of the coke oven plant.

Reference plants: All operators in the world are trained to operate their coke oven plant as smoothly as possible. However, economic incentives may induce higher production levels and increase the chances of less stable plant operation.

Cross media effects: no significant cross media effects are known

Operational data: not available

Economics: not available

Reference literature: [InfoMil, 1997]

PI.2 Maintenance of coke ovens

Description: The maintenance of coke ovens is one of the most important process-integrated measures and is a decisive factor for smooth and undisturbed operation (see PI.1).

The maintenance should not be performed in campaigns but continuously. No fixed or minimum period of maintenance can be set. It should follow a systematic programme and be carried out by specially-trained maintenance personnel (e.g. during two shifts). Following the maintenance programme at SIDMAR, B-Gent is described below as an example. There each oven receives a complete overhaul every 3 – 3.5 years. This entails leaving the oven empty for a week in order for the following operations to be carried out:

- degreasing of all deposits within the chamber (walls, ceiling)
- oxythermic welding of cracks, holes and surface damage of the refractory brickwork
- repair of the floor of the oven chamber by flooding with cement
- air born dust injection into the fine cracks
- overhaul of the sealing surface of the door frame by fraising and readjustment of the doorframe in the oven
- complete overhaul of the doors; complete dismantling of all individual parts, cleaning and reassembling; readjustment of the flexible sealing. Damaged door bricks are replaced; in many cases the door is completely rebricked.

In addition to this three-yearly overhaul, the bracing system of the ovens (springs, anchoring etc.) is regularly checked and adjusted.

Applicability: Can be applied both at new and existing plants.

Main achieved emission level: Good maintenance prevents cracks in the refractory brick work and minimises leakage and consequently COG emissions. It is achievable that this prevents black smoke being visible at the stacks discharging flue gas from coke oven firing. In addition, the maintenance, readjustment and overhaul of the doors and frames prevents leaks.

Reference plants: Sidmar coke oven plant, B-Gent; coke oven plant 2 of Hoogovens IJmuiden, NL-IJmuiden.

Cross media effects: no significant cross media effects are known

Operational data: The maintenance programme described has been applied at Sidmar, B-Gent since 1986, for instance, with considerable success.

Economics: Personnel costs for skilled workers carrying out the maintenance programme are considerable (about 1.5 Ecu₁₉₉₇/t coke in case of Sidmar, B-Gent) but as part of the measures of smooth and undisturbed operation better skilled operators contribute to higher productivity, thus compensating the costs.

Driving force for implementation: Intention to operate the coke oven plant as smooth as possible with optimised productivity and minimised emissions.

Reference literature: not available

PI.3 Improvement of oven door and frame seals

Description: Oven door gas-tightness is essential and can be achieved by applying the following measures:

1. use of spring-loaded, flexible sealing doors;
2. careful cleaning of the door and its frame at every handling (see PI.4).

Existing batteries can be equipped with new spring-loaded sealing doors if door frames and armour plates are not too deformed. In this respect, the strength of the buck-stays plays an important role, because the armour plates (holding the brickwork in place) are preferably held in place by springs, which are mounted to the buck-stays.

It must be noted that the situation differs for small and large ovens. For ovens less than 5m high knife edge doors in combination with good maintenance may be sufficient to prevent door emissions.

Applicability: Applicable at new and in some cases existing plants.

Main achieved emission level: The specific emission values from flexible sealing doors are much lower than those from conventional doors. Provided they are kept clean, "new generation" doors make it possible to keep visible emissions below 5% of all the coke oven doors per battery, both on the pusher side and on the coke side. However, examples can be found of good results with traditional (knife edge) doors on well maintained small ovens and bad results with flexible sealing doors on badly maintained large ovens. But flexible sealing offers much better opportunity for tightness especially for large ovens. Table 6.3 indicates possible improvement in emissions reduction.

Reference plants: Many new and rebuilt plants apply spring-loaded flexible sealing doors.
Coke oven 1, Hoogovens IJmuiden, NL-IJmuiden
Coke oven Hüttenwerke Krupp Mannesmann GmbH, D-Huckingen

Cross-media effects: No cross-media effects occur

Economics: not available

Operational data: not available

Reference literature: [Vos, 1995]

PI.4 *Cleaning of oven doors and frame seals*

Description: Many older European coke oven plants still have original non-spring-loaded knife-edge doors. In these plants, leaking doors can be a serious problem. However, with good maintenance (see PI.1 and PI.2) any visible emissions from existing doors can be under 10% [Vos, 1995]. The success of the maintenance plan depends to a large extent on a stable coking process, permanent staffing for maintenance, continuous monitoring and feedback regarding achievements. An indoor maintenance workshop for doors is highly recommended.

An applied method of cleaning coke oven doors is to use a high-pressure water-jet. This has proved to be very successful. High-pressure water-jet coke oven door cleaning however can not be performed every cycle. Advanced door cleaners using scrapers at every cycle also have good results.

Applicability: Applicable both to existing and new plants.

Main achieved emission levels: The high-pressure water-jet cleaner system makes it possible to virtually eliminate visible emissions – a reduction to 95% of the time can be achieved (according to the EPA method).

Reference plants:

Coke oven 2, Hoogovens IJmuiden, NL- IJmuiden

Redcar coke oven plant No. 1, British Steel Teeside Works, United Kingdom

Cross-media effects: Cleaning with a high-pressure water-jet generates a contaminated wastewater flow, which can be treated along with the wastewater from the coke oven batteries.

Economics: not available

Operational data: not available

Reference literature: [Vos, 1995; Murphy, 1991]

PI.5 *Maintaining free gas flow in the coke oven*

Description: The oven chamber is usually kept under slightly positive pressure during coking. Negative pressure would allow air to penetrate the oven chamber and would partially burn coke, leading to destruction of the oven. The oven bottom should be at atmospheric pressure. As a rule of thumb, the over pressure (in mm water column) in the collecting main is maintained at twice the height of the oven in metres. Hence, for a modern 7m high oven, the corresponding over pressure would be 14 mm water column. For older 4m high ovens, the over pressure in the collecting main would be 8 mm water column.

This pressure difference is necessary to remove the gases and tar from the coke oven chamber. The pressure drop is created by means of a throttle valve at the outlet of the collecting main where the pressure is -80 mm water column.

A free space is maintained at the top of the oven chamber to allow the gases and volatilised tar to flow toward the ascension pipe, which, depending on the design, is usually situated either on the pusher machine side of the oven or on both sides. The gas flow may be hampered by charged coal reaching the oven top and by the top of the oven becoming clogged with graphite. Whenever the gas flow in the oven is obstructed, door and charging hole leakage will occur because the over pressure behind the obstruction increases.

This situation can be prevented by adequate levelling of the charged coal, by periodic degraphitising of the oven roof and by periodic goose neck cleaning (see also PI.2).

The build up of graphite on the oven roof can be minimised by adequate heat distribution over the oven walls.

Applicability: Applicable at new and existing coke ovens

Main achieved emission level: A good pressure distribution in the coke oven chamber significantly reduces diffuse emissions and leakage. Furthermore, the risk of the coke charge jamming during pushing operations is reduced.

Reference plants: All plants with a proper maintenance programme (see PI.2) and which aim to minimise door leakage apply this technique.

Cross-media effects: No cross-media effects occur

Operational data: not available

Economics: not available

Reference literature: [InfoMil, 1997]

PI.6 Emission reduction at coke oven firing

Description: The heat for the coking process is provided by firing a gaseous fuel in the heating chambers. Heat is transferred to the coke oven chamber by heat conduction through the refractory brick wall. A higher temperature in the oven chamber gives in a shorter coking time. The most important pollutants from coke oven firing are NO_x , SO_2 and particulate matter.

The level of SO_2 emissions is strongly related to the sulphur content of the fuel. Thus, emissions of SO_2 can be minimised by minimising the sulphur content of the fuel. Usually, (enriched) blast furnace gas or coke oven gas is used to fire the coke oven. The sulphur content of the coke oven gas depends on the desulphurisation performance of the coke oven gas treatment plant. The H_2S content in coke oven gas may vary from around 50 mg/Nm^3 to 1000 mg/Nm^3 , depending on the desulphurisation process and on the efficiency. If no desulphurisation is applied (which is still the case at some plants in the EU) the H_2S content can be as high as $8000 \text{ mg H}_2\text{S /Nm}^3$. Enriched blast furnace gas has a low sulphur content. One of the main parameters of the coke oven gas desulphurisation process is the temperature of the gas.

Emissions of SO_2 and particulate matter may be significantly increased when crude coke oven gas from the oven chambers leaks through cracks in the heating walls and is combusted together with the heating fuel (see also EP.3).

The case of NO_x is somewhat more complicated. The NO_x generated consists almost entirely of thermal NO_x , which is formed by the reaction between N_2 and O_2 in the flame. Thermal NO_x formation is strongly related to peak temperatures and O_2 concentrations in the flame. Indirectly, NO_x emissions are also related to the fuel (enriched blast furnace gas or COG) and to the type of coal used, the specific charge weight of the coal, the coking time and the dimensions of the coke oven chamber.

The most effective way to reduce NO_x formation is to reduce the flame temperature in the heating chamber. The aim is therefore to burn with a cool flame. Three methods have shown themselves effective at doing this:

- Waste gas recirculation. The waste gas from the coke oven is admixed with fuel and combustion air. The lower O_2 and higher CO_2 concentrations reduce the flame temperature. However, the preheat effect of waste gas recirculation may counteract the temperature reducing effect.
- Staged air combustion. By adding the combustion air in several stages, combustion conditions become more moderate, and NO_x formation is reduced.
- Lower coking temperature. Lower coking temperature have an influence on economics and the energy efficiency of the coke ovens. A lower coking temperature requires a lower heating chamber temperature, which results in less NO_x formation.

Furthermore, the heating chamber temperature (and thus NO_x formation) can be reduced while normal coking temperature is maintained by decreasing the temperature gradient over the refractory brick wall from the heating chamber side to the coke oven chamber side. This can be done by using thinner bricks and a refractory with a better thermal conductivity. Formerly, a heating chamber temperature of 1320°C would lead to a coke oven chamber temperature of 1180°C . Nowadays, a coke oven chamber temperature of 1200°C is achieved at the same heating chamber temperature due to thinner bricks.

Applicability: Process integrated de- NO_x measures are applicable in new plants. Lowering the temperature in an existing plant will result in longer coking time and operation below nominal capacity.

Main achieved emission levels: For existing plants without process-integrated de-NO_x measures, such as staged air combustion, achievable levels for NO_x are in the range 1300 g/t coke to 1900 g/t coke (concentrations: 600 – 1500 mg/Nm³ at 5% O₂). Table 6.2 shows emission factors between 700 to 1800 g/t coke. Plants which have implemented process integrated de-NO_x measures emit NO_x of 450-700 g NO_x/t coke (concentrations: 500 – 770 mg/Nm³ at 5% O₂).

Reference plants: Most modern plants are equipped with low-NO_x firing systems. For example:

Coke oven Hüttenwerke Krupp Mannesmann GmbH, D-Huckingen;

Coke oven Prosper, D-Bottrop;

Coke oven battery No. 5 of Ruhrkohle Hassel, D-Gelsenkirchen;

Cross-media effects: When the cooling capacity of the crude gas coolers is increased to improve desulphurisation efficiency, the energy consumption increases and possible thermal emissions are increased. No cross-media effects have been identified as a result of lowering NO_x emissions from coke oven firing.

Economics: not available

Operational data: not available

Reference literature: [Eisenhut, 1988; Stalherm, 1995]

PI.7 Coke dry quenching (CDQ)

Description: Based on a Swiss patent, the CDQ process was originally developed on an industrial scale in the former Soviet Union at the beginning of the 1960s (the so called Giprokoks process). It was intended for application in coke oven plants located in regions which suffer from long periods of severe cold, such as for example: Siberia, Finland, Poland, where wet quenching of coke is difficult or even impossible. In addition, plants in these regions need considerable quantities of energy for heating purposes (steam and/or electricity) pipe tracings and defrost equipment [Bussmann, 1985]. Later on, because of the special structure of the Japanese energy supply market the Giprokoks process was applied there and underwent further systematic development from around 1973 onwards. Figure 6.11 shows a scheme of a current CDQ plant design consisting of the shaft-like cooling unit, the waste heat boiler and the gas recycling system.

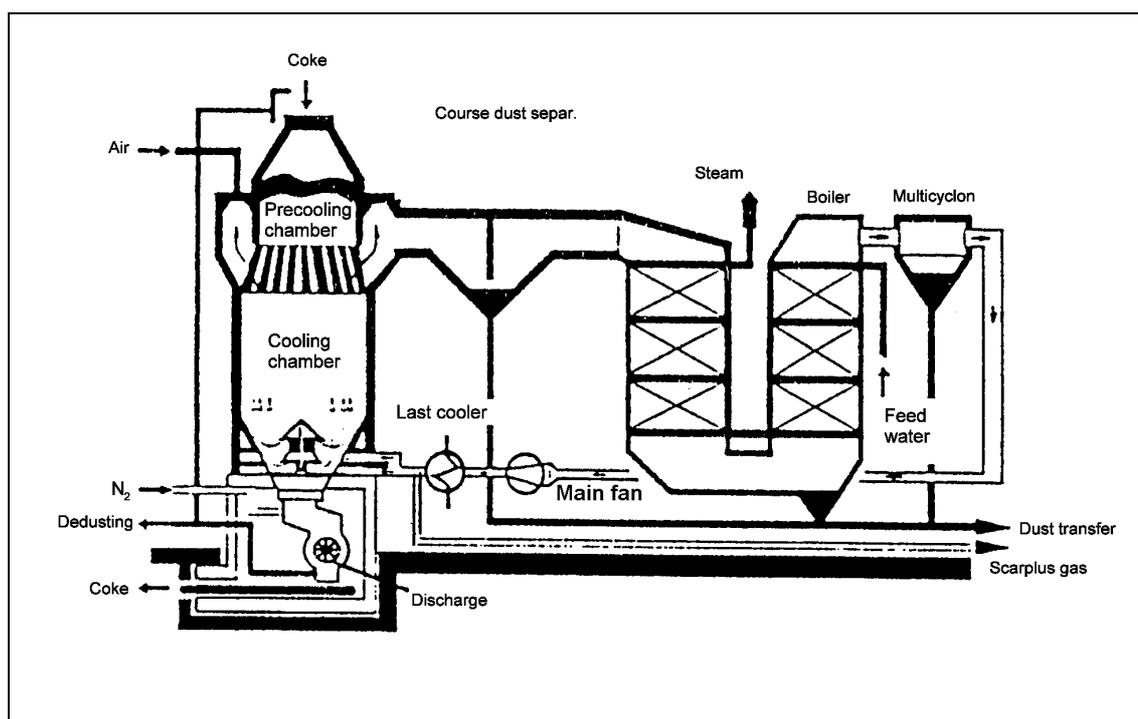


Figure 6.11 : Scheme of a coke dry quenching plant -[Schönmuth, 1994]

The carbonised coke initially passes from the battery directly, or via a container on a hoist, to the cooling unit where the coke is emptied downwards through an aperture into the shaft. As the coke column descends at a constant rate it emits its sensible heat into a largely inert, counter-flowing gas. The cooled coke (cooled to 180 – 200 °C) is discharged at the bottom of the shaft by way of sluices and conveyed away by suitable equipment. The gas, which is recycled by a blower, has a temperature of 750 – 800 °C and is relieved of the absorbed heat in a downstream waste heat boiler used for steam generation (about 0.5 t steam (480 °C, 60 bar)/t coke). It is then fed back into the cooling shaft. Coarse and fine dust precipitators ensure that the boiler and blower are protected against entrained coke dust. Because of after-gassing of coke the inert gas enriches with carbon monoxide and other compounds which makes gas exclusion necessary from time to time. This excess gas is treated in de-dusting devices, preferably in a bag filter with < 5 mg/Nm³ residual dust content. It is subsequently fed to the heating gas of the coke oven battery [Schönmuth, 1994; Bussmann, 19985]. At around 50 Nm³/t coke the flow of excess gas is relatively low.

Applicability: Technically speaking, CDQ can in principle be applied at new and existing plants. Due to limited utilisation rate of CDQ every coke oven plant with CDQ also needs a wet quenching station.

Main achieved emission levels: The advantages of CDQ compared with wet quenching are energy recovery and better environmental performance (reduced emissions of dust, carbon monoxide and hydrogen sulphide).

With CDQ about 0.5 t steam (480 °C, 60 bar)/t coke corresponding with 1.5 GJ/t coke can be recovered and plumes can be avoided. Meanwhile the severe problem of boiler erosion could be solved [Ritamäki, 1996].

When considering dust emissions both quenching and the subsequent steps have to be taken into account, particularly coke handling and sieving. At the Kaiserstuhl coke plant in Germany for example, the dust emissions from coke handling and sieving made it necessary to install an additional cooling step (reducing coke temperatures to below 80 °C). Dry coke has to be wetted, resulting in 1% moisture content. Overall, there is no significant difference in dust emissions between CDQ with these measures and emission-minimised wet quenching (see EP.5).

There is also no significant difference in CO emissions because of exclusion of excess gas from the gas circuit. The lower emissions of H₂S from CDQ are not significant in relation to the overall sulphur emissions from an integrated steel works.

Although it has been claimed elsewhere [Ritamäki, 1996; Wenecki, 1996], according to German CDQ operators there is no significant difference in coke quality after screening.

Reference plants: According to [Ritamäki, 1996] worldwide there are about 60 coke oven plants in 18 different countries operating CDQ. Most of them are located in the CIS countries (Commonwealth of Independent States) because of cold climatic conditions (25 plants with 109 units) and in Japan (20 plants with about 33 units) because of high energy prices. In Japan CDQ is installed at 80% of plants [Arimitsu, 1995].

There are several CDQ units in the EU 15: two at Thyssen Stahl, D-Duisburg [Bussmann, 1985], the world's biggest unit (250 t/h) at Kokerei Kaiserstuhl, D-Dortmund [Schönmath, 1994] and three units at Raahe Steel, FIN-Raahe [Ritamäki, 1996]

Cross-media effects: A disadvantage of CDQ is the emission of particulate matter at the places where dry quenched coke is handled [Eisenhut, 1988; Schönmath, 1994]. These emissions can be avoided by spraying and/or transport in closed conveyors. Especially when transport over longer distances is necessary (for example at non-integrated coke oven plants) high levels of emissions of particulate matter may occur.

The electrical power consumption of fans, operation of various de-dusting devices etc. is not negligible.

Operational data: Positive operational results were reported by Thyssen, D-Duisburg [Bussmann, 1985], Raahe Steel, FIN-Raahe [Ritamäki, 1996] and Przyjazn Coke Oven Plant, Poland [Wenecki, 1996]. But utilisation rates vary considerably. The best rate is reported for the CDQ at Raahe Steel (99.9%) [Ritamäki, 1996] because there are three CDQ units with one of them on stand-by all the time. Average utilisation rates for all the existing plants are reported to be between 80 and 90%. The lowest rates are 40 - 60% [Ritamäki, 1996]. Therefore every coke oven plant using CDQ also has a wet quenching station.

Economics: The economic aspect is the crucial point of CDQ. Investment and operation costs are very high. The capital investment of a CDQ for a 2 Mt/a coke oven plant is about 110 MEcu₁₉₉₆. This is about 10 to 15 times higher than the cost of running a wet quenching station (including quenching tower, sedimentation tanks, pumps, etc). Moreover a wet quenching station has to be installed in addition because of the limited utilisation rate of CDQ. It is also reported that investment cost can be 15 to 20 times higher. This equates with 40 – 60 Ecu₁₉₉₆ /t coke. The operation costs (not including capital costs) are 7 – 8 Ecu₁₉₉₆/t coke. The value of produced steam (480 °C, 60 bar) is also 7 – 8 Ecu₁₉₉₆/t coke calculated on the basis of natural gas prices in Germany. Thus in the EU a CDQ can not be operated economically. This is why the technique is only applied in a few cases. Economic efficiency depends directly on energy price levels. In Japan energy is significantly more expensive. Hence CDQ is considered in Japan an energy production unit and is widely applied.

Reference literature: [Arimitsu, 1995; Bussmann, 1985; Eisenhut, 1988; Ritamäki, 1996; Schönmath, 1994; Wenecki, 1996]

PI.8 Larger coke oven chambers

Description: The development of wider and higher coke oven chambers is based on two main principles: reduction of the number of ovens pushed per day and reduction of the length of the sealing faces.

The main characteristics of the high or wide chamber coke ovens is the large oven volume compared to conventional ovens. Thus for a given output, the door seal is decreased in length and the frequency of pushing is reduced. But special attention has to be paid to the sealing because it is more difficult to keep such ovens gas-tight, especially at the top and bottom ends (see PI.3).

Applicability: Only applicable to new plant concept. In some cases, a higher coke oven chamber can be chosen at a complete rebuild of the plant on the old foundations.

Main achieved emission levels: When properly maintained and when spring-loaded flexible sealing doors are used under comparable operational circumstances, the total (fugitive) emissions per tonne of coke from the door and frame seals can be expected to be directly proportional to the reduction of the seal length compared to conventional coke ovens. High oven doors demand a more intensive maintenance.

A reduction in emissions from pushing operations can be expected, given that fewer pushing operations are required per tonne of coke and that emissions are directly proportional to the number of pushes. Nevertheless emission factors (emission per tonne coke) are not influenced by the application of larger coke oven chambers.

Reference plants:

Coke oven battery, D-Huckingen;
Prosper No.3 coke oven battery, D-Bottrop
Kokerei Kaiserstuhl, D-Dortmund

Cross-media effects: Visible fugitive door emissions may occur.

Operational data:

Table 6.7 shows the characteristics of several coke ovens.

Aspect	Unit	Small	Medium	Tall		
				Huckingen	Prosper	Kaiserstuhl
Dimensions (useful)						
Height	[m]	4.5	6.0	7.85	7.1	7.63
Length	[m]	11.7	14.2	17.2	15.9	18.0
Width	[m]	0.45	0.45	0.55	0.59	0.61
Useful volume	[m ³]	22.1	36.4	70.0	62.3	78.9
Productivity	[t coke/oven]	12.7	21.3	43.0	39.8	48.7
Number of ovens	[#]	322	187	120	142	120
Total oven openings	[#]	2898	1496	1080	1278	1080
Length of sealing faces	[km]	10.5	6.9	6.0	6.2	5.5
Oven pushes	[#/d]	430	257	128	138	115
Total opening actions	[#/d]	3870	2056	1152	1242	1035
Length of sealing faces to be cleaned	[km/d]	14.0	9.5	5.6	6.0	5.3

Table 6.7 : Characteristics of several coke oven types – [Eisenhut,1988]

No specific problems are known with the plants with wide and high chambers. Greater demands are placed on wall strength.

Economics: not available

Reference literature: [Eisenhut, 1988; Stalherm, 1990]

PI.9 Non-recovery coking

Description: In the non-recovery coking process, essentially all the tar and gases released from coking process are combusted within the oven and the sole flue. The non-recovery coking process requires a different oven design from that traditionally used. A coke oven gas treatment plant and wastewater treatment plant are not needed.

Primary air for partial combustion is introduced into the coke oven chamber above the charge through ports located in the doors. This partial combustion supplies the heat for coking in the top of the oven ("oven crown"). The amount of primary air is controlled to maintain the desired temperature in the oven crown.

The partially combusted gases exit the oven chamber through passages in the oven wall and enter the sole flue. In the sole flue, secondary air is added to complete the combustion. The heat from secondary combustion is transferred to the oven chamber by heat conduction through the refractory floor.

Waste gases are conducted to a collecting main and are lead through a waste heat boiler before being emitted to the air. The whole system operates at sub-atmospheric pressure.

The oven is much broader and lower than in conventional by-product coke oven designs. However, oven charge weights are comparable. Table 6.8 shows the typical design features of non-recovery coke ovens.

Feature	Value
Oven length	15.6 m
Oven width	4.2 m
Centreline distance	5.2 m
Charge weight	23-43 t
Coking time (nominal)	24-48 hours

Table 6.8 : Design features of non-recovery coke oven – based on [Knoerzer, 1991]

The usual method of charging a coke oven is to drop coal from a larry car into the oven chamber through charging holes. In the non-recovery coking system, charging is performed through the pusher side door opening by a combined charging/pushing machine.

The plant at Inland Steel Company's Indiana Harbour Works went into operation in May 1998. It serves to feed one of the largest blast furnaces in the world. The feedstock is limited to certain types of coal [Eisenhut, 1992; Nashan, 1997]. The horizontal design of the ovens means that coal that expands can be used.

Applicability: Only applicable as a whole new plant concept.

Main achieved emission levels:

Table 6.9 gives figures for the emissions from a non-recovery coking plant. The values refer to emissions without the application of an emission abatement system. At Vansant, Virginia, a portion of the flue gas is scrubbed in a thermal dryer, but the remainder of the ovens operate without flue gas purification devices.

Component	Unit	Value	Unit	Value
Particulate matter	mg/Nm ³	n/a	g/t coke	1960
SO ₂	mg/Nm ³	n/a	g/t coke	7000
NO _x	mg/Nm ³	n/a	g/t coke	380
CO	mg/Nm ³	n/a	g/t coke	77

Table 6.9 : Emissions from non-recovery coking without emission abatement – based on [Knoerzer, 1991]; calculated from g/t coal, based on assumption: 1 tonne coal yields 0.78 tonnes coke (see 6.1.2.3).

Because the coke oven operates at sub-atmospheric pressure, emissions from the doors during the coking operation are negligible. Emissions from charging and pushing are given in Table 6.10. Note that the two existing batteries are not equipped with de-dusting facilities during charging and pushing.

Component	Unit*	Charging	Pushing
Particulate matter	g/t coke	7.35	276**
Benzene soluble organics	g/t coke	Not detected	0.65
Benzo(a)pyrene	g/t coke	1.3·10 ⁻⁵	not detected

*Based on EPA-USA emission factors

Table 6.10 : Emissions from non-recovery coke oven charging and pushing operation – based on [Knoerzer, 1991]; calculated from g/t coal, based on assumption: 1 tonne coal yields 0.78 tonne coke (see 6.1.2.3).

Reference plants: Non-recovery coke ovens have been operated since 1962. Currently plants are in operation at: - Battery 3B and 3C, Jewell Coal and Coke Company, Vansant, Virginia, USA.

A non-recovery coking plant is also under construction at Inland Steel Company's Indiana Harbor Works, USA. This plant was due to start production in May 1998.

There are currently no non-recovery coking plants in operation or planned in Europe.

Cross-media effects: This plant concept causes significantly fewer cross-media effects than that of the conventional coke oven. However, SO₂ and particulate emissions from the coke oven firing stack are higher than in conventional coking. Whenever a scrubber is used to reduce emissions to air of SO₂ and particulates, a contaminated wastewater stream is generated. The plant which is currently under construction will be equipped with a flue gas desulphurisation device.

A disadvantage of the non-recovery coking is the large area occupied by the ovens, due to their horizontal configuration.

Partial combustion results in a smaller output of blast furnace coke.

In integrated steelworks it seems to be very difficult to achieve an economically profitable energy link between a non-recovery coking plant to the existing energy network.

Operational data:

Battery 3B of the Jewell Coal and Coke Company was commissioned in 1989. Battery 3C was commissioned in 1990. There are no specific problems known.

Economics: The cost for a greenfield non-recovery coke oven plant producing 1.2 million tonnes per year is 185 million USD₁₉₉₆, excluding coal handling and power production costs. This corresponds to approximately 147 million Ecu₁₉₉₆. The operational costs for the greenfield coke oven plant are projected to be approximately 16.6 USD₁₉₉₆/t coke, corresponding to 13.2 Ecu₁₉₉₆/t coke.

Reference literature: [Knoerzer, 1991; Shoup, 1991; Eisenhut, 1992; Nashan, 1997]

EP.1 Minimising oven charging emissions

Description: Oven charging is most commonly performed by gravity charging with charging cars. Three basic techniques are used for charging in this way:

1. "Smokeless" charging. This system uses gas-tight connections between the coke oven and the charging car. The chambers are rapidly filled through usually four or five charging holes. Suction is generated by steam or water injection in the goose neck of the ascension pipe.
2. Sequential charging or stage charging. In sequential or stage charging, the charging holes are charged one after the other. This type of charging takes a relatively long time. Suction is generated on both sides of the oven, either by the use of two ascension pipes (when present) or the use of the ascension pipe and a jumper pipe to the adjacent oven. The connections between the charging car and the oven are not gas-tight, but due to the suction, virtually no emissions occur when only one opening to the atmosphere is present.
3. Charging with telescope sleeves also known as "Japanese charging"; this kind of charging is done by simultaneous charging through the (usually) four charging holes. The connections between the charging car and the coke oven are not gas tight, but are enclosed by "telescope sleeves" from which the gases are extracted and led to a collecting main via a connection between collecting main and charging car. The extracted gases are combusted and subsequently led through a particulate matter arresting device, which is stationed on the ground. In some cases, the extracted gases are treated on the charging car.

Charging can also be carried out by transfer of coal through pipelines. Two systems for pipeline charging have been developed:

1. A central pipeline system with connections to all ovens.
2. A pipeline with a connection to a charging car, through which the coal is charged.

Coal preheating enables pipeline charging.

During the levelling process after charging a packing sealing of the levelling doors against the levelling bar can minimise emissions from levelling doors.

Applicability: Can be applied both to new and existing plants.

Main achieved emission level: Charging emissions can be very low in all of these systems. The main determining factor is the over pressure in the oven chamber and the charging telescopes. Some of the systems are more vulnerable to operational problems than others.

Emissions from charging are hard to quantify but < 30 seconds of visible emissions per charge is generally achievable. Emissions of less than 10 seconds are achievable under optimum conditions.

Reported particulate matter emissions from charging with telescope sleeves with a dust collecting unit on the charging car are less than 5 g/t coke (concentration < 50 mg/Nm³). In this case, the evacuated gas is burnt before discharge.

Reference plant: Emission-minimised charging is applied at many plants all over the world. The following examples are given:

"Smokeless" charging:

Coke oven 2, Hoogovens IJmuiden, NL-IJmuiden.

Sequential/stage charging:

Many plants in the USA;

Coke oven Mannesmann Hüttenwerke Krupp, D-Huckingen;

Coke oven Prosper, D-Bottrop,.

"Japanese" charging:

- Coke oven 1, Kawasaki Steel Corporation, Chiba Works, Japan.
- Coke oven, Nippon Steel Corporation, Kimitsu Works, Japan;
- Coke oven, Sumitomo Metal Industries, Wakayama Works, Japan;
- Coke oven 1, Hoogovens IJmuiden, NL-IJmuiden ;
- Coke oven, ZK Saar & Gneisenau, Germany

Operational data: not available

Economics: not available

Reference literature: [Eisenhut, 1988]

EP.2 Sealing of ascension pipes and charging holes

Description: During the coking period diffuse emissions from coke oven openings can be minimised by sealing these openings efficiently after the pushing and charging operation. However, such measures can only be successful when accompanied by careful maintenance and cleaning [Eisenhut, 1988].

Water-sealed ascension pipes are standard equipment at most new coke oven plants (Figure 6.12).

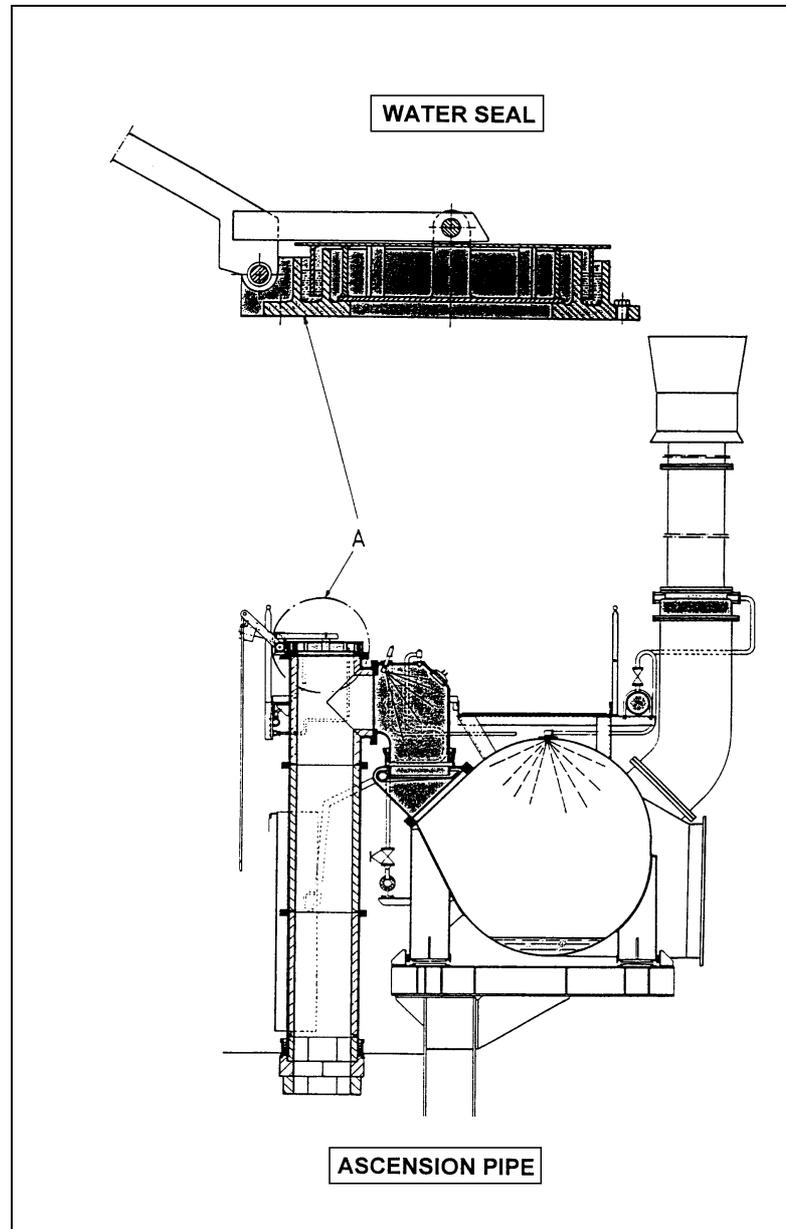


Figure 6.12 : Ascension pipe of a coke oven chamber

Additionally many older plants are also fitted with water-sealed ascension pipes. The water sealing can be very effective, providing that the water supply and run-off pipes are kept free of obstructions.

Nowadays, the best way to keep the charging holes leak-proof is to lute them carefully with a clay-water suspension.

Applicability: Applicable both at new and existing plants. In new plants the design of the ascension pipes and the charging holes can be optimised in order to reduce fugitive emissions.

Main achieved emission levels: : Water sealed ascension pipes will significantly reduce emissions of particulate matter, CO and hydrocarbons (see also Table 6.3). Expressed as a frequency of leaking (% of the total number of charging holes) possible achievements with luted lids are a maximum of 1% of charging holes showing visible leaks (see also Table 6.3).

Reference plants: Many plants in the world have water sealed ascension pipes. Furthermore, almost all plants lute the charging holes, either automatically or by hand.

Cross-media effects: Water sealed ascension pipes generate a wastewater flow. However, this wastewater can in some cases be recycled to the ammonia liquor tank or led to the ammonia stripper/still. Electricity is consumed by the pumps to circulate the water. Luting of charging holes causes no significant cross-media effects.

Economics: not available

Operational data: not available

Reference literature: [Eisenhut, 1988]

EP.3 Minimising leakage between coke oven chamber and heating chamber

Description: By performing systematic continuous maintenance of the coke oven, in accordance with PI.2, leakage through the brickwork can be avoided. Leaks through cracks in the brickwork allow COG to reach the flue gas from coke oven firing. This causes additional emissions of SO₂, particulate matter and hydrocarbons. The presence of cracks is easily detectable by the visible black smoke emissions from the coke oven stack during firing. However, it is not easy to identify which coke oven chamber is leaking. One technique for locating cracks is to fire the ovens on both sides of an empty oven. The position of the cracks is revealed by flames coming through the walls into the oven chamber.

Cracks, holes and other damage to the surface of the refractory brickwork can be efficiently repaired by oxythermic welding, silicon welding, and by wet or dry spraying with refractory cement. In some extreme cases it may be necessary to renew the brickwork.

Applicability: Technique which has to be used only at existing plants.

Main achieved emission levels: Emissions can be reduced to nearly zero when the spraying is properly applied and the occurrence of cracks is monitored. The quality and condition of the coke oven refractory walls are also very important in this respect.

Reference plants: Many plants in the EU.

Cross-media effects: not available

Operational data: not available

Economics: not available

Reference literature: [InfoMil, 1997]

EP.4 De-dusting of coke pushing

Description: Several systems have been developed to minimise coke side particulate matter emissions from coke pushing:

1. Coke side sheds, including evacuation and de-dusting.
The particulate matter is evacuated by means of a coke side shed and de-dusted in a fabric filter.
2. Draught-hood-system
Operates on the wet scrubber principle and utilises the thermal draft of the dust-laden gases, while keeping the coke dry and thus avoiding hydrogen sulphide formation.
3. Container car.
The coke is pressed directly from the coke oven chamber in a container car. The coke does not get into contact with oxygen and only small amounts of particulate matter are generated. Usually applied in combination with coke dry quenching (see PI.7)
4. Coke transfer machine with (integrated) hood and mobile de-dusting device.
The particulate matter is evacuated by means of integrated hoods on the coke transfer machine
5. Coke transfer machine with (integrated) hood, stationary duct and stationary gas cleaning, preferably by fabric filtration (Figure 6.13), so-called “Minister Stein System”.

During the whole pushing process the quenching car shall be positioned in the catchment area of the de-dusting collection system (use of one-point quenching car with (integrated) hoods on the coke transfer machine).

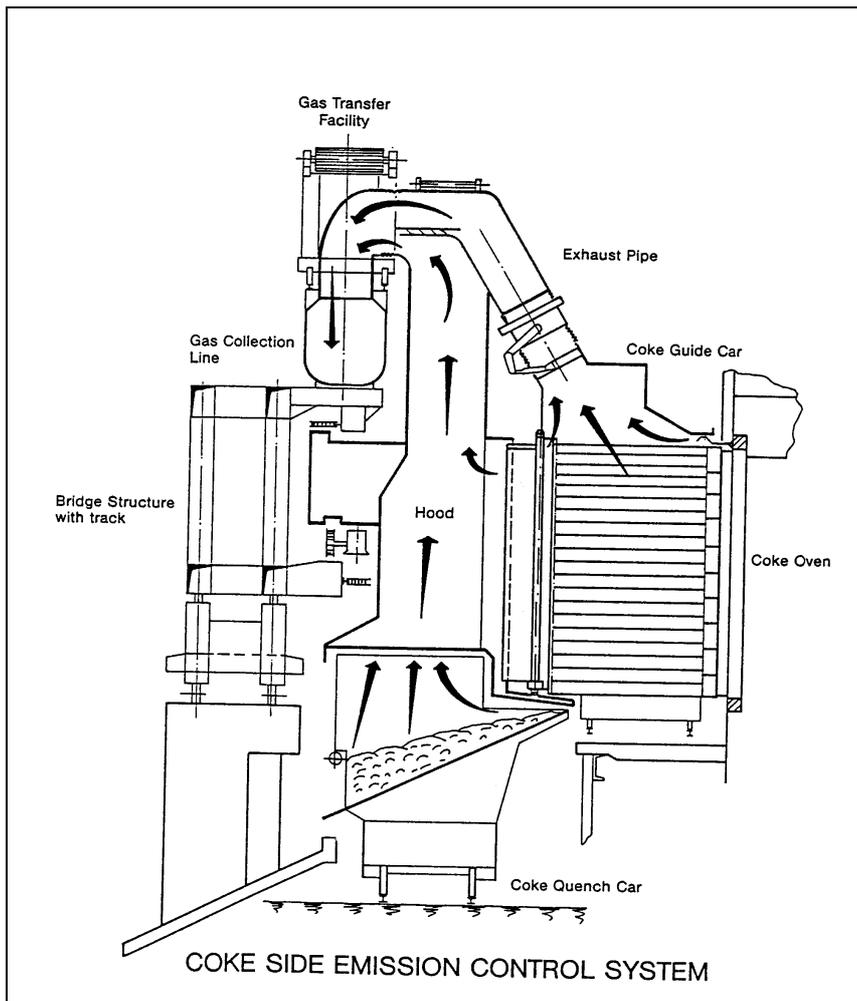


Figure 6.13 : Example for a de-dusting system for dust from coke pushing

Applicability: Coke side de-dusting is applicable both at new and existing plants. At existing plants, a tailor-made solution frequently has to be engineered. Sometimes lack of space at the plant is a constraint.

Main achieved emission level: Emission of particulate matter without abatement is about 500 g/t coke. Of the five techniques mentioned above, the "Minister Stein System" gives the best performance, achieving >99% collection efficiency, combined with good working conditions for operators (in contrast to coke side sheds). Emission factors (at the stack) below 5 g particulate matter/t coke can be achieved (see also Table 6.3).

At existing plants, a particulate matter collection rate of >99% is also achievable, as has been shown at the Ruhrkohle Hassel coking plant, Germany. A particulate matter concentration in the exhaust air of <30 mg/Nm³ was achieved. The system was also based on evacuation through a stationary duct.

Air extraction capacity is often in the order of 200000 Nm³/h in plants using the Minister Stein System but is depending on chamber size. A fabric filter is used to minimise particulate emissions.

Reported dust capture and removal efficiency of the Draught-hood-system was 95-96%.

Reference plants:

"Minister Stein" System:

Coke oven, United States Steel, Clairton Works, Pittsburgh, PA, USA;

Coke oven battery at Sidmar, B-Gent

Coke oven battery No.5 Ruhrkohle Hassel, Germany.

Coke oven Mannesmann Hüttenwerke Krupp, D-Huckingen;

Coke oven Prosper, D-Bottrop,;

Coke oven 1, Hoogovens IJmuiden, NL-IJmuiden;

Coke oven 2, Hoogovens IJmuiden, NL-IJmuiden .

Draft-hood-system:

ACZ de Carbonsiation, Sluiskil, The Netherlands

Cross-media effects: The operation of a de-dusting device requires energy to drive the vents for air evacuation. The collected solids can be recycled into the process.

Economics: not available

Operational data: not available

Reference literature: [Eisenhut, 1988; InfoMil, 1997]

EP.5 Emissions minimised wet quenching

Description: When coke is quenched in the quenching tower particulate matter and water mist are generated by the evaporation of quenching water from the glowing coke and emitted with the plumes. The quantities of entrained particulate matter depends on the operating conditions, coke properties and the type of water addition. Attempts have been made to reduce particulate matter and water vapour emissions by means of constructional and other measures; for example, by spraying the plume with water.

Optimum solutions include the use of lamella stack baffles and favourable design of the quenching tower (Figure 6.14).

In addition “Swamp”/top quenching can be applied as a modification to quench the coke with water. Then, the water is partly injected through a piping system in the bottom of the quenching car (drowning the coke) and partly sprayed on the top of the coke (whereas in most wet quenching systems the water is only sprayed on the top of the coke). Emissions of particulate matter is thus reduced. Nevertheless the quenching tower itself is the same like for top quenching with the same dust arrestment device (Figure 6.14). One disadvantage of “swamp” quenching is the emission of pieces of coke and of the quenching car due to “explosive” formation of steam under and inside the coke mass in the quenching car.

A special type of wood is used for the load-bearing structure. The actual dust catching equipment consists of individual frames in which the louvre-shaped plastic lamellas are installed.

Applicability: Applicable at both new and existing plants. Existing quenching towers can be equipped with emission reduction baffles. A minimum tower height of at least 30m is necessary in order to ensure sufficient draft conditions.

Main achieved emission level: Emission of particulate matter during wet quenching without reduction measures is about 200 – 400 g/t coke. With the system described it can be reduced at least to 50 g /t coke (with an emission factor before abatement of maximum 250 g/t coke and a solids content of quenching water below 50 mg/l). This emission factor is guaranteed by the only European supplier [Nathaus, 1997]. In practice emissions of less than 25 g/t coke are normally achieved. It must be noted that representative measurements are difficult. The mentioned emission factors have been determined by the VDI 2303 method (Guideline for Sampling and Measurement of Dust Emissions from Wet Quenching).

Reference plants: examples of recently built or retrofitted quenching towers with emission reduction baffles are located at:

- Sidmar, B-Gent
- Hüttenwerke Kruppmanneßmann GmbH, D-Duisburg
- Kokerei Kaiserstuhl, D-Dortmund
- Kokerei Hassel, D-Gelsenkirchen
- Preussag Stahl AG, D-Salzgitter

All these plants achieve emission factors less than 25 g dust/t coke.

Cross-media effect: Additional energy is consumed in water spraying, although this is not significant.

Operational experience: There are many quenching towers round the world equipped with emission reduction baffles. Good operational experience is reported.

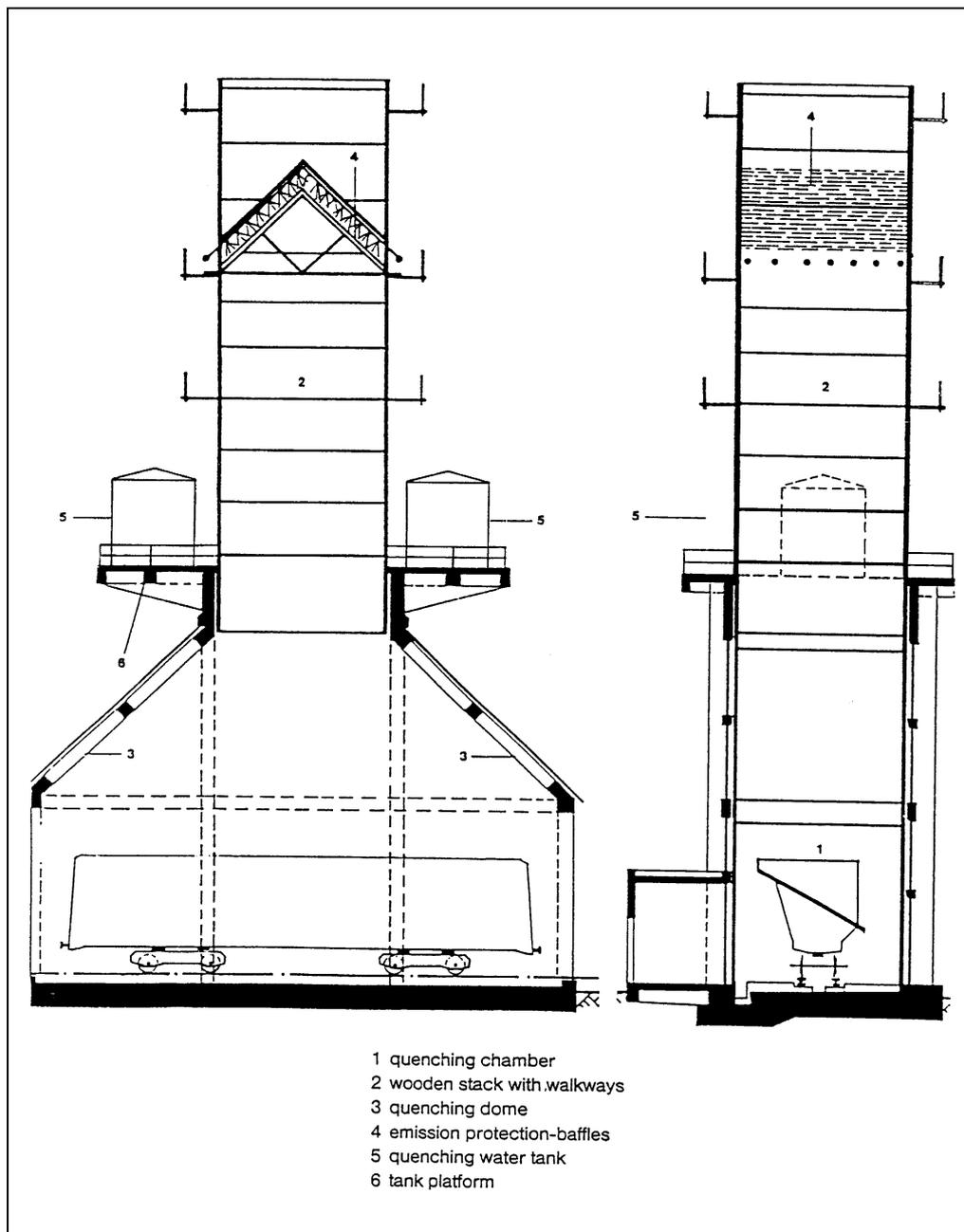


Figure 6.14 : Schematic diagram of a quenching tower with emission reduction baffles

Economics: The costs for the retrofitting of an existing quenching tower with emission reduction baffles are in the order of 150000 – 200000 Ecu₁₉₉₇. The investment costs for a complete quenching tower with this system are up to 11 MEcu₁₉₉₇ (tower of the Kokerei Kaiserstuhl plant, D-Dortmund, which is the biggest in the world (15 x 15 x 50 m)).

Driving force for implementation: Normally legal requirements and enforcement by local authorities initiate the retrofitting of existing quenching towers.

Literature: [Nathaus,1997]

EP.6 DeNO_x of waste gas from coke oven firing

Description: The NO_x emissions from the coke oven firing are preferably minimised by process-integrated measures, but end-of-pipe techniques may also be applied.

In the SCR process, NO_x in the flue gas is catalytically reduced by ammonia (NH₃) to N₂ and H₂O. Vanadium pentoxide (V₂O₅), tungsten oxide (WO₃) on a titanium oxide (TiO₂)-carrier are often used as a catalyst. Other possible catalysts are iron oxide and platinum. Optimal operating temperatures are in the 300 to 400°C range. Such high temperatures decrease the energy recovery in the regenerators (180-250°C is optimum) of the coke ovens or necessitate additional heating of the waste gas.

Special attention should be paid to de-activation of the catalyst, the accumulation of the explosive ammonium nitrate (NH₄NO₃), ammonia slip and the formation of corrosive SO₃.

Applicability: SCR is only applicable to new plants and, if the lay-out permits, to fully rebuilt plants.

Main achieved emission level: A de-NO_x efficiency of 90% can be achieved. There is not much experience in the sustainability in case of application to coke oven plants.

Reference plants: De-NO_x of waste gas from coke oven firing is only rarely applied. Only one case is known:

Coke oven battery 2, 3 and 6, Kawasaki Steel, Chiba Works, Japan.

Cross-media effects: An increase in energy consumption and ammonia consumption occurs. Part of the catalyst has to be deposited when it is not effective any more.

Operational data:

In 1976, an SCR was commissioned at Kawasaki Steel Chiba Works to treat the waste gas from coke oven firing. The SCR has a capacity of 500000 Nm³/h. The operation temperature is 240°C and the reduction medium is ammonia (NH₃). De-NO_x efficiency is 90%. The formation of ammonium sulphate and accumulation of dust means the catalyst had to be regenerated once a day by heating the waste gases to >260°C.

In 1992 the SCR was taken out of operation with the closing down of batteries 2 to 4.

Economics:

Investments: 47 million Ecu₁₉₉₆ in 1976

Operational costs: not available

Additional economics:

[InfoMil, 1997] reports the following costs for SCR in general:

Investment: 50 Ecu₁₉₉₆/(Nm³/h) ± 30%;

For a coke oven plant with a battery flue gas flow of 300000 Nm³/h and a production of 1 Mt coke per year this would lead to:

Investment: 15 million Ecu₁₉₉₆ ± 5 million

Operational: 0.17-0.51 Ecu₁₉₉₆/t coke.

Reference literature: [InfoMil, 1997]

EP.7 Coke oven gas desulphurisation

Description: Because of its hydrogen sulphide (H_2S) content (up to 8 g/Nm^3) unpurified coke oven gas is unsuited for use on many industrial applications. When the gas has been desulphurised, however, its use for a variety of applications becomes viable. Many plants now sell coke oven gas after desulphurisation at a profit. Desulphurisation for commercial reasons coincides with the need to protect the environment from the effect of "acid rain". because desulphurised coke oven gas decreases emissions of SO_2 at the site of coke oven gas combustion. In many cases, sulphur is removed in two stages: a low-pressure stage and a high-pressure stage. Although COG desulphurisation is not yet common practice in the EU 15 its implementation is becoming increasingly common.

Coke oven gas also contains a variety of organic sulphur compounds such as carbon disulphide (CS_2), carbon oxysulphide (COS), mercaptans, etc. (approximately 0.5 g/Nm^3 total). However, there is only limited knowledge so far about recovering organic sulphur compounds from coke oven gas.

As described in 6.1.3.3, there are two main types of coke oven gas desulphurisation processes: wet oxidative processes and absorptive processes. The absorptive processes combine H_2S removal and processing with ammonia (NH_3) removal and processing. Table 6.11 lists the different processes and their characteristics.

Wet oxidative processes		Absorption/stripping processes	
Name	Description	Name	Description
Stretford	H_2S is scrubbed from the coke oven gas by a sodium carbonate solution (Na_2CO_3) and elemental sulphur (S^0) is yielded using vanadate (VO_3) as an intermediate. Regeneration of the scrubbing liquid takes place by aeration (O_2), using anthraquinone disulphonic acid (ADA) as an intermediate.	Carl Still, Diamex or ASK*	H_2S is scrubbed from the coke oven gas by a NH_3 solution. The NH_3 solution is derived from the NH_3 scrubber. The H_2S and NH_3 are stripped from the washing liquor by steam stripping and the vapours are led to a Claus plant or a sulphuric acid plant.
Takahax	Similar to the Stretford process, except that 1,4-naphthoquinone-2-sulphonic acid is used as an intermediate for the regeneration.	Vacuum carbonate	H_2S (and also HCN and CO_2) is scrubbed from the COG with a sodium carbonate solution or a potassium carbonate solution (Na_2CO_3 or K_2CO_3). The potassium variant allows higher carbonate concentrations. The washing liquor is regenerated in a column, using high temperature and low pressure (0.12-0.14 bar). Acid gases are stripped from the liquor and can be treated in a Claus plant or a sulphuric acid plant.
Thylox	Sodium thioarsenate ($Na_4As_2S_5O_2$) binds the H_2S and regeneration is done by oxygen treatment. Elemental sulphur is yielded.	Sulfiban	Coke oven gas is scrubbed with monoethanolamine (MEA). NH_3 removal prior to H_2S removal is necessary to avoid pollution of the washing liquor. H_2S is stripped from the MEA solution using steam and can be treated in a Claus plant or a sulphuric acid plant. Insoluble organic S-compounds are removed from the MEA solution as solid waste.
Perox	The gas is scrubbed with an ammonia solution. Parabenzoquinone is used for sulphur oxidation and regeneration of the scrubbing liquor is done by oxygen.	DESULF	Virtually the same as the ASK-process, but the NH_3 is removed from the NH_3/H_2S vapour in saturators, producing ammonium sulphate ($(NH_4)_2SO_4$).
Fumaks-Rhodacs	H_2S is oxidised by picric acid in the Fumaks-phase, yielding elemental sulphur. Cyanides are recovered in the Rhodacs phase.		

- ASK: Ammoniumsulphide Kreislaufwäscher

Table 6.11 : Coke oven gas desulphurisation processes and their characteristics – based on [UN-ECE, 1990; EC Coke, 1996]

In Europe, the most commonly applied process is the absorptive process using an ammonia liquor to scrub the H_2S from the coke oven gas (Carl Still, Diamex or Ammoniumsulphide Kreislaufwäscher (ASK) process). Figure 6.15 shows a recently installed example of the (ASK) process.

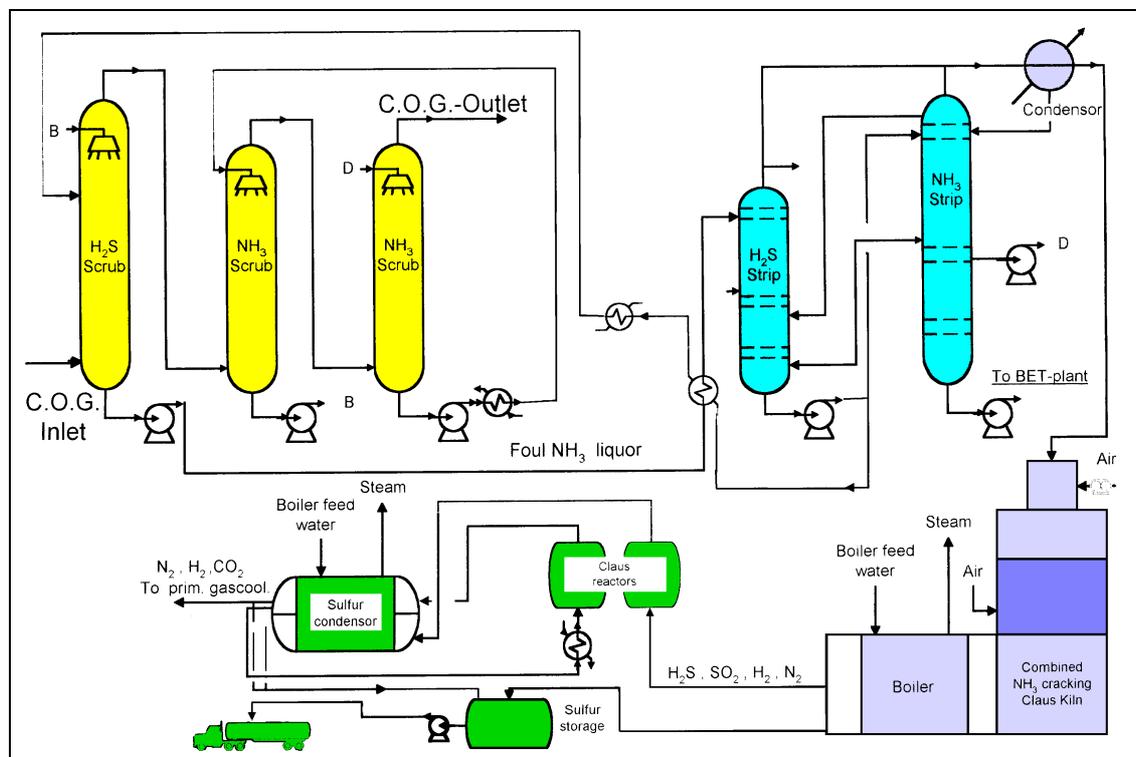


Figure 6.15 : Flow diagram of a COG desulphurisation plant (ASK-process) installed in 1997

The most commonly applied wet oxidative process is the "Stretford" process.

The Stretford process is applicable in a wide range of desulphurisation capacities. Reported coke oven gas desulphurisation design capacities range from 400 to 110000 Nm^3/h .

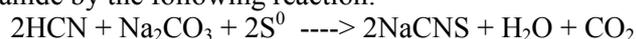
Applicability: Coke oven gas desulphurisation of both the wet oxidative and the absorptive type can be applied at new and existing plants. The choice depends on the cleaned coke oven gas specifications, environmental considerations, the integration within the gas cleaning plant, etc.

Main achieved emission level: Wet oxidative processes have a better desulphurisation efficiency than absorptive processes. Wet oxidative processes can have an efficiency of $>99.9\%$, achieving residual H_2S concentrations as low as $1 \text{ mg}/Nm^3$ in the coke oven gas. Absorptive processes do not usually exceed 95% desulphurisation efficiency and residual H_2S concentrations in the coke oven gas are usually between $500\text{-}1000 \text{ mg}/Nm^3$.

None of the available techniques remove organic sulphur compounds with a high efficiency. In the low pressure stage of gas cleaning, organic sulphur compounds are only reduced from $0.5 \text{ g}/Nm^3$ to $0.2\text{-}0.3 \text{ g}/Nm^3$ [Eisenhut, 1988].

Reference plants: Table 6.12 gives an overview of reference plants (not exhaustive).

Cross-media effects: Any wet oxidative process used for desulphurising coke oven gas will remove most of the hydrogen cyanide from the coke oven gas as well and form sodium thiocyanide by the following reaction:



The sodium thiocyanide and the small amounts of sodium sulphate and thiosulphate formed by side reactions are not regenerated by the process and build up in the circulating liquor.

It is therefore necessary to purge a liquid stream to prevent salting out of the chemicals. In the case of the Stretford process, this discharge flow contains vanadium compounds, quinone and hydroquinone compounds (from ADA), thiocyanide and thiosulphate. Discharge of these components is undesirable from an environmental and economic point of view (water pollution and loss of expensive chemicals).

To reduce chemical consumption, cyanic acid (HCN) can be removed prior to desulphurisation in a pre-washer, using a sodium polysulphide or ammonium polysulphide solution. Pre-removal of HCN does not reduce the total volume of effluent produced.

Economics: An overview of the economics of the two main types of coke oven gas desulphurisation in Europe (ASK and Stretford) is given in Table 6.13. [Rothery, 1987] reports that an increase in desulphurisation efficiency from 95 to 99.9% in the Stretford process only costs 10% extra. In the Stretford process, the most important operational cost items are the capital charges and the chemicals (see Table 6.11). Coke oven gas with high HCN concentrations ($> 2 \text{ g/Nm}^3$) consumes relatively large amounts of chemicals. In this case it might be beneficial to install a HCN prewash before the Stretford unit.

Reference literature: [Rothery, 1987; InfoMil, 1997]

Wet oxidative	
Stretford	Dofasco, Hamilton, Canada; British Steel, Orgreave, United Kingdom; British Steel, Redcar, United Kingdom; Metarom, Romania; Erdemir, Turkey; Sollac, France; Kobe Steel, Kakogawa Works, Japan; Posco, Korea;
Takahax	Nippon Steel, Yawata Works, Japan; Nippon Steel, Nagoya Works, Japan; Nippon Steel, Hirohata Works, Japan; Nippon Steel, Oita Works, Japan; Nippon Steel, Muroran Works, Japan; Nippon Kokan, Fukuyama Works, Japan; Nippon Kokan, Keihin Works, Japan;
Fumax	Nippon Steel, Kimitsu Works, Japan; Sumitomo Metal Industries, Wakayama Works, Japan; Sumitomo Metal Industries, Kashima Works, Japan; Kawasaki Steel, Chiba Works, Japan;
Thylox	Not available
Absorptive processes	
Carl Still or Diamex or ASK	Prosper, Bottrop, Germany; Thyssen Stahl, Duisburg, Germany; Zentral Kokerei Saar, Dillingen, Germany; Kawasaki Steel, Mizushima Works, Japan; Coke oven No.2, Hoogovens IJmuiden, The Netherlands; Coke oven of Sidmar, B-Gent
Vacuum carbonate	ACZ de Carbonisation, Sluiskil, The Netherlands;
Sulfiban	Coke oven No.1, Hoogovens IJmuiden, The Netherlands; Nippon Kokan, Keihin Works, Japan;

Table 6.12 : Table of reference plants for coke oven gas desulphurisation processes – [InfoMil, 1997]

Costs and efficiency	Unit	Stretford process	ASK process	Vacuum carbonate
H ₂ S removal efficiency	[%]	95-99.5	90-97	>90
Capital costs	(1·10 ⁶ Ecu ₁₉₉₆)	4.43-5.99	n/a	19.3*
Operating costs	(Ecu ₁₉₉₆ /1000 Nm ³ COG)			
Chemicals		1.06-1.92	-	
Electricity		0.46-0.60	-	
Steam & water		0.12	-	
Maintenance		0.25-0.30	-	
Labour		0.36	-	
Capital charges		2.25-3.04	-	1.90
Gross costs		4.49-5.61	n/a	2.38
Sulphur credit		0.33-0.57	n/a	0.48
Total net cost		3.92-5.28	n/a	1.90

* including sulphuric acid plants and calculated from case with 30000 Nm³/h and 6 g H₂S/Nm³

Table 6.13 : Typical operating and capital costs for the desulphurisation of 45000 Nm³/h coke oven gas, containing 8 g/Nm³ H₂S – [Rothery, 1987; InfoMil, 1997]

EP.8 Removing tar (and PAH) from the coal water

Description: The ammonia-still effluent normally does not contain tar but when it does this tar has an adverse effect on the operation of the biological wastewater treatment. In particular, the presence of polycyclic aromatic hydrocarbons (PAH) in the tar may cause problems as the PAH may have a toxic effect on the micro-organisms in the activated sludge and are relatively hard to degrade. Therefore, it is advisable to remove tar from the coal water prior to the biological treatment of the water.

The tar can be removed by adding coagulating chemicals and subsequent separation using a technique such as:

- gravitational sedimentation followed by filtration;
- centrifuging the coal water followed by filtration;
- Flotation followed by filtration;

This treatment removes most of the tar from the wastewater in the form of a highly concentrated filter cake or sludge. This has to be treated further, e.g. by recycling to the coke ovens.

Another method of removing tar from the wastewater is to use a rotary vacuum drum filter pre-coated with wood flour. This system is installed to decrease PAH concentrations in the coal water prior to biological treatment. Nevertheless the wood flour filter, which had been installed at Hoogovens IJmuiden, NL-IJmuiden, has been replaced in 1998 by sand filtration achieving the same PAH removal efficiency.

Applicability: Removal of tar prior to wastewater treatment is applicable both at new and existing plants

Main achieved emission level: Effluent concentrations from the wood flour filter are 700-800 µg/l (EPA-PAH) at a removal efficiency of 99%. Emissions after the biological treatment are drastically reduced. This would correspond to a EPA-PAH emission of the biological treatment of 150 µg/l in the effluent. The PAH-laden wood-flour is recycled to the coke oven.

Reference plants:

Sedimentation and filtration:

ACZ de Carbonisation, Sluiskil, The Netherlands

Wood-flour filter:

Coke oven, Hoogovens IJmuiden, NL- IJmuiden

Cross-media effects: All of these tar removal techniques generate a waste. However, this tar-laden waste can be recycled into the coke ovens.

Operational data: not available

Economics: not available

Reference literature: [InfoMil, 1997]

EP.9 Ammonia stripper

Description: Keeping the concentration of ammonia in the stripper and still effluent low benefits the operation of a biological wastewater treatment plant. The removal efficiency strongly depends on the alkaline and steam addition and on the design of the stripper (i.e. the number of trays). A larger dose of NaOH and an increase in the number of trays can significantly reduce the ammonia concentration in the effluent.

When plant effluent treatment involves nitrification and subsequent denitrification, ammonia stripping of the effluent is less critical. In this case an economic and environmental optimum between ammonia stripping and ammonia removal in the biological wastewater treatment plant should be found.

Applicability: Applicable both at new and existing plants

Main achieved emission level: Effluent ammonia concentrations may vary from 20 to 150 mg/l, depending on steam and alkali dosage and stripper design. Values between 20 and 40 mg/l are achievable but may not be required because of adjustment of an appropriate balance of BOD₅/P/N ratio in the wastewater prior to biological treatment.

Reference plants: Almost all coke oven plants around the world use an ammonia stripper.

Cross-media effects: Strippers consume energy in the form of steam (0.1-0.2 t steam/m³ wastewater) and consume alkalis (NaOH; 6-22 l/m³). Formerly, lime was used instead of NaOH. Higher doses of steam and alkalis lead to lower NH₃ concentrations in the effluent. Furthermore, an ammonia (and H₂S) laden steam is generated, which must be treated, for example in a sulphuric acid plant, a Claus plant, or in ammonium sulphate crystallisation units.

Operational data: not available

Economics:

Ammonia stripper, treating 150 m³ effluent per hour

Investments: 0.75-0.9 million Ecu₁₉₉₆ in 1993 for the columns

Operational costs: 0.18 Ecu₁₉₉₆/m³ in 1993

Reference literature: [InfoMil, 1997]

EP.10 Gas-tight operation of the gas treatment plant

Description: In the gas treatment plant the crude coke oven gas is cleaned in several stages for subsequent use as a fuel. The relatively volatile character of the crude coke oven gas components means emissions can occur at flanges, pressure valves, pumps etc. Apart from the negative environmental effects, occupational safety considerations are also important as some COG compounds are known carcinogens (e.g. PAH and benzene). The BTX plant incorporated in the coke oven gas treatment plant is particularly important in this respect. In this plant BTX (which mainly consists of benzene, toluene and xylenes) is scrubbed from the coke oven gas by means of a scrubbing liquid. Subsequently, the loaded scrubbing liquid is regenerated and the BTX is recovered and can be sold.

The gas-tight operation of a gas treatment plant is mainly a safety issue.

All measures to enable virtually gas-tight operation of the gas treatment plant should be considered:

- Minimising the number of flanges by welding piping wherever possible;
- Use of gas-tight pumps (e.g. magnetic pumps or integral pumps);
- Avoiding emissions from pressure valves in storage tanks. This is most commonly achieved by connecting the valve outlet to the coke oven gas collecting main (collection of the gases and subsequent combustion or gas blanketing or vent scrubbers could also be possible).

Applicability: Applicable both at new and existing plants. In new plants, a gas-tight design will probably be easier to achieve than at existing plants.

Reference plants: Voest Alpine Stahl AG, A-Linz, Hoogovens IJmuiden, NL-IJmuiden and Sidmar, B-Gent operate a virtually gas-tight gas treatment plant to treat the coke oven gas.

Cross-media effects: No cross-media effects occur.

Operational data: not available

Economics: not available, part of plant design.

Reference literature: [InfoMil, 1997]

EP.11 Wastewater treatment plant

The wastewater from a coke oven plant contains a mixture of hydrocarbons, cyanide-compounds and nitrogen compounds in relatively high concentrations. Several methods are available to treat this wastewater. In all cases, the wastewater is led through an ammonia stripper before further treatment (see 6.2.3.1.2).

The wastewater can be treated biologically and chemically. When biological treatment is applied, tar is often removed by a chemical/physical method (see EP.9) and the wastewater is often diluted in order to avoid the influent's having toxic effects on the micro-organisms, especially the inhibition of nitrifying bacteria.

The most commonly applied biological technique for the treatment of coke oven wastewater is the aerobic biological system with activated sludge. In some cases special attention has been paid to nitrification and (anoxic) denitrification.

In some cases a biological system based on a fluidised bed is used to treat the wastewater. In the UK there is one plant in which wastewater is treated in reed beds.

a. Aerobic system with activated sludge

Description: In the aerobic system with activated sludge, the biodegradable contaminants are mainly biologically degraded to CO₂, H₂O and minerals and the non-degradable, non-polar components (like most PAH and heavy metals) are removed from the water phase by partial adsorption to the activated sludge.

In practice, most of the potentially hazardous contaminants, such as phenols, cyanides and aromatic hydrocarbons, are biologically degraded and heavy metals are partially removed by adsorption to the activated sludge.

Activated sludge systems with a low food-microorganism ratio (F/M) are preferred from an environmental point of view. A low F/M ratio enables also biodegradation of heavily biodegradable organic compounds. The F/M ratio is the ratio of organic matter to activated sludge (as mixed-liquor suspended solids (MLSS)) and it is expressed in "kg COD/kg MLSS/d"; in which COD is the chemical oxygen demand.

Aeration may use oxygen instead of ambient air. This increases process control and reduces "stripping" of volatile components in the wastewater. For example, oxygen aeration is used at ACZC in NL-Sluiskil and at Sibmar, B-Gent.

Main achieved emission levels: Table 6.14 gives the performance figures for aerobic activated sludge systems at European coke oven plants.

Component	Concentration	Unit	Specific emission value	Unit
COD	140 – 700	[mg/l]	0.2 - 1	[kg/t coke]
N-Kjeldahl	20 – 120	[mg/l]	0.01 – 0.1	[kg/t coke]
NH ₃	<1 – 100	[mg/l]	0 – 0.1	[kg/t coke]
SCN ⁻	<0.1 – 35	[mg/l]	0 – 0.05	[kg/t coke]
Phenol	<0.1 – 10	[mg/l]	0 – 0.005	[kg/t coke]
PAH	0.003 – 0.2	[mg/l]	0 – 0.001	[kg/t coke]

Table 6.14 Effluent concentrations and specific emissions of European coke oven plants using aerobic activated sludge wastewater treatment (both with high and low F/M ratio) – [EC Coke, 1996]

b. Nitrification concept

Description: Some wastewater treatment plants are designed to remove ammonium (NH_4^+) efficiently by means of nitrification. The traditional design of an aerobic activated sludge system can be taken as a starting point for this kind of plant. The system should have a very low F/M ratio and a high recirculation rate in order to avoid the slow-growing nitrification bacteria being washed out. The nitrification bacteria convert the ammonium into nitrate (NO_3^-). Under such conditions heavily biodegradable organic compounds can also be mineralised with high removal efficiency.

Main achieved emission levels: Generally, systems with a low specific load have a better performance and allow the degradation of ammonia by means of nitrification. A low specific load also enhances the degradation of organic compounds with a low degradation rate. If "nitrification" is applied, the effluent nitrate (NO_3^-) concentrations will be relatively high (in the order of 200 mg/l).

Reference plants: The same effluent treatment plants mentioned under "c" (nitrification-denitrification concept)

c. Nitrification-denitrification concept

Description: In some cases, local authorities have demanded low discharges of all nitrogen compounds (including nitrates) from the effluent. This requires additional anoxic treatment of the wastewater. Several plant lay-outs are possible, but good results have been obtained at wastewater treatment plants with the so-called pre-denitrification-nitrification concept (pre-DN/N).

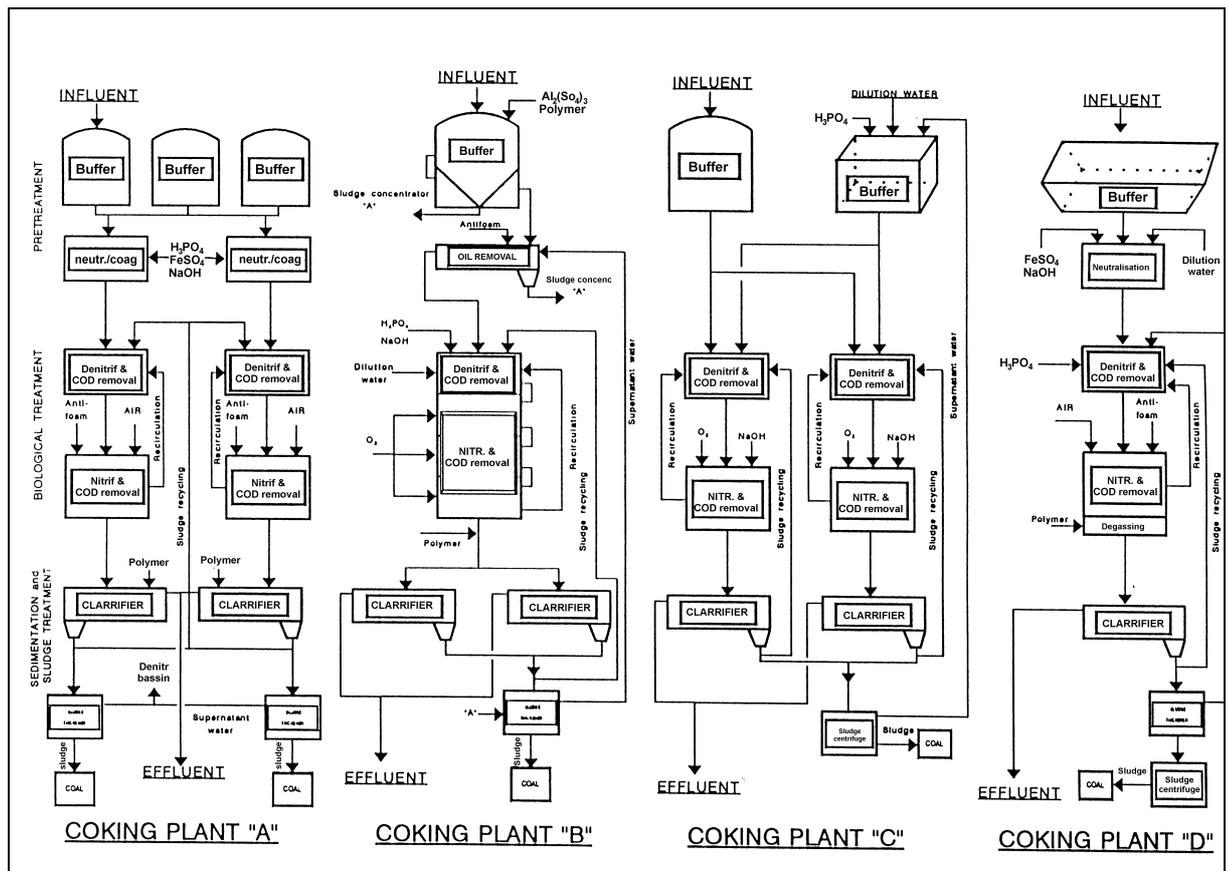
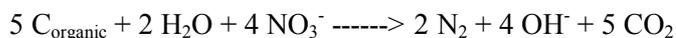


Figure 6.16 : Block diagrams of four different effluent treatment plants for wastewater from coke ovens with nitrification/denitrification – [Löhr, 1996];

Plant "A": Kokerei Kaiserstuhl, D-Dortmund; Plant "B": Kookfabriek Sidmar, B-Gent
Plant "C": Kookfabriek ACZC, Netherlands; Plant "D": Cokerie de Serémange, France

In the pre-DN/N-system, the aerobic activated sludge system is also used as a starting-point. However, before the wastewater is aerated, nitrate-rich water from the nitrification-step is added. Under anoxic conditions, bacteria use the nitrate as terminal electron acceptor instead of molecular oxygen (O₂). The nitrogen is emitted as molecular nitrogen (N₂). The overall reaction is:



Main achieved emission levels: Nitrification-denitrification systems have a very low F/M ratio (0.05-0.2 kg COD/kg MLSS/d) and achieve very good results in cleaning coke oven wastewater. Nitrogen emissions from this system are especially low compared to systems with a high F/M ratio or nitrification alone. Influent and effluent concentrations at four of these plants are given in Table 6.15.

Reference plants:

Coke oven wastewater treatment plants with the pre-DN/N system have been installed in the United Kingdom (British Steel Corp. Scunthorpe and Orgreave), Belgium (Sidmar, B-Gent), France (Lorfonte Serémange), Germany (Ruhrkohle AG Kaiserstuhl, D-Dortmund) and the Netherlands (ACZC Sluiskil) (see Figure 6.16).

Aspect/Component	Unit	Sidmar	Serémange	Kaiserstuhl	ACZC
Coke production	[Mt/a]	1.180	0.630	2.0	0.605
Coal water flow	[m ³ /h]	42	16	71	20
Dilution	[m ³ /h]	40	5	(15)***	40
Total Flow	[m ³ /h]	86	30	72	59
Specific effluent flow	[m ³ /t coke]	0.59	0.31	0.38	0.86
Food-microorganism (F/M) ratio	[kg COD/kg MLSS/d]	≤ 0.15	n/a	≤ 0.15	0.17
Influent					
pH	-	n/a	n/a	9.5	9.3
Suspended solids	[mg/l]	30-40	n/a	<10	n/a
COD (x±s)	[mg/l]	3650±310	4450±460	2250±590	3600±500
BOD ₅	[mg/l]	2100	n/a	1340	n/a
Phenol	[mg/l]	964	1000	350	900
SCN ⁻	[mg/l]	355	380	200-250	72.5**
N-Kjeldahl	[mg/l]	n/a	n/a	n/a	n/a
Ammonia (x±s)	[mg/l]	83±61	82±105	43±30	25±15
Nitrite	[mg/l]	n/a	n/a	-	n/a
Nitrate	[mg/l]	n/a	n/a	-	n/a
Oil and tar	[mg/l]	40	n/a	n/a	n/a
PAH (6 Borneff)	[µg/l]	200	n/a	106	n/a
Effluent					
pH	-	n/a	n/a	7.8	7.04
Suspended solids	[mg/l]	42	n/a	43	28.6
COD (x±s)	[mg/l]	189±30	486±315	228±48	117±29
BOD ₅ (x±s)	[mg/l]	19±2.8	19±?	4±32	5.4±1.9
Phenol	[mg/l]	0.06	<2	0.16	0.1
SCN ⁻	[mg/l]	3	<4	n/a	0.9
N-Kjeldahl	[mg/l]	n/a	n/a	n/a	7.0
Ammonia (x±s)	[mg/l]	0.7±0.8	32±53	1.3±0.2	0.05±0.6
Nitrite	[mg/l]	3	0	0.21	1.0
Nitrate	[mg/l]	65	0*	15.0	48.1
Oil and tar	[mg/l]	5	n/a	n/a	n/a
PAH (6 Borneff)	[µg/l]	<50	n/a	11.7	2.7
n/a	not available; x±s arithmetic average ± standard deviation				
*	Lorfonte reports this value, although in theory it is impossible; but high residual ammonia content and negligible nitrate concentration indicate nitrification inhibition.				
**	ACZC has indicated that this value might be too low, due to the sampling technique used.				
***	consists of rain water, thus not continuously added [Löhr, 1997]				

Table 6.15 : Influent and effluent concentrations and some aspects of wastewater treatment systems with the pre-DN/N system – [InfoMil, 1997; Löhr 1996; Löhr, 1997]

Cross-media effects: The energy consumption of the ACZC plant is 0.008 GJ/t coke.

Economics:

Investment: 0.6 million Ecu₁₉₉₆ in 1994 for an extension of a nitrification system to a Pre-DN/N-system. Total investments are 4.6 million Ecu₁₉₉₆.

Operational costs: 345000 Ecu₁₉₉₆ per year (0.57 Ecu₁₉₉₆/t coke), including wastewater levies.

Reference literature: [EC Coke, 1996; InfoMil, 1997; Löhr 1996; Löhr, 1997]

6.4 Conclusions

In understanding this chapter and its contents, the attention of the reader is drawn back to the preface of this document and in particular the fifth section of the preface: “How to understand and use this document”. The techniques and associated emission and/or consumption levels, or ranges of levels, presented in this chapter have been assessed through an iterative process involving the following steps:

- identification of the key environmental issues for the sector; for coke oven plants these are the prevention of fugitive smoke, the treatment of coke oven gas, and wastewater treatment, particular considering ammonia;
- examination of the techniques most relevant to address those key issues;
- identification of the best environmental performance levels, on the basis of the available data in the European Union and world-wide;
- examination of the conditions under which these performance levels were achieved; such as costs, cross-media effects, main driving forces involved in implementation of this techniques;
- selection of the best available techniques (BAT) and the associated emission and/or consumption levels for this sector in a general sense all according to Article 2(11) and Annex IV of the Directive.

Expert judgement by the European IPPC Bureau and the relevant Technical Working Group (TWG) has played a key role in each of these steps and in the way in which the information is presented here.

On the basis of this assessment, techniques, and as far as possible emission and consumption levels associated with the use of BAT, are presented in this chapter that are considered to be appropriate to the sector as a whole and in many cases reflect current performance of some installations within the sector. Where emission or consumption levels “associated with best available techniques” are presented, this is to be understood as meaning that those levels represent the environmental performance that could be anticipated as a result of the application, in this sector, of the techniques described, bearing in mind the balance of costs and advantages inherent within the definition of BAT. However, they are neither emission nor consumption limit values and should not be understood as such. In some cases it may be technically possible to achieve better emission or consumption levels but due to the costs involved or cross media considerations, they are not considered to be appropriate as BAT for the sector as a whole. However, such levels may be considered to be justified in more specific cases where there are special driving forces.

The emission and consumption levels associated with the use of BAT have to be seen together with any specified reference conditions (e.g. averaging periods).

The concept of “levels associated with BAT” described above is to be distinguished from the term “achievable level” used elsewhere in this document. Where a level is described as “achievable” using a particular technique or combination of techniques, this should be understood to mean that the level may be expected to be achieved over a substantial period of time in a well maintained and operated installation or process using those techniques.

Where available, data concerning costs have been given together with the description of the techniques presented in the previous chapter. These give a rough indication about the magnitude of costs involved. However, the actual cost of applying a technique will depend strongly on the specific situation regarding, for example, taxes, fees, and the technical characteristics of the installation concerned. It is not possible to evaluate such site-specific

factors fully in this document. In the absence of data concerning costs, conclusions on economic viability of techniques are drawn from observations on existing installations.

It is intended that the general "BAT" in this chapter could be used to judge the current performance of an existing installation or to judge a proposal for a new installation and thereby assist in the determination of appropriate "BAT" based conditions for that installation. It is foreseen that new installations could be designed to perform at or even better than the general "BAT" levels presented here. It is also considered that many existing installations could reasonably be expected, over time, to move towards the general "BAT" levels or do better.

While the BREFs do not set legally binding standards, they are meant to give information for the guidance of industry, Member States and the public on achievable emission and consumption levels when using specified techniques. The appropriate limit values for any specific case will need to be determined taking into account the objectives of the IPPC Directive and the local considerations.

For coke oven plants, the following techniques or combination of techniques are considered as BAT. The order of priority and the selection of the techniques will differ according to the local circumstances. Any other technique or combination of techniques achieving the same or better performance or efficiency can also be considered; such techniques may be under development, an emerging technique or a technique already available but not mentioned/described in this document.

1. General:
 - Extensive maintenance of oven chambers, oven doors and frame seals, ascension pipes, charging holes and other equipment (systematic programme carried out by specially trained maintenance personnel);
 - Cleaning of doors, frame seals, charging holes and lids and ascension pipes after handling.
 - Maintaining a free gas flow in the coke ovens.
2. Charging:
 - Charging with charging cars.
From an integrated point of view, "smokeless" charging or sequential charging with double ascension pipes or jumper pipes are the preferred types, because all gases and particulate matter are treated as part of coke oven gas treatment. If however the gases are extracted and treated outside the coke oven, charging with land-based treatment of the extracted gases is the preferred method. Treatment should consist of efficient evacuation and subsequent combustion and fabric filtration Emissions of particulate matter < 5 g/t coke are achievable.
3. Coking:

A combination of the following measures:

 - Smooth, undisturbed coke oven operation, avoiding strong temperature fluctuations;
 - Application of spring-loaded flexible-sealing doors or knife edged doors (in case of ovens ≤ 5 m high and good maintenance) achieving:
 - <5% visible emissions (frequency of any leaks compared to the total number of doors) from all doors in new plants *and*
 - <10% visible emissions from all doors in existing plants.
 - Water-sealed ascension pipes, achieving <1% visible emissions (frequency of any leaks compared to the total number of ascension pipes) from all pipes;
 - Luting charging holes with clay-suspension (or other suitable sealing material), achieving <1% visible emissions (frequency of any leaks compared to the total number of holes) from all holes;
 - Levelling doors equipped with sealing package achieving <5% visible emissions.

4. Firing:
 - Use of desulphurized COG
 - Prevention of leakage between oven chamber and heating chamber by means of regular coke oven operation *and*
 - Repair of leakage between oven chamber and heating chamber *and*
 - incorporation of low-NO_x techniques in the construction of new batteries, such as stage combustion (emissions in the order of 450 – 700 g/t coke and 500-770 mg/Nm³ respectively are achievable in new/modern plants).
 - Due to the high cost, flue gas denitrification (e.g. SCR) is not applied except in new plants under circumstances where environmental quality standards are not likely to be met.
5. Pushing:
 - Extraction with an (integrated) hood on coke transfer machine and land-based extraction gas treatment with fabric filter and usage of one point quenching car to achieve less than 5 g particulate matter/t coke (stack emission).
6. Quenching:
 - Emission minimised wet quenching with less than 50 g particulate matter/t coke (determined according VDI method). The use of process water with significant organic load (like raw coke oven wastewater, wastewater with high content of hydrocarbons etc.) as quenching water is avoided.
 - Coke dry quenching (CDQ) with recovery of sensible heat and removal of dust from charging, handling and sieving operations by means of fabric filtration. With respect to present energy prices in the EU, “instrument/operational cost-environmental benefit” consideration sets strong limitations on the applicability of CDQ. In addition a use of recovered energy must be available.
7. Coke oven gas desulphurisation:
 - Desulphurisation by absorption systems (H₂S content grid gas 500-1000 mg H₂S/Nm³) *or*
 - Oxidative desulphurisation (< 500 mg H₂S/Nm³), provided that cross-media effects of toxic compounds are abated to a large extent.
8. Gas-tight operation of gas treatment plant:

All measures to enable virtually gas-tight operation of the gas treatment plant should be considered like:

 - Minimising the number of flanges by welding piping connections wherever possible;
 - Use of gas-tight pumps (e.g. magnetic pumps);
 - Avoiding emissions from pressure valves in storage tanks, by means of connection of the valve outlet to the coke oven gas collecting main (or by means of collecting the gases and subsequent combustion).
9. Wastewater pre-treatment:
 - Efficient ammonia stripping, using alkalis. Stripping efficiency should be related to subsequent wastewater treatment. Stripper effluent NH₃ concentrations of 20 mg/l are achievable;
 - Tar removal.

10. Wastewater treatment:
Biological wastewater treatment with integrated nitrification/denitrification achieving:
- COD removal: > 90%
 - Sulphide: < 0.1 mg/l
 - PAH (6 Borneff): < 0.05 mg/l
 - CN⁻: < 0.1 mg/l
 - Phenols: < 0.5 mg/l
 - Sum of NH₄⁺, NO₃⁻ and NO₂⁻: < 30 mgN/l
 - Suspended solids: < 40 mg/l
- These concentrations are based on a specific wastewater flow of 0.4 m³/t coke.

In principle the techniques listed under points 1 - 10 are applicable to new as well as to existing installations considering the preface except low-NO_x techniques (only for new plants).

6.5 Emerging techniques and future developments

The substitution of part of the coke in the blast furnace by oil and, more recently, pulverised coal has played a major role in reducing fuel costs. The injection of pulverised coal is now at a level of approximately 180 kg/t pig iron, with a residual coke consumption of 300 kg/t pig iron. The theoretical maximum of blast furnace coal injection is approximately 270 kg/t pig iron, with a residual coke consumption of 220 kg/t pig iron. This is due to the carrying capacity of the coke and the thermochemical conditions in the blast furnace. Higher coal injection rates may be possible.

Apart from the fuel savings achieved, coal injection has a positive environmental effect because less coke is consumed and so emissions from coke oven plants are avoided. At a coal injection rate of 180 kg/t pig iron, which is achieved at many places already, approximately 30% less coke is consumed. It is expected that the coal injection rate will continue to increase over the coming years.

Furthermore, several new iron-making techniques are being developed and one technique is already in commercial operation (Corex). These new iron-making techniques use coal instead of coke as a fuel. It is expected that within the coming 25-50 years these new techniques will take over the role of the blast furnaces. This would make the metallurgical coke oven plant superfluous. The new iron-making techniques are described in chapter 7 "Alternative iron-making techniques".

Nevertheless, developments to decrease emissions from existing coke oven plants are still going on. New plant concepts with lower emissions and/or higher energy efficiency are operated (see PI.9 "non-recovery coking") or under development ("Jumbo coke oven"). Because the Jumbo coke oven is not yet applied on a commercial scale, a description is given here.

Single Chamber System (SCS)

Description: SCS coking reactors/Single Chamber System (formerly called JCR) are coke ovens with large coke oven volume and widths between 450-850 mm. The process includes the use of preheated coal. The reactors are separate process controlled modules with rigid, pressure stable, heating walls to absorb high coking pressure. It is expected that SCS coke ovens are able to take the place of current multi-chamber batteries with walls of limited flexibility. The system has the following reported advantages:

- Enables the use of wide range of coking coal (with regards to volatile matter values);
- Oven service schedule can be observed with greater ease, facilitating better heating control;
- Depending on chamber width reduced number of orifices and thus a reduced number of emission sources;
- Depending on chamber width reduced number of oven openings, (for example 55-102 coke pushes/day depending on the chamber width between 850 and 450 mm as compared with 115 pushes of the Kaiserstuhl plant);
- Improvement in thermal efficiency from 38% to 70%;
- Almost emission-free doors, operating on three different sealing levels.

Results

Studies on the economic optimisation of the overall system has yielded the result that coking plant with single reactors having the same throughput as the modern Kaiserstuhl coking plant requires about the same investment provided the chamber width is reduced from 850 mm to 450 mm.

Reference literature: [Nashan, 1997; EC Coke, 1993]

High performance wet desulphurisation of coke oven gas

Description: When desulphurising crude coke oven gas, the wet oxidative processes generally perform better than the absorption/stripping processes. However, a disadvantage of the wet oxidative processes is the use (and emission) of highly toxic chemicals.

It would be possible to increase the removal efficiency of the absorption processes by means of scrubbing with a caustic soda solution (NaOH), which is sprayed as pressurised mist into the crude gas stream.

Main expected achievements: It is expected that a grid gas sulphur content of 0.1 mg/Nm³ is achievable.

Status: unknown.

7 BLAST FURNACES

7.1 Applied processes

The first true coke-based blast furnace was introduced in 1735 [Ullmann's, 1994]. The blast furnace remains by far the most important process for the production of pig iron. The technique is likely to continue to dominate pig iron production for at least the next 20 years [Lüngen, 1995]. Figure 7.1 shows a general view of two blast furnaces.



Figure 7.1 : : General view of two blast furnaces with three hot stoves each and the stack for the waste gas from the hot stoves

A blast furnace is a closed system into which iron bearing materials (iron ore lump, sinter and/or pellets), additives (slag formers such as limestone) and reducing agents (coke) are continuously fed from the top of the furnace shaft through a charging system that prevents escape of blast furnace gas (BFgas). Figure 7.2 shows a simplified layout of a blast furnace consisting of the furnace itself, the cast house, the hot stoves and two-stage treatment of BFgas.

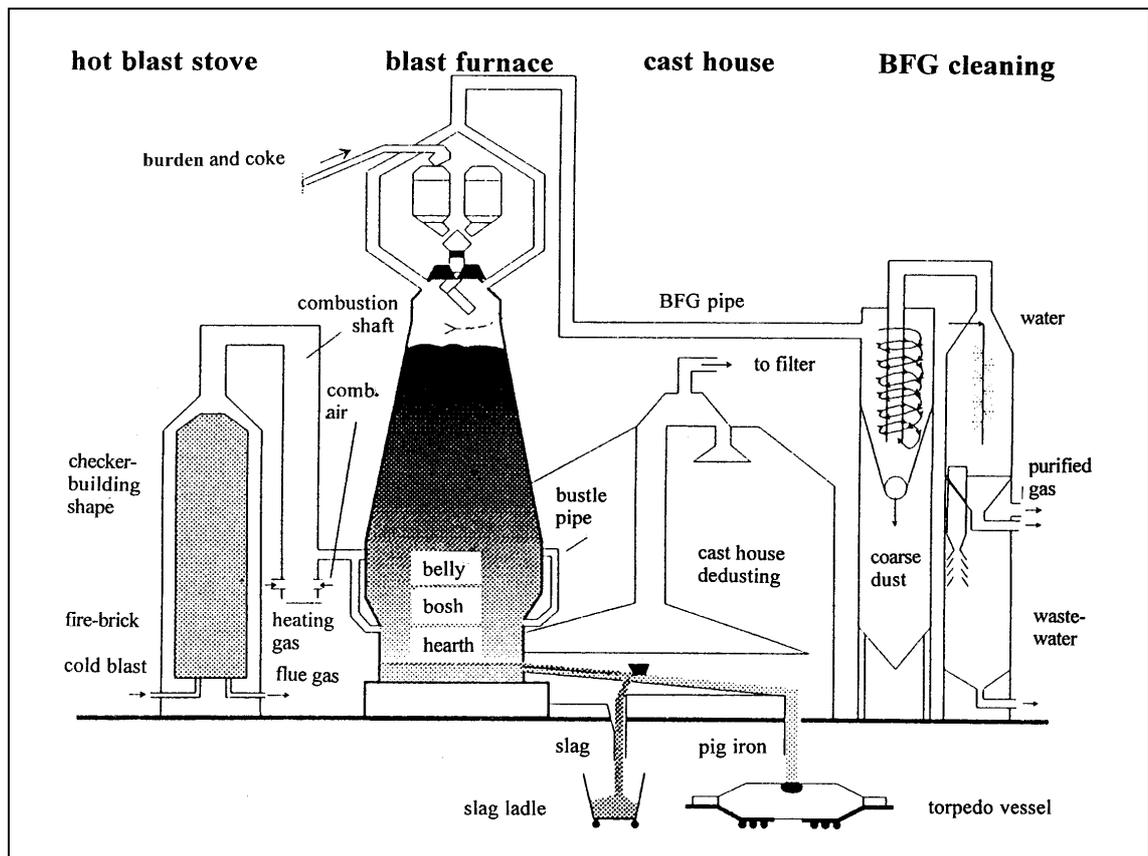


Figure 7.2 : Simplified scheme of a blast furnace - [UBA Rentz, 1996]

A hot air blast, enriched with oxygen and auxiliary reducing agents (coal powder, oil, natural gas and in a few cases plastics) is injected on the tuyère level providing a counter-current of reducing gases. The air blast reacts with the reducing agents to produce mainly Carbon monoxide (CO), which in turn reduces iron oxides to metal iron. The liquid iron is collected in the hearth along with the slag and both are cast on a regular basis. The liquid iron is transported in torpedo vessels to the steel plant and the slag is processed to produce aggregate, granulate or pellet for road construction and cement manufacture. The blast furnace gas (BFgas) is collected at the top of the furnace. It is treated and distributed around the works to be used as a fuel for heating or for electricity production.

Various reducing agents are available. Carbon/hydrocarbons in the form of coke, coal, oil, natural gas, or nowadays in some cases also plastics, are generally available in sufficient quantities at reasonable cost. However, the choice between several reducing agents is not determined by costs alone. Apart from being a reducing agent, coke also serves as a carrier of the bulk column in the blast furnace. Without this carrying capacity, blast furnace operation would not be possible.

Iron ore processed nowadays contains a large content of hematite (Fe_2O_3) and sometimes small amounts of magnetite (Fe_3O_4). In the blast furnace, these components become increasingly reduced, producing iron oxide (FeO) then a partially reduced and carburised form of solid iron. Finally, the iron charge melts, the reactions are completed and liquid hot metal and slag are collected in the bottom. The reducing carbons react to form CO and CO_2 . Fluxes and additives are added to lower the melting point of the gangue, improve sulphur uptake by slag, provide the required liquid pig iron quality and allow for further processing of the slag.

As the blast furnace burden moves down, its temperature increases, thus facilitating oxide reduction reactions and slag formation. The burden undergoes a series of composition changes as this happens:

- The iron oxide in the burden becomes increasingly reduced (forming sponge iron and finally molten pig iron).
- The oxygen from the iron ore reacts with the coke or the carbon monoxide, thus forming carbon monoxide or carbon dioxide, which is collected at the top.
- The gangue components combine with the fluxes to form slag. This slag is a complex mix of silicates of a lower density than the molten iron.
- The coke primarily serves as a reducing agent, but also as a fuel. It leaves the furnace as carbon monoxide, carbon dioxide or carbon in the pig iron.
- Any hydrogen present also acts as a reducing agent by reacting with oxygen to form water.

The main operations are as follows:

- Charging of raw materials
- Generation of hot blast
- Blast furnace process
- Direct injection of reducing agents
- Casting
- Slag processing

7.1.1 Charging

The mixture of iron bearing materials (iron ore rubble, sinter and/or pellets) and additives (flux material) are known collectively is called the "burden". The burden and the accompanying coke are charged into the top of the furnace either via skips or mechanical conveyor belts. It enters into the furnace via a sealed charging system which isolates the furnace gases from the atmosphere. This system is necessary because blast furnace pressure exceeds atmospheric pressure (0.25 – 2.5 bar gauge). Whilst many new large blast furnaces have high top pressures (up to 2.5 bar gauge) there are modern furnaces operating at pressures much lower than this. These pressures can be as low as 0.25 bar gauge, depending on the age of the furnace and other constraints such as available blast pressure and limitations due to the gas treatment plant construction.

The sealed charging system can be a bell charging system or a bell-less charging system. Some particulate matter and BFGas emissions may arise during charging. The evacuation of gas at the top of the furnace and connection to the BFGas treatment system can be used to control emissions at this stage of the process.

7.1.2 Hot stoves

The hot blast for the blast furnace operation is provided by the hot stoves (sometimes called "blast furnace cowpers"). Stoves are auxiliary installations used to heat the blast. Increased blast temperature results in reduction of carbon requirement. A hot blast is needed to transfer heat to the solid burden in order to raise the temperature for reaction. The blast also helps to provide the oxygen necessary for coke gasification, and transport the gas that, on contact with the burden, reduces iron oxides.

The stoves operate on a cyclical basis. They are heated up by burning gases (usually enriched BFGas) until the dome is at the correct temperature (approximately 1100 - 1500°C); combustion gas is then cut off and cold ambient air is forced through the stoves in the reverse direction. The cold air is heated by the hot bricks and thus forms the hot blast (900 - 1350°C), which is fed to

the blast furnace. The process continues until the stove can no longer generate the proper blast gas temperature, after which the initial heating cycle is started again. The duration of each cycle depends on individual site conditions such as energy source, system characteristics and conservation measures.

In principal hot stoves can be classified as being of either the internal or external combustion chamber type (Figure 7.3). This distinction is important for CO emissions (7.2.2.1.1).

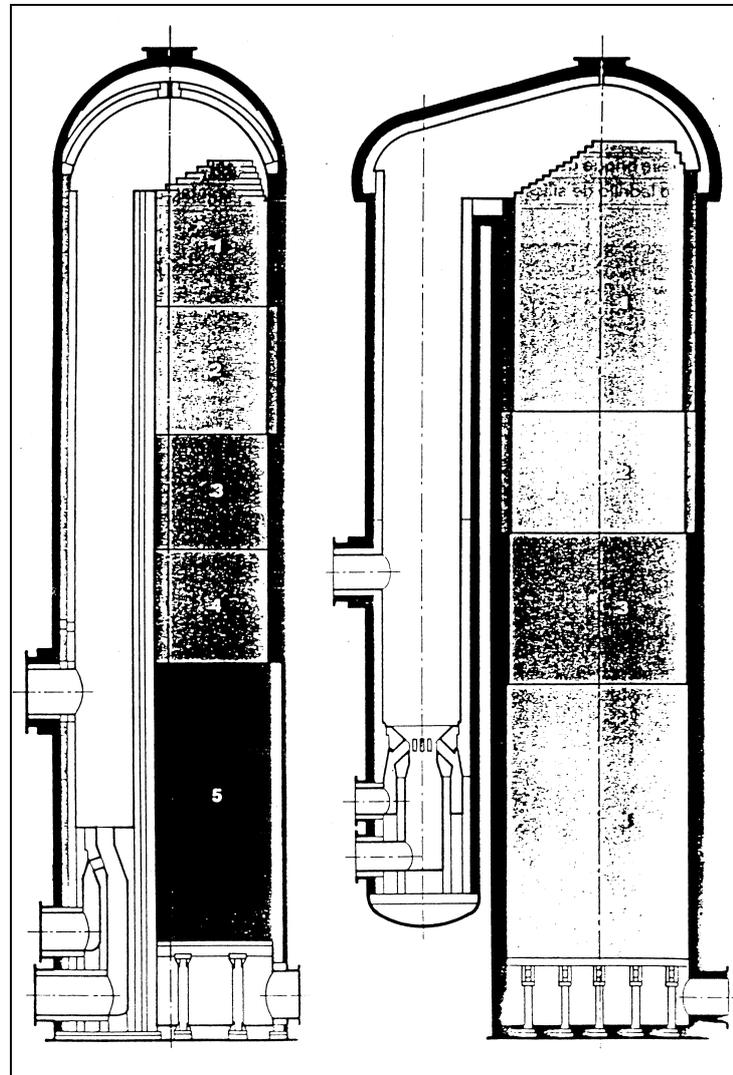


Figure 7.3 : Cross section of hot stoves (“cowpers”) with internal and external combustion chamber – [EC Sinter/BF, 1995]

Three or four hot stoves are necessary for each blast furnace. Emissions to air occur during the heating phase of the stove.

7.1.3 Blast furnace

7.1.3.1 General description

In a blast furnace, the raw materials enter at the top, while the products (molten iron and slag) are tapped from the bottom (the hearth). The solid burden moves downwards, meeting a rising stream of hot reducing gas. BFgas with residual calorific value is collected from the top of the furnace for treatment.

A blast furnace can be divided into six temperature zones:

- The Top: At the top, charging of the burden and evacuation of BFgas occurs.
- The Shaft: In the Shaft, the hot BFgas gives its heat to the solid burden. The temperature of the burden rises from ambient temperature to approximately 950°C and the iron oxide becomes partially reduced in this zone.
- The Belly: The belly links the shaft to the bosh. In this section the temperature rises further from 950°C to approximately 1250°C. Further reduction of the iron oxide takes place and coke reaction begins.
- The Bosh: Coke reactions continue to take place in the zone of the bosh. The iron melts and slag is formed.
- The Tuyères: In this zone the hot blast is introduced into the furnace by means of a series of tuyères (up to 42). The tuyères are located around the upper perimeter of the hearth and are fed by a large pipe (bustle pipe), circling the furnace at the height of the bosh. Temperatures here can exceed 2000°C and the oxides are completely reduced.
- The Hearth: The hearth collects the molten pig iron and slag. One to four tapholes are located around the hearth, with one or two in operation at any time.

The shaft, belly, bosh and tuyère belt are typically water-cooled with the hearth water, oil or air cooled. The furnace is lined with refractory material (the hearth being up to 1.5 m thick at least at the hearth level).

Pig iron production ranges from approximately 0.5 Mt/a for small blast furnaces to nearly 4 Mt/a for large blast furnaces.

7.1.3.2 Blast furnace top gas (BFgas)

BFgas contains about 20-28% CO, 1-5% H₂, inert compounds (50-55% N₂, 17-25% CO₂), some sulphur and cyanide compounds and large amounts of dust from the burden. The amount of cyanide may be especially high during blow down operations of the blast furnace; but this only occurs occasionally and then additions are made to the system to minimise cyanide formation. The heating value of BFgas is approximately 2.7 to 4.0 MJ/Nm³. The production of BFgas is approximately 1200 to 2000 Nm³/t pig iron.

After cleaning, the BFgas is often used as a fuel after enriching with coke oven gas or natural gas, which have a higher heating value. BFgas can also be used without upgrading, e.g. in the hot stoves, if modern burners and/or combustion air preheating are applied [UBA Comments, 1997]. BFgas can also be used without enrichment or modern burners/preheating if appropriate safety precautions are taken.

7.1.3.3 Zinc and lead

Blast furnace technology uses high-grade ore. Any elements accompanying the iron are distributed between the hot metal phase and slag. Those passing into the hot metal include phosphorous, sulphur, manganese and silicon. Titanium, aluminium, calcium, magnesium and the bulk of silicon and sulphur pass into the slag as oxides or metalloids. It is also possible for a variety of elements to volatilise and deposit on different parts of the blast furnace. This is especially the case for zinc (Zn) and lead (Pb) which reach the blast furnace via iron ores and recycled by-products via the sinter plant.

The total zinc in the charge generally varies from 100 to 250 g/t of pig iron produced. As a rule of thumb, most modern plants restrict zinc to 100-150 g/t pig iron. Recycled materials with high zinc or lead content are usually not accepted, or only used in limited amounts.

Removal of Zn and Pb from the furnace is done by operating the blast furnace in such a way that the temperature in the centre of the furnace is kept above 400°C. The zinc is then more likely to be removed with the top gas as fine ZnO particles, which are almost completely retained in the BFgas treatment [Pazdej, 1995].

7.1.4 Direct injection of reducing agents

Most blast furnace installations inject reducing agents into the furnace at the tuyère level. This partially replaces coke in the top charge. This practice enables the operator to optimise on the use of reducing agents. Other advantages are increased output and a reduction in the coke-making requirements, thereby decreasing the specific coke oven emissions per tonne of steel produced. Many plants inject powdered coal, oil or natural gas. Two companies have started to inject plastic waste using its high hydrocarbon content for reduction processes [UBA Comments, 1997].

7.1.5 Casting

The blast furnace is periodically cast to remove the molten pig iron and slag from the hearth. For this purpose one of the tap-holes is opened in the side wall of the hearth by means of a tap-hole drill or soaking bar. In some cases, an oxygen lance is used to open the tap-hole. Usually, a blast furnace has one to four tap-holes. The number of tap-holes is determined by capacity (size) of the blast furnace.

In modern blast furnaces, pig iron and slag are tapped together (typically slag starting to run after the hot metal). The slag and pig iron are subsequently separated at the skimmer in the cast house, after which each continues in a separate runner.

In modern blast furnaces, pig iron and slag are tapped together (typically slag starting to run after the hot metal). The slag and pig iron are subsequently separated at the skimmer in the cast house, after which each continues in a separate runner.

Molten pig iron tapped from the blast furnace flows along refractory or low cement covered runners, lined with a heat resistant alumina-carbon or silica-carbide mixtures, and is poured into ladles (directly or via tilted runners). These ladles can be open or closed top types, or torpedo cars. In this molten state, the metal has a temperature of approximately 1440-1500 °C. Slag tapped from the furnace flows in runners to a granulation plant, to slag ladles or to an open pit.

At the end of the casting cycle, the tap-hole is closed mechanically by injecting a specific heat resistant tap hole clay mixture, using a so-called "mudgun".

7.1.6 Slag processing

The amount of slag produced is determined by the iron ore used and the amount of flux required to achieve the necessary pig iron quality. Slag can be put to a variety of uses including material for road building, concrete aggregate, thermal insulation (mineral wool) and as a cement replacement. Total utilisation of blast furnace slag is a target which has already been met in many cases.

Currently, there are three processes in operation to treat blast furnace slag:

- slag granulation process;
- slag pit process;
- slag pelletising process.

All slag cooling processes may generate hydrogen sulphide, which can result in odour.

7.1.6.1 Slag granulation processing

Granulation is the most common process currently used in the EU 15 to treat blast furnace slag. The process involves pouring the molten slag through a high-pressure water spray in a granulation head, located in close proximity to the blast furnace.

After granulation, the slag/water slurry is typically transported to a drainage system, consisting of horizontal filtering basin (especially the OCP process), a vertical filtering hopper or a rotating de-watering drum (especially the INBA process). In several cases, the slag/water slurry is transported to a separation tank prior to water drainage. Here, water vapours are collected and condensed, or emitted via a stack.

After de-watering the residual moisture of the slag sand is generally around 10%. The filter bed is periodically back-washed with water and air to remove the fine particles.

Figure 7.4 and Figure 7.5 show the two most common granulation techniques: the OCP process and the INBA process.

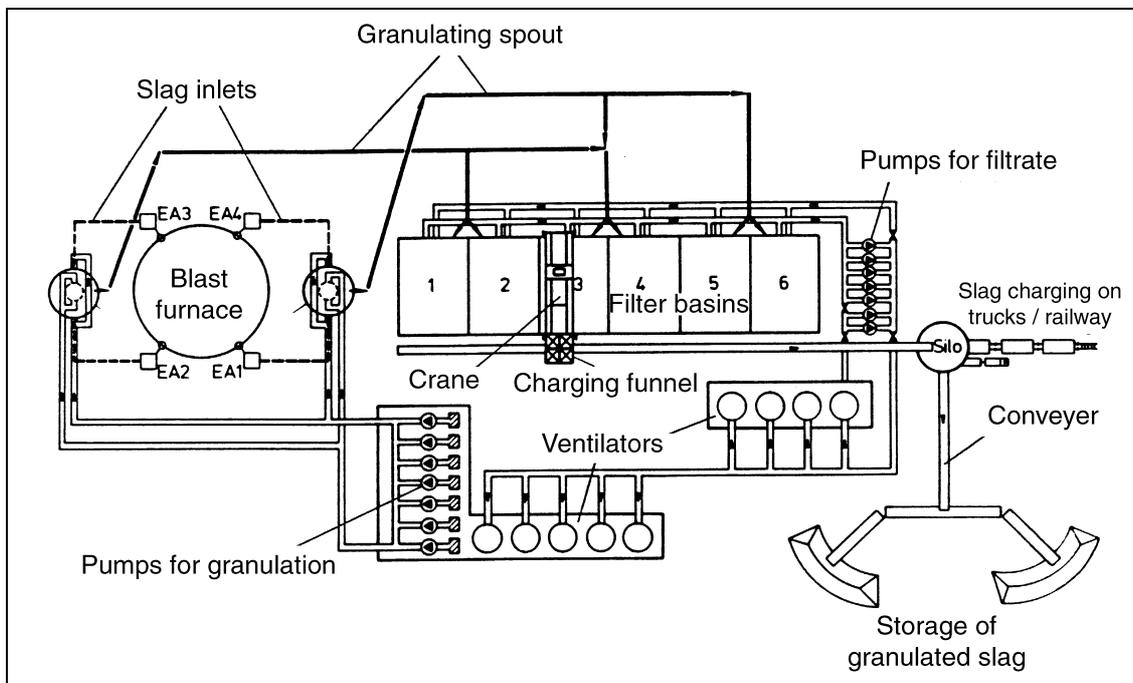


Figure 7.4 : Granulation of slag from blast furnace in the OCP process – [Poth, 1985]

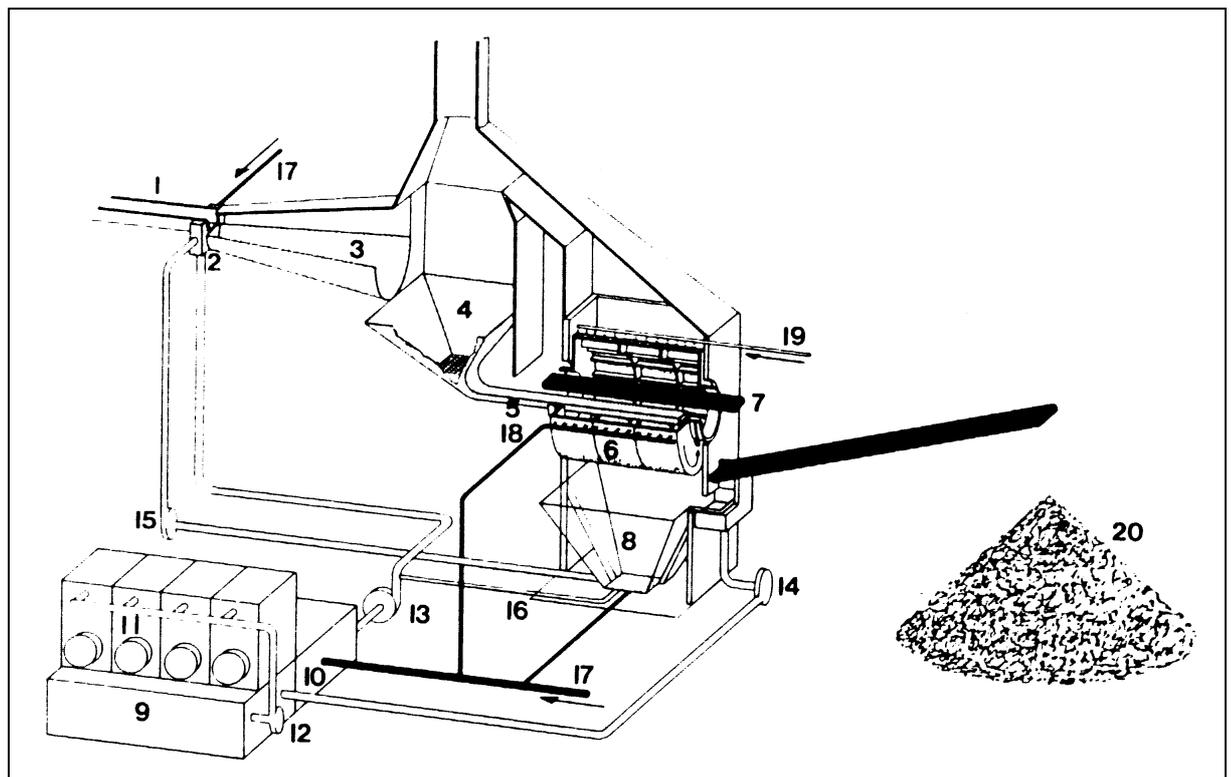


Figure 7.5 : Granulation of slag from blast furnace in the INBA process – [Radoux, 1982]

- | | |
|----------------------------|------------------------------|
| 1. slag channel. | 13. cold water. |
| 2. splash pot. | 14. pump for warm pot. |
| 3. cold channel. | 15. pump for sump water. |
| 4. collection tank . | 16. whirl water. |
| 5. distributor. | 17. addition water. |
| 6. filter drum. | 18. cleaning water. |
| 7. conveyer | 19. pressured air. |
| 8. water collection basic. | 20. granulated slag storage. |
| 9. cooling station. | |

7.1.6.2 Slag pit process

The slag pit process involves pouring thin layers of molten slag directly into slag pits adjacent to the furnaces. Alternatively, after collection of slag in ladles the molten slag is slowly cooled and crystallised in open air. The pits are alternately filled and excavated, and lump slag is broken up and crushed for use as coarse aggregate. In practice, cooling time can be reduced by spraying the hot slag with a controlled amount of water, although this increases the potential for odour emissions. When properly applied, the cooling water is totally consumed by evaporation.

The slag pit process produces lump slag that is a desirable raw material for road construction. The cooling time has a strong influence on the quality of lump slag produced. Water cooling also improves the micro-structure, achieving better mechanical properties.

7.1.6.3 Slag pelletising process

The pelletising process is only in use in a few plants in the EU 15 and in some plants in Canada. The molten slag is spread in a layer on a plate, which acts as a deflector. The sheet of slag is sheared by controlled water jets which initiate the swelling and cooling of the slag. The slag is then projected centrifugally into the air on a rotating drum to complete the blowing-up and cooling. The slag particles follow different trajectories according to their size, which ranges from granulated sand to expanded pellets.

When properly applied, process water is totally consumed by evaporation and as moisture in the product. Specific water consumption is less than that required for wet granulation.

7.2 Present emission and consumption levels

7.2.1 Mass stream overview and input/output data

Figure 7.6 provides an overview for the input and output mass streams of a blast furnace. This overview may be used for the collection of data from single blast furnaces.

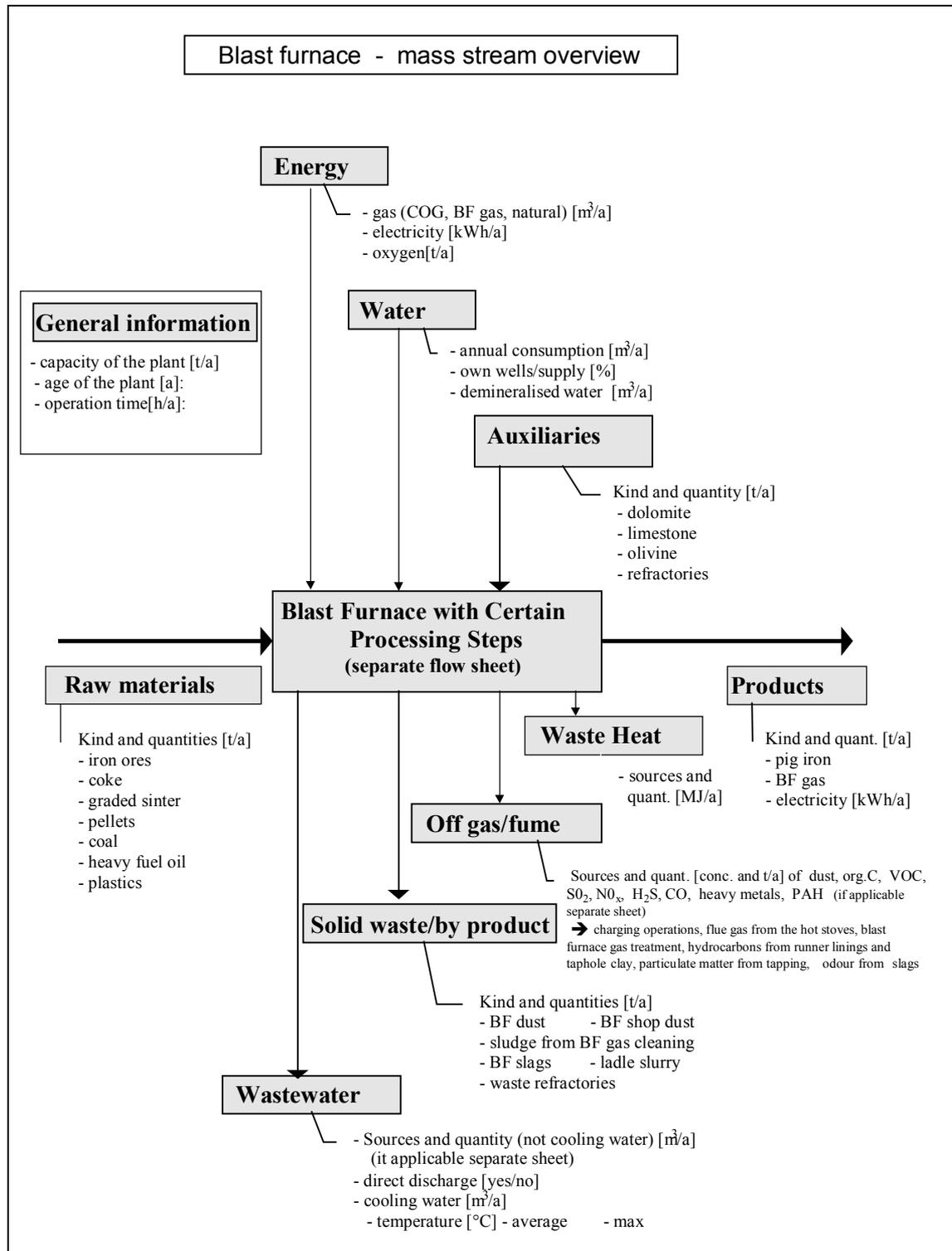


Figure 7.6 : Mass stream overview of a blast furnace

Figure 7.7 shows an example of the general process layout of a blast furnace along with the input and output mass streams.

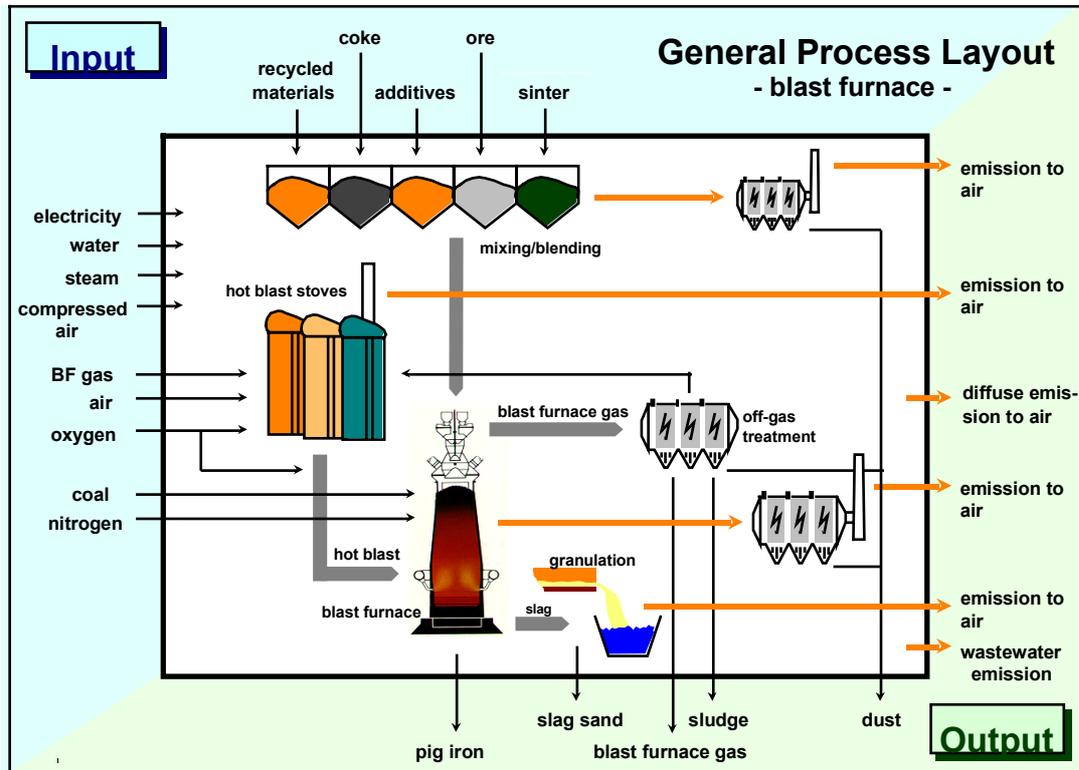


Figure 7.7 : General process layout of a blast furnace with indication of the single operations and the input and output mass streams

Subsequently both specific input factors and specific emission factors can be calculated. Table 7.1 gives figures for these factors taken from four blast furnaces from four different EU Member States, thus giving a representative picture of modern plants.

The emission factors quoted are per tonne of liquid steel in order to allow the addition of emissions from the various production units.

Input			Output		
Raw materials			Products		
Sinter ^{*1}	kg/t pig iron	720 – 1480	pig iron	kg/t	1000.0
iron ore ^{*1}	kg/t pig iron	25 – 350			
pellets ^{*1}	kg/t pig iron	100 – 770	Energy		
coke ^{*1}	kg/t pig iron	280 – 410	BFGas	MJ/t pig iron	4400 – 5000
coal ^{*2}	kg/t pig iron	0 – 180	electricity ^{*6}	MJ/t pig iron	ca. 750
heavy oil ^{*3}	kg/t pig iron	0 – 60			
Lime	kg/t pig iron	0 – 10	Gas. emissions ^{*7}		
recyc. Materials	kg/t pig iron	2 – 8	dust	g/t LS ^{*8}	10 – 50
plastics ^{*4}	kg/t pig iron	0 – 30	Mn	g/t LS	< 0.01 – 0.13
			Ni	g/t LS	< 0.01 – 0.02
			Pb	g/t LS	< 0.01 – 0.12
Energy			SO _x	g/t LS	20 – 230
BFGas	MJ/t pig iron	1050 – 2700	NO _x	g/t LS	30 – 120
COG	MJ/t pig iron	90 – 540	H ₂ S	g/t LS	0.2 – 20
natural gas	MJ/t pig iron	50 – 230	CO	g/t LS	770 – 1750
Electricity	MJ/t pig iron	270 – 370	CO ₂	kg/t LS	280 – 500
			PCDD/F	µg I-TEQ/t LS	<0.001–0.004
Oxygen ^{*5}	m ³ /t pig iron	25 – 55			
			Residues/ by-products		
Steam	MJ/t pig iron	22 – 30	slags	kg/t LS	200 – 290
			top gas dust	kg/t LS	6 – 16
Compressed air	m ³ /t pig iron	9 – 11	top gas sludge	kg/t LS	3 – 5
			dust from cast house de-dusting	kg/t LS	0.5 – 1.5
Water	m ³ /t pig iron	0.8 - 50	rubble	kg/t LS	14 – 25
			Wastewater	m ³ /t LS	0.1 – 3.3 ^{*9}

Legend: LS = liquid steel (crude steel)

*1 depending on individual site conditions

*2 in the case of coal powder injection – which is increasingly frequent but not universal (if practised: 140-180 kg coal/t pig iron)

*3 in the case of oil injection – at other blast injection rate can be upto 140 kg/t pig iron

*4 as of 1998 injection of plastics is practised at only two sites in the EU 15

*5 additional oxygen is not always required, particularly at low tuyere injection rates it can be “0”, which is not the case at the four blast furnaces reported

*6 in the case of a top gas turbine

*7 overall emissions from the blast furnace not including the combustion of BFGas at other plants like BOF or coke ovens

*8 conversion factors used (weighted average of all European blast furnaces): 940 kg pig iron/t LS

*9 can be higher in case of high salt content raw materials

Table 7.1 : Input/output data from four existing blast furnaces in four different EU Member States;

Data from 1996; the emission data represent the emissions after abatement;

No information is available concerning how the data were obtained, such as the methods used for sampling, analysis, and computation of the time intervals and reference conditions .

Table 7.2 complements the information given in Table 7.1 with more details on emission factors to air (after abatement) for the main individual operations associated with a blast furnace.

operation/emission source		dust [g/t LS]	H ₂ S [g/t LS]	SO ₂ [g/t LS]	NO _x [g/t LS]	CO [g/t LS]
charging zone	n/r x±s	25/5-38 ^{*3} 14±13	l.s.	l.s.	l.s.	l.s.
coal preparation for inject.	n/r x±s	15/2-54 12±16	l.s.	l.s.	l.s.	l.s.
BF cast house	n/r x±s	12/2-79 ^{*4} 30±24	5/0.3-4 ^{*5} -	12/2-250 ^{*6} 70±34	4/1-27 ^{*7} -	l.s.
slag granulation	n/r x±s	n/a	14/1-300 ^{*8} 63±95	13/1-142 ^{*8} 31±42	l.s.	l.s.
hot stoves	n/r x±s	3-6 -	l.s.	45/15-375 ^{*9} 120±100	41/10-550 ^{*10} 82±102	29/50-2700 ^{*11,12} 700±735

Legend: LS = liquid steel (crude steel); x±s = mean value and standard deviation (only calculated if enough data are available); n = number of data; r = range of data (min-max); l.s. = low significance; n/a = not available

*1 data from [EC Study, 1996]

*2 conversion factors used (weighted average of all European basic oxygen steelworks): 940 kg pig iron/t LS

*3 unreasonably low emissions factors (< 1 g/t LS) have not been taken into account; nor have data from wet scrubbers, which have much higher emission factors (2-20 times higher)

*4 data from plants with continuous measurement

*5 not taking into account one extreme value of 64 g H₂S/t LS

*6 including two high values of 180 and 250 g SO₂/t LS, all other the values are below 100 g SO₂/t LS

*7 not taking into account one extreme value of 2450 g NO_x/t LS

*8 low values (around 1 g/t LS) are for slag granulation plants with fume condensation

*9 8 values are significantly higher (> 200 g SO₂/t LS) due to the SO₂ content of the used coke oven gas

*10 3 values are significantly higher (> 300 g NO_x/t LS) due to the combustion conditions

*11 high values can occur in case of hot stoves with internal combustion chambers

*12 it is not clear whether the emissions during burden charging are included or not

Table 7.2 : Emission factors for emissions to air for blast furnaces^{*1,2}

7.2.2 Information about single emission mass streams and energy demand

The following emissions of off-gas, wastewater and solid waste/by-products can be observed in the blast furnace process:

7.2.2.1 Waste gas emissions

- flue gas from the hot stoves
- emissions from charging
- blast furnace gas (as an indirect emission)
- emissions from cast house
- emissions from slag processing

7.2.2.2 Solid wastes/by-products emissions

- particulate matter from casting
- dust and sludge from BFgas treatment
- slag from blast furnace

7.2.2.3 Wastewater emissions

- Overflow water from BFgas scrubbing
- Wastewater from slag granulation
- Blow down from cooling water circuits

7.2.2.1 Waste gas emissions

7.2.2.1.1 Waste gas emissions from the hot stoves

Hot stoves are usually fired with BFgas often in combination with coke oven gas (COG) and/or natural gas. Coke oven gas contains sulphur compounds, which are emitted as SO₂ at the hot stove when the gas is used for firing. [UBA Comments, 1997] reports that several hot stoves with modern burners and combustion air preheating are operated using BFgas without enrichment.

Flue gas flow from the hot stoves is approximately 100000 to 240000 Nm³/h per blast furnace. Emissions of SO₂ are in the range 20 to 250 g/t pig iron produced (Table 7.1) and 160 to 400 mg/Nm³ when the stoves are fired with enriched BFgas. In case of a mixture of BFgas and non-desulphurised COG the emission factor can be up to 400 g SO₂/t pig iron (Table 7.2).

The hot stoves are the main source of emissions of NO_x in the blast furnace process. NO_x is formed as a result of high temperatures in the stove. Emissions are in the range of 10 to 580 g/t pig iron produced (Table 7.2). Emission concentrations may vary from 70 to 400 mg/Nm³.

Particulate matter emissions from the hot stoves total less than 10 mg/Nm³. This is equivalent to approximately 3-6 g/t pig iron produced (Table 7.2).

CO emissions are considerable in the case of internal combustion chambers (see Figure 7.3) Cracks in the brick work, which are very difficult to avoid, mean BFgas can move without combustion to the waste gas and can be emitted in concentrations of up to 2500 mg CO/Nm³ equivalent to as much as 2700 g CO/t pig iron (Table 7.2). In the case of an external combustion chamber with controlled combustion, the concentration is about 50 mg CO/Nm³.

7.2.2.1.2 Emissions from charging and conveying

Since the furnace pressure is higher than atmospheric pressure a sealed charging system based on "bell-shaped" or "bell-less type" charging is used.

All components present in the BFGas may be emitted here, but the main components are carbon monoxide (CO) and particulate matter. When using modern systems, much lower charging and conveying emissions are possible [UBA Comments, 1997].

7.2.2.1.3 Blast furnace gas (as an indirect emission)

Raw BFGas contains particulate matter (including heavy metals and carbon), carbon monoxide, carbon dioxide, sulphur compounds, ammonia, cyanide compounds, hydrocarbons and PAH. The BFGas is contained and used as an energy source. The gas is purified in order to meet quality specifications and is reused for various firing processes such as blast generation in the hot stoves (see 7.1.2) or for coke oven firing (see 6.1.2.2). Thus (indirect) emissions of combusted BFGas occur.

BFGas treatment usually consists of pre-treatment for the removal of coarse particulate matter and subsequently wet scrubbing for the removal of fine particulate matter (and thus heavy metals), SO₂ and cyanide compounds. In some plants electrostatic precipitation is applied.

The particulate matter content of raw BFGas is very different from one plant to another and is also highly dependent on process conditions, varying from 7 to 40 kg/t pig iron produced. After cleaning, the BFGas contains normally less than 10 mg/Nm³ particulate matter. This is emitted at the site of combustion.

In Table 7.3 specific factors of several components in raw untreated BFGas are given.

Raw BFGas component	Value	Unit	Specific factor	Unit
BFGas production	1.0 - 7.0	[1·10 ⁵ Nm ³ /h]	1200 – 2000	[Nm ³ /t pig iron]
Particulate matter	3500 - 30000	[mg/Nm ³]	7000 – 40000	[g/t pig iron]
Hydrocarbons (C _x H _y)	67 - 250	[mg/Nm ³]	130 – 330	[g/t pig iron]
Cyanide compounds (as CN ⁻)	0.26 - 1.0*	[mg/Nm ³]	0.5 - 1.3	[g/t pig iron]
Ammonia (NH ₃)	10 - 40	[mg/Nm ³]	20 - 50	[g/t pig iron]
PAH**				
Benzo(a)pyrene	0.08 - 0.28	[mg/Nm ³]	0.15 - 0.36	[g/t pig iron]
Fluoranthene	0.15 - 0.56	[mg/Nm ³]	0.30 - 0.72	[g/t pig iron]
Carbon monoxide (CO)	20 - 28	[vol.%]	300 – 700	[kg/t pig iron]
Carbon dioxide (CO ₂)	17 - 25	[vol.%]	400 – 900	[kg/t pig iron]
Hydrogen (H ₂)	1 - 5	[vol.%]	1 – 7.5	[kg/t pig iron]

* emissions during blowdown may be significantly higher

** many other polycyclic aromatic hydrocarbons (PAH) are also present

Table 7.3 : Raw blast furnace gas composition (before treatment) – based on [InfoMil, 1997]

During the two stage treatment of BFGas, dust is removed with a high degree of efficiency as are compounds associated with particulate matter such as most heavy metals and PAH. Table 7.4 shows the BFGas composition after treatment. The gas still contains some heavy metals but compared with sinter plants (see 4.2.1) the load is much lower.

Treated BFgas component	Concentration	Unit	Specific factor	Unit
BFgas production	1.0 - 7.0	$1 \cdot 10^5 \text{ Nm}^3/\text{h}$	1200 - 2000	$[\text{Nm}^3/\text{t pig iron}]$
Particulate matter	1 - 10	$[\text{mg}/\text{Nm}^3]$	1 - 20	$[\text{g}/\text{t pig iron}]$
Hydrocarbons (C_xH_y)	n/a	$[\text{mg}/\text{Nm}^3]$	n/a	$[\text{g}/\text{t pig iron}]$
H_2S	14	$[\text{mg}/\text{Nm}^3]$	17-26	$[\text{g}/\text{t pig iron}]$
Cyanide compounds (as CN^-)	n/a	$[\text{mg}/\text{Nm}^3]$	n/a	$[\text{g}/\text{t pig iron}]$
Ammonia (NH_3)	n/a	$[\text{mg}/\text{Nm}^3]$	n/a	$[\text{g}/\text{t pig iron}]$
Heavy metals**				
Mn	0.10 - 0.29	$[\text{mg}/\text{Nm}^3]$	0.22 - 0.37	$[\text{g}/\text{t pig iron}]$
Pb	0.01 - 0.05	$[\text{mg}/\text{Nm}^3]$	0.02 - 0.07	$[\text{g}/\text{t pig iron}]$
Zn	0.03 - 0.17	$[\text{mg}/\text{Nm}^3]$	0.07 - 0.22	$[\text{g}/\text{t pig iron}]$
Carbon monoxide (CO)	20 - 28	$[\text{vol.}\%]$	300 - 700	$[\text{kg}/\text{t pig iron}]$
Carbon dioxide (CO_2)	17 - 25	$[\text{vol.}\%]$	400 - 900	$[\text{kg}/\text{t pig iron}]$
Hydrogen (H_2)	1 - 5	$[\text{vol.}\%]$	1 - 7.5	$[\text{kg}/\text{t pig iron}]$

n/a = data not available

Table 7.4 : Blast furnace gas composition (after two stage treatment) – based on [InfoMil, 1997]

In the EU wet scrubbing is the technique most commonly applied as a second step in BFgas treatment. Scrubbing generates a contaminated wastewater flow, containing suspended solids (e.g. carbon and heavy metals), cyanide compounds, nitrogen compounds, etc. The separated solids generate a waste problem because of heavy metals, especially zinc. Whereas coarse dust is normally recycled to the sinter plant, sludge from scrubbing is usually de-watered and either recycled to the sinter plant (normally via hydrocyclone) or put to secure landfill.

7.2.2.1.4 Emissions from the cast house

The casting of pig iron generates particulate matter emissions. On average unabated emissions are in the range 400 to 1500 g/t pig iron produced. These emissions mainly arise from contact between the hot metal and slag and ambient oxygen. In order to catch the dust formed during casting in many blast furnaces in the EU cast house de-dusting systems are used (dust extraction at tap-hole, skimmer and pig iron charging to the torpedo ladle) with flows of between 200000 and 700000 m^3/h . Dust emissions depend on applied abatement technique (in some cases there is still none) and dust collection efficiency. In many cases bag filters are applied, achieving less than 10 mg dust/ Nm^3 . According to Table 7.2 dust emission factors vary between 2 to 85 g/t pig iron with an average of 32 g dust/t pig iron.

Furthermore, a certain amount of SO_2 is emitted from the liquid slag and iron during casting (2-270 g/t pig iron – see Table 7.2).

7.2.2.1.5 Emissions from slag processing

The reaction of water with molten slag, particularly with sulphur compounds (essentially CaS and MnS), generates both steam and diffuse H_2S and SO_2 emissions. These emissions cause potential odour and corrosion problems. Their importance varies according to the slag processing technique used.

Emissions can vary greatly from one plant to another, from one slag treatment cycle to another and within the slag treatment cycle itself. Therefore the range of available emission factors is wide. Table 7.2 contains figures varying from 1 – 320 g $\text{H}_2\text{S}/\text{t}$ pig iron and 1 – 150 g SO_2/t pig iron for slag granulation. If slag is not exposed to water but air-cooled, longlasting small

emissions of mainly SO₂ will occur. From the perspective of nuisance abatement this can be considered an advantage.

The water used in the granulation and the pelletising process can largely be collected and reused. These systems can be operated so as to generate very small amounts of wastewater. The steam generated during this process contains particulate matter, SO₂ and H₂S, which is usually emitted to the atmosphere. Tests have been carried out to reuse the sensible heat of the slag, but no system is commercially available at the moment. The potential for energy recovery is approximately 0.35 GJ/t pig iron.

The production of lump slag from pits usually leads to larger emissions of SO₂ and H₂S, which can be more difficult to control. Conditioning with water can influence the emissions of H₂S.

7.2.2.2 Solid wastes/by-products emissions

7.2.2.2.1 Particulate matter from casting

Between 0.5 and 1.5 kg of dust can be extracted from the cast house (see 7.2.2.1.4) per tonne of pig iron (Table 7.1). This dust can be separated in a bag filter for instance and can easily be recycled to the sinter strand. Recycling the dust in this way is common practice.

7.2.2.2.2 Dust and sludge from BFGas treatment

BFGas is usually treated in two stages: separation of coarse dusts in cyclones followed by fine dust separation in a wet scrubber. This produces 6 – 17 kg of dry dust per tonne of pig iron and 3 – 5 kg of sludge /t pig iron (Table 7.1).

Table 7.5 shows the typical composition of dry coarse dust. This dust mainly contains carbon and iron from coke and sinter abrasion respectively. This coarse dust is normally recycled to the sinter strand. This is much more difficult for the sludge because its zinc content is 10 to 20 times higher and lead content is 20 to 30 times higher.

C 25 - 40	Fe 15 - 40	Pb 0.02 - 0.07	Zn 0.1 - 0.5	Mn 0.1 - 0.5	Al ₂ O ₃ 0.2 - 3.7	Ti 0.02 - 0.2
S 0.2 - 1.3	SiO ₂ 4 - 8	P ₂ O ₅ 0.04 - 0.26	CaO 2 - 8	MgO 0.3 - 2	Na ₂ O 0.03 - 0.64	K ₂ O 0.24 - 0.96

Table 7.5 : Typical composition in [weight-%] of dry coarse dust from BFGas treatment – based on [IISI, 1987; Mertins, 1986; data from European blast furnaces from 1997]

The zinc and lead compounds are mostly passed on to the cyclone and are mainly separated in the scrubber. Most of the particles associated with zinc and lead compounds or these heavy metals themselves have grain sizes of less than 25 µm and concentrate in this fraction of the sludge.

C 15 – 47	Fe 7 - 35	Pb 0.8 – 2.0	Zn 1 - 10	Mn 0.12 – 0.14	Al ₂ O ₃ 0.8 – 4.6	S 2.4 – 2.5
SiO ₂ 3 – 9	P ₂ O ₅ 0.1 – 0.44	CaO 3.5 – 18	MgO 3.5 - 17	Na ₂ O 0.15 – 0.24	K ₂ O 0.08 – 0.36	

Table 7.6 : Typical composition in [weight-%] of sludge from BFGas treatment – based on [IISI, 1987; Mertins, 1986; data from European blast furnaces from 1997]

In most cases this sludge is landfilled. Figure 7.8 shows the fate of dust and sludge from BFGas treatment without distinguishing them. The percentage of 33% landfilling is a result of sludge dumping.

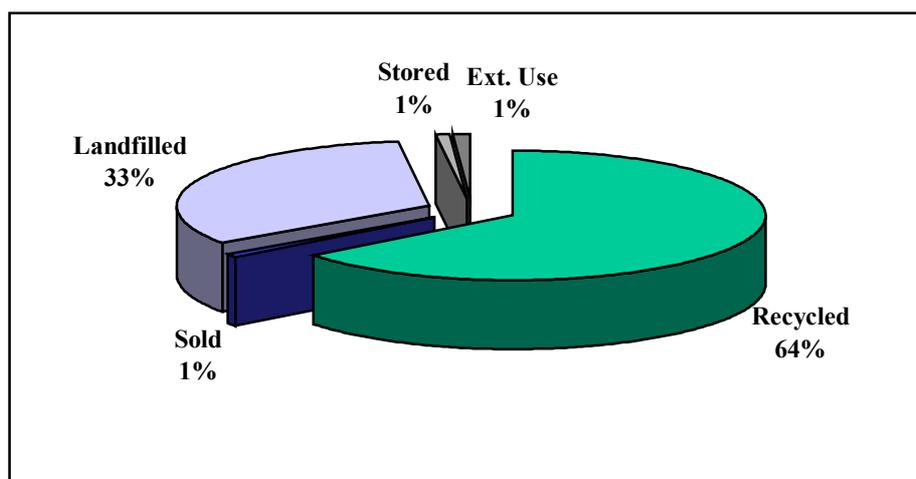


Figure 7.8 : Fate of dust and sludge from BFGas treatment in the EU – [EC Study, 1996]

7.2.2.2.3 Slag from blast furnaces

The specific quantity of slag mainly depends on the raw materials used, but lies in the range 210 – 310 kg/t pig iron produced (Table 7.1). Its composition for basicity below and above 1.0 is given in Table 7.7.

Classification CaO/SiO ₂ MgO-content	Blast furnace slag	
	> 1.0 middle	< 1.0 high
Fe _{total}	0.2 – 0.6	0.4
Mn _{total}	0.2 – 0.7	0.3
TiO ₂	0.5 – 2.7	0.7
Al ₂ O ₃	9.0 – 14.0	9.2
S _{total} mainly CaS	1.1 – 2.0	1.6
SiO ₂	33.2 - 37.0	38.4
CaO	38.1 – 41.7	35.6
MgO	7.0 – 11.0	18.0
Na ₂ O	0.3 – 0.6	0.5
K ₂ O	0.6 – 0.8	0.8
CaO/SiO ₂	1.1 – 1.2	0.9
(CaO+MgO)/ SiO ₂	1.3 – 1.5	1.2

Table 7.7 : Chemical composition in [% weight] of blast furnace slag for basicity below and above 1.0 – [Geiseler, 1992]

The slag is normally reused for various purposes (Table 7.9).

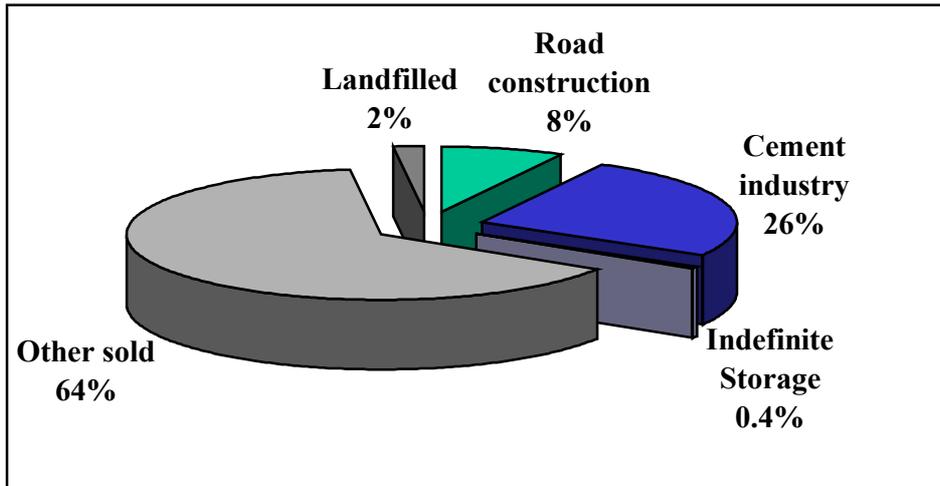


Figure 7.9 : Final use of blast furnace slag in the EU – [EC Study, 1996]

7.2.2.3 Wastewater emissions

7.2.2.3.1 Overflow water from BFGas treatment

Water from BFGas scrubbing is normally treated, cooled and recycled to the scrubber (Figure 7.10). Treatment usually takes place in circular settling tanks.

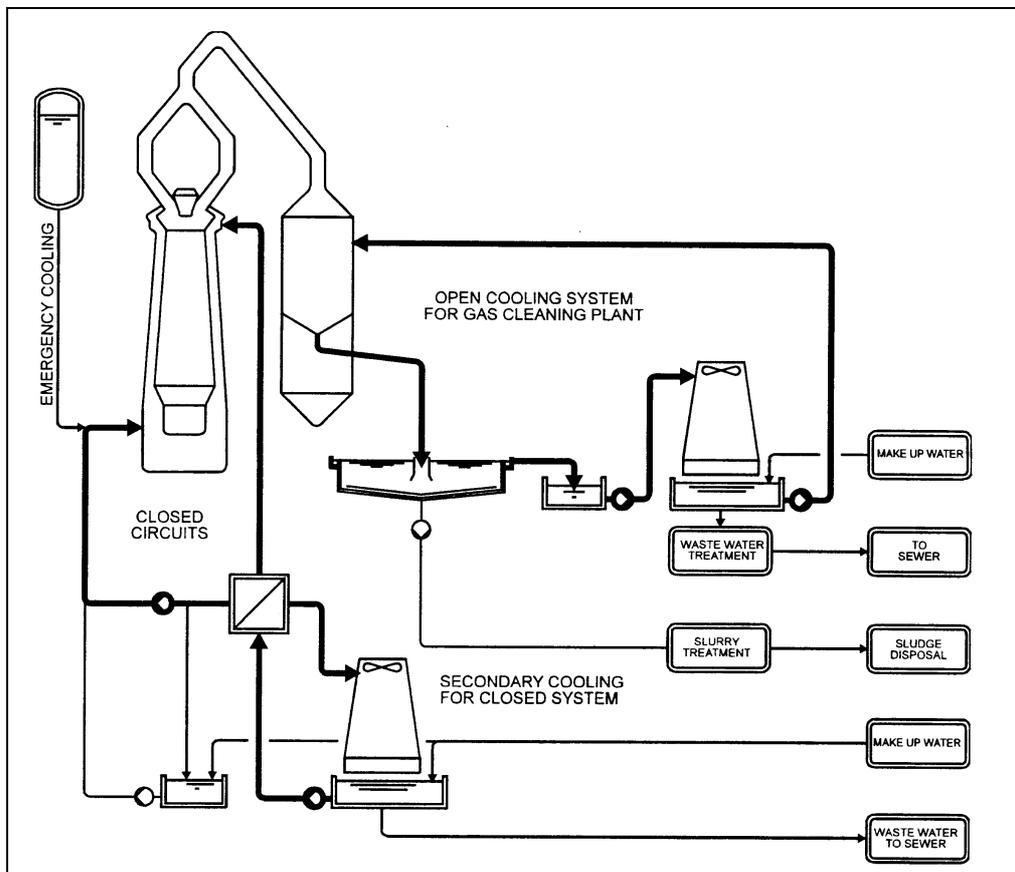


Figure 7.10 : Schematic diagram of water management at a blast furnace

The overflow of the circuit is normally 0.1 – 3.5 m³/t pig iron depending on raw material quality/specification and water availability which influences the measures taken to optimise water recycling. Especially high salt content raw materials can require significant higher volumes of wash water.

7.2.2.3.2 Wastewater from slag granulation

Overflow of water from slag granulation primarily depends on water availability and is in the range 0.125 – 10 m³/t pig iron produced. Information on chemical composition is available from one German plant (Table 7.8).

Parameter	Unit	Number of measur.	Average concentration	Median concentration	Emission factor*	Unit
Flow	[m ³ /d]		16000		1.96	[m ³ /t]
Pb	[µg/l]	19	3.45	2.0	3.92	[mg/t pig iron]
Cr	[µg/l]	19	3.32	2.0	3.92	[mg/t pig iron]
Cu	[µg/l]	19	5.99	4.20	8.232	[mg/t pig iron]
Zn	[µg/l]	19	37.63	20.00	39.2	[mg/t pig iron]
Ni	[µg/l]	19	4.91	3.00	5.88	[mg/t pig iron]
TOC	[mg/l]	19	4.40	4.10	8.036	[g/t pig iron]
COD	[mg/l]	19	20.62	20.30	39.788	[g/t pig iron]

*based on the median

Table 7.8 : Example from Stahlwerke Bremen GmbH, D-Bremen for the composition of wastewater from slag granulation in 1996/1997 – [Weigel, 1998]

7.2.2.3.3 Blow down from cooling water circuit

Blow downs of the cooling water circuit take place (see Figure 7.10). Information on representative quantities and chemical composition is not available.

7.2.2.4 Energy and reductant demand

The blast furnace process is the most energy-consuming process unit (inform of reducing agents) in iron and steel production (see Figure 3.4). Table 7.9 gives figures for the energy input/output of a blast furnace using coal injection and top gas pressure recovery for electricity generation.

Energy carrier	Specific energy value (GJ/t pig iron)
Input:	
- coke	12.4
- powdered coal	1.63
- hot blast (from stoves)	4.52
- electricity	0.12
- Total (gross)	18.67
Output	
- electricity	0.35
- BFgas	5.15

Table 7.9 : Example of energy input/output of a blast furnace with high top pressure using coal injection and top gas pressure recovery in 1986 - [InfoMil, 1997]

Many plants with adequate top gas pressure (> 1.5 bar gauge) and sufficient pressure drop are suitable for installation of expansion turbines for the recovery of top gas pressure energy.

Process improvements have meant that the specific coal consumption in European steelworks has decreased considerably over the last 30 years.

A trend towards direct reducing agent injection will probably further decrease specific coke consumption. Direct reducing agent injection in the blast furnace replaces the use of coke, thus saving energy in coke production. Also, many European blast furnaces use oil injection instead rather than coal injection.

7.3 Techniques to consider in the determination of BAT

Process-integrated measures

- PI.1 Direct injection of reducing agents
- PI.2 Energy recovery from blast furnace gas
- PI.3 Energy recovery from top gas pressure
- PI.4 Energy savings at the hot stove
- PI.5 Use of tar-free runner linings

End-of-pipe techniques

- EP.1 Blast furnace gas treatment
- EP.2 De-dusting of tap holes and runners
- EP.3 Fume suppression during casting
- EP.4 Hydro-cyclonage of blast furnace sludge
- EP.5 Treatment and reuse of scrubbing water
- EP.6 Condensation of fume from slag granulation

PI.1 Direct injection of reducing agents

Description: Direct injection of reducing agents means replacing part of the coke by another hydrocarbon source, which is injected in the furnace at the tuyère level. These hydrocarbons may be in the form of heavy fuel oil, oil residues, granular or pulverised coal, natural gas or plastic waste. Granular or pulverised coal and oil are currently used. By reducing the need for coke overall pollution and energy demand decrease.

It should be stressed, however, that a certain amount of coke is still necessary in the blast furnace to allow proper blast furnace operation. The coke provides the required carrying capacity to sustain the blast furnace charge and ensures sufficient gas penetration.

The theoretical maximum for coal injection at the tuyère level is thought to be 270 kg/t pig iron. This limit is set by the carrying capacity of the coke and the thermochemical conditions in the furnace. Tuyère injection of hydrocarbons requires additional injection of oxygen (at increasing levels as tuyère injection rates increase) in order to achieve the required temperatures within the furnace itself. Little experience has been gained at high tuyère injection rates (and thus elevated oxygen levels), and here, among other things, safety also becomes a more important issue.

The type of coal and the effects of coal injection on the quality of pig iron also influence the maximum input rate [Campbell, 1992; InfoMil, 1997].

Type	Injection rate (kg/t pig iron)	Residual coke rate (kg/t pig iron)	Total coal consumption* (kg/t pig iron)
None	0	500	625
Typical	180	310	568
Best practice	210	270-300	547-585

* it is assumed that 1 kg of coke is produced from 1.25 kg of coal.

Table 7.10 : Comparison of residual coke consumption and total coal consumption at different levels of direct injection pulverised coal

Main achieved energy saving: The net energy savings of coal injection have been calculated at 3.76 GJ/t coal injected. At an injection rate of 180 kg/t pig iron, energy savings amount to 0.68 GJ/t pig iron or 3.6% of the gross energy consumption of the blast furnace (see Table 7.9). This energy saving is achieved indirectly as a consequence of reduced coke consumption. Higher input rates will enable higher energy savings.

Cross-media effects: Direct injection of reducing agents decreases the need for coke production. Thus, emissions at the coke oven plant are avoided. For every kg of coal injected, approximately 0.85-0.95 kg of coke production is avoided.

Applicability: Direct injection of reducing agents is applicable both at new and existing blast furnaces. It should be noted that this highly process-integrated measure is closely related to the operation of the blast furnace itself and the resulting stability and the pig iron and slag quality.

Reference plants: Coal or oil injection into blast furnaces is a technique now widely applied in Europe and world-wide (e.g. coal injection into blast furnaces at Hoogovens IJmuiden, NL-IJmuiden; Sidmar, B-Gent; Thyssen AG, D-Duisburg and oil injection at British Steel, UK-Teesside; Voest Alpine, A-Linz).

Operational data: At Hoogovens IJmuiden pulverised coal is injected on a commercial scale. The standard injection rate was 170 kg coal/t hot metal in blast furnace 6 in 1993, but tests in the blast furnace have shown successful injection rates of 210 kg coal/t pig iron with a coke rate of 270 kg/t pig iron. However, these rates do not yet seem to be possible for long periods. Measurements have indicated that less than 1% of the injected coal escapes through the blast furnace top [Steeghs, 1994].

At high coal injection rates, the furnace tends toward low wall working and an increase in blast furnace resistance. This indicates that careful control of the burden distribution with low coke rates and a delicate balance between wall and central gas flow are necessary [Steeghs, 1994].

Economics: There are economic incentives for using high rates of coal injection to achieve greater cost savings, particularly at plants which might otherwise face capital expenditure on rebuilding coke ovens or may have to purchase coke. Furthermore, coal injection can allow the use of coals of a lower quality compared to coking coals. This may also reduce costs.

Costs at 10 Ecu₁₉₉₆/GJ in 1988 are reported [InfoMil, 1997]. However, [Campbell, 1992] indicate that costs are saved, due to the lower coke consumption. The capital cost of installing coal injection at British Steel, UK - Port Talbot No. 4 blast furnace in 1997 was approximately 24 MEcu (including some redundant part from another site).

Reference literature: [Campbell, 1992; Steeghs, 1994; InfoMil, 1997]

PI.2 Energy recovery from blast furnace gas (BFgas)

Description: A typical blast furnace produces approximately 1200 to 2000 Nm³ of BFgas per tonne of pig iron. The BFgas consists of 20-28% of carbon monoxide (CO) and 1-5% hydrogen (see Table 7.3 and Table 7.4). The carbon monoxide is formed during oxidation of carbon in the blast furnace. Much of the CO is further oxidised to CO₂ in the blast furnace. Carbon monoxide and hydrogen represent a potential energy source and measures are taken at all blast furnaces around the world to recover this energy.

Blast furnace top gas is therefore cleaned and stored in gasholders for subsequent use as a fuel. Given the low energy content of BFgas per Nm³ it is often enriched with coke oven gas or natural gas prior to use as a fuel.

Main achieved energy saving: The energy content of BFgas typically varies from 2.7-4.0 MJ/Nm³, depending on its carbon monoxide concentration. This is only 10% of the energy content of natural gas. Nevertheless, the large amounts of BFgas generated mean the energy recovering potential is very high. Total export from the blast furnace is approximately 5 GJ/t pig iron, which equals 30% of the gross energy consumption of the blast furnace.

Applicability: Applicable at all new and existing plants.

Cross-media effects: BFgas cleaning is unavoidable and results in wastewater and solid residues.

Reference plants: Applied at many new and existing blast furnaces in the world.

Operational data: No specific operational data required.

Economics: Significant economies are achieved through energy savings.

Reference literature: not available

PI.3 Energy recovery from top gas pressure

Description: High top pressure blast furnaces provide an ideal opportunity for recovering energy from the large volumes of pressurised top gas which they generate. Energy is recovered by means of an expansion turbine, which is installed after the top gas cleaning device.

Main achieved energy saving: The amount of energy that can be recovered from the top gas pressure depends on the top gas volume, the pressure gradient and the admission temperature. Energy recovery in this way is viable when the BFgas cleaning device and distribution network has a low pressure drop.

Top gas pressure in modern blast furnaces is approximately 0.25-2.5 bar gauge. The pressure of the BFgas collecting main is approximately 0.05-0.1 bar. Part of the top gas pressure is "consumed" by the gas cleaning device.

The electricity generated is reported to be as much as 15 MW in a modern blast furnace with a top gas pressure of 2-2.5 bar.

Energy savings are estimated at up to 0.4 GJ/t pig iron for a 15 MW turbine. The savings amount to 2% of the gross blast furnace energy demand. Application of top gas pressure recovery at blast furnaces is common in furnaces with high top pressure.

Applicability: Top gas pressure recovery can be applied at new plants and in some circumstances at existing plants, albeit with more difficulties and additional cost.

At new plants, the top gas turbine and the BFgas cleaning facility can be adapted to each other in order to achieve a high efficiency of both scrubbing and energy recovery.

Cross-media effects: It should be noted that the use of turbines requires a thorough cleaning of the BFgas. First of all a high content of alkalis causes technical problems (corrosion).

Reference plants: Top gas pressure recovery is applied worldwide at modern blast furnaces with a high top gas pressure and volume.

Operational data: Usually operates automatically without problems. Axial turbines can be used. These are more efficient than radial turbines. The critical point is when the turbine has to be shut down and the pressure gradient is transferred to the gas cleaning device. Special "safety" measures are always taken to avoid damage to the gas cleaning device and/or the collecting main.

Economics: The main drive to install a top gas pressure recovery turbine is the economic benefits. Profitability of the turbine increases with increasing volume and pressure gradient of the top gas and with increasing energy costs. In a modern blast furnace, a pay-back period of less than three years is possible, but, dependent on local circumstances and top gas pressure it can be more than 10 years.

Reference literature: [Arimitsu, 1995; Fisher, 1988; Joksch, 1995; InfoMil, 1997]

PI.4 Energy savings at the hot stoves

Description: The hot stoves are fired with (often enriched) BFGas. Several techniques are available to optimise the energy efficiency of the hot stove:

1. Computer-aided hot stove operation; avoids unnecessary reserves by adapting the energy supply to the actual demand and minimises the amount of enriching gas added (in cases where enrichment takes place).
2. Preheating of the fuel in conjunction with insulation of the cold blast line and waste gas flue. Sensible heat from the flue gas can be used to preheat the fuel media. The feasibility of this depends on the efficiency of the stoves as this determines the waste gas temperature; e.g. at waste gas temperatures below 250°C heat recovery may not be a technically or economically attractive option. The heat exchanger preferably consists of a heating oil circuit, for economic reasons. In some cases, imported heat may be used, e.g. sinter cooler heat, if the distances are reasonable. A preheated fuel medium reduces energy consumption. At plants which use enriched blast furnace gas, preheating the fuel could mean that enrichment would no longer be necessary.
3. Improvement of combustion through more suitable burners;
4. Rapid O₂ measurement and subsequent adaptation of combustion conditions.

Main achieved energy saving:

[Joksch, 1995] reported the following energy savings:

1. Computer-aided hot stove operation leads to an efficiency improvement of the hot stove of more than 5%. This equals an energy saving of approximately 0.1 GJ/t pig iron.
2. Preheating of the fuel media can lead to an energy saving of approximately 0.3 GJ/t pig iron as well.
- 3/4. Another 0.04 GJ/t pig iron may be saved by improved combustion and adaptation of combustion conditions.

The total energy savings possible by a combination of techniques is of the order of 0.5 GJ/t pig iron produced.

Applicability: Applicable both at new and existing plants.

Cross-media effects: no cross-media effects were reported by [Joksch, 1995]. It might be expected that preheating of the fuel media and a reported increase of the flue gas temperature leads to higher NO_x emissions from the hot stoves. The application of modern burners may reduce the NO_x emissions.

Reference plants:

Thyssen Stahl AG, D-Duisburg;
Blast Furnace No.7; Hoogovens IJmuiden, NL- IJmuiden;
Sidmar, B-Gent;
Aceralia, E-Gijon.

Operational data: not available

Economics: These measures might be attractive from an economic point of view, because energy consumption is reduced and thus money is saved. The profitability depends on the amount of energy saved and on the investment and operational costs of the measures. Typical installation costs for heat recovery are 6 MEcu₁₉₉₇ per set of stoves, i.e. per blast furnace. Implementation of computer-aided control could require the construction of a fourth stove in case of three stoves blast furnaces (if possible at all) in order to maximise benefits.

Reference literature: [Joksch, 1995]

PI.5 Use of tar-free runner linings

Description: The runner system in the cast house of a blast furnace is made of a refractory outer layer (e.g. concrete). The runners are lined with a heat resistant material, which is based on alumina embedded in a carbon-matrix. Coal tar pitch can serve as a binder.

The linings protect the outer layer against the heat of the liquid iron and (especially) the liquid slag. The linings are subject to wear and are renewed every few weeks. The quality of the carbon matrix is an important factor in the durability of runner linings. A weak matrix does not hold the alumina and cause the linings to wear more rapidly.

The required strength of the mixture is only achieved after heating for several hours with burners. Tar decomposition during heating gives rise to emissions of hydrocarbons (and PAH). A minor part of the emissions occur during casting of slag and pig iron.

A new type of runner lining which does not contain tar has been developed and is being applied successfully. With the new runner lining emissions of volatile organic compounds (VOC) and polycyclic aromatic hydrocarbons (PAH) are much lower. The new material has the same slag resistance and the durability can be even better than that of traditional runner linings.

Main achieved emission level:

Table 7.11 shows figures for the emission reduction resulting from use of tar free runner linings. The figures shown are estimates based on a theoretical model.

Component	Traditional runner lining (g/t pig iron)	Tar-free runner lining (g/t pig iron)	Emission reduction (%)
Volatile organic compound (VOC)	100	1	99
Polycyclic aromatic hydrocarbons (PAH)	3.5	0.03	99

Table 7.11 : Emissions from runner linings – [InfoMil, 1997]

Applicability: In principle, tar-free runner linings are applicable at both new and existing blast furnaces.

Cross-media effects: Application of this measure has a positive effect on the health of employees, because they are less exposed to harmful components. No other cross-media effects are known.

Reference plant: Hoogovens IJmuiden, NL-IJmuiden

Operational data: At Hoogovens IJmuiden, the tar-free runner linings are currently applied without problems and even lead to improved runner lining durability.

Economics: Not available

Reference literature: [InfoMil, 1997]

EP.1 Blast furnace gas treatment

Description: Blast furnace gas (BFgas) must be cleaned in order to meet grid gas requirements. When leaving the blast furnace, the BFgas (or "top gas") contains particulate matter, cyanides (HCN), ammonia (NH₃) and sulphur compounds (see Table 7.3 and Table 7.4).

BFgas is usually treated in two stages. In the first stage, coarse particulate matter is removed by means of dry cyclonage, a deflector etc. The coarse particulates collected in this way have a high iron content and can be recycled to the sinter plant (see Table 7.5).

In the second stage, particulate matter (incl. zinc oxide (ZnO) and carbon (C)), cyanide and ammonia are removed by wet scrubbing. The scrubbers are specially designed hurdle-type, venturi or annular gap scrubbers. In some cases wet electrostatic precipitation is applied.

In modern plants, special attention is paid to the pressure drop of the gas cleaning system, because a high pressure drop has a negative effect on the energy efficiency of top gas pressure recovery turbine (see PI.3). The aim is therefore to construct gas cleaning systems with a low pressure drop and a high gas cleaning efficiency.

Main achieved emission level: BFgas cleaning systems are usually highly efficient, achieving a clean gas particulate matter concentration of <10 mg/Nm³. Values as low as 1 mg/Nm³ have been achieved as well. Particulate matter not caught by the gas cleaning system is emitted or combusted at the site of combustion of the BFgas.

The pressure drop of the gas cleaning system depends on the type of equipment used. Reported pressure drops of two modern systems were between 0.07 and 0.14 bar. Pressure drops in older systems range from 0.15-0.5 bar.

Applicability: Blast furnaces around the world apply BFgas cleaning systems. At new plants, a modern system with a low pressure drop and low water and energy consumption can be applied. Many old blast furnaces are equipped with "old-fashioned" scrubbers. These scrubbers are able to achieve the grid gas requirements, but consume considerable amounts of water and energy and have a relatively large pressure drop. It is sometimes possible to replace the scrubber by a more modern type, but this should be considered in conjunction with the application of a top gas pressure turbine (see chapter: PI.3), which depends on the operating pressure of the furnace amongst other factors.

Cross-media effects: It should be noted that the scrubbers generate a contaminated wastewater flow. Gross water consumption of the blast furnace systems is 0.1 – 3.5 m³/t pig iron (see 7.2.2.3.1 and Table 7.1). Raw materials with a higher salt content can make greater flows necessary in order to clean the BFgas. The water flow generated contains suspended solids (e.g. carbon and heavy metals (Zn, Pb)), cyanide compounds and ammonia. Usually, the wastewater is treated by means of heavy metal precipitation and a solid waste (sludge) is generated.

This blast furnace sludge contains relatively high concentrations of zinc (Zn) and lead (Pb) (see Table 7.6). This hampers recycling of the sludge into the production process. Zinc in particular is a "poison" to the blast furnace. Therefore, some plants apply hydrocyclonage to separate the sludge flow into two flows; one with a low zinc content, which can be recycled to the sinter plant, and a second with a high zinc content, which can be stored or disposed of (see EP.4).

Reference plant: BFgas treatment is widely applied at blast furnaces around the world.

Operational data: not available

Economics: not available

Reference literature: [Pazdej, 1995; InfoMil, 1997]

EP.2 De-dusting of tapholes and runners

Description: Pig iron and slag are cast from the blast furnace and run through the runners to the ladle and the slag treatment unit respectively. During tapping/casting the liquid metal comes into contact with atmospheric oxygen (O_2) when flowing along the runners. As a result of the high temperature of the iron (1300 - 1500°C) it reacts with the oxygen to form iron oxides (e.g. Fe_2O_3), the so-called brown fume.

Slag does not react with atmospheric oxygen because most of its components are already oxidised. However, alkaline oxides (e.g. Na_2O and K_2O) can vaporise out of the slag forming particulate emissions.

Generally, two types of measures can be applied to reduce dust emissions from casting:

1. Covering the runners with movable lids,
2. Dissipating oxygen from the tapped hot metal by means of covering the hot metal with nitrogen (N_2). In this way, the formation of iron oxides is avoided (see EP.3).

Main achieved emission level: During tapping, approximately 400 - 1500 g particulate matter is generated per tonne of pig iron when no abatement measures are applied. Evacuation of air from above the runners essentially leads to greater particulate matter generation, due to the increased availability of oxygen.

In cases where the runners are covered by lids and particulate matter is evacuated and treated, the most important parameter is evacuation efficiency. Special attention should be paid to the covering of the runners. The runner covers should be tightly connected, aiming for a leak-free system. At the tap-hole, efficient evacuation of the generated particulate matter is difficult, due to the space required for operation of the tap-hole drill and the mudgun and due to the presence of the bustle pipe, which serves the tuyères and circles the blast furnace at the height of the bosh. Usually, a very large flow is needed to efficiently evacuate the particulate matter generated in the vicinity of the tap-hole.

The main evacuation points in a cast house are

- the tap-hole,
- the skimmer and
- the tilted runner (charging the torpedo ladle)

The specific flow of evacuated gas (waste gas) is 1200-3300 Nm^3/t pig iron. With an effective dust collection and abatement system (e.g. bag filter) specific emission factors of less than 10 g/t pig iron can be achieved [InfoMil, 1997]. The dust collection efficiency can exceed 99% and the dust removal efficiency of fabric filters can exceed 99% also [InfoMil, 1997].

Applicability: Covering of runners and subsequent evacuation and cleaning can be applied at new and existing plants.

Cross-media effects: The application of evacuation and treatment of gases consumes additional energy as it requires powerful fans. From [InfoMil, 1997], an energy consumption of approximately 0.007 GJ/t pig iron can be calculated, based on an annual pig iron production of 3 Mt by BF-7 at 8640 operational hours.

The collected dust has a high iron content and can be recycled to the sinter plant.

Reference plant:

Evacuation and treatment of gases:

Blast furnaces 7, Hoogovens IJmuiden, NL-IJmuiden; blast furnace A, Voest Alpine, A-Linz; blast furnace Schwelgern, Thyssen AG, D-Duisburg

Operational data: The described technique is operated in practice without significant problems.

Economics: At Hoogovens investments for a fabric filter system treating 690000 Nm³/h is reported to be in the range of 1 to 2.3 million Ecu₁₉₉₆. This only includes the fabric filter equipment. Runner covering and evacuation lay-out are not included in this figure.

Operational costs can be calculated as 0.5 to 2.8 Ecu₁₉₉₆/t pig iron, based on an annual production of 3 Mt pig iron by BF-7 and 8640 operational hours [InfoMil, 1997].

Installation cost for British Steel, UK-Scunthorpe (1 million t/a BF) for one cast house de-dusting system was approximately 4.0 M Ecu₁₉₉₇.

Installation cost for the cast house de-dusting system at blast furnace A of Voest Alpine, A-Linz (about 3 million t/a BF) was approximately 14.5 M Ecu₁₉₉₆. Operation costs (without energy) are about 0.42 million Ecu per year₁₉₉₆.

Reference literature: [InfoMil, 1997]

EP.3 Fume suppression during casting

Description: EP.2 describes conventional de-dusting systems of cast house emissions. These systems are rather complex and costly. New approaches prevent the molten iron from reacting with atmospheric oxygen to form “brown fume” (fume suppression). To do so the whole transport route for the hot metal, from the tap-hole via various distribution and transfer points to the torpedo ladle, is enclosed by means of carefully designed screening structures. The space between the molten metal and the covers is kept as small as possible, and it is (if necessary) flooded with nitrogen (inert gas). In integrated steel works, the nitrogen yielded in air separation for oxygen generation may be available for this purpose.

This new method eliminates the installation and operation of complex and expensive exhaust and filter systems which was necessary previously, and thus leads to considerable cost savings. The cost of recycling of filter dust is also reduced.

At the tap-hole, an exhaust system is necessary.

Figure 7.11 shows the quantity of dust generated during charging of hot metal with and without dust suppression. These figures are about 100 times lower when nitrogen inertisation is used.

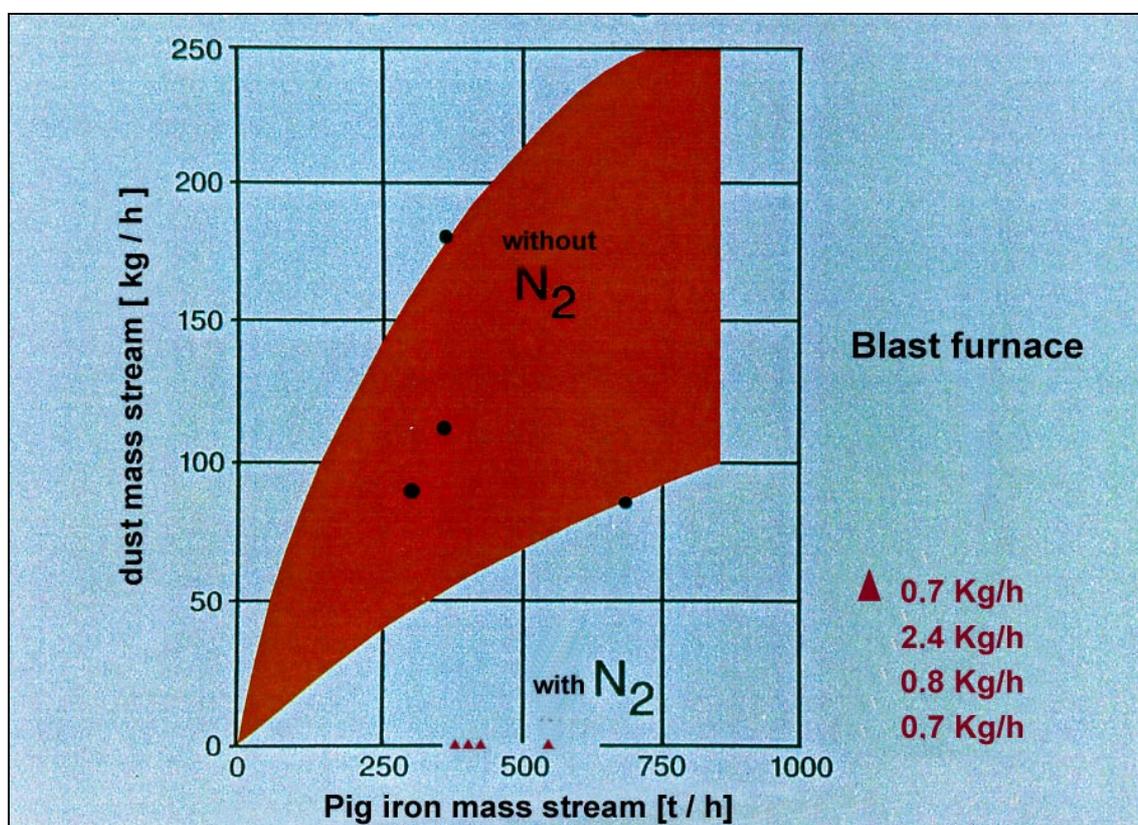


Figure 7.11 : Dust generation with and without nitrogen inertisation during hot metal charging (to the torpedo level) depending on the flow of pig iron – [de Haas, 1997]

Main achieved emission levels: During conventional casting 0.4–1.5 kg dust/t pig iron is generated (Table 7.1). This quantity is reduced by dust suppression to about 0.012 kg dust/t pig iron [de Haas, 1997]. Figure 7.12 shows the effect of dust suppression during charging of hot metal to the torpedo ladle.



Figure 7.12 : Charging of hot metal into torpedo ladle with dust suppression with inert gas – [Haas, 1997].

Applicability: Applicable both at new and existing plants.

Cross-media effects: There are no significant cross-media effects if the system is compared with conventional de-dusting systems, described in EP.2.

Reference plants: At Stahlwerke Bremen, D-Bremen, this fume suppression technique has been in operation since 1991.

Operational data: Experience with fume suppression at Stahlwerke Bremen shows constant operation conditions without significant problems.

Economics: A comparison of costs is shown in Figure 7.13 shows the comparison of costs. The new fume suppression technique is considerably cheaper. The installation at Stahlwerke Bremen with a production of 3 Mt pig iron/a required an investment of 6.8 million Ecu₁₉₉₆ including dust suppression and tap hole de-dusting with a subsequent bag filter.

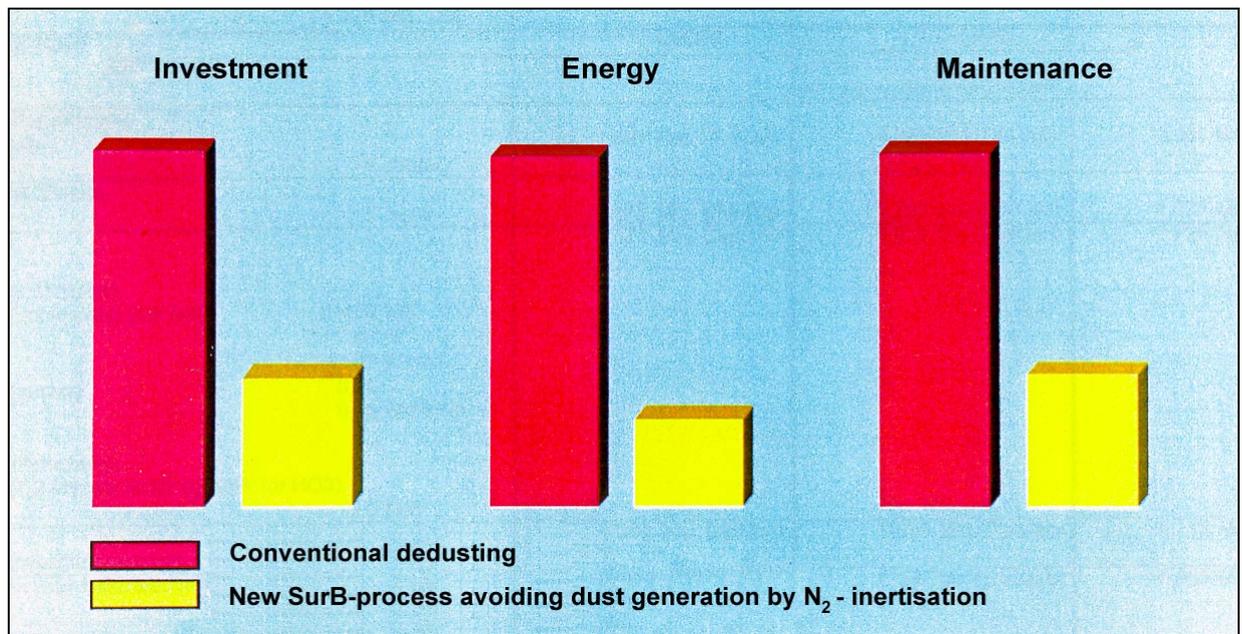


Figure 7.13 : The installation at Stahlwerke Bremen with a production of 3 Mt pig iron/a required an investment of 6.8 million Ecu₁₉₉₆ including dust suppression and tap hole de-dusting with a subsequent bag filter.

Energy costs are about 190000 EURO/a and maintenance costs about 170000 EURO/a. This is also much lower than conventional systems. Nitrogen cost can however be variable, depending on local circumstances.

Reference literature: [Grützmacher, 1991; de Haas, 1997]

EP.4 Hydrocyclonage of blast furnace sludge

Description: Blast furnace top gas contains large amounts of particulate matter (7-40 kg/t pig iron – see Table 7.3). A large part of this particulate matter is removed in the dry first step of the BFgas treatment system. This part mainly consists of relatively coarse material with a high iron and carbon content and is recycled to the sinter plant.

The remainder (1-10 kg/t pig iron) is scrubbed from the BFgas by means of wet scrubbing. After precipitation 3-5kg of sludge per tonne of pig iron is generated. This sludge has a relatively high zinc (Zn)-content. This is an obstacle to reuse of the sludge in the sinter plant.

By means of hydrocyclonage of the sludge, a zinc-rich and a zinc-poor sludge can be generated out of the total sludge. The zinc is mainly present in the form of zinc oxide (ZnO), which appears as very small particles (see 7.2.2.2.2). Hydrocyclonage concentrates these small particles in the overflow, whereas the zinc-poor fraction is leaving the cyclones through the underflow. It should be noted that the efficiency of hydrocyclonage depends on the sludge characteristics.

The sludge from the underflow is reused in the sinter plant. Note that this reuse should be seen in relation to the total zinc input in the blast furnace. The zinc-rich sludge from the overflow is stored, awaiting future possibilities for processing, or landfilled. Addition problems may arise from radioactive components which preferably remain in the fine fraction.

Main achieved emission level: In Table 7.1 the zinc and lead contents of the sludges are given.

	Specific sludge generation (kg/t pig iron)	Zn content (%)	Weight on dry basis (%)	Treatment
Untreated sludge	1.0 - 9.7	0.1 – 2.5	100	Hydrocyclonage
Hydrocyclonage overflow	0.2 – 2.7	1 - 10	20 - 40	Stored on stock pile/landfill
Hydrocyclonage underflow	0.8 - 7.8	0.2 - 0.6	60 - 80	Recycled to sinter plant

Table 7.12 : Example for the zinc content of hydrocyclonaged blast furnace sludge – based on [Pazdej, 1995; InfoMil, 1997]

Applicability: Hydrocyclonage can be applied at new and existing plants as an end-of-pipe technology, whenever wet de-dusting is applied and where grainsize distribution allows reasonable separation.

Cross-media effects: A small but unquantified amount of energy is consumed to operate the cyclones.

Reference plant: Hoogovens IJmuiden, NL- IJmuiden; Thyssen AG, D-Duisburg

Operational data: The system operates without problems

Economics: [UBA Rentz, 1996] reports investment cost for a three-stage plant with a capacity of 20000 t/a is 2 million Ecu₁₉₉₄ and operation cost is 25 Ecu/t.

Reference literature: [Pazdej, 1995; UBA Rentz, 1996]

EP.5 Treatment and reuse of scrubbing water

Description: BFGas is usually cleaned in specially designed hurdle type, venturi or annular gap scrubbers (see also EP.1). This generates a contaminated water flow containing suspended solids (1-10 kg/t pig iron; including (heavy) metals), cyanides and phenols. Measures can be taken to minimise discharges to water and to minimise water consumption.

To scrub the pollutants from the BFGas, approximately 0.3-4.0 l/Nm³ are needed. This figure corresponds to a gross water consumption of 0.4-8 m³/t pig iron. A large part of this water can be treated and recycled.

The treatment is usually performed in circular settling tanks. The sedimentation properties of the sludge is very often improved by dosage of flocculation agents (anionic polyelectrolytes, mixed polymers or activated silicic acids) or by sludge contact plants. Attention must be paid to pH value and water hardness. Operational data from 7 German sludge contact plants can be seen from Table 7.13.

Plant-no.	BFGas Flow [10 ³ Nm ³ /h]	Water flow [m ³ /h]	Content of suspended solids in [mg/l]		Flocculation agent		Water content of the sludge [%]	Electricity consumption [kWh/1000m ³]
			before treatment	after treatment	kind	dosage [mg/l]		
1	50	225-375	1000-3000	10-50	activated silicic acid	5-10	60-85	40
2	250	1500	500	50	mixed polymer	0.4	85-90	17.8
3	55	500	300-600	20	activated silicic acid	6	75-85	-
4	550	50	100-300	20-30	activated silicic acid	-	85-95	8
5	225	1100	260	10	anionic polyelectr.	0.13	63	27.5
6	320	1400	430	13	anionic polyelectr.	0.13	60	21.5
7	125	800	150	20	mixed polymer	10	60	-

Table 7.13 : Operational data from 7 German sludge contact plants for the treatment of BFGas scrubbing water – [Theobald, 1988]

Depending on the operational conditions of the blast furnace, cyanide treatment may be necessary, particularly during blowdowns. Meanwhile this is mainly performed by formaldehyde addition to the water circuit (Figure 7.14). Available experience indicates that:

- optimum pH is between 8-9
- there is no reaction below pH 7
- glyconitrile is decomposed at pH above 10 into cyanide and formaldehyde.

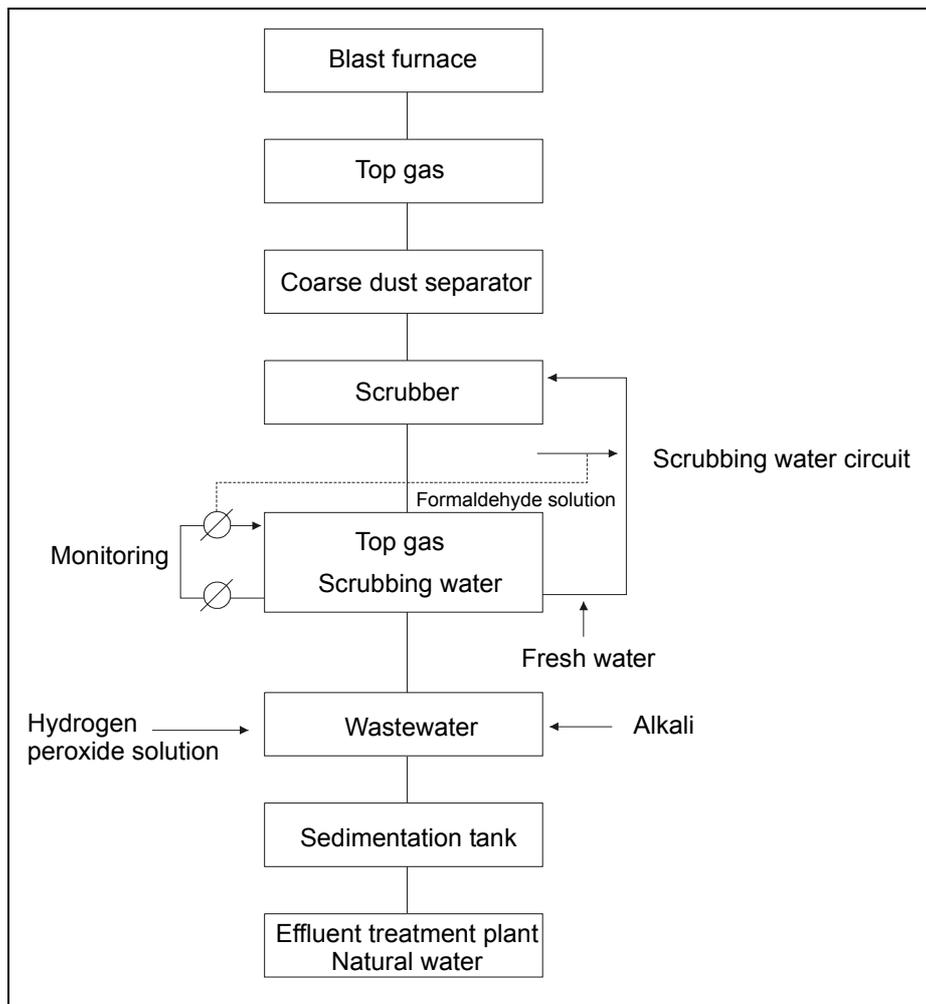


Figure 7.14 : Example process for cyanide treatment of scrubbing water circuit at blast furnaces – [Theobald, 1997]

The sludge from the fine particle precipitation has a relatively high zinc-content (see Table 7.6) and can be treated by means of hydrocyclonage (see also EP.4).

A new concept of wastewater treatment is going to be implemented at Hoogovens IJmuiden, NL-IJmuiden and will go into operation in 1999. It consists of the combined treatment of wastewater from the coke oven plant, the blast furnaces and from the fine scrubber at the pellet plant in an activated sludge system with pre-denitrification and nitrification in order to minimise COD and nitrogen compounds emissions.

Main achieved emission level: High recycling efficiency of the scrubbing water can be achieved with an overflow of only 0.1 m³/t pig iron. This water is removed from the system with the blast furnace sludge and may undergo further treatment.

Parameter	Unit	Average value	Unit	Emission factor
Flow	[m ³ /d]	3387	[m ³ /t pig iron]	0.23
COD	[mg/l]	51	[g/t pig iron]	11.7
CN ^{*1}	[mg/l]	0.7	[g/t pig iron]	0.06
Kjeldahl-N	[mg/l]	133	[g/t pig iron]	30
Sulphide (H ₂ S)	[mg/l]	2.2	[g/t pig iron]	0.5
Suspended solids	[mg/l]	16.1	[g/t pig iron]	3.7
Zinc (Zn)	[µg/l]	1051	[mg/t pig iron]	242
Copper (Cu)	[µg/l]	12.7	[mg/t pig iron]	3.0
Chromium (Cr)	[µg/l]	33.4	[mg/t pig iron]	7.6
Cadmium (Cd)	[µg/l]	0	[mg/t pig iron]	0
Nickel (Ni)	[µg/l]	39	[mg/t pig iron]	8.9
Lead (Pb)	[µg/l]	89	[mg/t pig iron]	18.3
Mercury (Hg)	[µg/l]	< 0.1	[mg/t pig iron]	0.02
Arsenic (As)	[µg/l]	5.7	[mg/t pig iron]	1.3
PAH-EPA	[µg/l]	3.1	[mg/t pig iron]	0.71

*1 free cyanide;
values from [Weigel, 1998]

Table 7.14 : Example for wastewater discharged from BFgas scrubbing water circuit after treatment – [InfoMil, 1997]

Applicability: Both new and existing plants can apply treatment and recycling of scrubbing water. Modern plants have a significant advantage, because the plant can be designed with an efficient water circuit. The salt content of the wash water influences the recycling rate. However, it is also possible to install an efficient water circuit in older plants.

Cross-media effects: When large amounts of water are recirculated, an efficient water treatment system must be used. Otherwise, operational problems will occur affecting the scrubbers (clogging etc.) and scrubbing efficiency will drop. Water treatment and recycling generates a (zinc-rich) sludge. A small overflow from the circuit is necessary to avoid accumulation of minerals/salts.

Scrubbing water recycling requires a significant amount of energy (see Table 7.13) is required. In addition the dosage of flocculation agents has to be considered.

Reference plant: Krupp Hoesch Stahl, D-Dortmund; Hoogovens IJmuiden, NL- IJmuiden.

Economics: Total investment for the blast furnace wastewater treatment at Hoogovens IJmuiden was reported to be approximately 18 million Ecu₁₉₉₆. The wastewater treatment includes suspended solid separation (two stages) and pH correction. Operational costs are not stated.

Reference literature: [InfoMil, 1997; Theobald, 1997; Theobald, 1988]

EP.6 Condensation of fume from slag processing

Description: As mentioned in 7.2.2.1.5 there are H₂S and SO₂ emissions from slag processing. This may lead to odour problems. In order to solve this problem some granulation plants are operated with fume condensation (Figure 7.15). The condensate and the slag de-watering water as well are circulated after cooling.

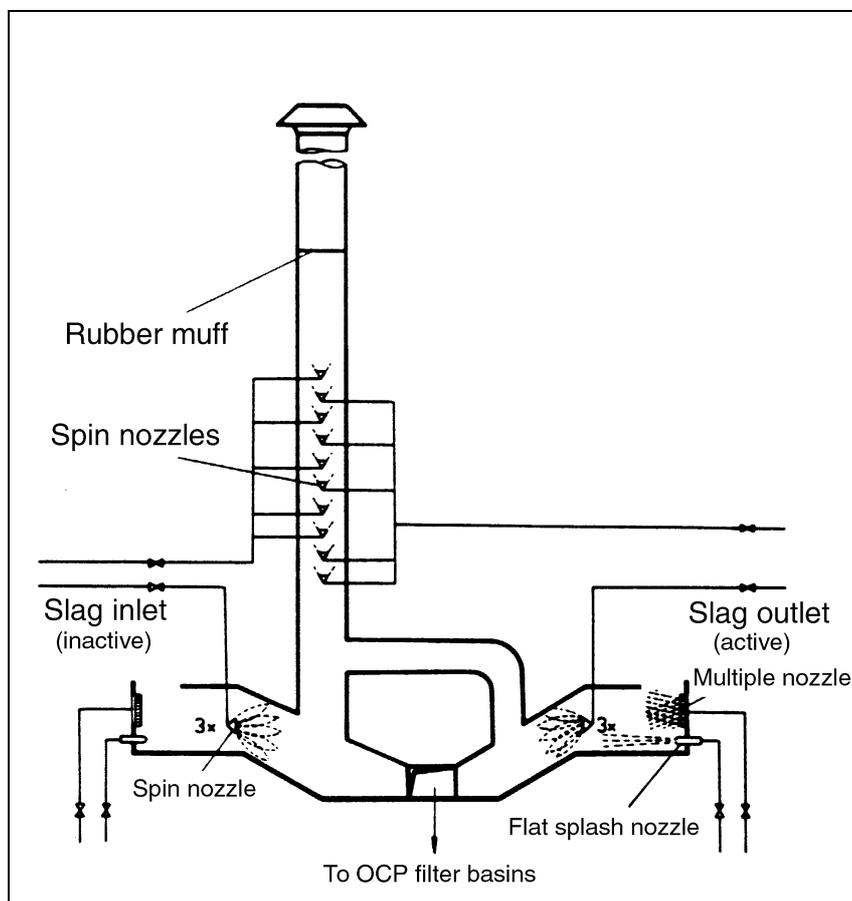


Figure 7.15 : Granulation of blast furnace slag with fume condensation – [Poth, 1985]

Main achieved emission levels: With fume condensation H₂S emissions are below 10 g H₂S/ t pig iron produced. According to Table 7.2 emissions of 1 g H₂S/ t pig iron are achievable.

Applicability: Applicable both at new and existing plants.

Cross-media effects: Cooling the circulated water needs considerable amounts of energy. Energy generation itself is very often linked with sulphur emissions. During fume condensation the absolute quantity of reduced sulphur is relatively small and may be related to the quantity emitted during energy generation.

Reference plants: Several slag granulation plants in Germany are equipped with fume condensation system, e.g. at Thyssen AG, D-Duisburg

Economics: not available

Reference literature: not available

7.4 Conclusions

In understanding this chapter and its contents, the attention of the reader is drawn back to the preface of this document and in particular the fifth section of the preface: “How to understand and use this document”. The techniques and associated emission and/or consumption levels, or ranges of levels, presented in this chapter have been assessed through an iterative process involving the following steps:

- identification of the key environmental issues for the sector; for blast furnaces these are BFgas containment, treatment and utilisation, fumes from casting and dealing with slags;
- examination of the techniques most relevant to address those key issues;
- identification of the best environmental performance levels, on the basis of the available data in the European Union and world-wide;
- examination of the conditions under which these performance levels were achieved; such as costs, cross-media effects, main driving forces involved in implementation of this techniques;
- selection of the best available techniques (BAT) and the associated emission and/or consumption levels for this sector in a general sense all according to Article 2(11) and Annex IV of the Directive.

Expert judgement by the European IPPC Bureau and the relevant Technical Working Group (TWG) has played a key role in each of these steps and in the way in which the information is presented here.

On the basis of this assessment, techniques, and as far as possible emission and consumption levels associated with the use of BAT, are presented in this chapter that are considered to be appropriate to the sector as a whole and in many cases reflect current performance of some installations within the sector. Where emission or consumption levels “associated with best available techniques” are presented, this is to be understood as meaning that those levels represent the environmental performance that could be anticipated as a result of the application, in this sector, of the techniques described, bearing in mind the balance of costs and advantages inherent within the definition of BAT. However, they are neither emission nor consumption limit values and should not be understood as such. In some cases it may be technically possible to achieve better emission or consumption levels but due to the costs involved or cross media considerations, they are not considered to be appropriate as BAT for the sector as a whole. However, such levels may be considered to be justified in more specific cases where there are special driving forces.

The emission and consumption levels associated with the use of BAT have to be seen together with any specified reference conditions (e.g. averaging periods).

The concept of “levels associated with BAT” described above is to be distinguished from the term “achievable level” used elsewhere in this document. Where a level is described as “achievable” using a particular technique or combination of techniques, this should be understood to mean that the level may be expected to be achieved over a substantial period of time in a well maintained and operated installation or process using those techniques.

Where available, data concerning costs have been given together with the description of the techniques presented in the previous chapter. These give a rough indication about the magnitude of costs involved. However, the actual cost of applying a technique will depend strongly on the specific situation regarding, for example, taxes, fees, and the technical characteristics of the installation concerned. It is not possible to evaluate such site-specific factors fully in this document. In the absence of data concerning costs, conclusions on economic viability of techniques are drawn from observations on existing installations.

It is intended that the general "BAT" in this chapter could be used to judge the current performance of an existing installation or to judge a proposal for a new installation and thereby assist in the determination of appropriate "BAT" based conditions for that installation. It is foreseen that new installations could be designed to perform at or even better than the general "BAT" levels presented here. It is also considered that many existing installations could reasonably be expected, over time, to move towards the general "BAT" levels or do better.

While the BREFs do not set legally binding standards, they are meant to give information for the guidance of industry, Member States and the public on achievable emission and consumption levels when using specified techniques. The appropriate limit values for any specific case will need to be determined taking into account the objectives of the IPPC Directive and the local considerations.

For blast furnaces, the following techniques or combination of techniques are considered as BAT. The order of priority and the selection of the techniques will differ according to local circumstances. Any other technique or combination of techniques achieving the same or better performance or efficiency can also be considered; such techniques may be under development or an emerging technique or already available but not mentioned/described in this document.

1. Blast furnace gas recovery;
2. Direct injection of reducing agents;
e.g. a pulverised coal injection of 180 kg/t pig iron is already proven, but higher injection rates could be possible.
3. Energy recovery of top BF gas pressure where prerequisites are present;
4. Hot stoves
 - emission concentration of dust <10 mg/Nm³ and of NO_x<350 mg/Nm³ (related to an oxygen content of 3%) can be achieved
 - energy savings where design permits
5. Use of tar-free runner linings;
6. Blast furnace gas treatment with efficient de-dusting;
Coarse particulate matter is preferably removed by means of dry separation techniques (e.g. deflector) and should be reused. Subsequently fine particulate matter is removed by means of:
 - a scrubber *or*
 - a wet electrostatic precipitator *or*
 - any other technique achieving the same removal efficiency;A residual particulate matter concentration of < 10 mg/Nm³ is possible.
7. Cast house de-dusting (tap-holes, runners, skimmers, torpedo ladle charging points);
Emissions should be minimised by covering the runners and evacuation of the mentioned emission sources and purification by means of fabric filtration or electrostatic precipitation. Dust emission concentrations of 1-15 mg/Nm³ can be achieved. Regarding fugitive emissions 5-15 g dust/t pig iron can be achieved; thereby the capture efficiency of fumes is important.
Fume suppression using nitrogen (in specific circumstances, e.g. where the design of the casthouse allows and nitrogen is available).
8. Treatment of blast furnace gas scrubbing wastewater:
 - a. Reuse of scrubbing water as much as possible;
 - b. Coagulation/sedimentation of suspended solids (residual suspended solids < 20 mg/l can be achieved as an annual average, single daily values up to 50 mg/l may occur);
 - c. Hydrocyclonage of sludge with subsequent reuse of the coarse fraction when grain size distribution allows reasonable separation.

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9. Minimising slag treatment emissions and slag to landfill;
Slag treatment preferably by means of granulation where market conditions allow.
Condensation of fume if odour reduction is required.
Whenever pit slag is produced, forced cooling with water should be minimised or avoided where possible and where space restrictions allow.
 10. Minimising solid waste/by-products.
For solid wastes, the following techniques are considered BAT in descending order of priority:
 - a. Minimising solid waste generation
 - b. Effective utilisation (recycling or reuse) of solid wastes/by-products; especially recycling of coarse dust from BFGas treatment and dust from cast house de-dusting, complete reuse of slag (e.g. in the cement industry or for road construction)
 - c. Controlled disposal of unavoidable wastes/by-products (fine fraction of sludge from BFGas treatment, part of the rubble)

In principle the techniques listed as points 1 - 10 are applicable to both new and existing installations if the mentioned prerequisites are met and considering the preface.

7.5 Emerging techniques and future developments

Although the blast furnace route is the main process for iron production, several other production routes for pig iron are currently being developed and one technique is already applied commercially (Corex). These so-called "smelting reduction" techniques invariably use coal instead of coke as the main fuel. Some of the new techniques also replace pellets and sinter by pulverised iron ore. If these techniques prove to be reliable, high-quality and cheap primary iron production units, the look of primary iron production will change drastically. These alternative technologies are described in more detail below.

Nevertheless, the blast furnace is still the dominant pig iron production unit, with several hundred units in operation worldwide. The blast furnace has a long history, and modern blast furnaces are highly effective and energy efficient reactors. The injection of carbon at the tuyère level has given new impetus to blast furnace operational practice. As an example of future possibilities for blast furnace operation, the development of oxy-coal techniques can be mentioned.

High oxy-coal techniques

Description: Coal injection tends to decrease raceway temperatures and, unless measures are taken to counteract this effect, the efficiency of combustion and utilisation decreases at higher coal injection rates. To allow the maintenance of suitable raceway conditions to promote effective coal utilisation and coke replacement, it is necessary to employ either progressively higher blast temperatures or higher levels of oxygen enrichment of the blast as coal injection rates are increased.

Conventional blast heating in regenerative stoves is limited by engineering considerations to temperatures of about 1200°C, which alone would allow coal injection up to 150 kg/t pig iron.

Two methods can be used to allow increased coal injection rates:

1. Higher blast temperatures using electrically powered plasma blast superheating.
This is economically feasible only in locations where cheap electricity is available. Tests have been carried out in France, where cheap electricity is available due to nuclear power generation.
2. Oxygen addition to the blast.
Oxygen from an air separation plant can be used to enrich the blast before the hot blast stoves or can be injected at the tuyère level, together with the coal (oxy-coal injection). Oxygen enrichment before the hot stove could lead to engineering and safety problems and thus it would be preferable to inject the oxygen at the tuyère level.

Main achievements: Theoretically, coal injection can be as high as 400 kg/t pig iron when using a highly enriched blast. In this case, the blast should be enriched with at least 30% oxygen (51% in blast). Coke consumption can be significantly reduced with respect to current consumption levels.

Status: Pilot plant operations and tests at commercial blast furnaces have been carried out. The principle is already proven. Experiments aim at the highest possible input at stable blast furnace operation and sufficient coal gasification.

Reference literature: [Campbell, 1992; Ponghis, 1993].

Reduction of CO emissions from hot stoves having an internal combustion chamber

Description: In 7.1.2 the two basic designs of hot stoves (with internal or with external combustion chamber) are described. In case of internal combustion chamber high emissions of CO occur (see 7.2.2.1.1) as a result of leaks from cracks in the refractory mass. This leakage seems to be inevitable and leads to emissions of unburned gas. It is, however, possible to reduce leakage by inserting steel sheets of an appropriate grade into the refractory wall during relining.

Main achieved emission levels: The impact of cracks (high CO emissions) can be significantly reduced. Measurement before and after inserting steel sheets are yet not available.

Status: This measure has already been introduced at one integrated steelworks in the EU 15.

Slag heat recovery

Description: Liquid slag from the blast furnace contains a large amount of sensible heat. Its temperature is approximately 1450°C and around 250-300 kg/t pig iron is produced in modern blast furnaces. None of the commercially applied systems in the world utilise this potential energy source. This is mainly caused by the technical difficulties to develop a safe, reliable and energy efficient system, which in addition does not influence the slag quality.

Energy savings: Estimated savings are approximately 0.35 GJ/t pig iron

Status: Tests have been carried out, but slag heat recovery is probably not likely to be operated on commercial scale in the near future.

Reference literature: [InfoMil, 1997]

8 BASIC OXYGEN STEELMAKING AND CASTING

The replacement of air with oxygen in steelmaking was originally suggested by Henry Bessemer. Since 1950, oxygen has been used in steelmaking irrespective of the specific production method. A prerequisite for the cost-effective use of the basic oxygen furnace (BOF) process on an industrial scale was the availability of the required tonnage of technically pure oxygen, as well as the water-cooled lance technology necessary for introducing the oxygen into the converter.

The first production-scale BOF (also called an oxygen converter) was erected at Linz in 1953.

The BOF process and the electric arc furnace (EAF) have since replaced less energy efficient existing steel making processes such as the Thomas process and open-hearth process (Bessemer, Siemens-Martin). In the EU the last Siemens-Martin ovens were taken out of operation at the end of 1993. The BOF process and the electric arc furnace are the only processes being used to produce steel in the EU. The BOF process accounts for two thirds of production and the EAF process for the remaining third (EU 15 in 1996 -see also Figure 1.2).

Figure 8.1 shows a basic oxygen furnace at the moment of charging hot metal.



Figure 8.1 : Basic oxygen furnace at the moment of charging hot metal

8.1 Applied processes and techniques

The objective in oxygen steelmaking is to burn (i.e., oxidise) the undesirable impurities contained in the metallic feedstock. The main elements thus converted into oxides are carbon, silicon, manganese, phosphorus, and sulphur. The purpose of this oxidation process, therefore, is:

- to reduce the carbon content to a specified level (from approximately 4% to less than 1 %, but often lower)
- to adjust the contents of desirable foreign elements
- to remove undesirable impurities to the greatest possible extent

The production of steel by the BOF process is a discontinuous process which involves the following steps:

- transfer and storage of hot metal
- pre-treatment of hot metal (desulphurisation)
- oxidation in the BOF (decarburisation and oxidation of impurities)
- secondary metallurgical treatment
- casting (continuous or/and ingot)

The individual steps and their associated emissions are summarised in Figure 8.2.

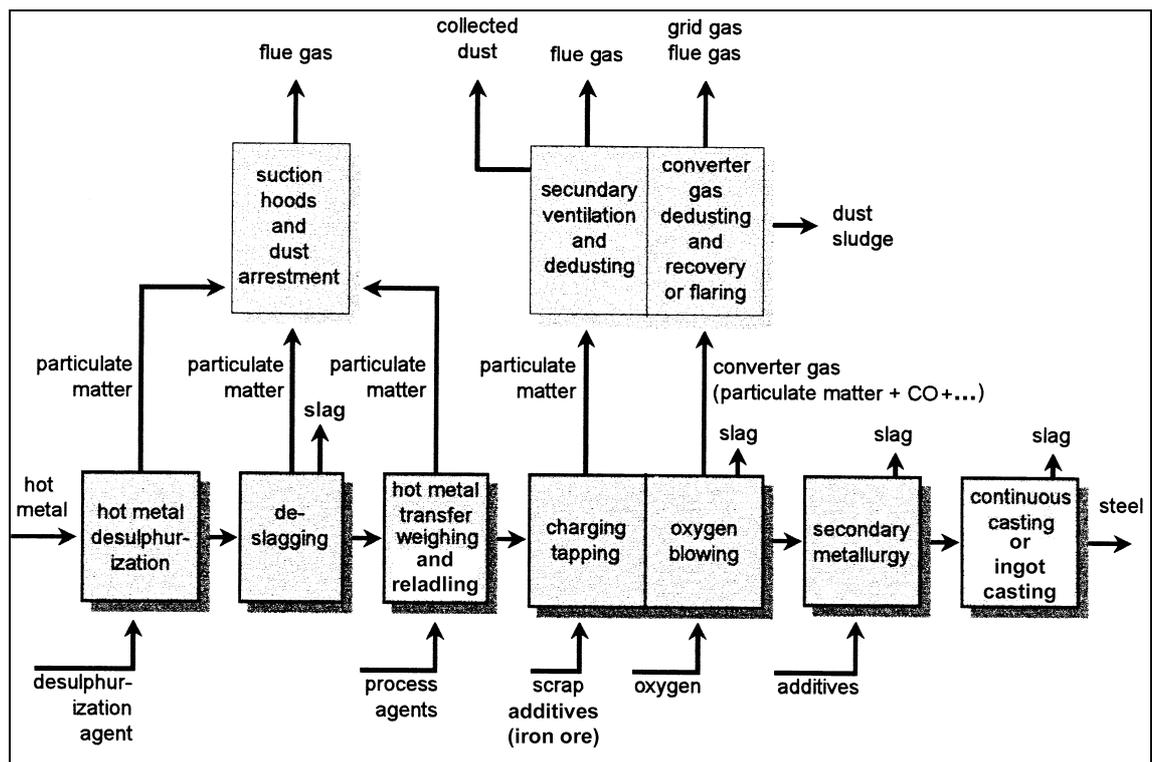


Figure 8.2 : : The sequence of oxygen steelmaking indicating the individual emission sources

8.1.1 Transfer and storage of hot metal

Hot metal is supplied from the blast furnace to the steelmaking plant using transfer cars or torpedo ladles. The ladles are lined with corundum, mullite, bauxite or dolomite brick with an additional insulating interlayer between the steel and the refractory material. The torpedo ladle, a form of hot metal mixer travelling on rails, has become the predominant system. The mixing vessel is supported on either side and can be rotated to discharge its contents. Torpedo ladles are commonly designed with capacities between 100 and 300 tonnes, with the largest units holding up to 400 tonnes. The design of the torpedo ladle minimises heat loss. The fact that the torpedo ladle doubles as a hot metal mixer eliminates the need for a separate hot metal storage system.

The service life of torpedo ladles in normal use varies between 150000 and 400000 tonnes. Desulphurisation in the torpedo ladle shortens its potential utilisation cycle between re-linings and requires special care in the selection of the refractory materials.

While transporting in an open-top ladle, hot metal is in some cases stored in mixers. These are rotatable horizontal steel containers lined with refractory brick. They serve to compensate for production fluctuations of the blast furnace and the steelmaking plant, to equalise the chemical composition of individual blast furnace heats, and to ensure homogeneous temperatures. Modern hot metal mixers have capacities of up to 2000 tonnes.

8.1.2 Pre-treatment of hot metal

Classic hot metal pre-treatment comprises the following steps:

- Desulphurisation
- Dephosphorisation
- Desiliconisation

In Europe only the desulphurisation step is commonly used when preparing hot metal for the BOF process. Dephosphorising and desiliconising the feedstock involves costly and sophisticated process technology which appears unwarranted in the foreseeable future, given current requirements.

Improved blast furnace metallurgy and a reduction in the amount of sulphur introduced via the reducing agents have resulted in lower hot metal sulphur levels. Today specified sulphur concentrations (between 0.001 and 0.020%) for charging in the converter are commonly adjusted in a hot metal desulphurisation facility located away from the blast furnace. External desulphurisation also implies benefits in terms of environmental protection. With an upstream blast-furnace process, these generally include reduced consumption of coke and sinter, decreased output of hot metal and steelmaking slag, improved quality of the metallurgical slag, prolonged service life of the refractory linings, and reduced oxygen consumption.

Known desulphurisation agents include calcium carbide, caustic soda, soda ash, lime and magnesium impregnated materials. Soda desulphurisation is a straightforward process due to the low melting temperature and the resulting easy miscibility of the product with the hot metal. Its drawbacks include the low specificity and the need to find a disposal route for the soda slag produced. The sulphur content of these slags varies between 1 and 15% and their Na_2O content between 5 and 40%, depending on the process employed. Recycling soda slag within the plant is an impractical proposition because of its high alkalinity. A cost-efficient alternative recycling method has not been devised to date. When dumped, sodium sulphite decomposes into sodium hydroxide solution and hydrogen sulphide due to the presence of water. The use of soda ash is restricted. Small amounts of soda ash may be utilised to a limited extent during the skimming process.

The most widespread hot metal desulphurisation method used in Europe today is that based on calcium carbide, which has superseded the previous soda process for waste disposal and air quality management reasons. The use of a mixture of calcium carbide, magnesium and lime allows the hot metal to be desulphurised to final levels below 0.001%, regardless of the initial sulphur content. Disadvantages lie in the fairly low exhaustion of the desulphurising agent and the need for intense mixing of the desulphurising agent with the hot metal. One specific benefit is that the process gives rise to a crumbly slag which can be easily removed. The use of magnesium in addition to calcium carbide is approximately just as common as the use of calcium carbide alone. Other desulphurising agents include lime powder, lime in conjunction with natural gas, and magnesium.

The desulphurisation process is carried out by a number of different methods and systems. In the more common variants, desulphurisation takes place

- in the blast furnace launder
- in the pouring stream
- in the transfer ladle, or
- in purpose designed metallurgical vessels.

Known desulphurisation equipment includes the immersion lance, the siphon ladle, rotating and oscillating vessels, and agitating equipment for use in the ladle. Calcium carbide is most commonly used in conjunction with the immersion lance and the stirring method. Magnesium is added in powdered form in a carrier gas via an immersed lance. The desulphurisation process is performed at separate treatment stations.

An example of practice is as follows: A desulphurising agent is blown through a lance into the hot metal with the aid of nitrogen. The sulphur is bound in the slag, which floats to the top of the hot metal. The slag is then removed in the slag separation unit and the liquid iron is fed from the ladle into the weighing pit. If necessary, process agents are added in these pits. In some cases, a second slag removal is performed here using slag scrapers. After weighing, the pig iron is charged into the converter.

8.1.3 Oxidation in the BOF

In order to meet the objectives mentioned above, undesired impurities are oxidised with subsequent removal with the off-gas or slag. Figure 8.3 lists main oxidation processes occurring in the BOF furnace. Undesirable impurities are removed with the off-gas or the liquid slag. The energy required to raise the temperature and melt the input materials is supplied by the exothermic oxidation reactions, so that no additional heat input is required, on the one hand, and scrap or ore have to be added to balance heat on the other hand. In some BOF and combined blowing processes gaseous hydrocarbons (e.g. natural gas), is injected as tuyère coolant (see Figure 8.5).

Carbon elimination	$[C] + [O]$	$\leftrightarrow CO$ (off-gas)
	$[CO] + [O]$	$\leftrightarrow CO_2$ (off-gas)
Oxidation of accompanying and tramp elements		
- Desiliconisation	$[Si] + 2[O] + 2(CaO)$	$\leftrightarrow (2CaO \cdot SiO_2)$
- Manganese reaction	$[Mn] + [O]$	$\leftrightarrow (MnO)$
- Dephosphorisation	$2[P] + 5 [O] + 3 [CaO]$	$\leftrightarrow (3CaO \cdot P_2O_5)$
- Desulphurisation	$[S] + [CaO]$	$\leftrightarrow (CaS) + [O]$
Deoxidation		
Removal of residual oxygen through ferro-silicon	$[Si] + 2[O]$	$\leftrightarrow (SiO_2)$
Aluminium	$2[Al] + 3[O]$	$\leftrightarrow (Al_2O_3)$
Notes: [] solute in the iron () contained in the slag		

Figure 8.3 : Chemical reactions taking place during the oxidation process

The operation of a basic oxygen furnace (BOF) is semi-continuous. A complete cycle consists of the following phases: charging scrap and molten pig iron, oxygen blowing, sampling and temperature recording and tapping. In a modern steelworks, approximately 300 tonnes of steel are produced in a 30-40 minute cycle. During the process a number of additives are used to adapt the steel quality and to form slag. During charging and tapping, the converter is tilted. During oxygen blowing, the converter is placed in the upright position.

There are several types of reactors used for the basic oxygen steel making process. The most commonly used type is the LD converter (Linz-Donawitz) applied for pig iron with low phosphorous content; in the case of high phosphorous content a modified process is used (LD/AC process = Linz-Donawitz/Arbed-CRM-process). The converter is a pear-shaped, refractory-lined reactor into which a water-cooled oxygen lance is lowered. Through this lance pure oxygen (>99%) from an air separation plant is blown onto the liquid pig iron (see Figure 8.4).

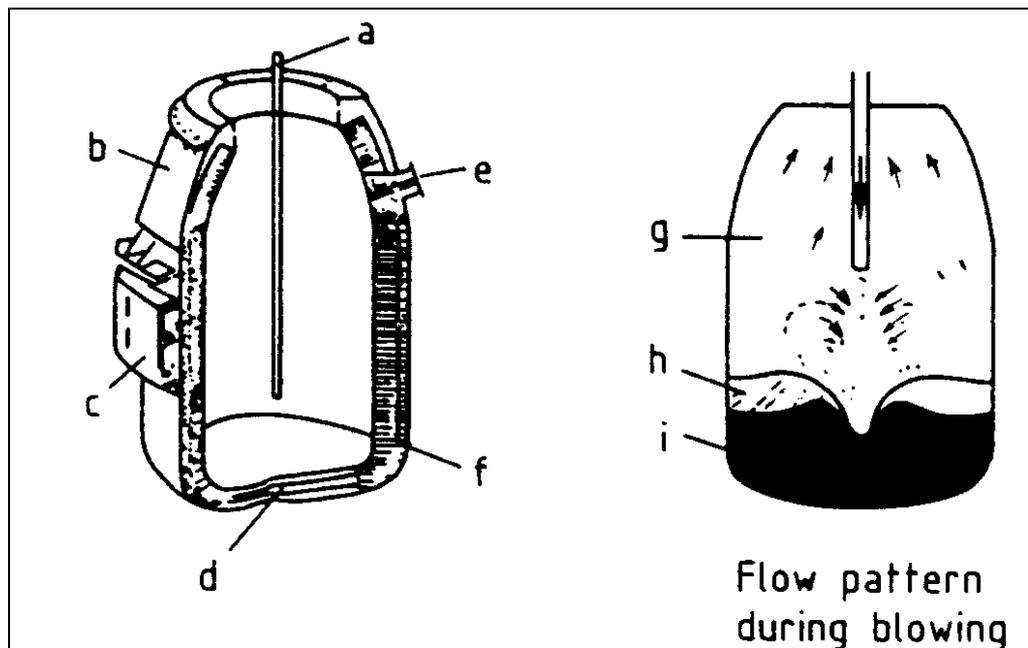


Figure 8.4 : Top-blown oxygen converter – [Ullmann's, 1994]

a – oxygen lance; b – converter top; c – supporting ring; d - converter bottom; e – tapping hole; f – refractory lining; g – gas space; h – slag layer; i – molten metal

Other types of steel making reactors are the OBM (Oxygen-Bottom-Maxhuette process) or Q-BOP process and the LWS process (Loire-Wendel-Sprunch process). These processes differ from the LD-converter in that instead of top blowing oxygen through a retractable lance, oxygen and fluxes are blown through submerged tuyères in the furnace bottom (Figure 8.5) [EC BOF, 1995].

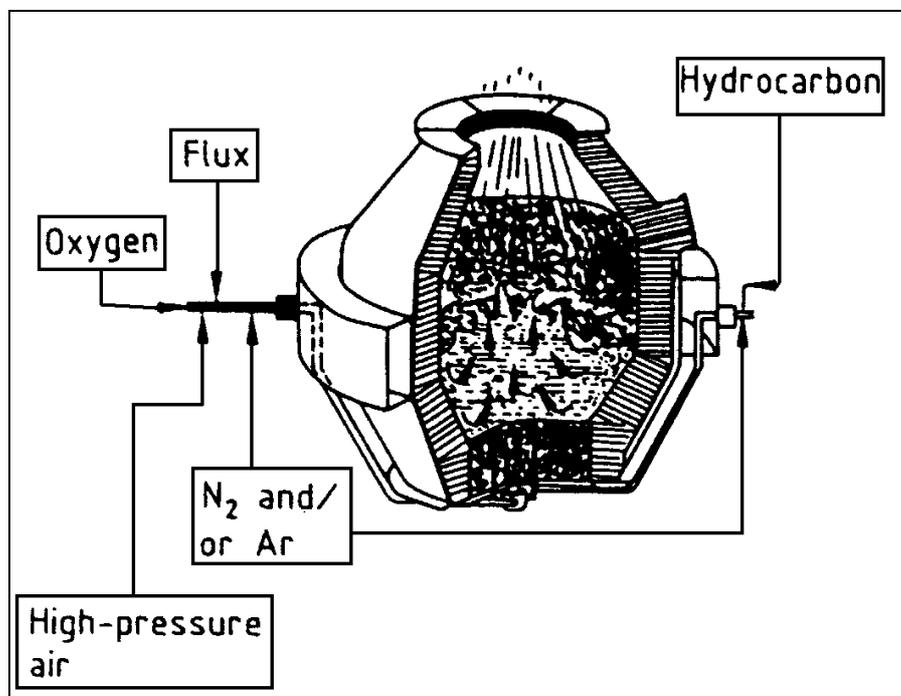


Figure 8.5 : Section through an OBM converter – [Ullmann's, 1994]

In these converters oxygen is injected from the bottom, through tuyères cooled by hydrocarbons blown into the melt. Combined blowing techniques have also been developed. Where necessary, the process can be enhanced by “bottom stirring” with argon gas (Ar) or nitrogen (N₂) through porous bricks in the bottom lining in certain phases of the process. Alternatively bottom tuyères may be used to inject pure oxygen or other gases during the blowing process. This produces a more intensive circulation of the molten steel and improves the reaction between the oxygen and the molten metal. The most frequent types are the LBE process (Lance-Bubbling-Equilibrium process) and the TBM process (Thyssen-Blowing-Metallurgy process) (Figure 8.6). A special version is the KMS process (Klöckner-Maxhütte-Steel Making process) in which oxygen is injected from the bottom together with lime and coal.

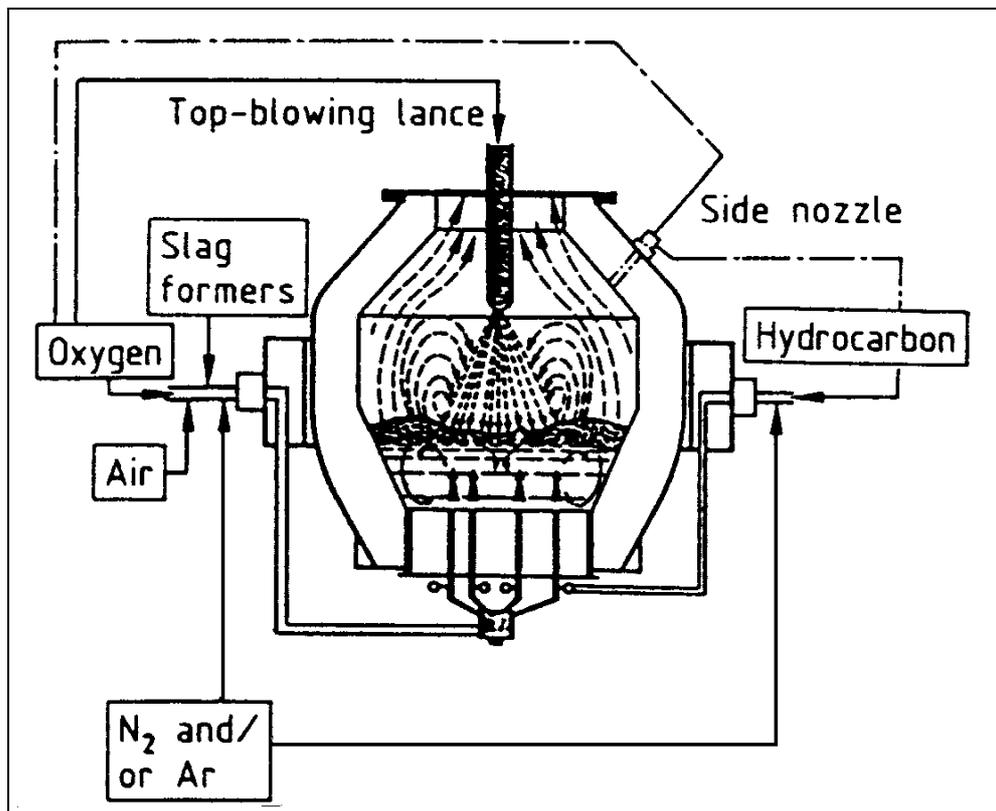


Figure 8.6 : Combined blowing technique with top blowing lance or side tuyère – [Ullmann's, 1994]

Table 8.1 gives an overview of numbers of oxygen converters in use in the EU, arranged by process types. Obviously the LD-process is the dominant one.

Process		Rated capacity [1000 t/a]
LD	17	12400
LD with bottom stirring	44	64960
LBE	22	27550
OBM	5	2780
K-OBM	1	2200
EOF	1	400
LWS	2	2900
Total	92	113210

Table 8.1 : Types, numbers and rated capacities of basic oxygen furnaces (oxygen converters) in operation in the EU

The amount of oxygen consumed depends on the composition of the hot metal (i.e. C, Si, P content).

Progress of the steel making process is measured by taking samples of the molten metal. In modern plants sampling is performed without interrupting the oxygen blowing, using a substance. The same result is obtained by standardising the process procedures and/or by using adequate dynamic modelling and monitoring. Those practices sustain quality, productivity and reduce the fume emissions during former converter tilting. When the steel quality meets the demands, the oxygen blowing is stopped and the crude steel is tapped from the converter into a ladle. The molten steel is then transported, after secondary metallurgy, to the casting machine.

The oxidising reactions are exothermic thus increasing the temperature of the molten iron. Scrap, iron ore or other coolants are added to cool down the reaction and maintain the temperature at approximately 1600-1650°C. Usually, approximately 10-20% of the converter charge is scrap, but values up to 40% are sometimes used. The amount of scrap charged depends on the pre-treatment given to the pig iron and the required liquid steel tapping temperature [UBA Comments, 1997]. Variations in the market value of scrap and the required steel specifications also have an influence.

The gases produced during oxygen blowing (converter gas) contain large amounts of carbon monoxide. In many steelmaking plants, measures have been taken to recover the converter gas and use it as an energy source. Both "open combustion" and "suppressed combustion" systems are in use. Open combustion systems introduce air into the converter flue gas duct, thus combusting the carbon monoxide. The heat generated is later recovered in a waste heat boiler. In suppressed combustion, a skirt is lowered over the converter mouth during oxygen blowing. Thus, ambient oxygen can not enter the flue gas duct and the combustion of carbon monoxide is prevented. The CO-rich flue gas can be collected, cleaned and stored for subsequent use as fuel. A main advantage of suppressed combustion is the smaller flue gas flow since no combustion occurs and no additional air-nitrogen is introduced. This results in higher productivity since oxygen blowing speed can be increased.

The oxygen steelmaking process also generates considerable quantities of particulate matter, during charging of scrap and hot metal, blowing and during tapping of slag and liquid steel. All steelmaking shops in the EU have taken measures to reduce particulate matter emissions.

During the steelmaking process, slag is formed. Slag control is intended to effectively reduce the amount of undesirable substances contained in the hot metal and to generate slag of high quality that will be suitable for subsequent processing and usage.

Usually, the slag is cooled and crushed, after which metallic iron is recovered by magnetic separation. The technical properties of the slag make it suitable for many kinds of application in civil and hydraulic engineering [Geiseler, 1991; Geiseler, 1992]

Because of its structure, LD slag has high abrasion resistance and is therefore often used for road construction [Köller, 1995]. It is also put to other uses or disposed of in landfills.

8.1.4 Secondary metallurgy

The oxidizing process in the converter is usually followed by post-treatment comprising a number of diverse metallurgical operations. Referred to as "secondary metallurgy", this treatment was developed in response to ever increasing quality requirements and has led to substantial productivity increases by shifting the burden of metallurgical refining processes away from the converter. The main objectives of secondary metallurgy are:

- mixing and homogenising
- adjustment of chemical compositions to close analysis tolerances
- temperature adjustment in time for the downstream casting process
- deoxidation
- removal of undesirable gases such as hydrogen and nitrogen
- improvement of the oxidic purity by separating non-metallic inclusions.

An overview of secondary metallurgy operations is given in Figure 8.7. These steps are performed in the ladle or ladle furnace, in a vacuum system, or in purpose designed furnaces.

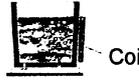
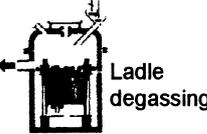
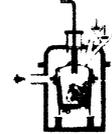
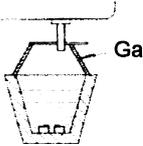
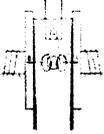
Avoiding slag-overflow			
Mixing/Homogenizing /Additives	<p>Gas blowing</p>  <p>Bottom porous plug</p>	<p>Lance</p> 	<p>Electromagnetic</p>  <p>Coil</p>
Injection of solid substances	<p>Alloying additions</p>  <p>Gas</p>	<p>Powder / Gas</p> 	<p>Wire</p>  <p>Gas</p>
Vacuum-Treatment	<p>Ladle degassing</p> 	<p>RH / DH</p> 	<p>VOD</p> 
Heating	<p>Ladle furnace</p> 	<p>VAD</p> 	
Pouring stream protection ladle/tundish	<p>Gas</p> 	<p>Gas</p> 	<p>Gas</p> 
Electromagnetic stirring linked with continuous casting	<p>MS</p> 	<p>EMBR</p> 	<p>SS / FS</p> 

Figure 8.7 : Overview of secondary metallurgy operations

An important step in secondary metallurgy is vacuum treatment. This mainly serves to remove gaseous hydrogen, oxygen, nitrogen or residual carbon concentrations from the steel at a vacuum of up to 50 Pa. The purpose of this operation is decarburisation and to free the molten steel from gases dissolved in the heat during the blowing cycle. Thus, the mass content of oxygen and nitrogen can to be lowered to 0.0002% and 0.005%, respectively, by reducing pressure to 10 mbar. Today, vacuum treatment operations include the precision decarburization

and deoxidation of unalloyed steels, the decarburisation of chromium-alloyed grades, the removal of sulphur and inclusions, as well as various alloying, homogenisation, temperature management and reoxidation prevention steps. Vacuum metallurgy has given us steels of improved purity, lower gas content, and tighter alloying tolerances.

The following vacuum treatment methods are used:

- ladle stand or tank degassing and
- recirculating degassing

Of these, recirculating degassing is the most common process today, although ladle stand degassing is coming back into widespread use.

In ladle degassing the ladle containing the oxidised metal is placed in a vacuumised container. An input of additional energy ensures higher reaction speeds and reduces the final concentration of undesirable bath constituents. This agitation may be accomplished by injecting argon through one or more porous plugs in the ladle bottom, by homogenising the melt via a lance, or by an inductive stirring process.

Depending on individual requirements, it is possible to add high-purity scrap (cooling scrap) to adjust the temperature of melts or to introduce alloying agents to obtain a precision adjustment of the steel composition. Alloying agents are added to the melt in solid form, or enclosed in a hollow wire which is unwound from a reel system, or else by powder injection via lances. Before the end of the stirring cycle, the oxygen level may be determined with the aid of a special probe and adjusted by adding deoxidants. Throughout the stirring process, the dust/gas mixture rising up from the ladle is drawn off by a movable fume hood.

In recirculating degassing, the molten metal is vacuum treated continuously or in separate portions. Depending on the process design, a distinction is made between vacuum lift degassing (DH) and vacuum recirculating degassing (RH).

The vacuum treatment is a key consideration in plant water management, since the vacuum is generated by a steam and water based process. Smaller systems use water ring pumps for this purpose, whereas larger facilities rely on multi-stage steam jet vacuum pumps or on a combination of water ring pumps and multi-stage jet vacuum pumps. Gas from the vacuum treatment chamber is drawn into the water by an under-pressure generated on the water jet pump principle. The water flow rates necessary for this evacuation process are considerable and may reach approx. 5 m³/t LS.

8.1.5 Casting

Once the final steel quality has been achieved the steel is conveyed in a casting ladle to the casting machines. Until a few years ago, the standard method was to pour the molten steel into permanent moulds (permanent mould or ingot casting) by a discontinuous process. Today, the method of choice is continuous casting, whereby the steel is cast in a continuous strand.

8.1.5.1 Continuous casting

Figure 8.8 shows the schematic diagram of a continuous casting plant.

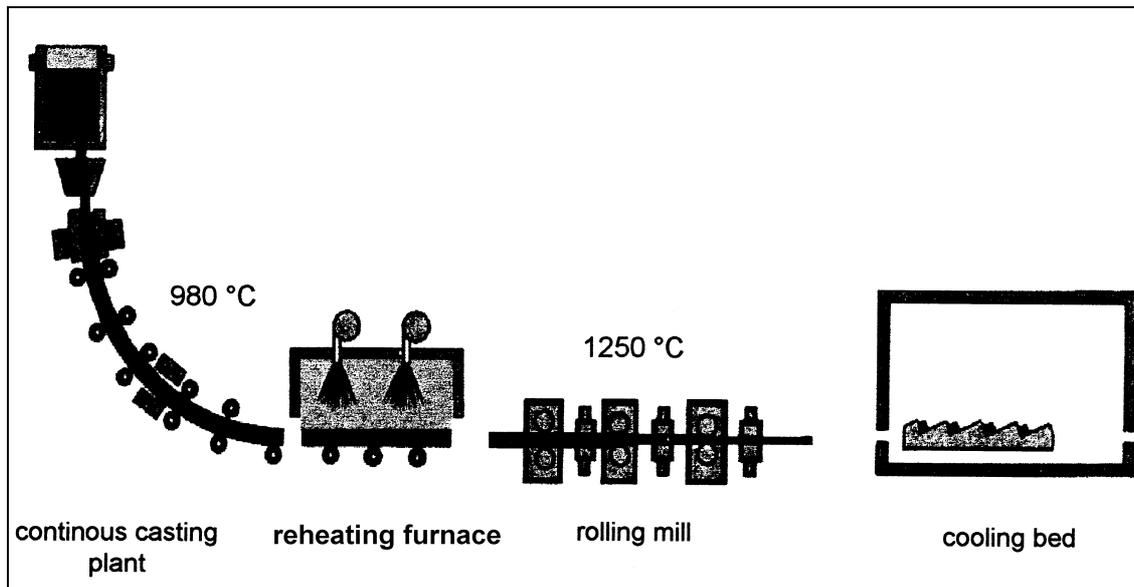


Figure 8.8 : Scheme of a continuous casting plant with soaking furnace and rolling mill with hot charging

Continuous casting offers several important benefits:

- energy savings, reduced emissions and reduced water utilisation due to the elimination of slabbing mills and billet mills
- improved working conditions
- high yield rates, in excess of 95%
- high productivity

Since continuous casting was first introduced on an industrial scale in the late '60s, its share in overall EU steel output has risen to approx. 95.4%. Worldwide, about 75% of all steel is now cast by the continuous method. Apart from the traditional ingot casting process, continuous casters have also replaced the blooming-slabbing and semifinishing operations of conventional hot-rolling mills. Today, almost all steel grades for rolled products can be handled by the continuous casting route, since the necessary preconditions such as deoxidation and degassing are provided by modern secondary metallurgy.

There exist various types of continuous casters referred to as vertical type, bending-and-straightening type, bow type and oval bow type machines, depending on their configuration.

The liquid steel is poured from the converter into a ladle, which transports the steel after secondary metallurgy to the so called tundish of the continuous casting machine (CCM). This is an intermediate ladle with controllable outlet. The ladles are preheated prior to accepting a liquid steel charge in order to avoid temperature stratification in the tundish.

When the liquid steel has reached the desired temperature, it is poured into the tundish. From here it passes to a short water-cooled copper mould in which no air is present and which performs oscillating up and down movements to prevent the steel from sticking. The mould gives the metal the desired shape. When the metal leaves the casting mould, a "skin" of solidified steel has formed and a large number of trundles (the "pinch-roll") guide the cast steel with a gentle curve toward a horizontal position. Here, the endless casting is cut in pieces with a torch cutter. Slabs, blooms and billets are cast in this way.

In the case of non-self-supporting sections the red-hot strand, with its solidified surface zone, moves through a number of driven and undriven roll pairs which support its shell against ferrostatic pressure. As the core is still liquid it is sprayed carefully with water and cooled until fully solidified (secondary cooling). This process prevents cracks in the strand surface zone, which is still fairly thin, and also protects the rolls from overheating. The supporting, conveying and drive elements are commonly rolls equipped with interior and exterior cooling. In the secondary cooling zone, interior roll cooling becomes dispensable as soon as the temperature is reduced sufficiently by the water spray. A number of bearings are connected to an automatic grease lubricating system. Once the strand has fully solidified, it can be cut to size by cutting torches moving with the strand or by shears. The rapid cooling process gives the steel a uniform solidification microstructure with favourable technological properties. The solidification microstructure of the strand can be influenced by downstream air or water cooling.

The shape of the strand is determined by the mould geometry. Current mould types include rectangular, square, round or polygonal sections. For the production of steel shapes, it is possible to use moulds resembling the approximate cross section of the intended product. Typical strand dimensions in continuous casting vary between 80 x 80 mm and about 310 x 310 mm, 600 mm (round) in billet and 450 x 650 mm in bloom systems, while slab casters produce sizes of up to 350 mm in thickness and up to 2720 mm in width. Billet casters can handle several (currently up to eight) strands at the same time, while the number of strands in slab casting is limited to two.

8.1.5.2 Ingot casting

In ingot casting, the liquid steel is cast into casting moulds. Depending on the desired surface quality, degassing agents (such as NaF) can be added during casting in the ingot mould. After cooling the ingots are taken out of the casting mould and transported to the rolling mills. Subsequently, after pre-heating the ingots are rolled into slabs, blooms or billets. In many places, ingot casting has been replaced by continuous casting. It is expected that ingot casting will eventually be almost completely replaced by continuous casting except in the case of those products which require ingot casting to achieve the necessary quality, such as producing heavy weights for forging.

8.2 Present emission and consumption level

8.2.1 Mass stream overview and input/output data

Figure 8.9 provides an overview of the input and output mass streams of basic oxygen steelmaking. This overview may be used for the collection of data from a single oxygen steelmaking plant.

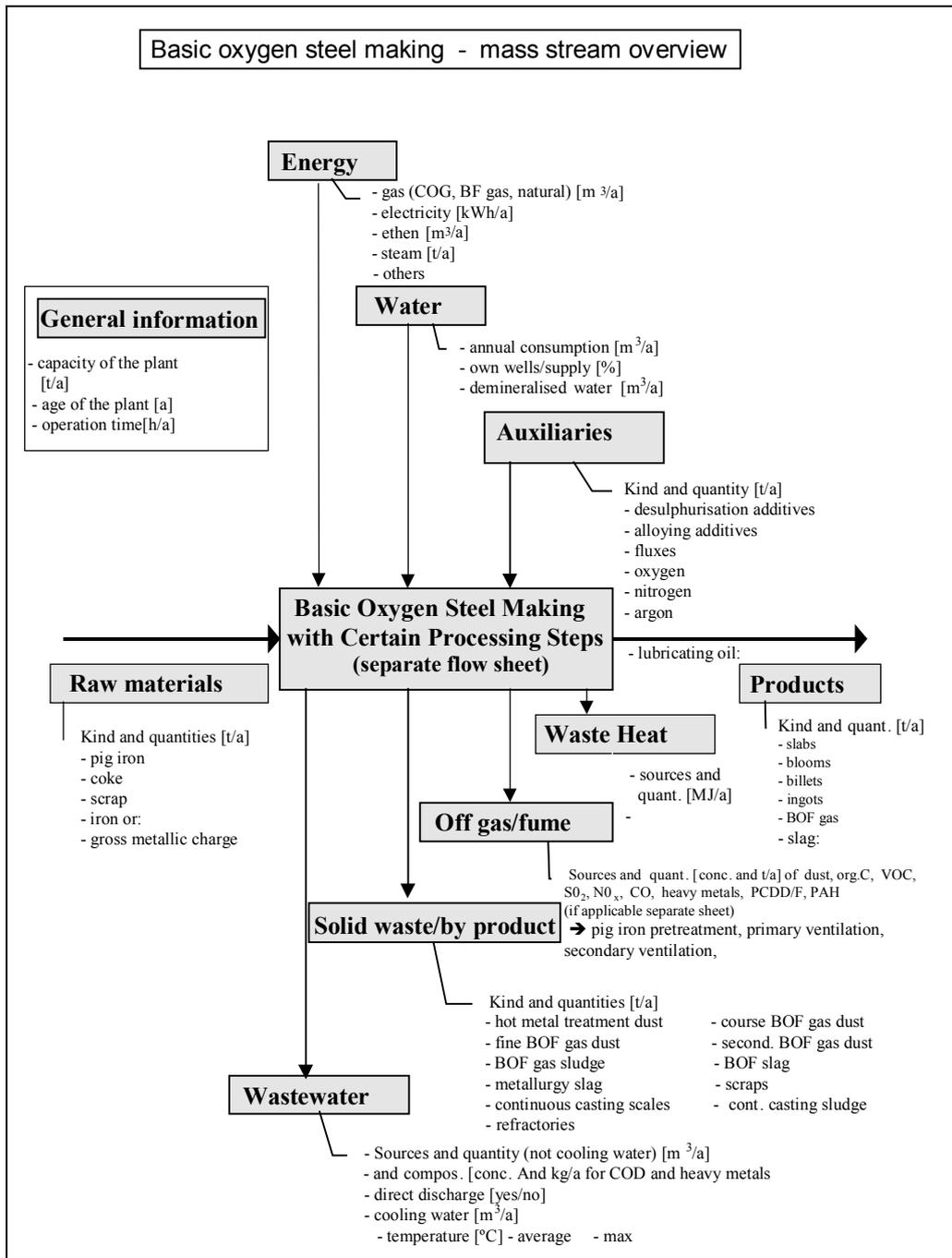


Figure 8.9 : Mass stream overview of a basic oxygen steel making plant

Figure 8.10 presents the general process layout of basic oxygen steelmaking and visualises the input and output mass streams.

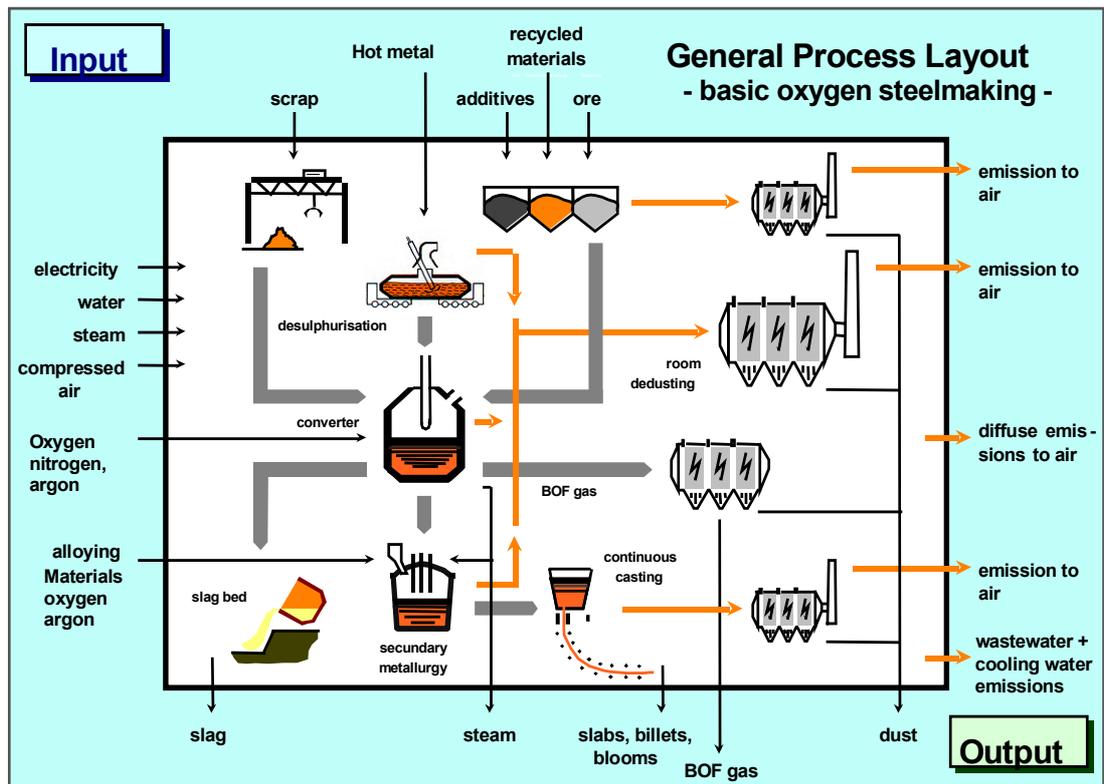


Figure 8.10 : General process layout of basic oxygen steelmaking indicating the individual operations and the input and output mass streams

Subsequently specific input factors and specific emission factors can be calculated. Values for these factors are shown in Table 8.2 from four basic oxygen steelmaking plants located in four different EU Member States.

Input			Output		
Raw materials			Products ^{*3}	kg/t LS	1000.0
pig iron ^{*1}	kg/t LS	820 – 980	Slabs }		
scrap	kg/t LS	170 – 255	Blooms }		
iron ore	kg/t LS	7 – 20	Billets }		
other Fe-material	kg/t LS	7 – 10	Ingots }		
coke	kg/t LS	0.02 – 0.48			
lime	kg/t LS	30 – 55	Energy		
dolomite	kg/t LS	1.5 – 4	BOF gas ^{*4}	MJ/t LS	(0)-650 – 840
alloys ^{*2}	kg/t LS	3 – 9	Steam ^{*5}	MJ/t LS	(0)-20 – 270
			Gas. Emissions		
Oxygen	m ³ /t LS	45 – 55	Dust	g/t LS	15 – 80
			Cr ^{*6}	g/t LS	0.01 – 0.36
			Cu ^{*6}	g/t LS	0.01 – 0.04
Energy			Pb ^{*6}	g/t LS	0.13 – 0.9
natural gas	MJ/t LS	20 – 55	Mn ^{*6}	g/t LS	<0.01 – 1.2
electricity	MJ/t LS	38 – 120	NO _x	g/t LS	5 – 20
			CO	g/t LS	1500 – 7960
			CO ₂ ^{*7}	kg/t LS	11.2 – 140
Steam	MJ/t LS	30 – 140	PAH ^{*8}	mg/t LS	0.08 - 0.16
			PCDD/F	µg I-TEQ/t LS	<0.001 – 0.06
Compressed air	Nm ³ /t LS	4 – 18			
			Residues/ By-products		
			Desulphur. slag	kg/t LS	2.2 – 19.2
Water	m ³ /t LS	0.4 – 5	BOF-slag	kg/t LS	85 – 110
			Slag from second. Metallurgy	kg/t LS	2 – 16
			Spittings	kg/t LS	4 – 5
			Dusts	kg/t LS	1.5 – 7
			Slag from contin. Casting	kg/t LS	4 – 5
			Mill scale	kg/t LS	1.2 – 6
			Rubble	kg/t LS	0.8 - 5
			Wastewater	m ³ /t LS	?

Legend: LS = liquid steel (crude steel)

^{*1} distinction can be drawn between high phosphorus (1.5-2.2% P) and low phosphorus hot metal (0.08-0.25% P)

^{*2} important alloying additions are: Fe-Ti, Fe-W, Fe-Ni, Fe-V, Fe-Si and Fe-Mo

^{*3} sum of products (slabs, blooms, billets or ingots)

^{*4} zero in case of non-recovery of BOF gas

^{*5} high value in case of non-suppressed combustion and heat recovery from flue gases in form of steam; zero in case of quantitative recovery of BOF gas without any heat recovery (no steam generation)

^{*6} higher value in case of less sufficient secondary de-dusting

^{*7} high value in case of partial to full combustion of the BOF gas

^{*8} PAH as Borneff 6; data available from two plants only

Table 8.2 Input/output-data from four existing basic oxygen steelmaking plants in four different EU Member States;

Information about the determination of the data like sampling methods, analysis methods, time intervals, computation methods and reference conditions is not available. Detailed data from other plants concerned are not available. Data refer to 1996. Emission data reflect the situation after abatement.

Table 8.3 complements Table 8.2 by presenting dust emission factors to air (after abatement) for the main individual operations/sources of oxygen steelmaking.

	Operation/emission source	dust [g/t LS]	
		n/r	x±s
converter gas	full combustion ^{*2}	13/10-200	66±78
	Suppressed combustion <u>without</u> fuel gas recovery ^{*3}	17/15-190	74±65
	Suppressed combustion <u>with</u> fuel gas recovery	13/1.5-16	8±4
other than converter emission sources	hot metal desulphurisation ^{*4}	26/1-7	
	hot metal handling (reladling) ^{*5}	1-17	
	BOF charging, tapping, deslagging, and sec. emissions during blowing ^{*5}	1-30	
	Secondary metallurgy ^{*6}	0.1-10	
	Continuous casting ^{*5}	0.5-4	
	sum of "other than BOF emissions" ^{*7}	20-80	

Legend: LS = liquid steel (crude steel); x±s = mean value and standard deviation (only calculated if enough data are available); n = number of data; r = range of data (min-max); n.r. = not relevant; n/a = not available

*1 data from [EC Study, 1996] except there is another indication

*2 three converters with 200 g dust/t LS, the residual with < 50 g dust/t LS

*3 three converters with 190 g dust/t LS, one with 140 g dust/t LS, the residual with < 100 g dust/t LS

*4 two plants using wet scrubbers or dry electric state precipitator exceed the given range (15-20 g dust/t LS); individual data are not available

*5 individual data are not available

*6 secondary metallurgy comprises operations in ladles, ladle furnaces, BOF and other equipment including charging and tapping; five BOF plants are reported to have dust emission factors between 15 and 20 g dust/t LS

*7 data from [EC Study, 1996] but corrected with data from [EUROFER BOF, 1997]

Table 8.3 : Emission factors for dust emissions to air (after abatement) for basic oxygen steelmaking plants^{*1}

8.2.2 Information about the individual emission mass streams and energy demand

The following emissions of off-gases, solid wastes/by-products and wastewater can be recognised in oxygen steelmaking.

8.2.2.1 Off-gas emissions

8.2.2.1.1 Primary off-gases from

- pig iron pre-treatment
- oxygen blowing and BOF gas (converter gas)
- ladles, ladle furnaces, converters and other equipment used in secondary metallurgy

8.2.2.1.2 Secondary off-gases from

- reladling and deslagging of hot metal
- BOF charging
- tapping of liquid steel and slag from BOF (converters) and ladles
- secondary metallurgy and tapping operations
- handling of additives
- continuous casting

8.2.2.2 Solid wastes/by-products

- desulphurisation slag
- BOF slag
- slag from secondary metallurgy
- spittings
- dusts from dry BOF gas treatment (when applied) or from treatment of other off gases (e.g. off gas from desulphurisation, off gas from secondary dedusting, off gas from secondary metallurgy)
- sludge from wet BOF gas treatment (when applied)
- slag from continuous casting
- mill scale from continuous casting
- rubble

8.2.2.3 Wastewater emissions from

- wet BOF gas treatment
- continuous casting

8.2.2.1 Off-gas emissions

8.2.2.1.1 Primary off-gases

8.2.2.1.1.1 Emissions from pig iron pre-treatment

In each of the three stages of the pig iron pre-treatment, emissions of particulate matter occur. The exhaust air generated in the desulphurisation process, subsequent slag separation and weighing is laden with up to 10000 mg/Nm³ or 1000 g/t steel of particulate matter [Koeller, 1995, EC BOF 1995]. Emissions can be abated by means of well-positioned suction hoods and subsequent treatment in a fabric filter or other appropriately efficient filter such as ESP.

8.2.2.1.2 Emissions during oxygen blowing and converter gas

During oxygen blowing, converter gas is released from the converter. This gas contains carbon monoxide (CO) and large amounts of particulate matter (mainly consisting of metal oxides, including heavy metals), relatively small amounts of sulphur oxides (SO₂) and nitrogen oxides (NO_x). In addition very small amounts of PCDD/F and PAH are emitted (see Table 8.2).

Generally, two systems can be used to recover energy from the converter gas:

- a) partial/full combustion;
- b) suppressed combustion.

The chosen type of recovery influences the emissions.

In the full (or open) combustion system, the process gas from the converter furnace is combusted in the flue gas duct. An opening between the converter furnace and the primary (or converter gas) ventilation allows the entrance of ambient air and thus partial or full combustion of the converter gas. In this case, the process gas contains approximately 15-20 kg particulate matter per tonne liquid steel and approximately 7 kg carbon monoxide per tonne liquid steel. Energy is recovered by using the sensible heat in a waste heat boiler. Note that open combustion systems have a large flow (2000-3000 Nm³/t LS) compared to suppressed combustion systems (50-100 Nm³/t LS). This is due to the introduction of air into the BOF gas (converter gas) duct.

When suppressed combustion is applied, a retractable water-cooled skirt is lowered over the mouth of the converter. In this way, carbon monoxide combustion in the flue gas duct is suppressed and the carbon monoxide can be recovered. The absence of nitrogen (given that air is excluded) means higher oxygen blowing speeds are possible and thus process time is reduced.

BOF gas (converter gas) must be classified as a lean gas in terms of its calorific value and Wobbe index, but belongs to the group of rich gases when considered in terms of its combustion properties (and particularly, its combustion temperature). The composition of the converter gas varies with the process used, the recovery method and, specifically, the oxygen volume (Table 8.4).

Parameter	Unit	Average value ¹⁾	Variation range
Composition			
CO	[vol.-%]	72.5	55 - 80
H ₂	[vol.-%]	3.3	2 - 10
CO ₂	[vol.-%]	16.2	10 - 18
N ₂ + Ar	[vol.-%]	8.0	8 - 26
Characteristic			
Density	[kg/Nm ³]	1.33	1.32 - 1.38
Gross calorific value	[kJ/Nm ³]	9515	
Net calorific value	[kJ/Nm ³]	9580	7100 - 10100
Theoretical flame temperature	[°C]	2079 ²⁾	
Specific air demand (wet)	[Nm ³ /Nm ³]	1.81	1.34 - 1.90
Specific off-gas quantity (wet)	[Nm ³ /Nm ³]	2.43	
1) combustion gas figures refer to conditions downstream of the gas-holder at a temperature of 15° C, a gauge pressure of 60 mbar, barometer level 1013 Pa and a humidity of 100% 2) with total humidity and 0° C			

Table 8.4 : Composition and characteristics of BOF gas

Figure 8.11 shows the CO content (a key parameter for the utilisation of BOF gas) as a function of oxidation time in case of suppressed combustion. On account of its low CO content the BOF gas generated during start and end of blowing (a few minutes each) is not collected but flared after de-dusting.

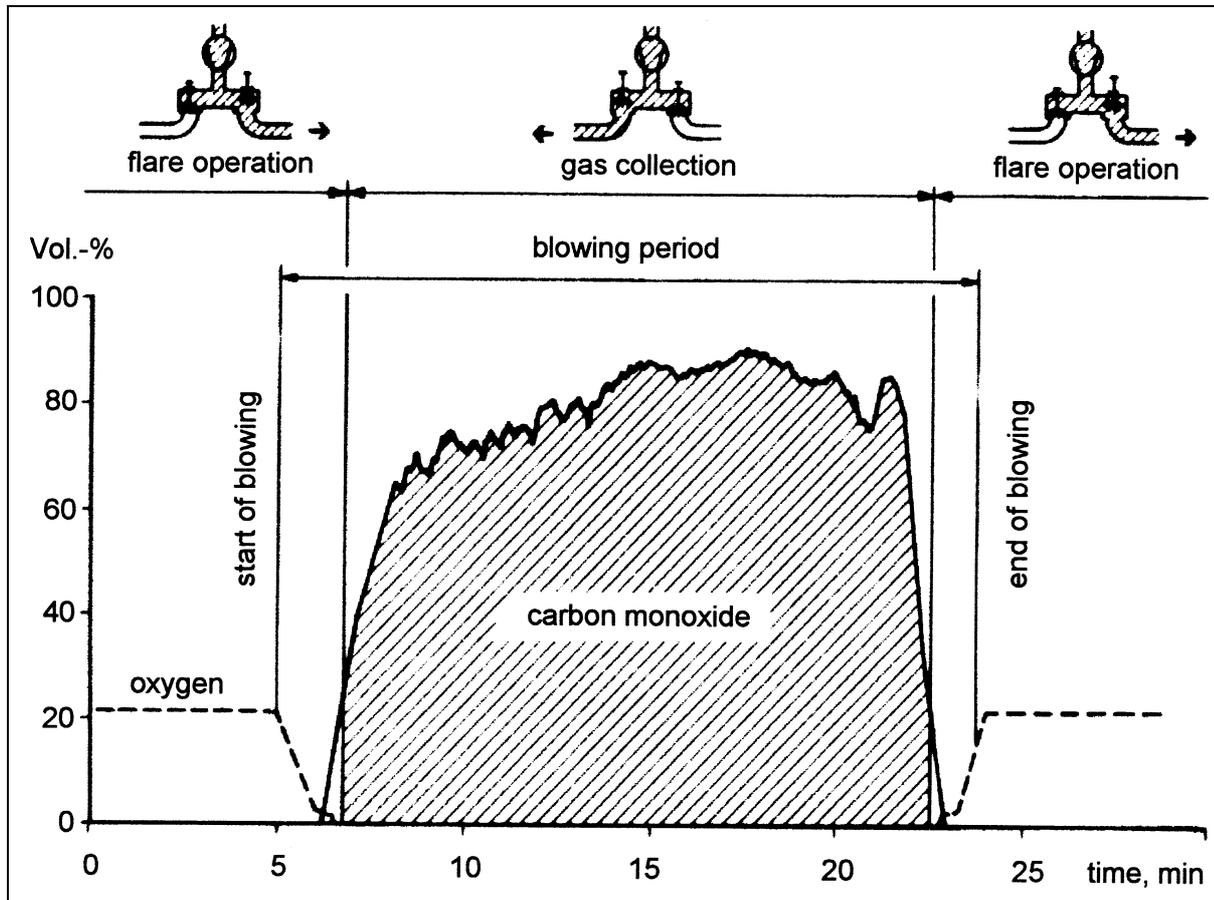


Figure 8.11 : BOF gas collection in the case of suppressed combustion

Particulate matter is usually removed from BOF gas by means of venturi scrubbers or dry electrostatic precipitators. When suppressed combustion is employed, the venturi scrubbers may achieve a particulate matter concentration of 5-10 mg/Nm³ (but concentrations up to 50 mg/Nm³ are also possible) in the grid gas. This corresponds to 1 g/t LS. The iron content of the particulate matter recovered is 42-65%. The particulate matter in the grid gas is emitted at the site of incineration of the grid gas.

When full combustion is employed, particulate matter emissions to the atmosphere are in the range 25 to 100 mg/Nm³ after treatment. As a result of the much higher flue gas flow of open combustion systems, this figure corresponds to particulate matter emissions of up to 180 g/t LS. Table 8.5 summarises the emissions to air from a BOF.

Component	Specific emission value	Unit
Flow of primary (BOF gas) ventilation		
- Full combustion	2000-3000	[Nm ³ /t LS]
- Suppressed combustion	50-120	"
Flow of secondary ventilation	1300-4800	[Nm ³ /t LS]
Particulate matter from oxygen blowing		
- unabated	15-20	[kg/t LS]
- after primary (BOF gas) de-dusting	0.5 – 200	[g/t LS]
Particulate matter from charging and tapping		
- unabated	200-1000	[g/t LS]
- after secondary de-dusting	2-60	"
- not caught by enclosure	25-100	"
(Heavy) metals		
Al	0.60 - 0.68	[g/t LS]
As	0.00 - 0.02	"
Cd	0.07 - 0.20	"
Cr	0.00 - 0.04	"
Cu	0.04	"
Fe	2.8 – 83	"
Hg	0.00 -0.02	"
Mg	1.45 - 2.40	"
Mn	2.7 – 60	"
Pb	1.5 – 2.9	"
Zn	8.2	"
Sulphur oxides (SO ₂)	0.4 – 5.5	[g/t LS]
Nitrogen oxides (NO _x)	5.0 – 20	[g/t LS]
Carbon monoxide (CO)	7.0 – 16	[kg/t LS]
Hydrogen fluoride*	0.008 - 0.01	[g/t LS]
PAH (Borneff 6)	0.08 – 0.16	[mg/t LS]
PCDD/F	<0.001 – 0.06	[µg I-TEQ/t LS]

Legend: LS = (crude) liquid steel

* whenever fluorspar (CaF₂) is added as a flux in the pig iron desulphurisation, emissions of fluorides may be much higher.

Table 8.5 : Specific emission values to air from a basic oxygen furnace with suppressed combustion; after abatement, if there is no other indication - based on [InfoMil, 1997]

Note that gradations between suppressed and full combustion exist; some plants employ full combustion, some plants employ partially suppressed combustion and some plants fully suppress BOF gas combustion. In some cases the BOF gas is not recovered but flared. Nowadays, there is a tendency towards suppressed combustion and subsequent BOF gas recovery. This recovery makes a large gas tank necessary. There also has to be local use for the recovered gas. The absence of these in some cases means there are still basic oxygen steelmaking plants in the EU which do not employ BOF gas recovery.

The lance hole is an important aspect in particulate matter emissions. As the oxygen lance needs to be retractable, particulate matter in the flue gas duct may escape through the lance hole reaching the building atmosphere. Shielding and blowing of steam or inert gas can avoid these emissions.

8.2.2.1.1.3 Emissions from ladles, ladle furnaces, BOF and other equipment used in secondary metallurgy

Dust releases from the various processes are in the range of 1-275 g/t LS [EUROFER BOF, 1997]. Emissions to air after abatement are in the range of 0.1-10 g/t LS (see Table 8.3)

Concerning the production of leaded steel (see 1.1.5) emissions occur during addition of lead into the ladle containing liquid steel. The resulting off gas is sucked off and normally treated in a bag filter especially for this waste gas to achieve low residual dust contents ($< 5 \text{ mg/Nm}^3$). But the ladle itself is not gas-tight and thus there are also fugitive emissions which are not caught but can be of considerable significance. Information on flow and lead emissions is not available.

8.2.2.1.2 Secondary off-gases

Secondary off-gases comprise emissions from following operations:

- reladling and deslagging of hot metal
- BOF charging (hot metal and scrap)
- tapping of liquid steel and slag from BOF (converters) and ladles
- secondary metallurgy and tapping operations
- handling of additives
- continuous casting

The emissions to air from all these processes/emission sources are summarised in Table 8.3. The emissions from charging and tapping of the BOF (converter) are of relevance and are described in more detail:

Particulate matter is emitted both during charging of scrap and hot metal and during tapping from the BOF. During charging or tapping operations, the converter is tilted. A so-called secondary ventilation system, is often installed to abate the particulate matter emissions that occur. The secondary ventilation system usually consists of a canopy hood just above the converter in tilted position and a dog house around the remaining 3/4 of the converter. Subsequent treatment of the evacuated gases is usually performed by means of a fabric filter or ESP.

During charging and tapping, not all of the particulate matter is captured by the secondary ventilation. A small amount of particulate matter is therefore emitted through the roof of the converter building (25-100 g/t liquid (crude) steel). The actual emission depends mainly on the evacuation efficiency and the charging sequence of scrap and liquid pig iron.

8.2.2.2 Solid wastes/by-products

The various solid wastes/by-products resulting from oxygen steelmaking are compiled with their specific quantity in Table 8.6.

Solid waste/by-product	Specific quantity (range) [kg/t LS]
Desulphurisation slag	2 – 25
BOF slag	100 – 130
Spittings	4 – 10
Coarse dusts and sludges from BOF gas treatment	3 – 12
Fine dusts and sludges from BOF gas treatment	9 – 15
Dust from secondary de-dusting	0.2 – 3
Slag from secondary metallurgy	2 – 16
Slag from continuous casting	4 – 5
Mill scale from continuous casting	1.2 – 6
Rubble	0.8 – 6

Legend: LS = liquid steel

Table 8.6 : Kind and specific quantity of solid residues/by-products resulting from oxygen steelmaking – after [EUROFER BOF, 1997; Rentz, 1996]

The figures in Table 8.6 confirm those given in Table 8.2 originating from four integrated steelworks in the EU. As far as it is available, information on composition and fate of the single solid wastes/residues are given here.

8.2.2.2.1 Desulphurisation slag

Desulphurisation slag is a heterogeneous slag which is only partially melted. The composition of desulphurisation slags strongly depends on the used desulphurisation agents. The typical composition of such a slag contains Table 8.7.

CaO	SiO ₂	Al ₂ O ₃	MgO	Total Fe	Metallic Fe	MnO	P ₂ O ₅	Cr ₂ O ₃	Free CaO	S	CaO/SiO ₂
27.0	18.0	8.0	10.0	20.0	15.0	≤0.5	≤0.2	≤0.1	≤5	≤4	1.5

Table 8.7 : Chemical composition of slag from pig iron desulphurisation in [weight-%] – [Geiseler, 1991]

The relatively high sulphur content and unsatisfactory mechanical properties do not make desulphurisation slag ideal for reuse. Partly they are used for landfill construction or for noise protection barriers. They are also landfilled (Figure 8.12).

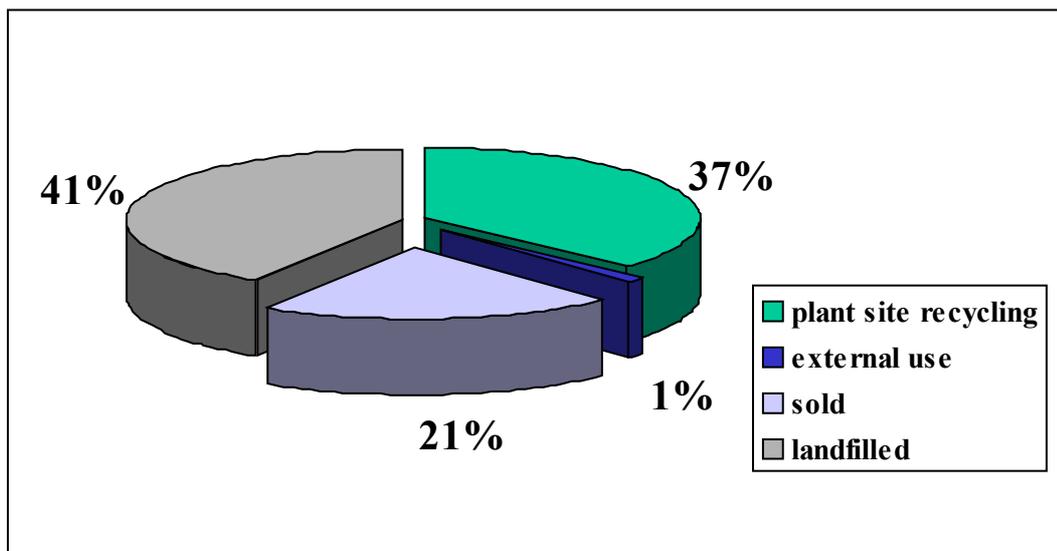


Figure 8.12 : Fate of slag from pig iron desulphurisation in the EU – [EC Study, 1996]

8.2.2.2.2 BOF slag

The slag from basic oxygen steelmaking make up the largest share of residues. The chemical composition of this slag depends on the processes employed (Table 8.8).

Process	LD/AC	LD	AOD
CaO	50.0	50.0	53.0
SiO ₂	9.0	15.0	28.0
Al ₂ O ₃	≤ 2	≤ 2	3.0
MgO	≤ 3	≤ 3	5.0
Total Fe	12.0	16.0	≤ 2
Metallic Fe	≤ 1	≤ 1	≤ 1
MnO	2.0	≤ 4	≤ 1
P ₂ O ₅	15.0	≤ 2	≤ 0.5
Cr ₂ O ₃	≤ 1	≤ 1	≤ 2
Free CaO	≤ 7	≤ 10	≤ 5
S	-	-	≤ 1
CaO/SiO ₂	4	2.5	1.8

Legend: LD/AC = Linz-Donawitz/Arbed-CRM process; LD = Linz-Donawitz process;
AOD = Argon-Oxygen-Decarburization

Table 8.8 : Chemical composition of BOF slags in [weight-%] - [Geiseler, 1991]

BOF slag can be reused by returning it to the iron-making process. It can also be used for fertilizer manufacture or for road construction. Lime phosphate slag (from LD/AC or OBM processes) is used entirely for fertilizer production. BOF slags are normally used in civil and hydraulic engineering, road construction and cement industry. Thereby the relatively high free CaO content has to be considered. Nevertheless, in the EU a considerable percentage of BOF slag is still put to landfill (Figure 8.13).

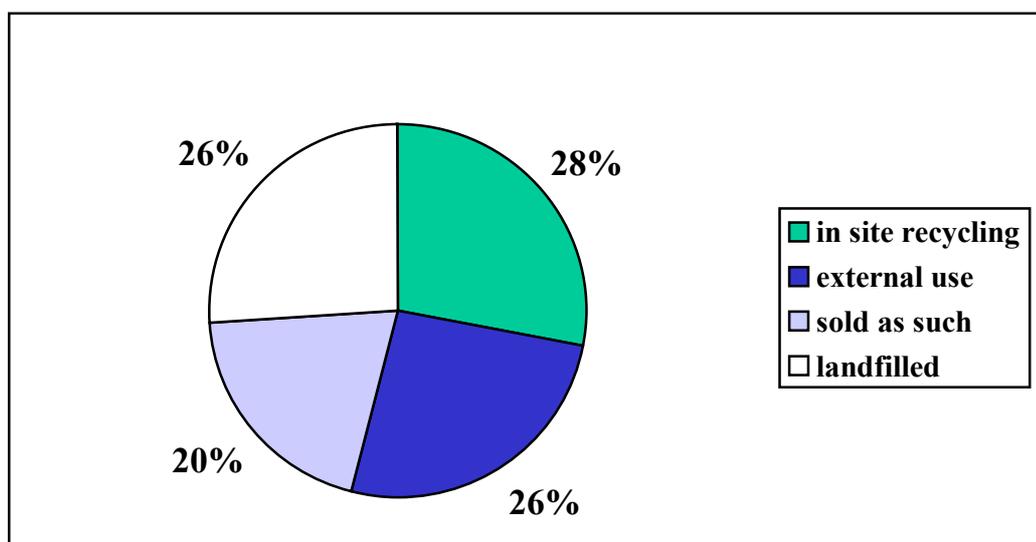


Figure 8.13 : Fate of BOF slags in the EU – [EC Study, 1996]

8.2.2.2.3 Spittings

Spittings are normally recycled to the sinter strand.

8.2.2.2.4 Coarse dust from BOF gas treatment

Coarse dust is separated from the BOF gas in the case of dry BOF gas treatment or as sludge in case of wet treatment (e.g. in a venturi scrubber). The composition of coarse dust can be seen from Table 8.9 in comparison with the one for fine dust. The composition of coarse sludge and fine sludge has the similar relations.

Parameter	Coarse dust	Fine dust
Total Fe	30 – 85	54 – 70
Metallic Fe	72	20
CaO	8 – 21	3 – 11
Zn	0.01 – 0.4	1.4 – 3.2
Pb	0.01 – 0.04	0.2 – 1.0
S	0.02 – 0.06	0.07 – 0.12
C	1.4	0.7

Table 8.9 : Composition of coarse and fine dust in [weight-%] – [Harp, 1990; IISI, 1987; data from one integrated steelworks in the EU]

The coarse dust after preparation is usually returned to the oxygen steelmaking process or is recycled to the sinter strand. In the EU only a smaller part is put to landfill (Figure 8.14).

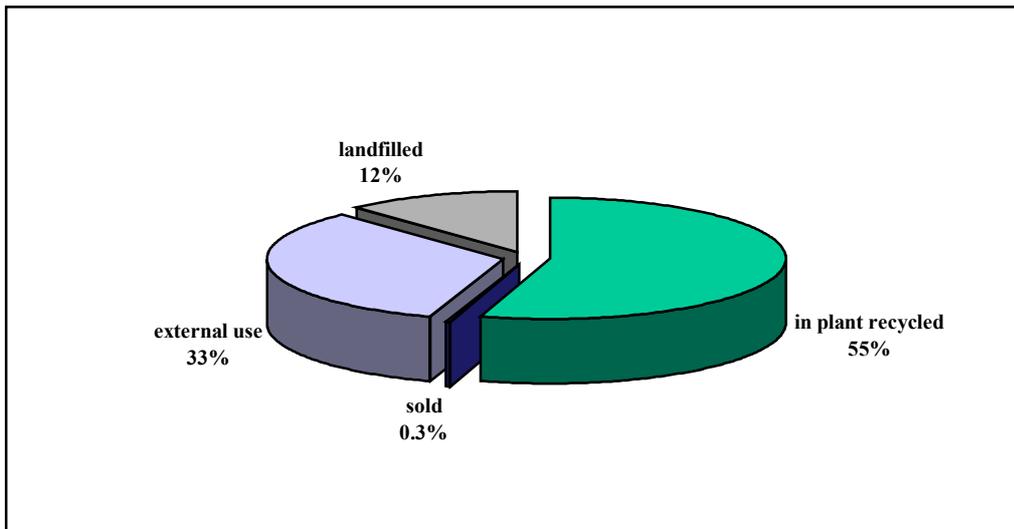


Figure 8.14 : Fate of dust from dry BOF gas treatment – [EC Study, 1996]

8.2.2.2.5 Fine dust and sludge from BOF gas treatment

Table 8.9 indicates that fine dust, compared to coarse dust, contains significantly larger amounts of lead and zinc. The main source of these heavy metals usually is scrap charged to the BOF. In some cases it is possible to control the lead and particularly the zinc input with the scrap. This reduces zinc content to below 1% which is target of the measure. Because of the zinc content, very often the fine dust or sludge cannot be recycled but is put to landfill (Figure 8.15).

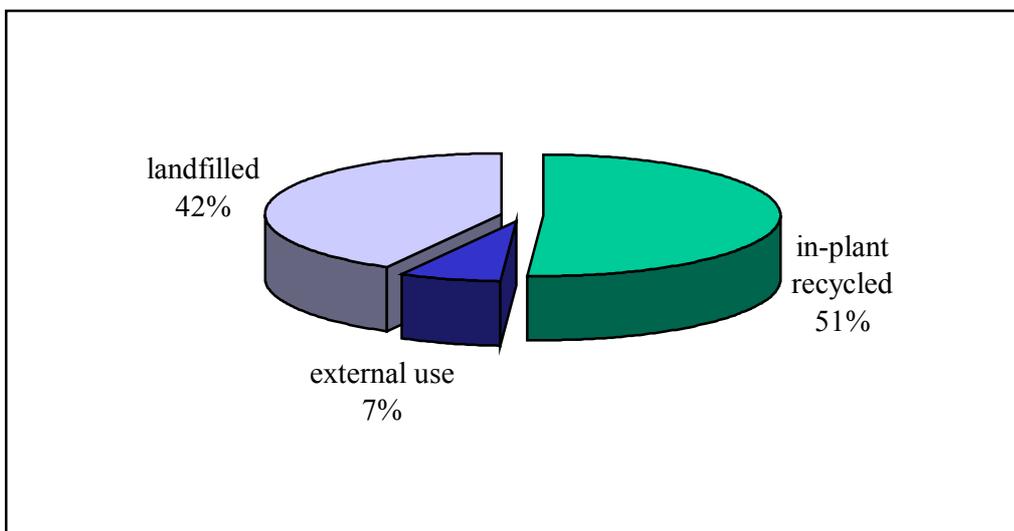


Figure 8.15 : Fate of sludge from (wet) BOF gas treatment in the EU – [EC Study, 1996];

This figure is for fine sludge as venturi scrubbers or wet ESP are in use as a second treatment step, thus creating sludges.

8.2.2.2.6 Slag and mill scale from continuous casting

These by-products are normally recycled to the sinter strand.

8.2.2.2.7 Rubble

There is no representative information available on the fate of these solid wastes/by-products. In some steelworks it is partly recycled to the BOF or reused for the production of new refractory material. In other cases it is put to landfill.

8.2.2.3 Wastewater emissions

The relevant sources of wastewater are following:

- scrubbing water from BOF gas treatment
- water from direct cooling in continuous casting.

In addition cooling water mainly derives from BOF cooling and continuous casting or ingot casting.

8.2.2.3.1 Wastewater from BOF gas treatment

BOF gas is treated either wet or dry. In the case of wet cleaning wastewater is produced, which is normally recycled after treatment. This treatment is very often performed in two steps: separation of coarse particles ($> 200 \mu\text{m}$ grain size) followed by sedimentation in circular settling tanks. Flocculating agents are added to improve sedimentation. The sludge is de-watered by means of rotary vacuum filters, chamber filter presses or centrifuges. Representative data about flow and quality of the treated wastewater discharged from the circuit (overflow) are not available.

8.2.2.3.2 Wastewater from vacuum generation

The specific wastewater flow from vacuum generation is about $5 \text{ m}^3/\text{t LS}$ (see 8.1.4). Information on composition and treatment or recycling is not available.

8.2.2.3.3 Wastewater from continuous casting

Emissions to water from continuous casting machines are generated by the direct cooling system. This is used for direct cooling of slabs, blooms, billets and the machines. The discharge water is contaminated with metal oxides and with hydrocarbons (oil). This water is very often treated together with wastewater from rolling mills. No representative data are available concerning flow rates and quality.

8.2.2.4 Energy demand

8.2.2.4.1 Basic oxygen furnace (BOF)

In the BOF (converter), fuel is consumed to preheat and dry the converters after relining and repair. This thermal energy consumption totals approximately 0.051 GJ/t LS. Electricity consumption is estimated at 23 kWh/t LS or 0.08 GJ/t LS. This figure includes the production of oxygen and the operation of the converters.

The process gas from the converter contains large amounts of carbon monoxide (CO) and is hot. When the energy from the BOF gas is recovered (waste heat recovery and/or BOF gas recovery), the basic oxygen furnace becomes a net producer of energy. In a modern plant, energy recovery can be as high as 0.7 GJ/t LS.

8.2.2.4.2 Continuous casting

Fuel consumption for preheating the ladle containing liquid steel is estimated at 0.02 GJ/t LS. Electricity consumption of the casting machines is estimated at 0.04 GJ/t LS [InfoMil, 1997].

8.3 Techniques to consider in the determination of BAT

Process-integrated measures

- PI.1 Energy recovery from the BOF gas
- PI.2 Lowering the zinc-content of scrap
- PI.3 On-line sampling and analysis of steel

End-of-pipe techniques

- EP.1 Primary de-dusting
- EP.2 Particulate matter abatement from pig iron pre-treatment
- EP.3 Secondary de-dusting
- EP.4 Dust hot-briquetting and recycling
- EP.5 Treatment of wastewater from wet de-dusting
- EP.6 Treatment of wastewater from continuous casting

PI.1 Energy recovery from the BOF gas

Description: This measure involves making efficient use of both the sensible heat and the chemical energy in BOF gas. Previously, most of the chemical energy was dissipated by flaring.

BOF gas produced during oxygen blowing leaves the BOF through the converter mouth and is subsequently caught by the primary ventilation. This gas has a temperature of approximately 1200°C and a flow rate of approximately 50-100 Nm³/t steel. The gas contains approximately 70-80% carbon monoxide (CO) when leaving the BOF and has a heating value of approximately 8.8 MJ/Nm³.

Generally, two systems can be used to recover energy from the BOF gas:

1. **Combustion of BOF gas in the converter gas duct and subsequent recovery of the sensible heat in a waste heat boiler;**

This BOF gas can be combusted by allowing ambient air into the gas duct of the primary ventilation system. Thus, the sensible heat and the total gas flow in the primary ventilation system increases and more steam can be generated in the waste heat boiler. The amount of air admixed with the BOF gas determines the amount of steam generated. In a complete steelmaking cycle (approx. 30-40 minutes), oxygen blowing lasts for approximately 15 minutes. Steam generation, which is directly related to oxygen blowing, is therefore discontinuous.

2. Suppression of BOF gas combustion and storage of the BOF gas in a gasholder for subsequent use.

BOF gas combustion in the primary ventilation system can be suppressed by avoiding the supply of ambient air in the system. This is usually done by lowering a water-cooled retractable skirt over the mouth of the converter. In this way, the carbon monoxide is retained and the BOF gas can be used as an energy source at other locations. The gas is cleaned to meet grid gas requirements and can be stored in a gasholder. A waste heat boiler may be installed to recover the sensible heat which is present in the non-combusted BOF gas.

It should be noted that BOF gas is not collected during start and end of blowing on account of its low CO content. During these periods, which last a few minutes, it is flared instead (see Figure 8.11).

There is currently a trend toward suppressed combustion followed by BOF gas recovery. There are two main reasons for this:

- Suppressed combustion reduces the quantity of flue gas and thus reduces the cost of fans and dust removal. The reduced waste gas flow rate typical of suppressed combustion results in a raw gas with a higher mass concentration. Thus for an identical clean gas dust concentration a more efficient dust recovery system must be used (see also EP.1).
- Large volumes of steam are obtained from full combustion systems. However, as the steam is produced discontinuously it cannot always be fully utilised. The use of recovered BOF gas is much more flexible. Use of BOF gas in conjunction with blast furnace gas and coke oven gas as a third gas-phase furnace product brings substantial advantages if it allows the replacement of considerable amounts of bought-in energy, such as natural gas. At some plants, the BOF gas is primarily used in upgrading the blast furnace gas [Joks, 1995]. Coke oven gas and natural gas are only admixed in the mixing stations with second and third priority (cascade control) [Joks, 1995].

Table 8.10 shows the benefits and drawbacks of suppressed combustion designed for air/fuel ratios below 0.1.

Advantages	Disadvantages
<ul style="list-style-type: none"> ▪ Reduced volumetric waste gas flow rate ▪ Larger dimensions of gas-carrying components to achieve more uniform velocities in the hoods ▪ Reduced energy consumption of the waste gas fan ▪ Reduced water requirement for off-gas cooling ▪ Design of dust recovery systems for smaller volumetric off-gas flow rates ▪ Off-gas utilisation possibility ▪ Reduced gas release from effervescent melt conditions due to arrangement of sealing skirts of different design 	<ul style="list-style-type: none"> ▪ Complex equipment technology with regard to safety engineering standards ▪ Need for additional components ▪ Additional safety engineering measures

Table 8.10 : Advantages and disadvantages of suppressed combustion with particular consideration of BOF gas utilisation – [EUROFER BOF, 1997]

The BOF gas composition in case of suppressed combustion is shown in Table 8.5. When full combustion is applied the CO content is much lower and CO₂ content is correspondingly higher.

Main achieved energy savings:

In Table 8.11, examples are given for steam generation in waste heat boilers at basic oxygen steelmaking plants of Thyssen Stahl AG, Germany.

Parameter	Unit	Ruhrort	Beeckerwerth	Bruckhausen
Capacity	[t LS/cast]	4 x 140	3 x 250	2 x 380
Air introduction factor*	-	>2.0	0.4	0.15
Flue gas flow	[Nm ³ /t LS]	250	115	87
Steam generation	[kg steam/t LS]	380	130	80

- the air introduction factor indicates how much air is introduced in the flue gas duct. 0: no air is introduced; 2: amount of introduced air is two times the amount of original BOF gas.

Table 8.11 : Steam generation at basic oxygen steelmaking plants of Thyssen Stahl AG – based on [Joks, 1995]

Energy recovery from a full combustion system with a waste heat boiler is reported to be 80% of the total outgoing heat. When suppressed combustion is applied, only 10 - 30% (0.1 - 0.3 GJ/t LS) of the total energy output is recovered in the waste heat boiler [Joks, 1995]. When the BOF gas is recovered, another 50 -90% is recovered as chemical energy (CO) in the BOF gas, depending on the air introduction factor. When the gas is flared, this energy is lost.

Total energy recovery when applying suppressed combustion, BOF gas recovery and a waste heat boiler for the use of sensible heat can be as high as 90% [Arimitsu, 1995; Joks, 1995].

When BOF gas is recovered, energy savings amount to 0.6 - 1.0 GJ/t LS, compared to flaring. The leak-free OG system, which was developed by Nippon Steel Corporation, leads to an

energy saving of 0.98-1.08 GJ/t LS and an increased molten steel production of 0.4%, compared to flaring.

Applicability: Both waste heat recovery and BOF gas recovery can be applied at new and existing plants.

Cross-media effects: Recovering BOF gas requires proper cleaning of the crude gas in order to meet the grid gas requirements. When full combustion is applied the flue gas is emitted to the atmosphere. Requirements for emissions to the atmosphere are usually less strict than grid gas requirements.

Overall emissions to air are reduced when suppressed combustion is applied. In addition, the (much) larger flue gas flow from full combustion systems implies more expensive and relatively less efficient particulate matter abatement.

The energy recovery may result in savings of non-renewable energy sources.

Recovery of BOF gas is potentially hazardous and requires stricter safety precautions (explosions, carbon monoxide leakage).

Reference plants: Energy recovery by means of full combustion systems or suppressed combustion systems is widely applied at oxygen steel plants around the world. There is a tendency towards suppressed combustion systems, mainly because of logistic advantages compared to full combustion systems.

Operational data: Not available

Economics: Estimated investments of suppressed combustion systems were between 5 - 25 Ecu₁₉₉₆/GJ. The pay-back period can be as short as one year depending on the local circumstances.

Reference literature: [Arimitsu, 1995; Joksch, 1995; InfoMil, 1997]

PI.2 Lowering the zinc content of scrap

Description: A high zinc content in the blast furnace has an adverse affect on correct operation. Therefore, recycling of material with a high zinc content is restricted. The dusts and sludges collected from the BOF gas de-dusting device may contain relatively high concentrations of heavy metals, especially zinc (Zn) (see Table 8.9). This zinc mainly originates from the scrap charged into the basic oxygen furnace. The release of Zn can fluctuate strongly from one cast to another, according to the type of external scrap loaded and the blowing conditions. The same problem, but to a lesser extent, applies to lead (Pb), cadmium (Cd).

In order to meet dust recycling requirements scrap with a low Zn content may be used. This excludes scrap containing galvanised products.

Main achieved emission level: At some plants a strict policy on the use of scrap with a low Zn content is practised. The sludge from BOF gas de-dusting has a Zn content of approximately 0.1-0.3%, which allows 100% recycling of the dust in the sinter plant.

Applicability: Application is possible at both new and existing plants. However, this measure strongly depends on the availability of scarp which is low in Zn, Pb and Cd and the economics of using this type of scrap. It is therefore not applicable in all cases.

Cross-media effects: This solution should be seen as a local solution. The overall effect is probably zero, because large amounts of galvanised steel are produced world-wide, which will lead to the generation of sludges with a relatively high Zn content when the steel will become available as scrap in the steelmaking process. At most BOF zinc is mainly emitted from the converter in the first few minutes of oxygen blowing.

Reference plants: Hoogovens IJmuiden, NL-IJmuiden and British Steel, UK-Scunthorpe, Sidmar, B-Gent.

Operational data: Operates without problems.

Economics: Scrap with a low Zn content is more expensive and increases the cost price per tonne LS produced. It is expected that scrap with a low Zn content will become increasingly hard to obtain. On the other hand, the use of low-Zn scrap enables recycling of the BOF gas cleaning sludges and dusts.

Reference literature: [Deckers, 1995; Pazdej, 1995; InfoMil, 1997]

PI.3 On-line sampling and steel analysis

Description: Oxygen steelmaking is a batch process. Every charge of hot metal has to be refined until the required steel quality is achieved. In order to monitor progress samples are taken from the steel bath for analysis. The result of the analysis is used to determine the additional time of oxygen blowing needed to achieve the required steel quality.

The latest dynamic modelling and monitoring systems reach a precision that makes sampling during blowing unnecessary. A control sample is then taken during the pouring period. This technique reduces the emissions due to sampling to zero.

It used to be necessary to interrupt oxygen blowing and tilt the BOF in order to take the sample. This was a time consuming process and it increased emissions from the BOF. In a modern plant samples are taken on-line during oxygen blowing by means of a sub-lance. This enables the refining process to continue while the samples are being analysed. This shortens production cycle times and so increases productivity. Emissions are lower compared with the previous sampling method as the position of the BOF is not changed.

Main achievements: Production time per batch is reduced so productivity is increased. Emissions to air are reduced as it is not necessary to tilt the BOF.

Applicability: Can be applied at all new plants. Existing plants need a retrofit to install this sampling system.

Cross-media effects: No cross-media effects are known.

Reference plants: Modern plants apply on-line sampling; dynamic modelling is applied at Sidmar, B-Gent.

Operational data: Not available.

Economics: Costs are probably reduced as a result of the higher productivity.

Reference literature: [InfoMil, 1997]

EP.1 Primary de-dusting

Description: During oxygen blowing, BOF gas is generated. This gas is loaded with a large amount of particulate matter. When the BOF gas is recovered for use as a fuel (see PI.1), the gas has to meet certain requirements. When BOF gas is combusted in the flue gas duct, the flue gas is emitted and has to meet the local emission standards. Nowadays, most plants recover the BOF gas as a fuel. Full combustion systems introduce ambient air in the primary ventilation system to combust the BOF gas. This leads to a large flue gas flow (2000-3000 Nm³/t LS); suppressed combustion systems only generate BOF gas (50-100 Nm³/t LS) (see Table 8.5). This results in significant differences in the dimensions of the primary de-dusting facilities. The reduced waste gas flow rate characterising the suppressed combustion method results in a higher raw gas mass concentration, so that the efficiency of the dust recovery system must be increased for an identical clean gas dust load. From a dust recovery point of view, therefore, the suppressed combustion principle allows the use of de-dusting systems designed for smaller volumetric flow rates which must nevertheless achieve higher dust recovery rates.

Primary de-dusting is usually performed by venturi-type scrubbers (approximately 90% of the plants) or dry electrostatic precipitators. Prior to the venturi or the ESP, coarse particulates are usually removed by means of a deflector etc.

Special attention should be paid to the emission of particulate matter through the oxygen lance hole. Emissions from this hole can be as large as 50 g/t liquid steel. Emissions can be abated by means of a movable "mill stone", which covers the hole during oxygen blowing and/or injection of inert gas (N₂/CO₂) or steam in the lance hole to dissipate the particulate matter. Other designs of lance hole seals are also efficiently combined with lance cleaning devices.

Main achieved emission level:

Dry de-dusting and suppressed combustion: By applying ESP, residual dust concentrations in the BOF gas may be reduced to as little as 10 mg/Nm³ (equivalent to 0.5-1.0 g/t LS), at any rate below 50 mg/Nm³. Before the gas is treated in the ESP, coarse dust is removed in a deflection zone and the gas is conditioned in an evaporation cooler.

Dry de-dusting and open combustion: No operational data have been forthcoming.

Scrubbing and suppressed combustion: Applying a wet system, first, coarse particles are removed in a wet separator, then fine particles are removed by venturi scrubbers. The particulate matter concentration in the BOF gas after scrubbing is usually between 15 to 50 mg/Nm³, but can also be less than 10 mg/Nm³.

Scrubbing and open combustion: When BOF gas is combusted in the flue gas duct and cleaned by means of venturi scrubbers, the residual dust content is between 10 to 50 mg/Nm³.

Applicability: Can be applied at both new and existing plants. An increasing number of plants apply dry ESP. For instance, at Voest-Alpine Stahl A-Linz, the steel plant was retrofitted with a dry de-dusting system to replace an existing wet one.

Cross-media effects: The extracted dusts and sludges may contain high concentrations of zinc, thus hampering their reuse. The use of Zn-free scrap may enable recycling of the BOF-sludge/dust to the sinter plant (see PI.2). Plants using dry ESP may hot-briquette the solid waste and recycle the briquettes directly into the steelmaking process (see EP.4).

In addition, wet de-dusting implies the generation of a contaminated wastewater flow (see 8.2.2.3.1).

Furthermore, the operation of a de-dusting device consumes energy. In a suppressed combustion system this would correspond to 0.001 to 0.005 GJ/t steel for a venturi scrubber and < 0.001 GJ/t steel for a dry ESP.

In an open combustion system, this would correspond to 0.04 to 0.15 GJ/t pig iron for a venturi scrubber and <0.005 GJ/t steel for a dry ESP.

Reference plants:

Dry ESP and suppressed combustion: Thyssen Stahl AG, D-Duisburg;

LD Melt Shop 3, Voest Alpine Stahl AG, A-Linz;

Dry ESP and open combustion: Not available;

Scrubbing and suppressed combustion: Oxygen steel plant 2; Hoogovens IJmuiden, NL-IJmuiden;

Scrubbing and open combustion: Oxygen steel plant 1; Hoogovens IJmuiden, NL-IJmuiden.

Economics: Investments: 24 to 40 million Ecu₁₉₉₆ for a 1 Mt/a steelmaking plant

Operational: 2 to 4 Ecu₁₉₉₆/t LS

Driving force for implementation: In order to recover BOF gas highly efficient purification is necessary. In the case of non-recovery of BOF gas the BOF gas has to be treated in order to meet existing emission limit values.

Reference literature: [Joksch, 1995; Köller, 1995; EC BOF, 1995; InfoMil, 1997].

EP.2 Particulate matter abatement from pig iron pre-treatment

Description: During the three stages of pig iron pre-treatment (desulphurisation, slag separation and hot metal transfer and weighing), particulate matter emissions occur. The specific dust emission factor (before abatement) varies from 110 to 830 g/t LS [EUROFER BOF, 1997]. These emissions are captured (Figure 8.16) and are usually treated by means of fabric filters. The desulphurisation stands are largely of the enclosed type. Key dust recovery measures include the use of ladle lids, controlled introduction of desulphurising agents, integrated deslagging operation, use of an enclosure with extraction system and installation of a door moving with the process (Figure 8.16).

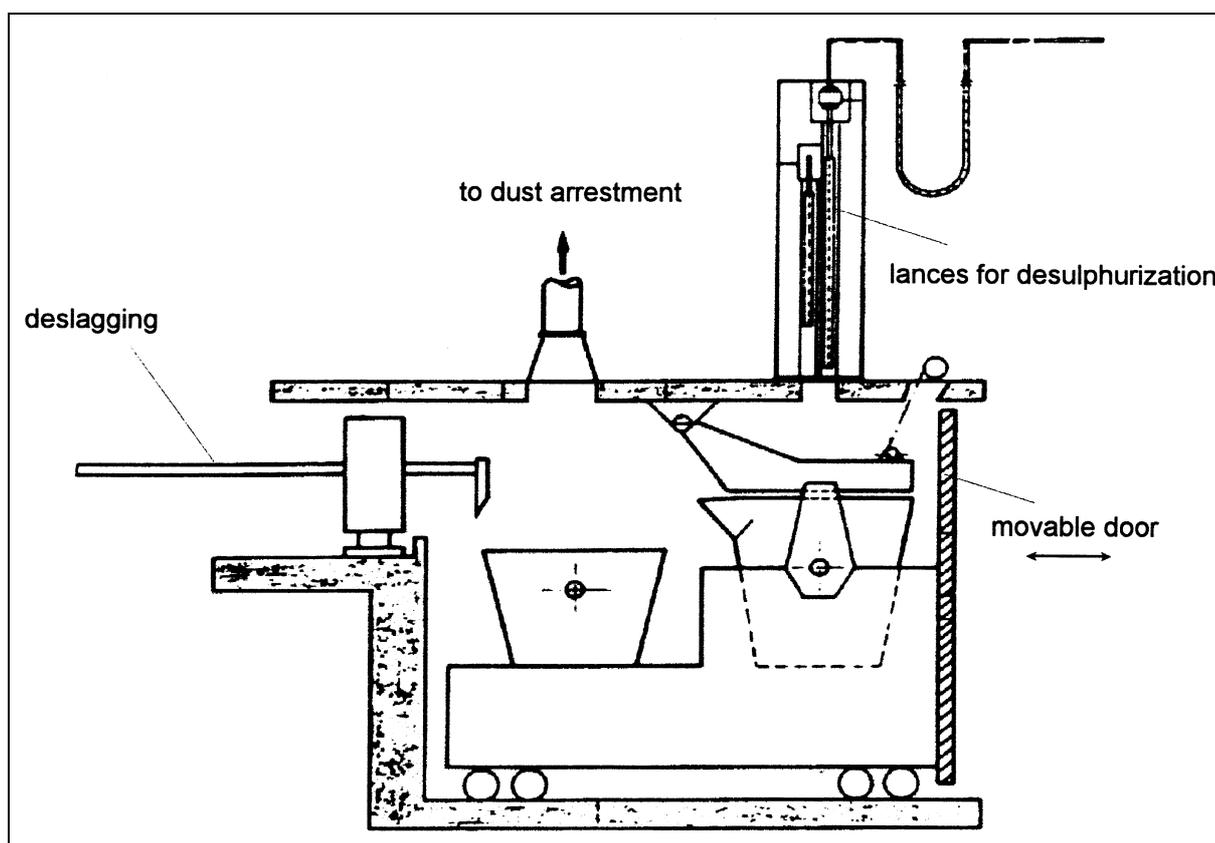


Figure 8.16 : Dust arrestment at a hot metal desulphurisation stand – [EUROFER BOF, 1997]

Extracted gases are loaded with up to 10000 mg/Nm³ of dust. In some cases, dry electrostatic precipitators are applied.

An important feature is the evacuation efficiency of the ventilation system. The position of the suction hoods has to be optimised to achieve a good extraction efficiency. Flue gas flow is in the range 30000 to 40000 Nm³/h.

Main achieved emission level: When the particulate matter is efficiently extracted and subsequently led through a fabric filter (or ESP), emissions less than 10 mg/Nm³ (around 1 g/t LS) can be achieved (see also Table 8.3).

Applicability: Can be applied both at new and existing plants.

Cross-media effects: The suction pumps consume energy. Furthermore, solid waste is generated. This can be recycled into the sintering process (high Fe content). The composition of

the dust from the pig iron desulphurisation unit is dependent heavily on the desulphurisation agent used.

Reference plants: Particulate matter abatement during pig iron pre-treatment is practised at many plants around the world.

Operational data: Both fabric filters and ESP can be operated without problems.

Economics:

Investments: approx. 10 million Ecu₁₉₉₆

Driving force for implementation: The main driving force has been emission limit values or other legal requirements.

Reference literature: [InfoMil, 1997; EC BOF, 1995]

EP.3 De-dusting of secondary off-gases

Description: Up until the early '70s, oxygen steelmaking plants were built without secondary dust collection equipment. As a result, most of today's secondary and subordinate primary source dust collecting installations are retrofits. The efficiency of such systems is highly dependent on local conditions. These play a particularly important role when it comes to the choice and design of the recovery system (enclosures, hoods, etc.). Determination of the waste gas flow rates often depends on local conditions and on the available space for installing piping systems, together with the possible size of the pipe cross-sections. For both new and existing installations it is virtually impossible to determine recovery efficiency in advance with any degree of certainty. Any evaluation attempt is further complicated by the extreme difficulty in measuring the results. The achievable recovery efficiencies shown in Figure 8.17 are derived from shop-floor and model tests, calculations and empirical observations and may vary according to the emission source considered and the specific process technology. Even an optimum design and high waste gas flow rates for a technical complete waste gas collection will not guarantee that a 100% recovery rate can be consistently achieved on a long-term basis throughout the oxygen steelmaking process. Changing or atypical operating conditions and environmental factors, such as unusual air flows in plant buildings, may lead to the release of unrecoverable dust streams which will escape as unavoidable secondary emissions through the roof monitors.

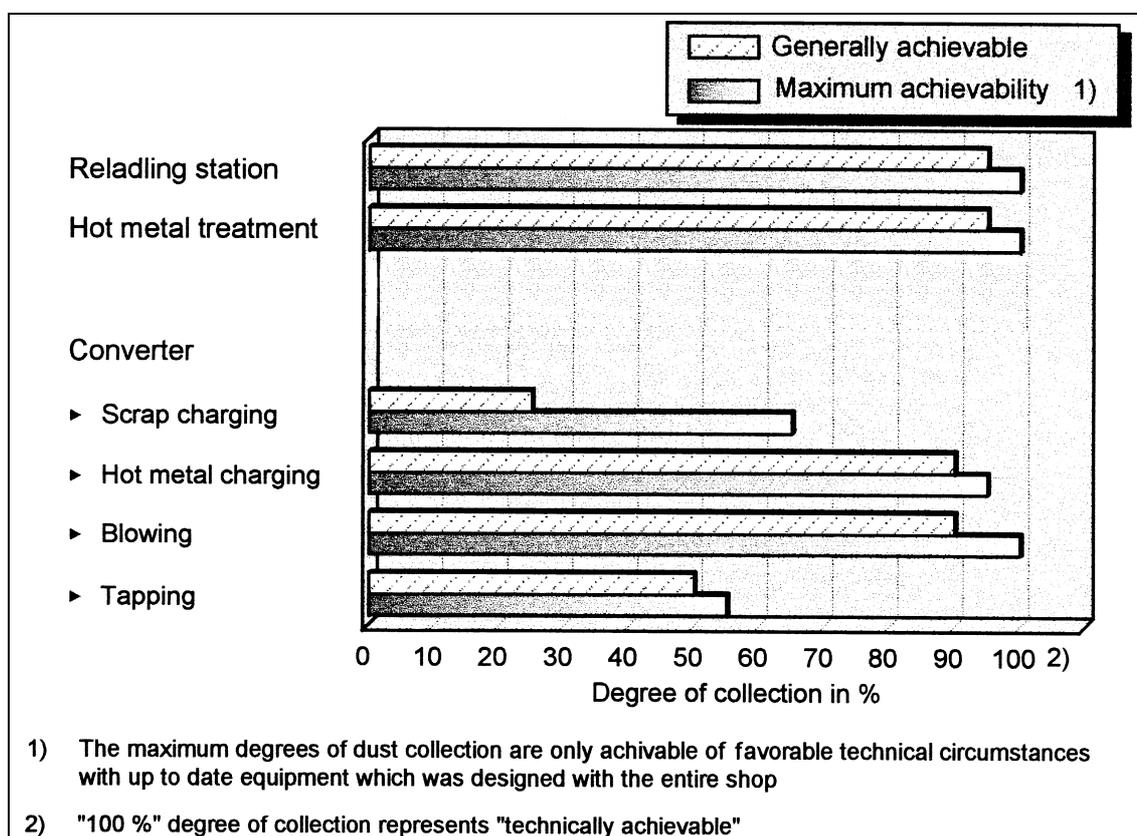


Figure 8.17 : Achievable degree of dust collection from secondary off-gases in basic oxygen steelmaking plants – [EUROFER BOF, 1997]

Secondary off-gases result from following operations

- reladling and deslagging of hot metal
- BOF charging
- tapping of liquid steel and slag from BOF (converters) and ladles
- secondary metallurgy and tapping operations
- handling of additives
- continuous casting

Estimates of the quantity of secondary emissions vary widely between 100 and 2000 g/t LS; BOF charging and tapping operations contribute most to these dust emissions (130 – 1230 g/t LS) [EUROFER BOF, 1997]. One explanation for this broad scatter range may be found in the fact that secondary emissions are difficult to delimit with any degree of certainty and that the recovery measures employed are often not adequately known. This results in a grey zone of overlap between data on substances released and data on actual emissions following the implementation of appropriate waste gas recovery and cleaning steps.

Charging and tapping

During charging of hot metal and scrap and tapping of the BOF (converter), unabated emissions of particulate matter occur (see figure above). The fact that the BOF is tilted means these emissions cannot be captured effectively by the primary ventilation system. Therefore, in most cases a secondary ventilation system is installed, which evacuates emissions generated during charging and tapping.

The secondary ventilation usually consists of a canopy hood just above the converter mouth in the tilted position and a doghouse around the remaining 3/4 of the converter (Figure 8.18). The canopy hood is preferably placed as close to the BOF as possible. In some existing plants the construction does not enable a canopy hood close to the BOF. In this case, the canopy hood may be installed near the roof, resulting in lower efficiency, depending on size of the hood and extracted volume.

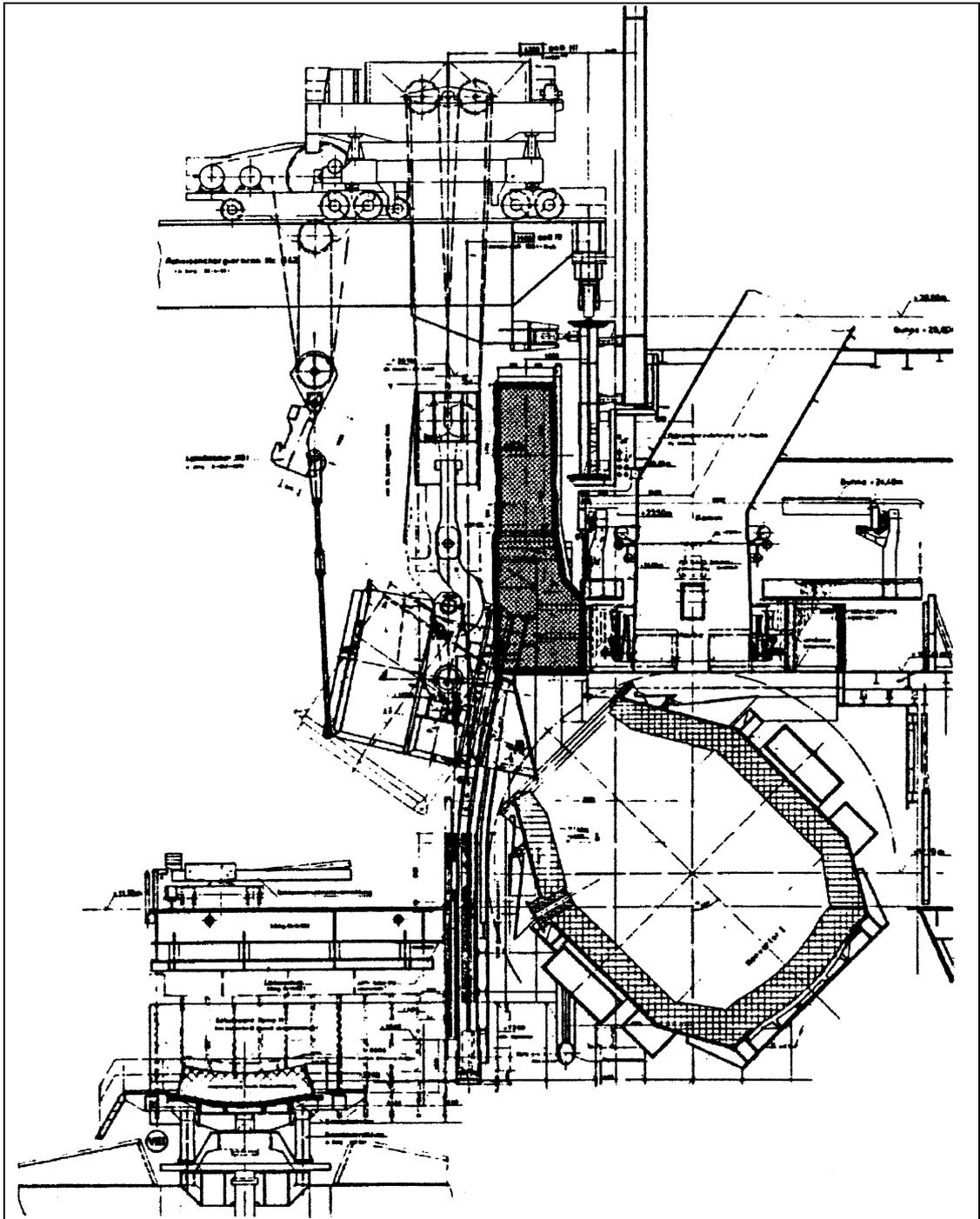


Figure 8.18 : Collection of secondary emissions during charging of hot metal to the BOF – [EUROFER BOF, 1997]; the same system is used for the charging of scrap

The flue gas flow from secondary de-dusting is in the range of 400000 to 1300000 Nm³/h [InfoMil, 1997] and cleaning is usually performed by means of a fabric filter, although dry electrostatic precipitators are also used. Some examples of design ratings of secondary de-dusting systems are summarised in Table 8.12. These examples show flows of between 650000 and 1000000 Nm³/h.

Plant	Capacity of the vessel [t]	BOF Number	Hot metal charge time [s]	Waste gas temperature [°C]	Waste gas Volume [m ³ /h]
OX 1	200	2	-	120	680000
OX 2	350	3	240	90	1000000
OX 3	300	2	40		950000
OX 4	200	2	120	200	1020000
OX 5	220	2	-	130	750000
OX 6	300	2	300	90	870000
OX 7	230	2	40	135	960000
OX 8	275	3 ¹⁾	90	150	650000
OX 9	210	3	240	150	800000

1) two-converter operation

Table 8.12 : Data from applied de-dusting systems for secondary emissions from charging and tapping of BOF – [EUROFER BOF, 1997]

Reladling of hot metal

The transfer of hot metal from the torpedo ladle to the charging ladles takes place in enclosed stands (Figure 8.19). An integrated control room arrangement permits direct monitoring of the transfer process. The hot metal ladle travels below mill floor level on a transfer car. This car carries a front sealing shield which seals off the vault at the hot metal pouring point, thereby forming an enclosed chamber. Where full enclosure is not feasible it is possible to mount a fume hood over the ladle (Figure 8.20).

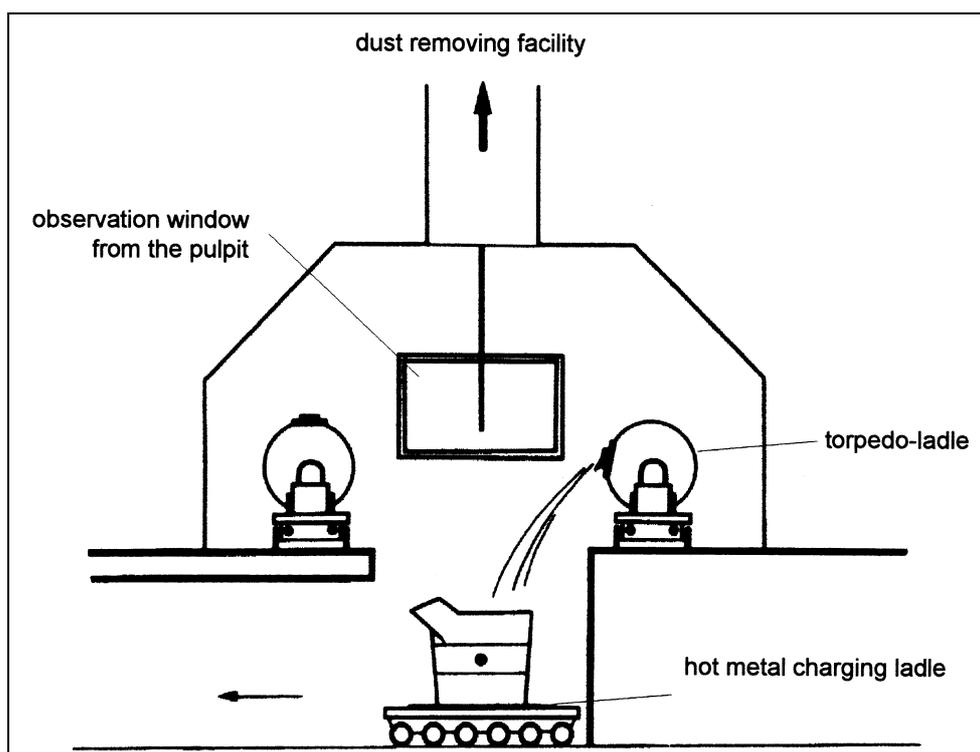


Figure 8.19 : Dust collection from reladling (torpedo ladle to charging ladle) of hot metal - [EUROFER BOF, 1997]

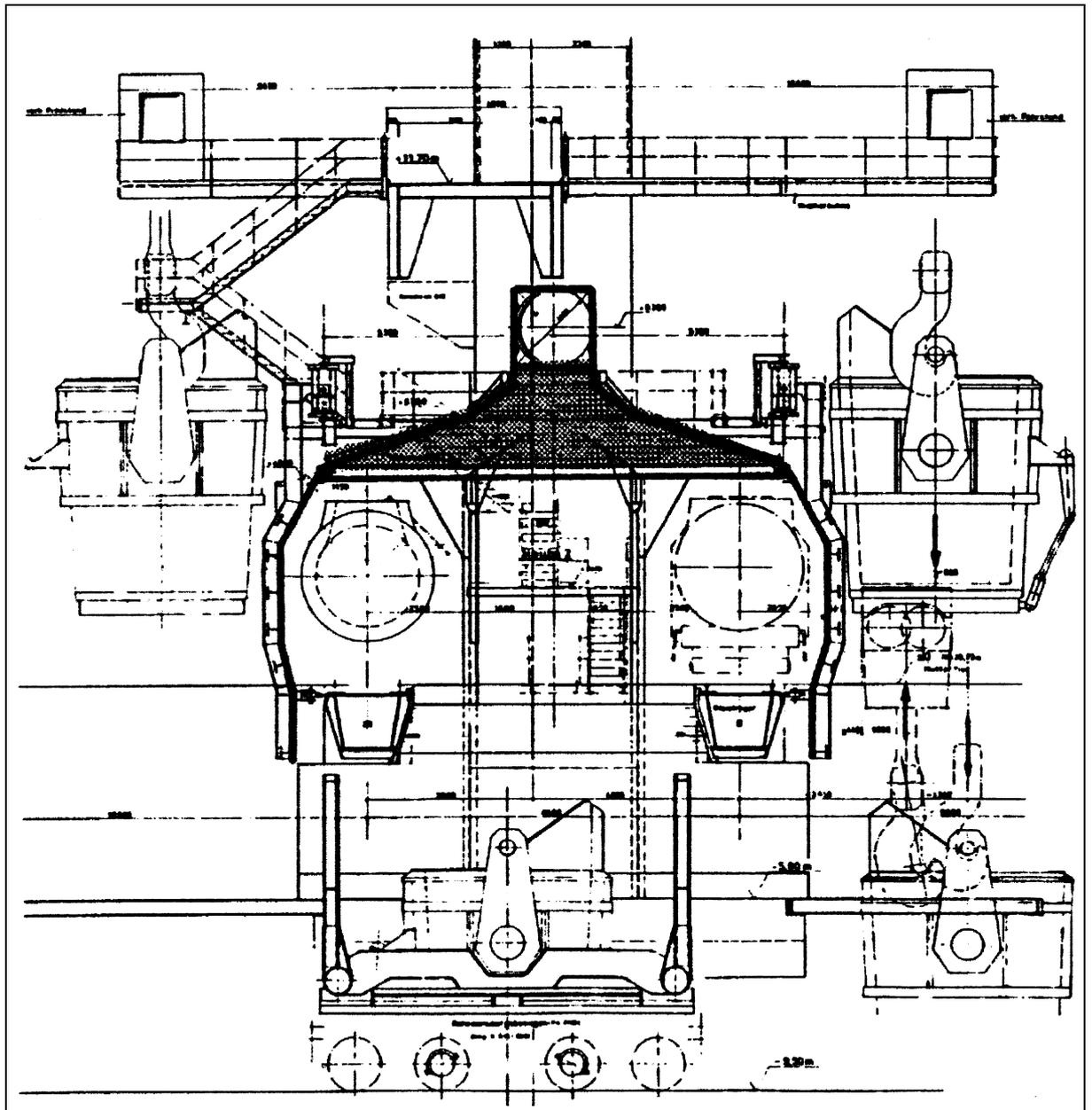


Figure 8.20 : Dust collection from a reladling station of hot metal - [EUROFER BOF, 1997]

A new technique to control dust emissions from reladling hot metal from torpedo ladle (or hot metal mixer) to charging ladle is to create an inert atmosphere by pouring solid carbon dioxide into the receiving ladle, in order to limit oxide dust formation. By vaporising the “dry ice“, continuously liberation of CO_2 gas occurs. Since this gas is heavier than air it forms an oxygen free layer (or blanket) on the bath surface, preventing iron oxidation. As the CO_2 is heated it will lift by thermal effect and shroud the liquid iron being poured [UNEP, 1997; Klein, 1993]. Figure 8.21 shows the reladling operation with an without fume/dust suppression.



Figure 8.21: Reladling of hot metal from torpedo ladle to charging ladle with and without dust/fume suppression with inert gas (CO₂) [UNEP, 1997]

The optimum procedure for producing an inert atmosphere at the hot iron transfer station is as follows:

- Render the inside of the empty ladle inert by injecting a maximum supply of CO₂ for about 30 seconds
- Maintain the inert conditions throughout the metal transfer using the minimum CO₂ flow

Although CO₂ is not a toxic gas, a CO₂ enriched atmosphere is a potential workplace safety problem because it can cause asphyxiation. Maximum allowable workplace concentration values have been established for CO₂ to limit the exposure of the personnel, and an appropriate ventilation and monitoring system has been installed in order to ensure no excessive exposure occurs [UNEP, 1997].

With a specific CO₂ consumption of 2.4 kg/t, the following results have been achieved:

- 87% reduction of dust emissions
- workplace limits for CO and CO₂ far from being exceeded [UNEP, 1997]

Deslagging of hot metal

For the deslagging process the ladle carrying the hot metal is tilted into its deslagging position while supported by a crane or a tilting cradle. The free cross-section of the hoods is restricted by suitable interior members to achieve higher intake velocities. The hoods can be of a travelling type so that they can serve several deslagging positions. The deslagging stands are normally separated by partitions which permit just enough travel for this operation. The openings are closed by sealing shields fitted on the car. Figure 8.22 shows the dust extraction system of a deslagging stand with desulphurisation.

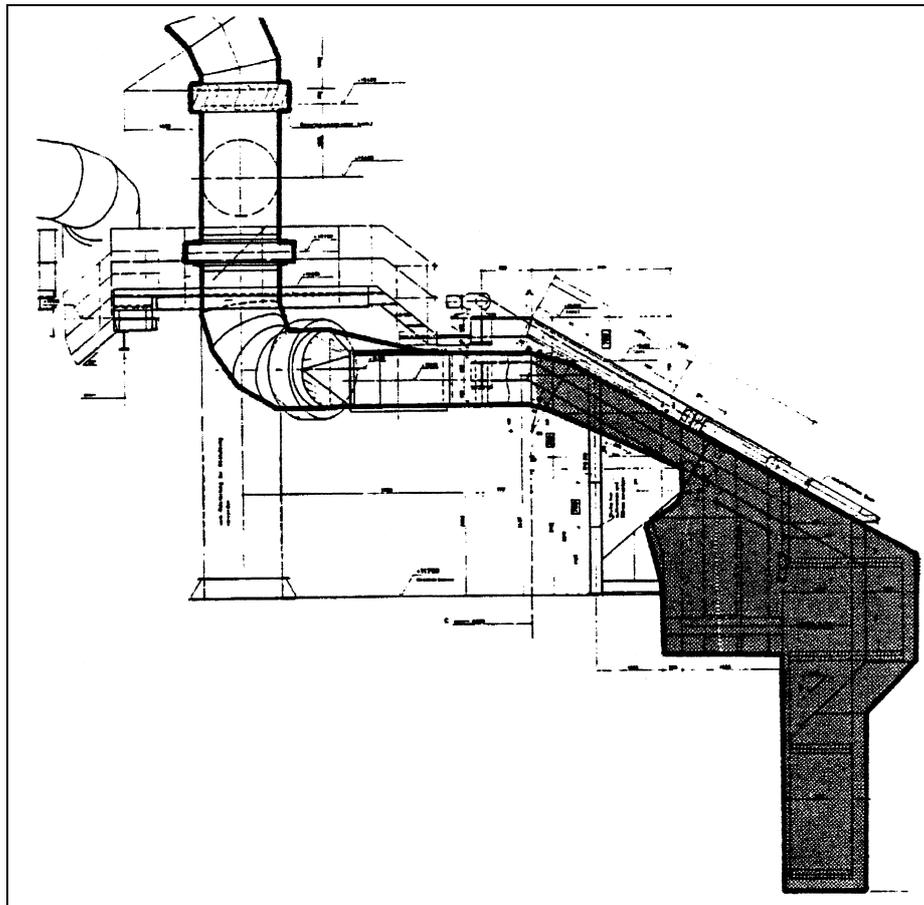


Figure 8.22 : Dust collection at a deslagging station – [EUROFER BOF, 1997]

Main achieved emission level: Emissions are mainly influenced by the efficiency with which the particulate matter generated during charging and tapping is evacuated. Emissions of particulate matter from the secondary ventilation system can be less than 10 mg/Nm^3 when the extracted gases are treated by means of a fabric filter. According to Table 8.3 dust emissions below 5 g/t LS can be achieved for each of the mentioned single emission to air.

The performance of an ESP will probably be somewhat lower, but should be seen in relation to the particulate matter capture efficiency of the secondary hoods.

In some plants (Japan), the entire roof is closed and evacuated, achieving a total efficiency of 100%.

Applicability: Secondary de-dusting can be applied at new and existing plants. In existing plants, the design of the plant may restrict the possibilities for proper evacuation.

Cross-media effects: During secondary de-dusting, a solid waste is generated (up to 1 kg/t LS). Reuse of this iron-rich solid waste is depending largely on the zinc content. Some plants may be able to reuse it and other may wish to dispose of it.

The operation of a ventilation and particulate matter cleaning device requires energy. Secondary ventilation requires an evacuation capacity of approximately 400000 to $1300000 \text{ Nm}^3/\text{h}$. This would correspond to energy consumption in the range 0.72 to $7.2 \text{ MJ}/(1000 \text{ Nm}^3)$ treated, when a fabric filter is applied. The energy consumption mainly depends on the pressure drop and the capacity of the associated fan. The specific energy consumption for secondary de-dusting is relatively high compared to other de-dusting operations (Figure 8.23).

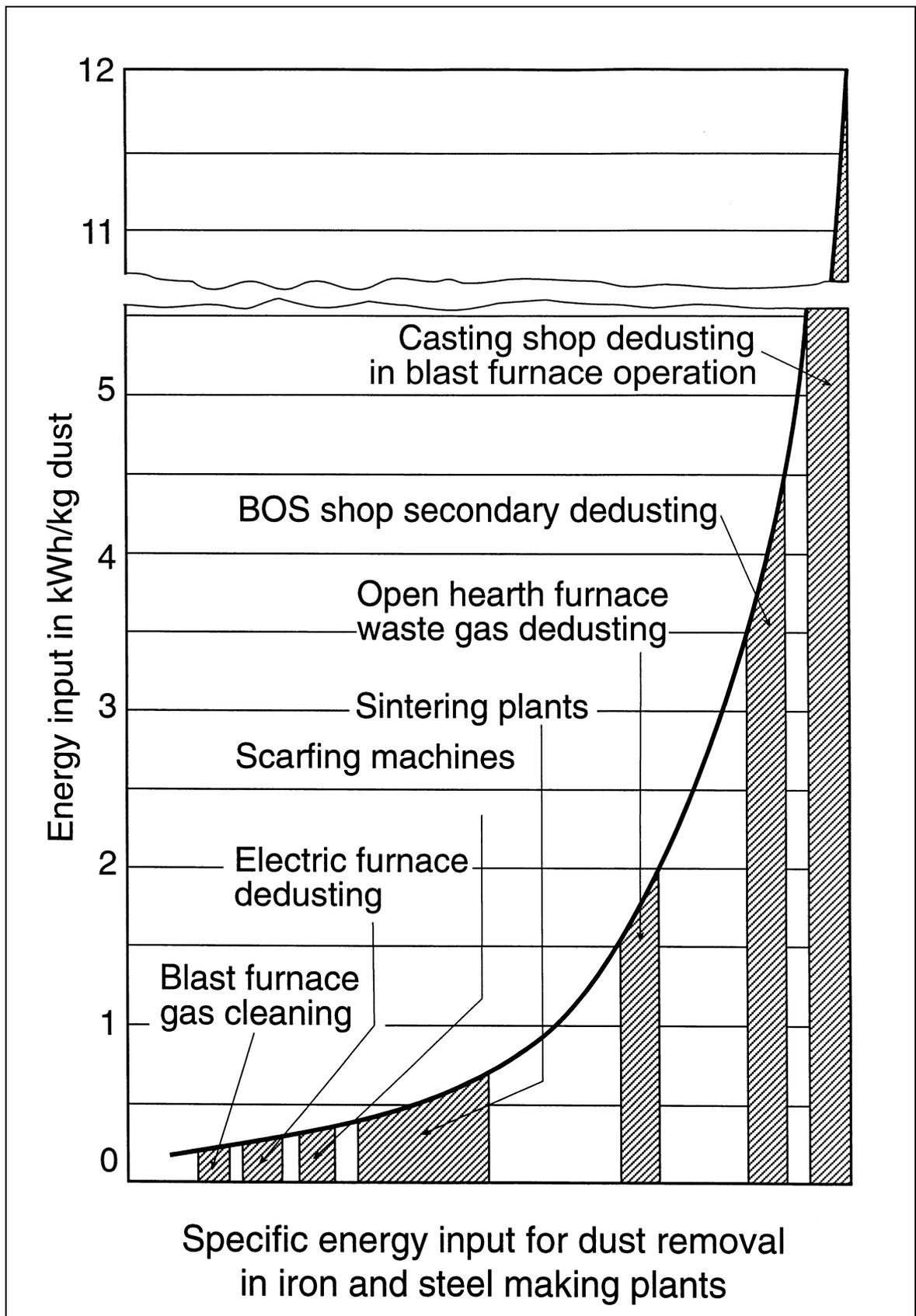


Figure 8.23 : Specific energy input for de-dusting operations in integrated steelworks - [Phillip, 1987]

Reference plants: Many plants worldwide apply secondary de-dusting.

Operational data: Both fabric filters and ESP can be operated without problems. The most difficult aspect of secondary de-dusting is evacuation efficiency and the recycling of the solid waste generated.

Economics:

Investments: 12 to 20 million Ecu₁₉₉₆

Operational: 0.8 to 4 Ecu₁₉₉₆/t LS

Driving force for implementation: The main driving force has been emission limit values or other legal requirements.

Reference literature: [InfoMil, 1997; EUROFER BOF, 1997; EC BOF, 1995]

EP.4 Dust hot-briquetting and recycling with recovery of high Zn concentrated pellets for external reuse

Description: Whenever dry ESP is applied as a means of cleaning of the BOF gas resulting from oxygen blowing, solid dust is generated. This dust has a high iron content (40-65%) and can be used as a valuable raw material when the dusts are pressed to briquettes. Although coarse dust and fine dust are briquetted in the same plants they are charged separately on account of their different properties (see Table 8.9), Briquettes from coarse dust contain about 70% metallic iron and can be used as a substitute for scrap in the BOF. Briquettes from fine dust have about 7% metallic iron and can be used as supplement for cooling ore [Auth, 1988]. Hot-briquetting is performed in a hot-briquetting plant (Figure 8.24). First the dusts are heated up to 750 °C in a moving bed reactor by hot air and autothermic reactions. In a second step briquettes are formed in a cylindrical press.

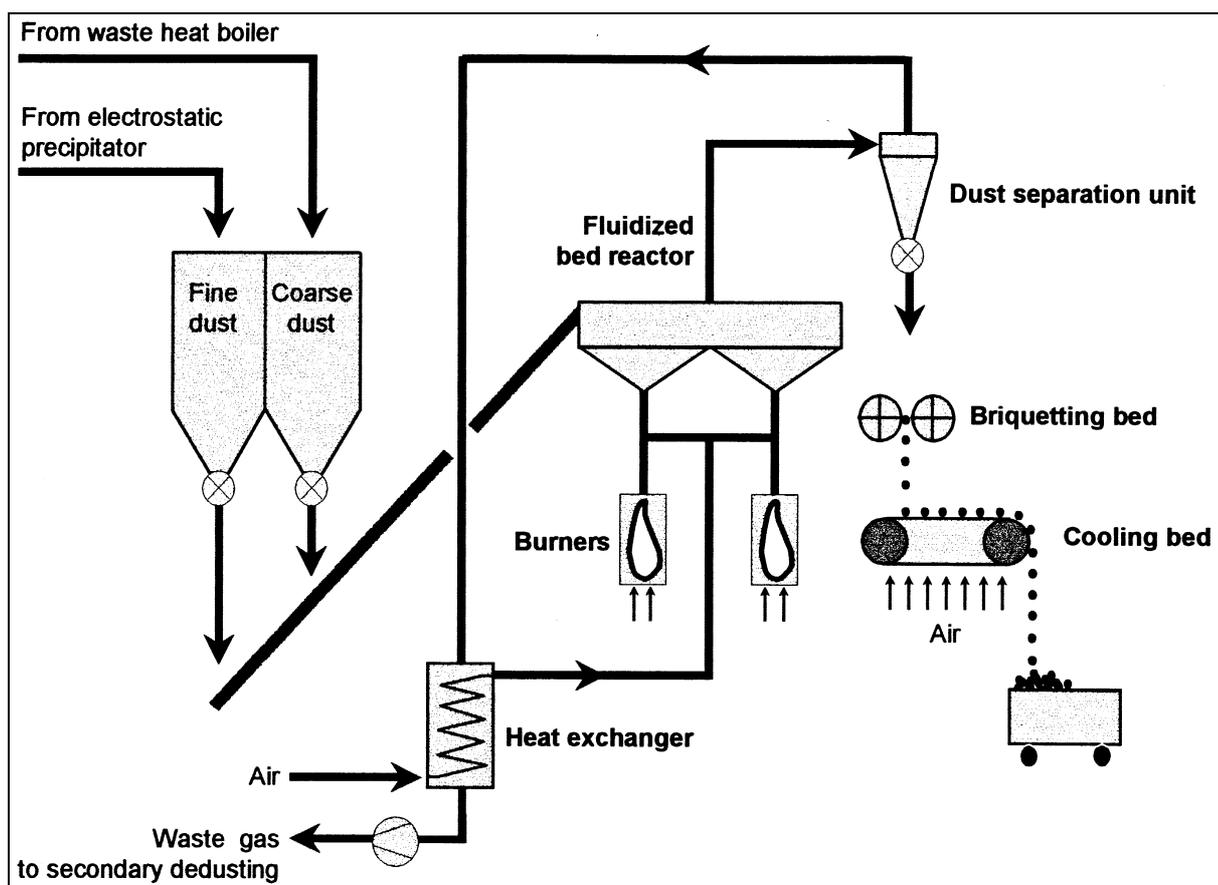


Figure 8.24 : Facility for briquetting dust from basic oxygen steelmaking plants – [EUROFER BOF, 1997]

Recycling dust progressively enriches the zinc concentration. When the dust briquettes have reached an average zinc content of at least 17% by weight, they are transported to external reprocessors for zinc recovery.

As the zinc in the dust within this closed system is very unevenly distributed, dusts with zinc contents significantly above 17% by weight also find their way into the cycle outlined here. This leads to considerable quantities of zinc being carried in the cycle unnecessarily, and being repeatedly reduced, vaporised, oxidised and briquetted in every successive cycle.

This sawtooth pattern of successive enriching and elimination of the dusts leads to a greatly fluctuating briquette charge in the BOF. This, in turn, not only influences the metallurgical work (slagging, build up of dust accretions in the waste-gas duct) but also has a significant impact on the thermal balance (hot metal/scrap). Regular analytic control is needed in order to ensure that the quality of the steel and slag produced is not impaired by any undue increase in the Zn content.

In order to optimise the process an on-line technique for determining the level of zinc in dust in real time has been developed. This new technology is referred to as LIBS (laser-induced breakdown spectroscopy). The device continuously measures the zinc content of the dust on the conveyor. Knowing the zinc content of the dust means it can be selected – when appropriate – for removal, pelletising and transport to the reprocessor.

Figure 8.25 gives a schematic overview of the optimised dust cycle at a oxygen steelmaking plant.

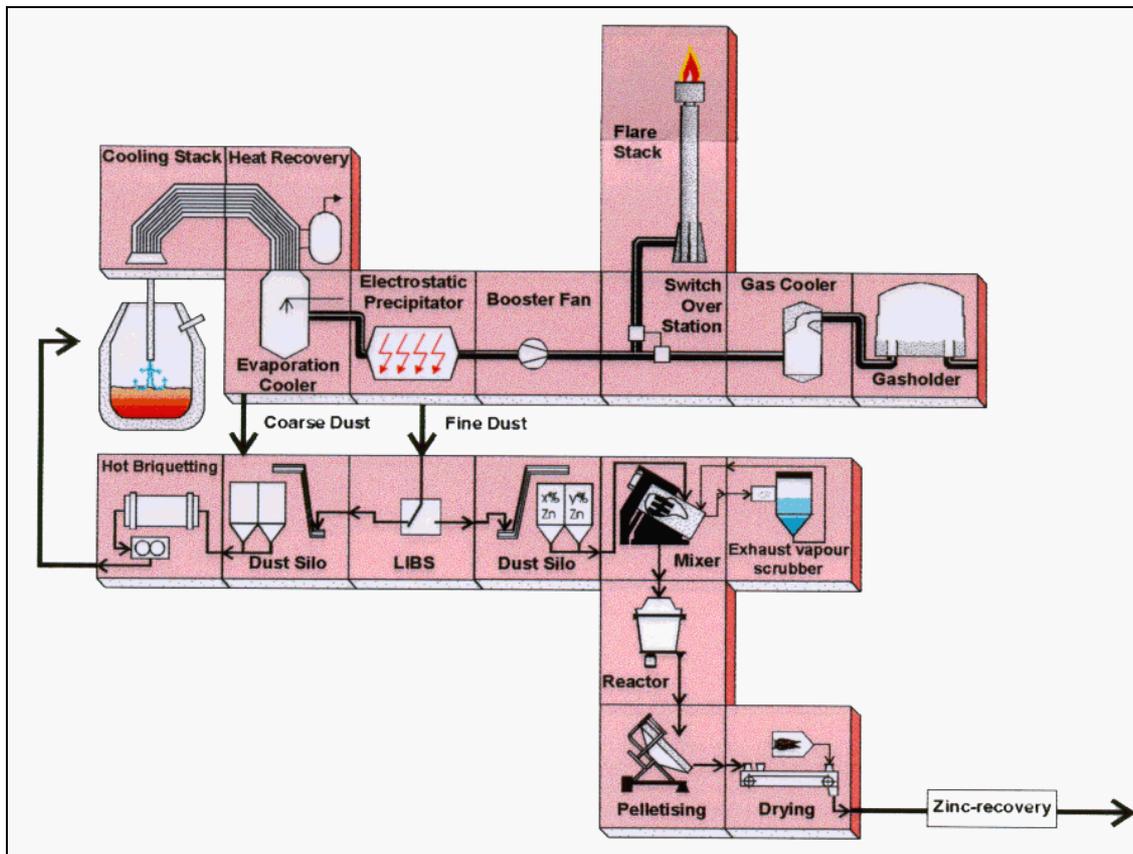


Figure 8.25 : Schematic overview of measures to optimise the recycling of dusts from BOF – [Heiss, 1997]

The reason for pelletising fine dust is that even by employing bonding agents it cannot be briquetted. Moreover, pellets generally better meet customer requirements such as analysis, storage stability, freedom from dust, transportability and ease of handling. Moreover, the dust pellets can be optimised for further processing by adding reducing agents, other additives etc.

Main achieved emission levels: Landfilling of solid wastes is avoided and valuable raw materials are saved. The amount of dust processed is approximately 10 to 20 kg per tonne of liquid steel produced. The overall iron yield is increased by approximately 1 %. A recycling rate of dusts of 100% can be achieved.

Applicability: This method is only applicable when dry electrostatic precipitation is used to clean the BOF gas. In the future it may be possible to treat sludges from venturi scrubbers as well, but this would require extra energy for the evaporation of water.

Cross-media effects: The hot-briquetting plant requires energy but saves raw materials.

Reference plants:

LD 3 Steelmaking plant, Voest-Alpine Stahl AG, A-Linz (at this plant the recovery of zinc in form of pellets for external reuse is practised);

Steel Plant Kwangyang Works, POSCO Iron and Steel Company, Republic of Korea;

Steel Plant Baoshan Iron and Steel Company, People's Republic of China;

Dneprovsky Metallurgical Combine (DMK), Ukraine.

LD 1, LD 2 steelmaking plants, Thyssen Krupp Stahl AG, D-Duisburg

BHP, Newcastle, Australia

Operational data: Not available

Economics: Not available

Driving force for implementation: The main driving forces are limited possibilities and high costs for/of dust disposal.

Reference literature: [Auth, 1988; UN-ECE, 1996; Rentz, 1996; Heiss, 1997]

EP.5 Treatment of wastewater from wet de-dusting

Description: In most oxygen steelmaking plants, scrubbers are used to reduce emissions to air from the primary gas flow (BOF gas) (see PI. 1). This potentially transfers pollution from air to water so the wastewater generated is usually recycled and treated before discharge.

The water from the scrubbers mainly contains suspended solids; Zinc and lead being the main heavy metals present.

A large part of the suspended solids in the scrubbing water circuit can be removed by means of hydrocyclonage and/or precipitation. After pH correction most of the water can be recycled (see also 8.2.2.3.1).

The bleed can be treated by means of precipitation and/or filtration prior to discharge.

Main achieved emission levels:

In Table 8.13 examples are given for the specific emissions to water from wet de-dusting systems at oxygen steelmaking plants.

Parameter		Suppressed combustion systems			Open combustion systems
		Hoogovens Oxygen Steelmaking Plant No.2*	Stelco LEW, Canada	LTV Steel Cleveland Works, USA	Hoogovens Oxygen Steelmaking Plant No.1*
Discharge flow	m ³ /t LS	0.52	1.1	0.002	0.65
Suspended solid	g/t LS	20	5.5	0.0083	9.4
Zinc (Zn)	mg/t LS	73	210	0.36	252
Lead (Pb)	mg/t LS	31	110	0.057	<74

*emissions at Hoogovens relate to 1994 values.

Table 8.13 : Examples for the specific emissions to water from wet de-dusting facilities at oxygen steelmaking plants – [InfoMil, 1997]

Most effective measures to minimise wastewater discharge are:

1. Increasing the recirculation rate of the scrubbing water;
A high recirculation can be achieved by means of a two-stage sedimentation in the scrubbing water flow, with carbon dioxide (CO₂) injection prior to the second sedimentation step, in order to enhance precipitation of carbonates. Note that injection of CO₂ is only possible in systems operating suppressed combustion.
2. Treatment of the bleed;
Although efficient recirculation can be achieved, a bleed is necessary to avoid accumulation of certain minerals/salts. The bleed contains suspended solids (including zinc, lead etc.) as the most important pollutant. The bleed is treated by sedimentation and filtration.

Applicability: A high recirculation efficiency and further treatment can be applied both at new and existing plants.

Cross-media effects: Sludge is generated during hydrocyclonage and/or the sedimentation of the suspended solids in the scrubbing water circuit. This sludge can be 100% recycled within the iron and steelmaking process. However, this is only possible if the zinc input via the scrap is strictly limited. At many other steelmaking plants in the world, the sludge from steelmaking plants cannot be used and is stored or disposed off (see also PI. 2 and EP.4).

Reference plants: High degree of recirculation and treatment of bleed:
Sidmar, B-Gent; Thyssen AG, D-Duisburg; LTV Steel Cleveland Works, USA

Operational data and economics: not available

Reference literature: [Theobald, 1997; InfoMil, 1997]

EP.6 Treatment of wastewater from continuous casting

Description: Water is used in continuous casting machines for direct cooling of the slabs, blooms and billets. A contaminated water flow is therefore generated. In many cases this wastewater is treated together with wastewater streams from the rolling mills. After treatment the water is recirculated.

The casting mould and the inner part of the rollers are usually cooled with water in a closed circuit and is not considered here.

The main pollutants are suspended solids and oil. The main measures to reduce discharges to water are a high rate of recirculation along with sedimentation and/or filtration of the bleed. Skimming tanks can be used to remove oil.

Main achieved emission levels: In Table 8.14 examples are given for specific emissions to water from continuous casting.

Parameter		Hoogovens Cont. Casting Machine OSF 1	Hoogovens Continuous Casting Machine OSF 2*	Stelco Lake Erie Works, Ontario, Canada	Inland Steel, Indiana Harbour Works, IN, USA
Discharge flow	[m ³ /t cast steel]	0.08	0.04	1.4	0.076
Recirculation rate	[%]	?	98	78	99
Suspended solid	[g/t cast steel]	0.8 - 10.7	0.11	26	1.4
Zinc (Zn)	[mg/t cast steel]	-	2.0	-	8.0
Lead (Pb)	[mg/t cast steel]	-	5.7	-	8.7
Oil	[mg/t cast steel]	30 - 365	41	2000	160

Table 8.14 : Overview of specific emissions to water from direct cooling systems at continuous casting – [InfoMil, 1997]

Applicability: A high recirculation rate and treatment of the bleed can be applied both at new and existing plants.

Cross-media effects: The sedimentation steps generate an iron-containing sludge which can be recycled into the sinter plant.

Reference plants:

Inland Steel, Indiana Harbour Works, Indiana, USA.

Hoogovens IJmuiden, NL-IJmuiden.

Sidmar, B-Gent.

Operational data and economics: not available

Reference literature: [InfoMil, 1997]

8.4 Conclusions

In understanding this chapter and its contents, the attention of the reader is drawn back to the preface of this document and in particular the fifth section of the preface: “How to understand and use this document”. The techniques and associated emission and/or consumption levels, or ranges of levels, presented in this chapter have been assessed through an iterative process involving the following steps:

- identification of the key environmental issues for the sector; for basic oxygen steelmaking and casting these are fume collection and treatment, BOF gas recovery and treatment, and zinc in collected dusts;
- examination of the techniques most relevant to address those key issues;
- identification of the best environmental performance levels, on the basis of the available data in the European Union and world-wide;
- examination of the conditions under which these performance levels were achieved; such as costs, cross-media effects, main driving forces involved in implementation of this techniques;
- selection of the best available techniques (BAT) and the associated emission and/or consumption levels for this sector in a general sense all according to Article 2(11) and Annex IV of the Directive.

Expert judgement by the European IPPC Bureau and the relevant Technical Working Group (TWG) has played a key role in each of these steps and in the way in which the information is presented here.

On the basis of this assessment, techniques, and as far as possible emission and consumption levels associated with the use of BAT, are presented in this chapter that are considered to be appropriate to the sector as a whole and in many cases reflect current performance of some installations within the sector. Where emission or consumption levels “associated with best available techniques” are presented, this is to be understood as meaning that those levels represent the environmental performance that could be anticipated as a result of the application, in this sector, of the techniques described, bearing in mind the balance of costs and advantages inherent within the definition of BAT. However, they are neither emission nor consumption limit values and should not be understood as such. In some cases it may be technically possible to achieve better emission or consumption levels but due to the costs involved or cross media considerations, they are not considered to be appropriate as BAT for the sector as a whole. However, such levels may be considered to be justified in more specific cases where there are special driving forces.

The emission and consumption levels associated with the use of BAT have to be seen together with any specified reference conditions (e.g. averaging periods).

The concept of “levels associated with BAT” described above is to be distinguished from the term “achievable level” used elsewhere in this document. Where a level is described as “achievable” using a particular technique or combination of techniques, this should be understood to mean that the level may be expected to be achieved over a substantial period of time in a well maintained and operated installation or process using those techniques.

Where available, data concerning costs have been given together with the description of the techniques presented in the previous chapter. These give a rough indication about the magnitude of costs involved. However, the actual cost of applying a technique will depend strongly on the specific situation regarding, for example, taxes, fees, and the technical characteristics of the installation concerned. It is not possible to evaluate such site-specific

factors fully in this document. In the absence of data concerning costs, conclusions on economic viability of techniques are drawn from observations on existing installations.

It is intended that the general "BAT" in this chapter could be used to judge the current performance of an existing installation or to judge a proposal for a new installation and thereby assist in the determination of appropriate "BAT" based conditions for that installation. It is foreseen that new installations could be designed to perform at or even better than the general "BAT" levels presented here. It is also considered that many existing installations could reasonably be expected, over time, to move towards the general "BAT" levels or do better.

While the BREFs do not set legally binding standards, they are meant to give information for the guidance of industry, Member States and the public on achievable emission and consumption levels when using specified techniques. The appropriate limit values for any specific case will need to be determined taking into account the objectives of the IPPC Directive and the local considerations.

For hot metal pre-treatment, oxygen steelmaking and continuous casting, the following techniques or combination of techniques are considered as BAT. The order of priority and the selection of the techniques will differ according to local circumstances. Any other technique or combination of techniques achieving the same or better performance or efficiency can also be considered; such techniques may be under development or are emerging techniques or are already available but not mentioned/described in this document.

1. Particulate matter abatement from hot metal pre-treatment (including hot metal transfer processes, desulphurisation and deslagging), by means of:

- Efficient evacuation;
- Subsequent purification by means of fabric filtration or ESP.

Emission concentrations of 5-15 mg/Nm³ are achievable with bag filters and 20-30 mg/Nm³ with ESP.

2. BOF gas recovery and primary de-dusting, applying:

- Suppressed combustion *and*
- Dry electrostatic precipitation (in new and existing situations) *or*
- Scrubbing (in existing situations).

Collected BOF gas is cleaned and stored for subsequent use as a fuel. In some cases, it may not be economical or, with regard to appropriate energy management, not feasible to recover the BOF gas. In these cases, the BOF gas may be combusted with generation of steam. The kind of combustion (full combustion or suppressed combustion) depends on the local energy management.

Collected dusts and/or sludges should be recycled as much as possible. Note the usually high zinc content of the dust/sludge. Special attention should be paid to the emissions of particulate matter from the lance hole. This hole should be covered during oxygen blowing and, if necessary, inert gas injected into the lance hole to dissipate the particulate matter.

3. Secondary de-dusting, applying:

- Efficient evacuation during charging and tapping with subsequent purification by means of fabric filtration or ESP or any other technique with the same removal efficiency. Capture efficiency of about 90% can be achieved. Residual dust content of 5-15 mg/Nm³ in case of bag filters and of 20-30 mg/Nm³ in case of ESP can be achieved. Note the usually high zinc content of the dust.

- Efficient evacuation during hot metal handling (reladling operations), deslagging of hot metal and secondary metallurgy with subsequent purification by means of fabric filtration or any other technique with the same removal efficiency. For these operations emission factors below 5 g/t LS are achievable.
Fume suppression with inert gas during reladling of hot metal from torpedo ladle (or hot metal mixer) to charging ladle in order to minimise fume/dust generation.
- 4. Minimisation/abatement of emissions to water from primary wet de-dusting of BOF gas applying the following measures:
 - Dry BOF gas cleaning can be applied when space permits;
 - Recycling of scrubbing water as much as possible (e.g. by means of CO₂ injection in case of suppressed combustion systems);
 - Coagulation and sedimentation of suspended solids; 20 mg/l suspended solids can be achieved.
- 5. Abatement of emissions to water from direct cooling at the continuous casting machines by:
 - Recycling of process and cooling water as much as possible;
 - Coagulation and sedimentation of suspended solids;
 - Removal of oil using skimming tanks or any other effective device;
- 6. Minimisation of solid waste
For solid waste generation, the following techniques are considered BAT in descending order of priority:
 - Minimising waste generation
 - Effective utilisation (recycling or reuse) of solid wastes/by-products; especially recycling of BOF slag and coarse and fine dust from BOF gas treatment
 - Controlled disposal of unavoidable wastes

In principle the techniques according to items 1 - 6 are applicable to new as well as to existing installations (if there are no other indications and mentioned prerequisites are met) considering the preface.

8.5 Emerging techniques and future developments

The following techniques have been identified as emerging techniques:

- Near net-shape casting and horizontal casting;
- Processing of Zn-rich sludges/dusts;
- New reagents in the desulphurisation process;
- Application of foaming techniques in pig iron pre-treatment and steel refining;
- Replacing air above the hot metal by inert gases (CO₂, N₂).

Near net-shape casting and horizontal casting

Description: Developments in continuous casting are still taking place. Near-net shape casting and horizontal casting are attractive processes for commercial application. These processes are able to connect directly with the downstream hot rolling process and hence have some advantages over conventional continuous casting of steel slabs and billets.

Status: already applied on a commercial scale at several plants worldwide. This technique should therefore be treated as an available technique in future references.

Main achievements: These modern types of continuous casting will entail smaller investments, simpler manufacturing processes, less energy consumption and labour savings. In addition, horizontal casting does not need the tall buildings currently needed for the conventional continuous casting process.

Reference literature: [UBA Comments, 1997]

Processing of Zn-rich sludges/dusts

Description: Zinc-rich sludges and dusts are generated during BOF gas cleaning and blast furnace gas cleaning. Nevertheless, the zinc content is not high enough to make reuse economic. Only some of these sludges and dusts can be recycled, so almost all iron and steel plants have a large deposit of zinc-rich sludges and wastes. In EP.4 hot briquetting and generation of pellets with high zinc content for external reuse are described. Technically it is possible to extract the non-ferrous metals from this sludge and dust, after which the valuable iron-containing "cleaned" solids can be recycled into the iron-making process. The non-ferrous metals extracted can be further processed by the non-ferrous metals industry. The reason that no method has been applied commercially until now is the high cost of the sludge/dust processing.

The following methods can be applied (in different stages of development):

- Rotating hearth furnace process (Inmetco);
- Fluidised bed process (Thyssen);
- Circulating fluidised bed reactor;
- High turbulence mixer process;
- Plasma process (Siromelt, Plasmelt);
- Multi-role oxygen cupola furnace.

Status: External processes for zinc-rich sludge/dust exist on a commercial scale [UBA Comments, 1997].

Reference literature: [Köller, 1995; UN-ECE, 1996; Rentz, 1996; EUROFER BOF, 1997]

New reagents in the desulphurisation process

Description: The use of new reagents in the desulphurisation process might lead to a decrease in particulate matter emissions and a different (more useful) composition of the generated dusts.

Status: Under development

Reference literature: [EC BOF, 1995]

Application of foaming techniques at pig iron pre-treatment and steel refining

Description: Foaming techniques are preferred for pig iron pre-treatment because the foam absorbs the particulate matter arising from the hot metal processing.

Status: Several foaming techniques are already available

Reference literature: [EC BOF, 1995]

Replacing air above the hot metal by inert gases (CO₂, N₂)

Description: Reducing the O₂ concentration above the hot metal during pig iron pre-treatment reduces the generation of oxides and thus particulate matter. The O₂ can be dissipated by means of an inert gas, such as CO₂ or N₂.

Status: Industrial-scale tests using CO₂ as inert gas have been carried out during pig iron tapping from the torpedo car into the steel ladle (Luxembourg) and during charging of pig iron into the BOF (France). In Germany tests have been carried out using N₂ as inert gas.

Cross-media effects: The use of N₂ may lead to emissions of NO_x.

Reference literature: [EC BOF, 1995]

9 ELECTRIC STEELMAKING AND CASTING

9.1 Applied processes and techniques

The direct smelting of iron-containing materials, such as scrap is usually performed in electric arc furnaces (EAF) which play an important and increasing role in modern steel works concepts (see Figure 1.2). Today the percentage of electric arc furnace steel of the overall steel production in the EU are 35.3 % [Stat. Stahl, 1997]. In Italy and Spain the production of electric arc furnace steel is significantly higher than steel production via the blast furnace-basic oxygen furnace route (not considering Member States having exclusively electric arc furnace steel production).

The major feed stock for the EAF is ferrous scrap, which may comprise of scrap from inside the steelworks (e.g. offcuts), cut-offs from steel product manufacturers (e.g. vehicle builders) and capital or post-consumer scrap (e.g. end of life products). Direct-reduced iron (DRI) is also increasingly being used as a feedstock due both to its low gangue content and variable scrap prices. As in the BOF, a slag is formed from lime to collect undesirable components in the steel.

Figure 9.1 shows an electric arc furnace plant. In this case the building containing a twin shell EAF is totally enclosed in order to minimise dust, gaseous and noise emissions.



Figure 9.1 : Electric arc furnace plant

An electric arc furnace with three electrodes and a shaft for scrap charging can be seen from Figure 9.2.

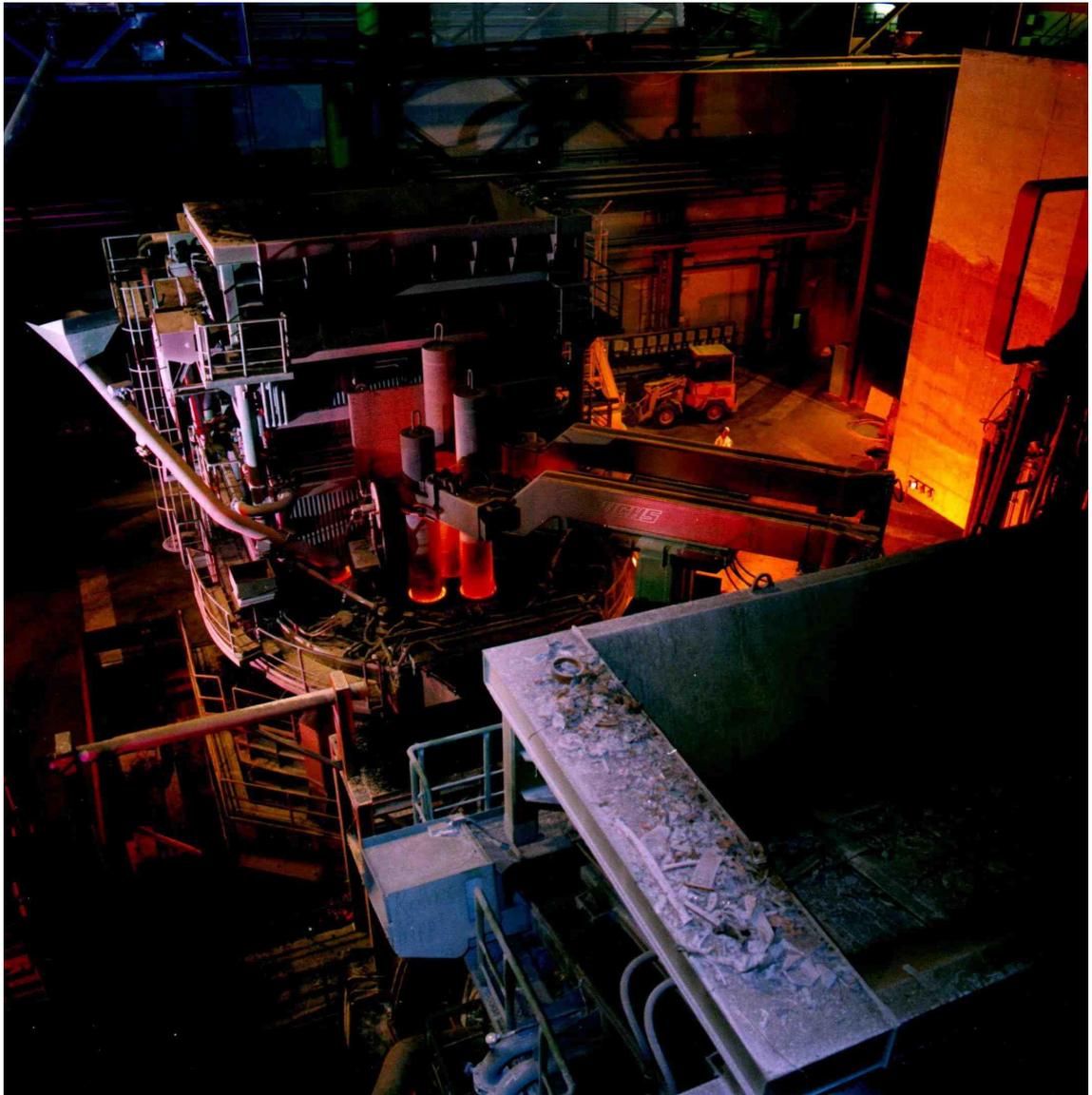


Figure 9.2 : Electric arc furnace with three electrodes and (in the foreground) a shaft for scrap charging

An overview of the electric steel making process is given in Figure 9.3.

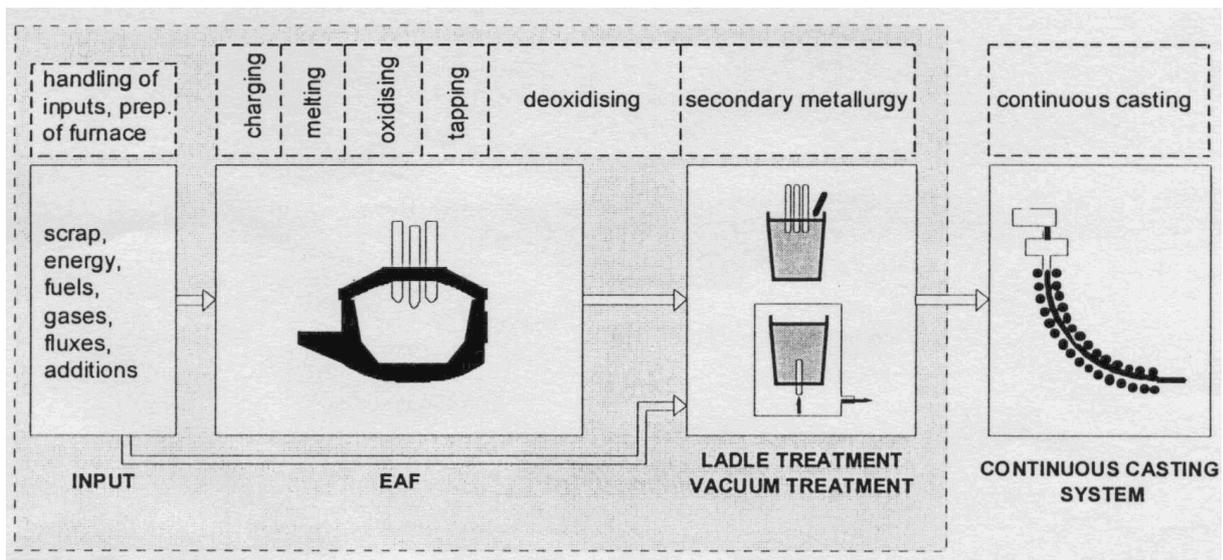


Figure 9.3 : Overview of the processes related to electric arc furnace steelmaking - [D Rentz, 1997]

With respect to the end-products distinction has to be made between production of ordinary, so-called carbon steel as well as low alloyed steel and high alloyed steels/stainless steels. In the EU about 85% of steel production is carbon or low alloyed steel [EC Study, 1996].

For the production of carbon steel and low alloyed steels, following main operations are performed:

- raw material handling and storage
- furnace charging with/without scrap preheating
- EAF scrap melting
- steel and slag tapping
- ladle furnace treatments for quality adjustment
- slag handling
- continuous casting

For high alloyed and special steels, the operation sequence is more complex and tailor-made for the end-products. In addition to the mentioned operations for carbon steels various ladle treatments (secondary metallurgy) are carried out like

- desulphurisation
- degassing for the elimination of dissolved gases like nitrogen and hydrogen
- decarburisation (AOD=Argon-Oxygen-Decarburisation or VOD=Vacuum-Oxygen-Decarburisation)

9.1.1 Raw materials handling and storage

The main scrap storage areas are usually outside in large uncovered and unpaved scrap-yards which may lead to soil pollution but there are also certain plants having covered and paved scrap-yards. Depending on weather conditions volatile inorganic and organic compounds may be emitted.

Today control of radioactivity of input scrap has become an important issue but this relevant problem is not covered by this document.

Some scrap sorting is carried out to reduce the risk of including hazardous contaminants. In-house generated scrap can be cut into handleable sizes using oxygen lancing. The scrap may be loaded into charging baskets in the scrap-yard or may be transferred to temporary scrap bays inside the melting shop. In some cases the scrap is preheated in a shaft or on a conveyor (see scrap preheating).

Other raw materials including fluxes in lump and powder, powdered lime and carbon, alloying additions, deoxidants and refractories are normally stored under cover. Following delivery, handling is kept to a minimum and where appropriate, dust extraction equipment may be used. Powdered materials can be stored in sealed silos (lime should be kept dry) and conveyed pneumatically or kept and handled in sealed bags.

9.1.2 Scrap preheating

In the past years more and more new as well as existing EAF have been equipped with a system for preheating the scrap by the off gas in order to recover energy. Today the so-called shaft technology and the Consteel Process are the two proven systems which have been successfully introduced into practice [Haissig, 1997].

The shaft technology has been developed in steps [Voss-Spilker, 1996]. With the single shaft furnace normally only about half of the charged scrap can be preheated, meanwhile with the finger shaft furnace (which means a shaft having a scrap retaining system) 100% of scrap can be preheated. The first basket is preheated during refining of the previous heat and the second during melt down of the first one. A further modification is the double shaft furnace which consists of two identical shaft furnaces (twin shell arrangement) are positioned next to each other and are serviced by a single set of electrode arms. The scrap is partly preheated by off gas and partly by side wall burners.

To date (October 1998) more than 20 shaft furnaces are worldwide in operation, eight in Europe.

Scrap preheating may result in higher emissions of aromatic organohalogen compounds such as polychlorinated dibenzo-p-dioxins and -furans (PCDD/F), chlorobenzenes, polychlorinated biphenyls (PCB) as well as polycyclic aromatic hydrocarbons (PAH) and other partial combustion products from scrap which is contaminated with paints, plastics, lubricants or other organic compounds. This formation can be minimised by post-combustion within the furnace by additional oxygen burners. They have been developed in order to (post-)combust CO (and hydrocarbons). Thus the chemical heat from this combustion can also be used for scrap preheating [Knapp, 1996]. But such a post-combustion is different from a post-combustion after the EAF in order to reduce emissions of organic compounds like PCDD/F etc. Such a post-combustion requires considerable quantity of energy.

9.1.3 Charging

The scrap is usually loaded into baskets together with lime or dolomite which is used as a flux for the slag formation. Lump coal is also charged at some plants with the result of relevant benzene (as well as toluene and xylenes) emissions. The furnace electrodes are raised in top position, the roof is then swung away from the furnace for charging. It is normal to charge about 50-60% of the scrap initially with the first scrap basket; the roof is then closed and the electrodes lowered to the scrap. Within 20-30 mm above the scrap they strike an arc. After the first charge has been melted the remainder of the scrap is added from a second or third basket.

A proprietary available system is known as the shaft furnace which allows part of the scrap to be preheated by charging it through a vertical shaft integrated in the furnace roof (see scrap preheating) [Voss-Spilker, 1996].

Another new charging systems have been developed. In the Consteel Process the scrap is continuously fed via a horizontal conveyor system into the arc furnace [Vallomy, 1992]. But this system is not generally considered as a proven technique.

9.1.4 Arc furnace melting and refining

During the initial period of melting, the applied power is kept low to prevent damage to the furnace walls and roof from radiation whilst allowing the electrodes to bore into the scrap. Once the arcs have become shielded by the surrounding scrap the power can be increased to complete melting. Oxygen lances and/or oxy-fuel burners are more and more used to assist in the early stages of melting. Fuels include natural gas and oil. Furthermore oxygen may be brought to the liquid steel by specific nozzles in the bottom or side wall of the EAF.

Oxygen in electric furnace steelmaking has found increasing considerations within the last 30 years not only for metallurgical reasons but also for increasing productivity requirements. The increase of oxygen usage can be attributed to today's availability of liquid oxygen and on-site oxygen plants [Knapp, 1996].

Oxygen for metallurgical reasons is used for decarburisation of the melt and removal of other undesired elements such as phosphorous, manganese, silicon and sulphur. In addition it reacts with hydrocarbons forming exothermic reactions. Oxygen injection result in a marked increase in gas and fume generation from the furnace. CO and CO₂ gases, extremely fine iron oxide particles and other product fume are formed. In case of post-combustion the CO content is below 0.5 vol%.

Argon or other inert gases may be injected into the melt to provide bath agitation and temperature balancing. The slag-metal equilibrium is also improved by this technique.

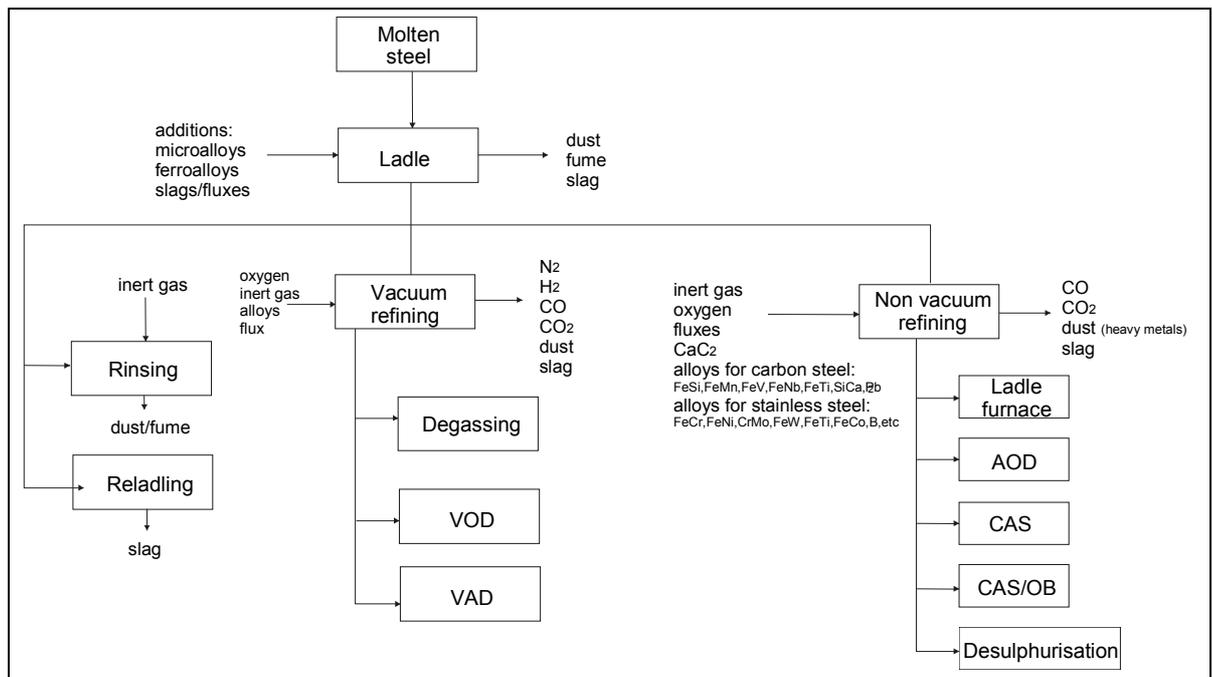
9.1.5 Steel and slag tapping

In plants without separate secondary metallurgy facilities, alloying elements and other additions are often given to the steel ladle before or during tapping. Such additions can noticeably increase the fume arising during tapping. Slag may need to be removed during heating and oxidising at the end of the heat prior to tapping. The furnace is tilted backwards towards the slagging door and the slag run off or raked into a pot or on the ground below the furnace resulting in dust and fume generation.

Today the steel is normally tapped through a bottom tapping system with minimum slag carry over into the ladle.

9.1.6 Secondary metallurgy

Secondary metallurgy which is carried out in ladles covers the processes and treatment of molten steel after the tapping of the primary steel making furnace up to the point of casting. It is typically carried out at ladle treatment stations. These stations in bulk steel production plants are usually located around a vacuum generation system or arc heating unit. Other minor stations are based on inert gas or powder injection equipment. These processes are shown schematically in Figure 9.4.



Legend: VOD = Vacuum-Oxygen-Decarburisation; VAD = Vacuum-Arc-Degassing; AOD = Argon-Oxygen-Decarburisation; CAS = Compositional Adjustment by Sealed Argon Bubbling; CAS/OB = CAS plus Oxygen Blowing

Figure 9.4: Secondary metallurgy/ladle treatments - [UK EAF, 1994]

In case of production of leaded steel, off gases containing lead have to undergo special treatment (see information under secondary metallurgy in basic oxygen steelmaking – 8.1.4 and 8.2.2.1.1.3)

9.1.7 Slag handling

Beside slag tapping further dust and fume are created during retrieval of the slag which may still be hot, using excavators. Outside the furnace building the slag may be cooled by water spraying before it is crushed and screened to allow metal recovery. In case of slag with free lime-alkaline fumes may be emitted. Slag breaking (or in some cases cutting with oxygen lances) and metal recovery can create dust emissions.

9.1.8 Continuous casting

The liquid steel is usually cast continuously. Ingot casting is also still applied for some grades and applications. Continuous casting is a process which enables the casting of one or a sequence of ladles of liquid steel into a continuous strand of billet, bloom, slab, beam blank or strip (see also 8.1.5.1). Steel is tapped from the ladle into a tundish from which it is distributed at a controlled rate into water-cooled copper moulds of appropriate dimensions.

To prevent the solidified shell from sticking, the mould is oscillated in the direction of casting at speed greater than the casting speed and a mould lubricant is added in powder form or vegetable oil. The strand is continuously withdrawn and is further cooled using direct water sprays. At a point where solidification is complete the strand is cut to required lengths using automatic oxy-gas cutters. In case of oxygen cutting or hydraulic shears of stainless steel iron powder injection is employed.

9.2 Present consumption and emission levels

9.2.1 Mass stream overview and input/output data

Figure 9.5 provides an overview for the input and output of electric arc furnaces. This overview may be used for the collection of data from an electric arc furnace.

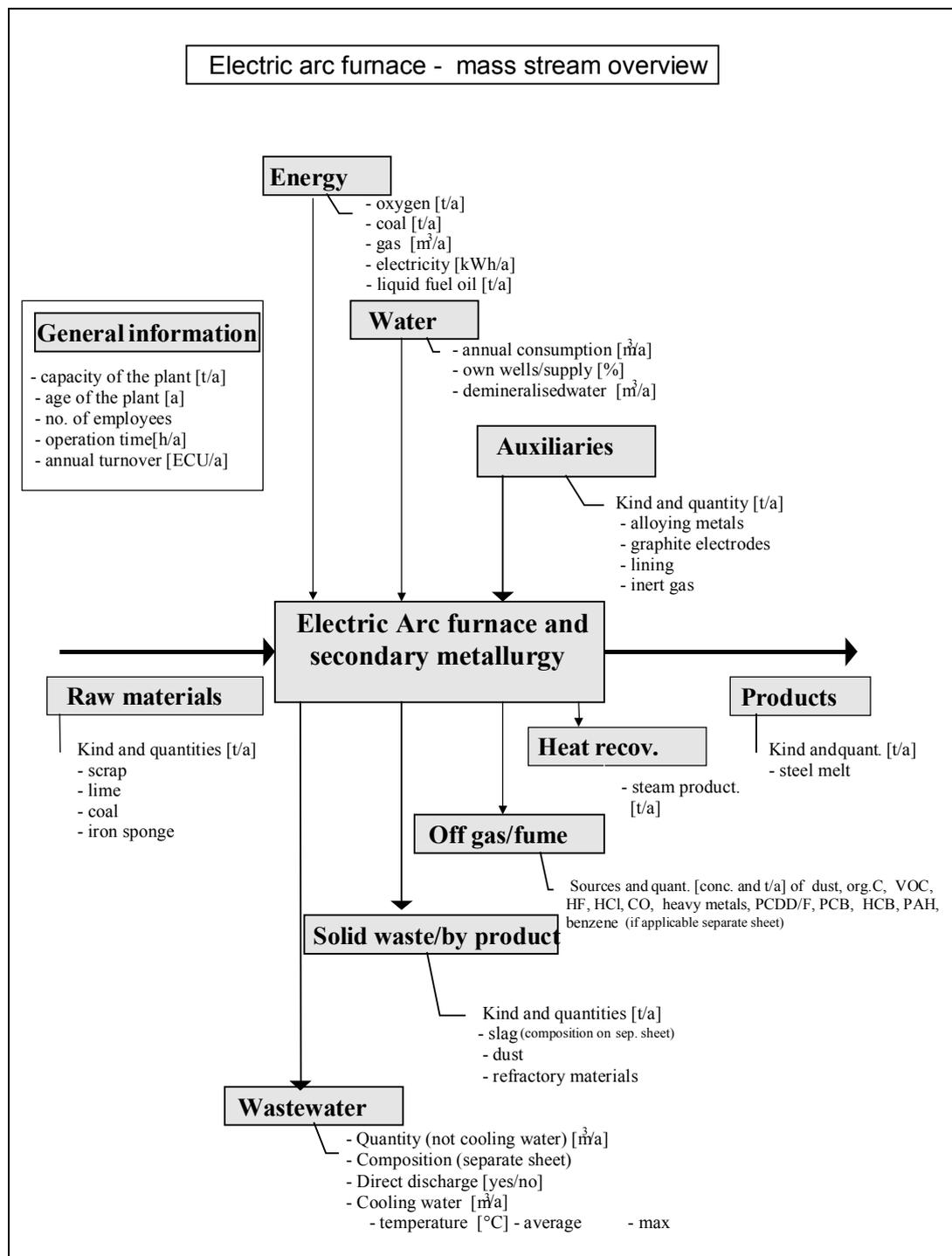


Figure 9.5 : Mass stream overview of an electric arc furnace

Subsequently specific input factors as well as specific emission factors can be calculated. Such factors are presented in Table 9.1. The data derive from various sources mentioned in the footnotes.

Input			Output		
Raw materials			Products		
scrap	kg/t	1080 - 1130	liquid steel (LS)	kg	1000.00
lime	kg/t	30 - 80	Emissions ^{*3}		
coal	kg/t	13 - 15	dust	g/t	1-780 ^{*4}
graphite electrodes	kg/t	1.5 - 4.5	Hg	mg/t	6 - 4470 ^{*5}
lining	kg/t	1.9 - 25.1 (aver. 8.1)	Pb	mg/t	16 - 3600 ^{*6}
liq. hot metal ^{*1}	kg/t		Cr	mg/t	8 - 2500 ^{*7}
DRI ^{*2}	kg/t		Ni	mg/t	1 - 1400 ^{*8}
pig iron ^{*2}	kg/t		Zn	mg/t	280-45600 ^{*9}
			Cd	mg/t	<1-72 ^{*10}
			Cu	mg/t	<1-460 ^{*11}
Energy			HF	mg/t	<700-4000 ^{*12}
total energy	MJ/t	2300 - 2700	HCl	mg/t	800-9600 ^{*12}
electricity	MJ/t	1250 - 1800	SO ₂	g/t	24-130 ^{*12}
oxygen	m ³ /t	24 - 47	NO _x	g/t	120-240 ^{*13}
			CO	g/t	740-3900 ^{*12}
Water		closed cooling cycle	TOC	g C/t	16-130 ^{*14}
			benzene	mg/t	170-4400 ^{*12,15}
			chlorobenzenes	mg/t	3-37 ^{*16}
			PAH ^{*17}	mg/t	3.5-71 ^{*18}
			PCB ^{*19}	mg/t	1.5-45 ^{*20}
			PCDD/F	µg I-TEQ/t	0.07-9 ^{*21}
			Solid wastes/by-pro.		
			slag from furnace	kg/t	100 - 150
			slag from ladle	kg/t	10 - 30
			dusts	kg/t	10 - 20
			refractory bricks	kg/t	2 - 8
			Noise	dB(A)	90 - 125

Legend: LS = liquid steel

^{*1}	hot metal is only used in very special cases (about 275 kg/t LS), then the quantity of scrap is lower
^{*2}	DRI (direct reduced iron) and pig iron are only used in special cases
^{*3}	in case of concentrations are only available, emission factors are calculated with 8000 Nm ³ /t LS [TWG, 1998]; in practice this specific flow can vary considerably from 6000 to 16000 Nm ³ /t LS which has to be considered
^{*4}	according to [EC Study, 1996] the average value and standard deviation for dust emissions (primary and secondary) of 38 plants is 124±166 g/t LS, plants with well-designed bag filters can achieve dust emission factors of <20 g dust/t LS [Theobald, 1995; UBA-BSW, 1996]; in first order heavy metal emissions directly correlate with the residual content of particulates in the off-gas (except heavy metals in the gas phase like Hg)
^{*5}	Hg emissions can strongly vary from charge to charge, data from [Theobald, 1995; UBA-BSW, 1996] for 4 German plants (average is 370 Hg mg/t LS); data from one Danish plant [DK EAF, 1997](average: 150 mg Hg/t LS); data from [Lindblad, 1998](16 measurements in 1994-96 with an average of 6 mg Hg/t LS)
^{*6}	data from 4 German plants [Theobald, 1995; UBA-BSW, 1996] (average is 450 mg Pb/t LS); data from one Danish plant [DK EAF, 1997](average: 700 mg Pb/t LS)
^{*7}	data from 4 German plants [Theobald, 1995; UBA-BSW, 1996] (average is 400 mg Cr/t LS)
^{*8}	data from 4 German plants [Theobald, 1995; UBA-BSW, 1996] (average is 140 mg Ni/t LS); data from one Danish plant [DK EAF, 1997](average: 280 mg Ni/t LS)
^{*9}	data from 4 German plants [Theobald, 1995; UBA-BSW, 1996] (average is 11400 mg Zn/t LS); data from one Danish plant [DK EAF, 1997](average: 5550 mg Zn/t LS)
^{*10}	data from [Theobald, 1995]: < 1 - 72 mg Cd/t LS (average: 16 mg Cd/t LS); data from [UBA-BSW, 1996]: 8 measurem. with 4-37 mg Cd/t LS (average: 25 mg Cd/t LS), one extreme value (180 mg Cd/t LS) was not taken into consideration; data from one Danish plant [DK EAF, 1997](average: 40 Cd mg/t LS)
^{*11}	the average from 4 plants is 80 mg Cu/t LS [Theobald, 1995]
^{*12}	data from 1 German plant (9 measurements) [UBA-BSW, 1996]
^{*13}	data from 1 German plant (9 measurements) [UBA-BSW, 1996]; data from Sweden [Lindblad, 1998] from several EAFs (17 measurements from 1885-93: 22 - 680 g NO ₂ / t LS)
^{*14}	TOC=total organic carbon; data from [Werner, 1997; Theobald, 1995]
^{*15}	9 measurements (average: 1920 mg benzene/t LS); benzene may derive from the coal input (degassing)
^{*16}	data from [Lindblad, 1992]: 20 measurem. from 9 plants (average: 22 mg/t LS) incl. all chlorob. except monochlorob.
^{*17}	sum of EPA 16
^{*18}	data from [Werner, 1997]: 9 measurements with 3.5-71 mg PAH/t (average: 35 mg PAH/t LS); data from [Lindblad, 1992]: 13 measurements from 7 plants (values: 8/23/84/120/180/240/920 mg PAH/t LS)
^{*19}	PCB as total PCB, calculated from (ΣPCB 28+52+101+153+138+180) x 5 (according to [UN-ECE, 1997])
^{*20}	data from [UBA-BSW, 1996]: 9 measurements with 1.5-16 mg PCB/t LS (average: 7.8 mg PCB/t LS); data from [Werner, 1997]: 9 measurements with 2-45 mg PCB/t LS (average: 17 mg PCB/t LS)
^{*21}	data from 8 Swedish plants: 0.2-9 µg I-TEQ/t with an average of 4 µg I-TEQ/t LS [Lindblad, 1992]; data from 4 German plants: 0.07-1.8 µg I-TEQ/t LS [Theobald, 1995]; data from [LUA NRW, 1997]: 0.3-5.7 µg I-TEQ/t LS; data from 1 Danish EAF with an average of 1.7 µg I-TEQ/t LS [EC EAF, 1997]

Table 9.1 : Input/output data for electric arc furnaces for the production of carbon steel compiled from various references indicated in the footnotes

9.2.2 Information about the single emission mass streams as well as about noise emissions and energy demand

The following emissions of off gases, solid wastes/by-products and wastewater can be recognised in electric arc furnace steelmaking.

9.2.2.1 Off gas emissions

9.2.2.1.1 Primary off gases

9.2.2.1.1.1 Off gas directly collected from the EAF

9.2.2.1.1.2 Off gas directly collected from secondary metallurgy processes

9.2.2.1.2 Secondary off gases from scrap handling and charging, steel tapping, secondary metallurgy with tapping operations and from continuous casting

9.2.2.1.3 Fumes from slag processing

9.2.2.2 Solid wastes/by-products

9.2.2.2.1 Slags from production of carbon steel/low alloyed steel/high alloyed steels

9.2.2.2.2 Dusts from off gas treatment

9.2.2.2.3 Refractory bricks

9.2.2.3 Wastewater emissions from

9.2.2.3.1 Drainage water from scrap-yard

9.2.2.3.2 Off gas scrubbing (exceptional)

9.2.2.3.3 Continuous casting

9.2.2.4 Soil contamination

9.2.2.5 Noise emissions

9.2.2.1 Off gas emissions

9.2.2.1.1 Primary off gases

9.2.2.1.1.1 Off gas directly collected from the EAF

Primary off gases represent approximately 95% of total emissions from an EAF [EC EAF, 1994]. Most of the existing plants extract the primary emissions by the 4th hole (in case of three electrodes) or by the 2nd hole (in case of one electrode) (Figure 9.6). Thus 85 – 90% of the total emissions during a complete cycle “tap-to-tap” can be collected [EC EAF, 1994]. There are still very few plants which do not have a 4th hole but only a dog house [EC Study, 1996]. More than 50% of the EAF in the EU have, in addition to the 4th hole, a system for evacuation the building atmosphere, especially hoods (see Figure 9.6).

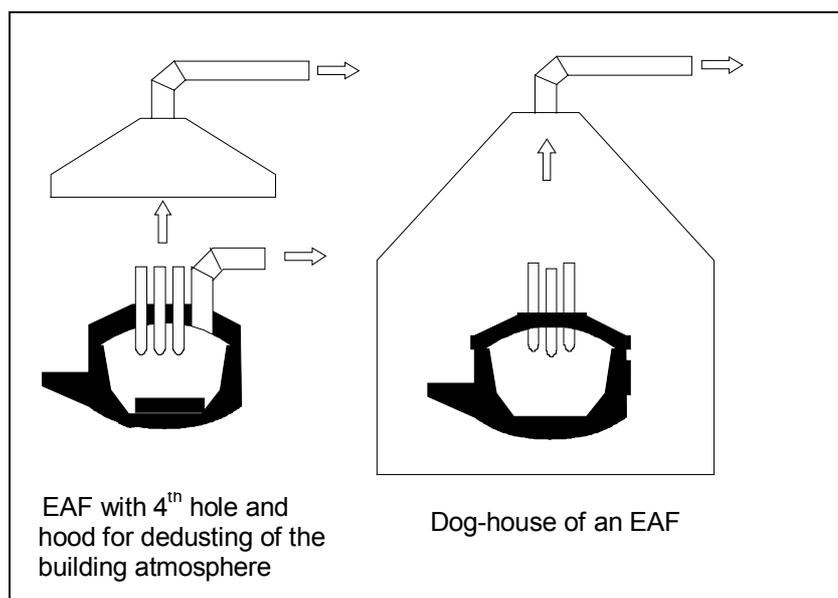


Figure 9.6 : Dust collection systems at EAF – based on [D Rentz, 1997]

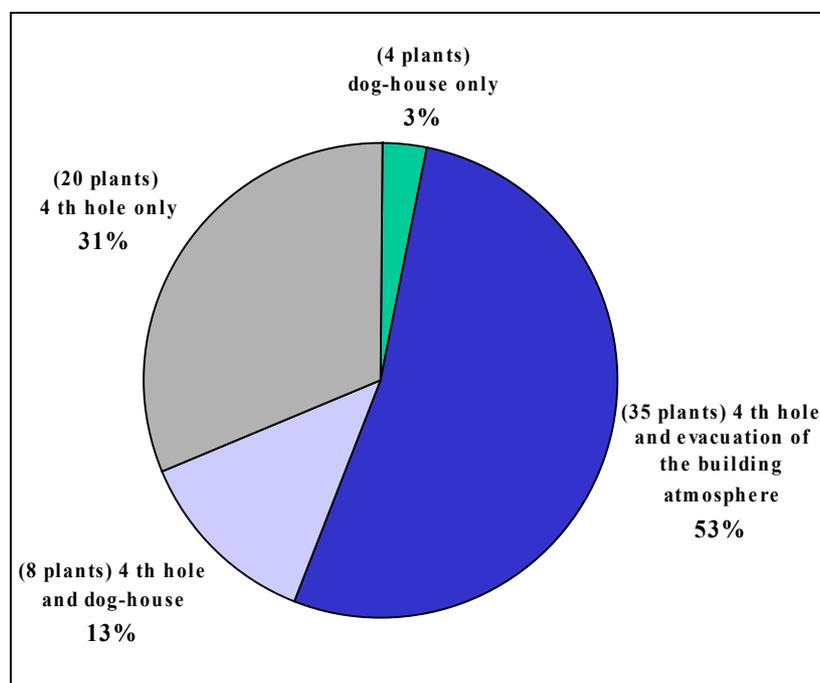
In this way also most of secondary emissions from charging and tapping as well as from EAF leakages during melting can be captured. If secondary metallurgy is carried out in the same building also these emissions can be collected. Very often the treatment of primary and secondary emissions are performed in the same device, mostly in bag filters. Table 9.2 summarises the qualitative efficiencies to collect emissions from the main operations of electric arc furnace steelmaking.

Sources of emissions	Scrap handling and tapping ^{*1}	Melting (in EAF)	Secondary metallurgy ^{*2}	Charging	Continuous casting ^{*1}
Collection System					
Fourth hole	no	yes	yes, if also equipped	no	no
Hood	yes, partly	yes	yes, if also equipped	Yes, partly	yes, partly
Dog-house	no	yes	yes, if also equipped	Only if closed ^{*3}	no
Total building evacuation	yes	yes	Yes	yes	yes

^{*1} if located in the same building; ^{*2} if secondary metallurgy is performed in separate ladles; ^{*3} usually the dog-houses are not closed during charging as they obstruct the access to the EAF

Table 9.2 : Systems for the collection of emissions from EAF plants – based on [EC EAF, 1994]

Figure 9.7 shows the percentages of the four existing emission collection systems in the EU, indicating that one third of the plants only have a 4th hole for the collection of primary emissions.



Legend: 4th hole = additional hole in the roof of an EAF in order to extract the off gas; in case of single electrode (usually there are three) this additional hole is called 2nd hole; dog-house = enclosure of the EAF;

Figure 9.7 : Percentages of existing dust collection systems in 67 EAF in the EU – [EC Study, 1996]

The primary off gas contain 14 – 20 kg dust/t liquid carbon/steel or low alloyed steel and 6 –15 kg dust/t in case of high alloyed steel [EC EAF, 1994]. The composition of the dust can be seen from the analysis of the dust separated from the off gas in the bag filters or electrostatic precipitators (ESP) (see Table 9.6). The heavy metals, especially mercury, which are present in the gas phase, are not associated with particulate matter. Thus they can not be eliminated by filtration or ESP. However most of the heavy metals are mainly associated with particulate matter and are removed from the off-gas with the separated dust.

The range of dust emission factors after abatement can be seen from Table 9.1. The range (1-780 g/t LS) is extremely wide indicating big difference in collection and abatement efficiency. In terms of concentration the emissions of most of the plants are around or below 10 mg dust/Nm³ but there are also plants with about 50 mg/Nm³ [EC Study, 1996]. Normally these emission factors or emission concentrations include secondary dust emissions because primary and secondary emissions are very often treated in the same equipment.

Heavy metals

Some emissions also show wide ranges (see Table 9.1). Higher values can be of high environmental relevance. Zinc is the metal with the highest emission factors. Mercury emissions can strongly vary from charge to charge depending on scrap composition/quality [Theobald, 1995; UBA-BSW, 1996]. The SO₂ emissions mainly depend on the quantity of coal and oil input but are not of high relevance. NO_x emissions also do not need special consideration.

VOC

VOC emissions, especially benzene can be remarkably high and correspond with the use of coal which degasses before being burnt off, especially when it is added as a 'nest' to the scrap basket. It can be expected, that benzene emissions correlate with the emission of toluene, xylenes and other hydrocarbons deriving from coal degassing.

In the nineties increasing note has been taken of organic pollutants. Only for a limited number of compounds analytical results are available. Organochlorine compounds, such as chlorobenzenes, PCB and PCDD/F have been measured.

Chlorobenzenes

Chlorobenzenes have been determined in Swedish EAF (1 – 37 mg/t LS – see Table 9.1). From one German plant it is known that hexachlorobenzene is present in the emitted off gas [UBA-BSW, 1996].

Polychlorinated biphenyls (PCB)

The measured PCB emissions vary considerably (15 - 45 mg/t LS – see Table 9.1). They are of environmental relevance. It is not known yet, whether PCB can be formed de novo during the process and/or within the off gas devices (like it is the case in sinter plants – see 4.2.2.4.2.10 and [Blahe, 1995; Scholz, 1997]. PCB are present in the scrap input which could be the dominant source for the measured emissions [Schiemann, 1995]. Especially PCB in small capacitors in several technical devices like washing machines, (hair)driers, cooker hoods, oil burners, fluorescent lamps etc. in the (shredded) scrap represent the main PCB input [Schiemann, 1995]. The so-called light fraction (if used as an input) can contain up to 140 ppm PCB (sum of all PCB congeners). One investigation showed that PCB emissions remained unchanged before and after a bag filter which achieved low residual dust concentrations (< 5 mg/Nm³) as daily mean value [Werner, 1997].

Polychlorinated dibenzo-p-dioxins and furans (PCDD/F)

Regarding PCDD/F there are many measurements available showing emission factors between 0.07 – 9 µg I-TEQ/t LS (see Table 9.1). Figure 9.8 presents an example of the distribution of PCDD/F homologues in the raw and cleaned off gas of an EAF.

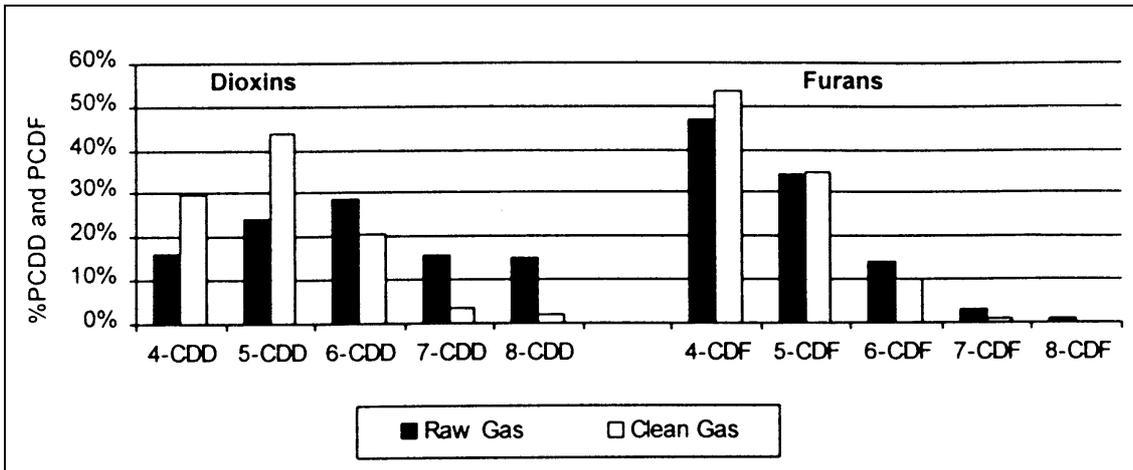


Figure 9.8 : Distribution of PCDD/F homologues in the off gas of a twin shell EAF with scrap preheating before and after abatement – [Werner, 1997]

The PCDD/F homologues with four and five chlorine atoms dominate. There is no reliable information available telling whether the input of PCDD/F or the de novo synthesis mainly cause the PCDD/F emissions. With respect to the absolute PCDD/F emissions there is a positive correlation between off gas temperature (Figure 9.9) and dust content (Figure 9.10).

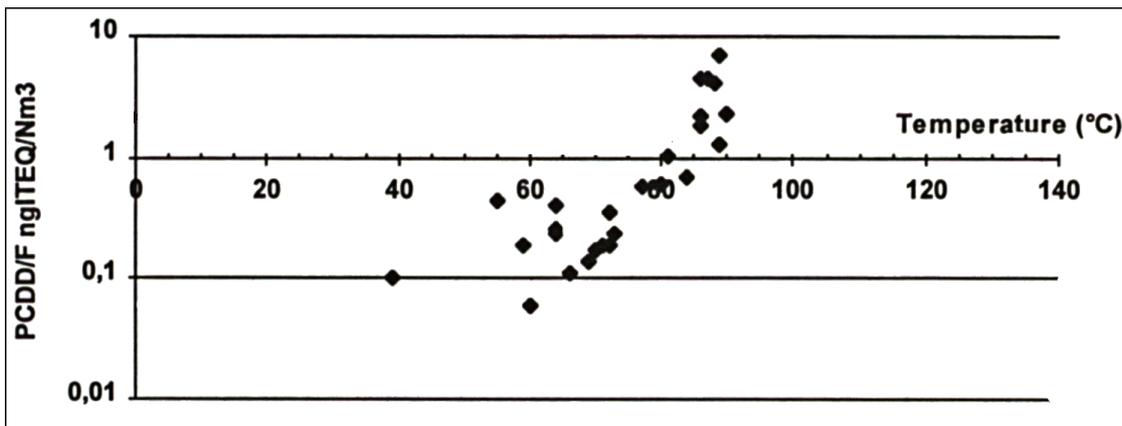


Figure 9.9 : Correlation of PCDD/F emissions and off gas temperature (after abatement in a bag filter) in the off gas of an EAF – [Werner, 1997]

Figure 9.9 indicates that as long as the clean gas temperature is below 75 °C PCDD/F emissions will stay below 1 ng I-TEQ/ Nm³. The physical explanation of this pertains to the decrease of volatility of PCDD/F with decreasing temperature [Spencer, 1992]. At low temperatures PCDD/F increasingly tend to adsorb to the filter dust.

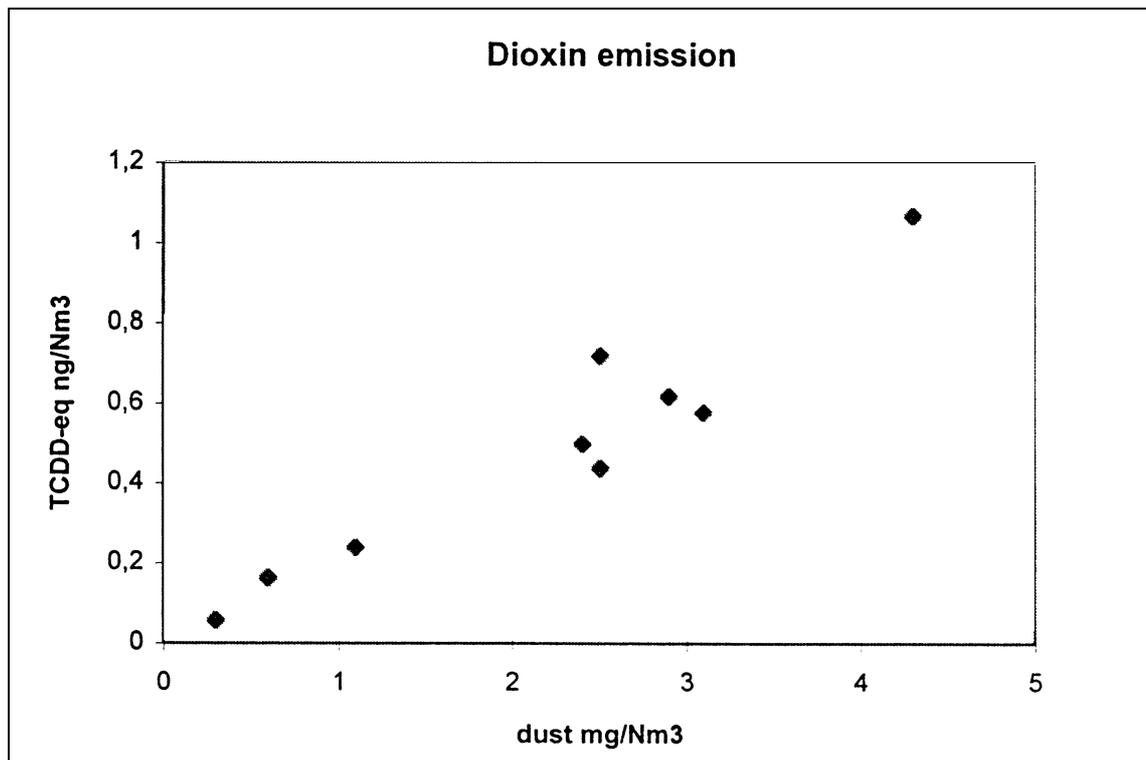


Figure 9.10 : Correlation of residual dust content and PCDD/F concentrations (after abatement in a bag filter) in the off gas of an EAF at temperatures below 85°C – [EC EAF, 1997; Pedersen, 1996]

The observation that there is a close connection between dust and PCDD/F emissions has to be related to off gas temperature. The dust content itself mainly depend on the dimension and quality of the bag filter but also on the relative humidity in the off gas which can be high in case of off gas quenching or location near to the sea (Figure 9.11).

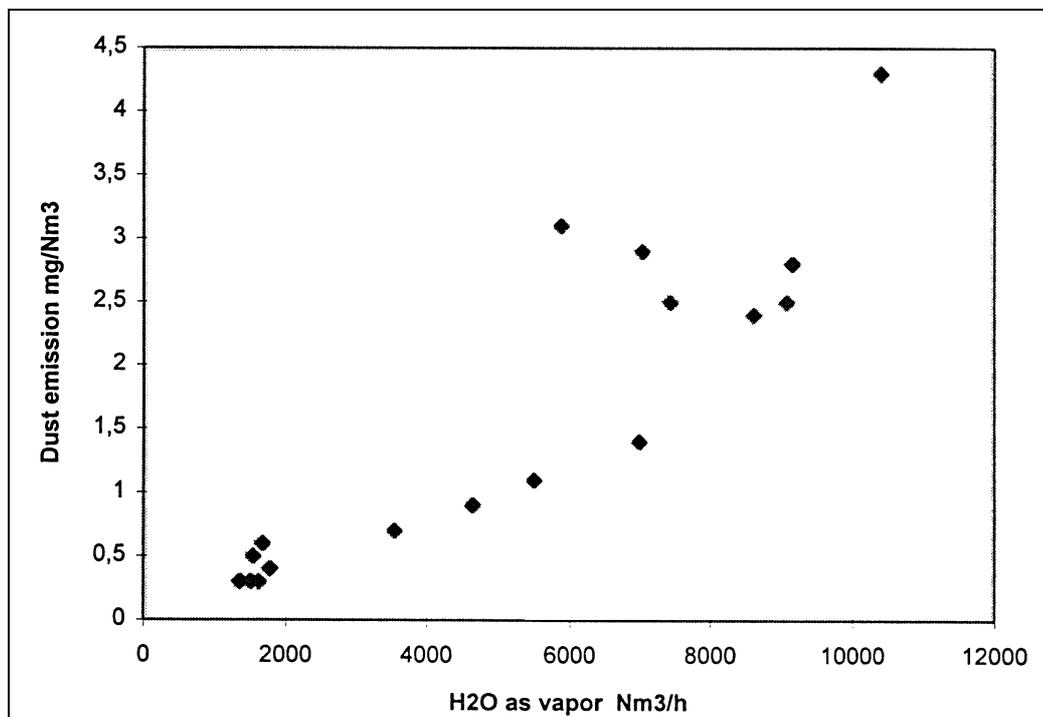


Figure 9.11 : Correlation of water vapour and residual dust content (after abatement in a bag filter) in the off gas of an EAF – [EC EAF, 1997; Pedersen, 1996]

Polycyclic aromatic hydrocarbons (PAH)

The emission factors for PAH are also relatively high (3.5 – 71 mg/t LS – see Table 9.1) but there are not many reported measurements. PAH are also already present in the scrap input [Schiemann, 1995] but may also be formed during EAF operation. The expectation that PAH adsorb to the filter dust to a high extent (also depending on the off gas temperature) could not be confirmed by investigations in Luxembourg, where PAH emissions remained unchanged before and after abatement in a bag filter which achieved low residual dust contents (< 5 mg/Nm³) as daily mean value [Werner, 1997].

9.2.2.1.1.2 Off gas directly collected from secondary metallurgy processes

Information about emissions from secondary metallurgy (mainly dust emissions) is very limited. [EC Study, 1996] reports dust emission factors before abatement from seven AOD/VOD refining installations between 6 – 15 kg dust/t LS and a single low figure of 1.35 kg dust/t LS. These seven installations have a de-dusting device independent from the de-dusting of EAF. Emission factors or concentrations after abatement are not reported.

9.2.2.1.2 Secondary off gases from scrap handling and charging, steel tapping, secondary metallurgy with tapping operations and from continuous casting

Secondary emissions mainly mean the emissions of dust except fume leakages from EAF which may contain all the pollutants described under primary emissions.

Information about secondary emissions is also limited. From charging the EAF usually 0.3 – 1 kg dust/t LS and from tapping 0.2 – 0.3 kg dust/t LS are emitted (emissions before abatement) [EC EAF, 1994]. For fume leakages during EAF operation dust emission factors between 0.5 – 2 kg dust /t LS are reported [EC Study, 1996].

Emission factors as sum of the mentioned three sources (charging, tapping, fume leakages) are between 1.4 – 3 kg dust/t LS [EC Study, 1996]. This can be considered as a confirmation that primary emissions are about ten times higher than secondary emissions.

Information about quantities of dust emissions from scrap handling as well as from continuous casting is not available.

Usually the secondary off gases are treated together with the primary ones. Bag filters are widely in use (about 90% of the installations [EC Study, 1996]) but in a few cases ESP and wet scrubbers are also applied.

With respect to micropollutants like organochlorine compounds, especially PCDD/F the contamination of secondary off gases (mainly the leakages from EAF) contribute to the overall emissions. When emission limit values of < 0.5 ng I-TEQ/Nm³ have to be complied with, secondary emissions have to be taken into consideration [Werner, 1997; Gerlafingen, 1998].

9.2.2.1.3 Fumes from slag processing

The processing of slags include cooling down by water spraying resulting in fumes. These fumes can be highly alkaline if the slag contains free CaO (see Table 9.4). This is very often the case. Alkaline depositions from the fumes may cause problems in the neighbourhood.

9.2.2.2 Solid wastes/by-products

The various solid wastes/by-products from electric arc furnace steelmaking are compiled together with their specific quantities in Table 9.3.

Solid waste/by-product	Specific quantity (range) [kg/t LS]
Slags from carbon steel/low alloyed steel production	
• Slag from EAF	100 – 150
• Slag from ladle	10 – 30
Slags from high alloyed steel production	
• Slag from EAF	100 – 135
• Slag from ladle	30 – 40
• AOD slag	ca. 160
Dusts from carbon steel/low alloyed/high alloyed steel production	10 – 20
Refractory bricks	2 – 8

Legend: LS = liquid steel

Table 9.3 : Kind and specific quantity of solid wastes/by-products from electric arc furnace steelmaking – [Geiseler, 1991, D Rentz, 1997]

9.2.2.2.1 Slags from production of carbon steel/low alloyed steel/high alloyed steels

The composition of slag from production of carbon and low alloyed steel can be seen from Table 9.4.

In addition, this table contains the slag composition from the production of stainless steel and from secondary metallurgy (AOD and VOD).

In traces, other elements than mentioned, such as Pb, As, Sb, Hg, Cl, F and hexavalent chromium may also be present.

Component [wt.-%]	Production of carbon/ low alloyed steel		Production of high alloyed steel	Secondary metallurgy	
	Slag from EAF	Slag from ladle	Slag from EAF ^{*2}	Slag from AOD	Slag from VOD
Fe _{tot}	10 – 32	≤ 2 – 5	≤ 2	≤ 1 – 2	max. 2
CaO	25 – 45	30 – 50	45	35 – 50	35 – 50
CaO _{free}	≤ 4	≤ 10	≤ 10	5 – max. 10	max. 5
SiO ₂	10 – 18	10 – 20	30	25 – 35	20 – 30
Al ₂ O ₃	3 – 8	3 – 12	5	1 – 10	1 – 10
MgO	4 – 13	7 – 18	7	4 – 7	5 – 15
MnO	4 – 12	≤ 1 – 5	2	1	n/a
Cr ₂ O ₃	1 – 2	≤ 0.5	3	1 – 5	1 – 5
TiO ₂	0.3	n/a	n/a	n/a	n/a
P ₂ O ₅	0.01 – 0.6	n/a	n/a	n/a	n/a
Na ₂ O	0.46 ^{*1}	n/a	n/a	n/a	n/a
K ₂ O	0.11 ^{*1}	n/a	n/a	n/a	n/a
V ₂ O ₅	0.11 – 0.25	n/a	n/a	n/a	n/a
ZnO	0.02 ^{*1}	n/a	n/a	n/a	n/a
CuO	0.03 ^{*1}	n/a	n/a	n/a	n/a
NiO	0.01 – 0.4	n/a	n/a	n/a	n/a
S	0.02 ^{*1}	n/a	n/a	n/a	n/a
C	0.33 ^{*1}	n/a	n/a	n/a	n/a

^{*1} data available only from one plant; ^{*2} data only from one plant; n/a = not available

Table 9.4 : Chemical composition of EAF slag from the production of carbon steel/low alloyed steel and high alloyed steel – based on [Geiseler, 1991; Plöckinger, 1979; D Rentz, 1997; Heinen, 1997]

In the EU, most of the slags from carbon and low alloyed steelmaking are still landfilled (Table 9.5) whereas the percentage of reuse of slags from production of high alloyed steels is significantly higher. But still one third is landfilled and stored.

Kind of steel	Total slag quantity	In-plant recycling		External use		Sold to another body		Landfilled and stored	
	[kt/a]	[kt/a]	[%]	[kt/a]	[%]	[kt/a]	[%]	[kt/a]	[%]
Carbon steels	1796	45.1	2.5	494.8	27.6	13.7	0.8	1242	69.2
Low alloyed steels	444	-	-	61.6	13.9	108.0	24.4	261	58.9
High alloyed steels	461	81.4	17.7	68.0	14.8	160.0	34.7	156	33.9
Total	2700	126.5	4.7	624.4	23.1	281.7	10.4	1659	61.4

Table 9.5 : Fate of EAF slags (reuse or disposal) in the EU; data from 57 plants producing 2.7 million t/a of slags (133 kg/t LS) – [EC Study, 1996]

Also most of the slags from ladle treatment and secondary metallurgy (also AOD and VOD slag) is mostly landfilled, with respect to the EU about 80 % [EC Study, 1996]. The rate of landfilling respectively reuse varies in the different Member States depending on legal requirements, availability of landfills, taxes, market situation, costs and possibilities to reuse processed slags.

9.2.2.2.2 Dusts from off gas treatment

As already mentioned the treatment of off gases (mostly primary off gases together with secondary off gases) is very often performed in bag filters. The composition of dusts from production of carbon, low alloyed and high alloyed steel can be seen from Table 9.6.

Component	Dust from carbon/low alloyed steel production [weight-%]	Dust from high alloyed/stainless steel production [weight-%]
Fe _{tot}	25 – 50	30 – 40
SiO ₂	1.5 – 5	7 – 10
CaO	4 – 15	5 – 17
Al ₂ O ₃	0.3 – 0.7	1 – 4
MgO	1 – 5	2 – 5
P ₂ O ₅	0.2 – 0.6	0.01 – 0.1
MnO	2.5 – 5.5	3 – 6
Cr ₂ O ₃	0.2 – 1	10 – 20
Na ₂ O	1.5 – 1.9	n/a
K ₂ O	1.2 – 1.5	n/a
Zn	10 – 35	2 – 10
Pb	0.8 – 6	0.5 – 2
Cd	0.02 – 0.1	0.01 – 0.08
Cu	0.15 – 0.4	0.01 – 0.3
Ni	0.02 – 0.04	2 – 4
V	0.02 – 0.05	0.1 – 0.3
Co	0.001 – 0.002	n/a
As	0.003 – 0.08	n/a
Hg	0.0001 – 0.001	n/a
Cl	1.5 – 4	n/a
F	0.02 – 0.9	0.01 – 0.05
S	0.5 – 1	0.1 – 0.3
C	0.5 – 2	0.5 – 1
Basicity	2.0 – 6.5	n/a
Moisture	6 – 16	n/a

Table 9.6 : Chemical composition of EAF dusts from the production of carbon steel/low alloyed steel and high alloyed steel – based on [EUROFER EAF, 1997; Hoffmann, 1997; Strohmeier, 1996]

In the EU, about two third of the dust is landfilled (Figure 9.12).

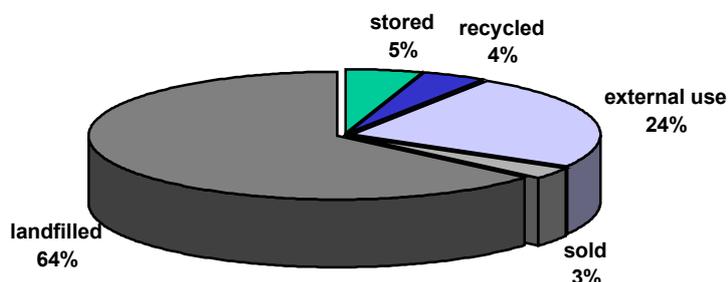


Figure 9.12 : Fate of dust collected from primary and secondary off gases of EAF; data from 67 plants – [EC Study, 1996]

In the single Member States the percentages of dusts which are reused respectively landfilled is very different depending on legal requirements, availability of landfills, taxes and other cost aspects.

Table 9.7 indicates, that in Austria, Germany and the Benelux States the dust recycling has achieved high rates whereas they are low in Southern Europe and in the UK. That means, that the data of the EC study (see Figure 9.12) are no more fully representative for the actual situation.

State(s)	Total amount of dust [t/a]	Amount of dust processed in the Waelz process [t/a]	Percentage [%]	Fate of the residual amount of dust
Austria and Switzerland	30000	25000	83	landfill
Benelux	65000	55000	85	landfill
Denmark	12000	-	100	-
France	90000	30000	33	landfill
Germany	150000	105000	70	landfill, filling of mines
Italy	180000	80000	44	landfill and recycling in a plant in I-Enirisorse
Scandinavia	30000	10000	33	landfill and storage for recycling in the future
Spain and Portugal	120000	25000	20	Landfill
UK	65000	0	0	Landfill
Total	730000	330000	45	

Table 9.7 : Percentages of filter dust from EAF (from carbon and low alloyed steel production) treated in the Waelz process for zinc recovery respectively landfilled in the EU Member States in 1997 – [Hoffmann, 1997]

The landfills for the dusts are equipped with different sealing systems. In [EC Study, 1996] the percentages of the various systems are reported (Figure 9.13).

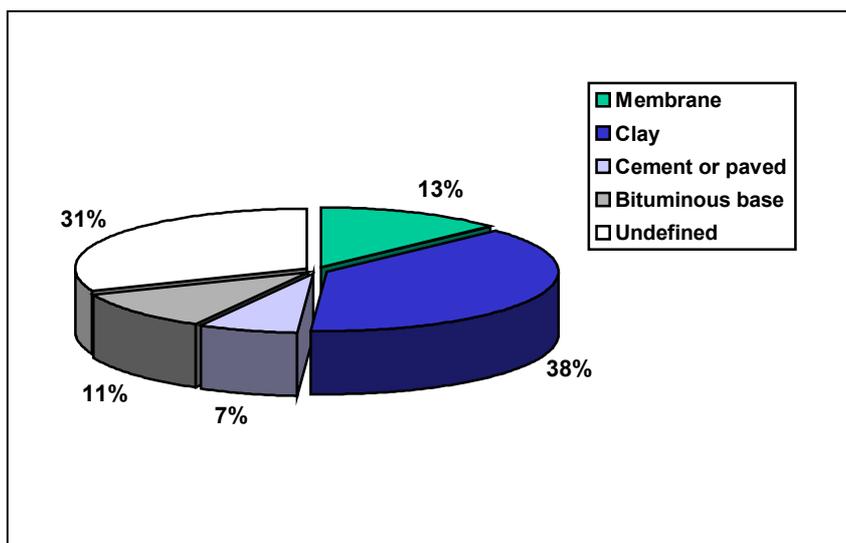


Figure 9.13 : Percentages of applied sealing systems for the landfill of filter dusts from EAF in the EU – [EC Study, 1996]

Filter dusts from the 14 production sites in the EU producing high alloyed steels/stainless steels are recycled to a much higher extent in order to recover nickel and/or chromium and/or molybdenum. About one third is still landfilled [EC Study, 1996] but the recycling percentage is steadily increasing [Kola, 1996].

9.2.2.2.3 Refractory bricks

In most cases refractory bricks are put to landfill [EC Study, 1996].

9.2.2.3 Wastewater emissions

9.2.2.3.1 Drainage water from scrap-yard

The main raw material of EAF, the different kinds of scrap are often stored on unpaved scrap-yards. Drainage water can be contaminated, especially in case of oil/emulsion containing scrap like turnings. There is no information available on quantities and pollution of drainage water. Usually it is at least treated in an oil separator prior to be discharged.

9.2.2.3.2 Waste water from off gas scrubbing

In the EU in some cases the off gases are treated in a wet scrubber. There is no information available on applied treatment techniques and discharged quantities and its pollution.

9.2.2.3.3 Wastewater from continuous casting

Wastewater results from direct cooling from continuous casting. The wastewater contains mill scale (1–3 g/l) and oil/grease. Usually this wastewater is treated together with other streams from the rolling mill(s).

9.2.2.4 Soil contamination

In many cases the scrap-yard is unpaved and uncovered. Contamination of soil may arise from storage of scrap contaminated with mineral oil/emulsions or other compounds. There is no information available about extent and impact of such soil contamination.

If the yard for slag processing is unpaved and the raw slag is containing free CaO alkaline water may enter the soil.

9.2.2.5 Noise emissions

The following noise sources are dominating in electric arc furnace steelworks:

- melting shop including EAF
- scrap yard
- primary de-dusting
- roof hood de-dusting
- water management equipment

Conventional EAF show average sound levels (melting and treating) of $L_{WA} = 118 - 133$ dB(A) for furnaces > 10 t and $L_{WA} = 108 - 115$ dB(A) for furnaces < 10 t; the specific transformer power determines the level of noise emissions. In electric steelworks sound levels of up to $L_{WA} = 127$ dB(A) can appear (measurement includes melting and treating). The main share of noise emissions contribute the melting shop including EAF, the scrap yard and the primary de-dusting.

9.3 Techniques to consider in the determination of BAT

Process-integrated measures

- PI.1 EAF process optimisation
- PI.2 Scrap preheating
- PI.3 Closed loop water cooling system

End-of-pipe techniques

- EP.1 Advanced emission collection systems
- EP.2 Efficient post-combustion in combination with advanced off gas treatment
- EP.3 Injection of lignite coke powder for off gas treatment
- EP.4 Recycling of EAF slags
- EP.5 Recycling of EAF dusts

PI.1 EAF process optimisation

Description: The EAF process has been steadily improved in order to optimise it and to increase productivity which correlates with the decrease of specific energy consumption.

Figure 9.14 indicates some of the most important measures/techniques which are briefly described in the following. These are:

- (Ultra) High power operation (UHP),
- Water cooled side walls and roofs,
- Oxy-fuel burners and oxygen lancing,
- Bottom tapping system,
- Foaming slag practice,
- Ladle or secondary metallurgy,
- Automation.

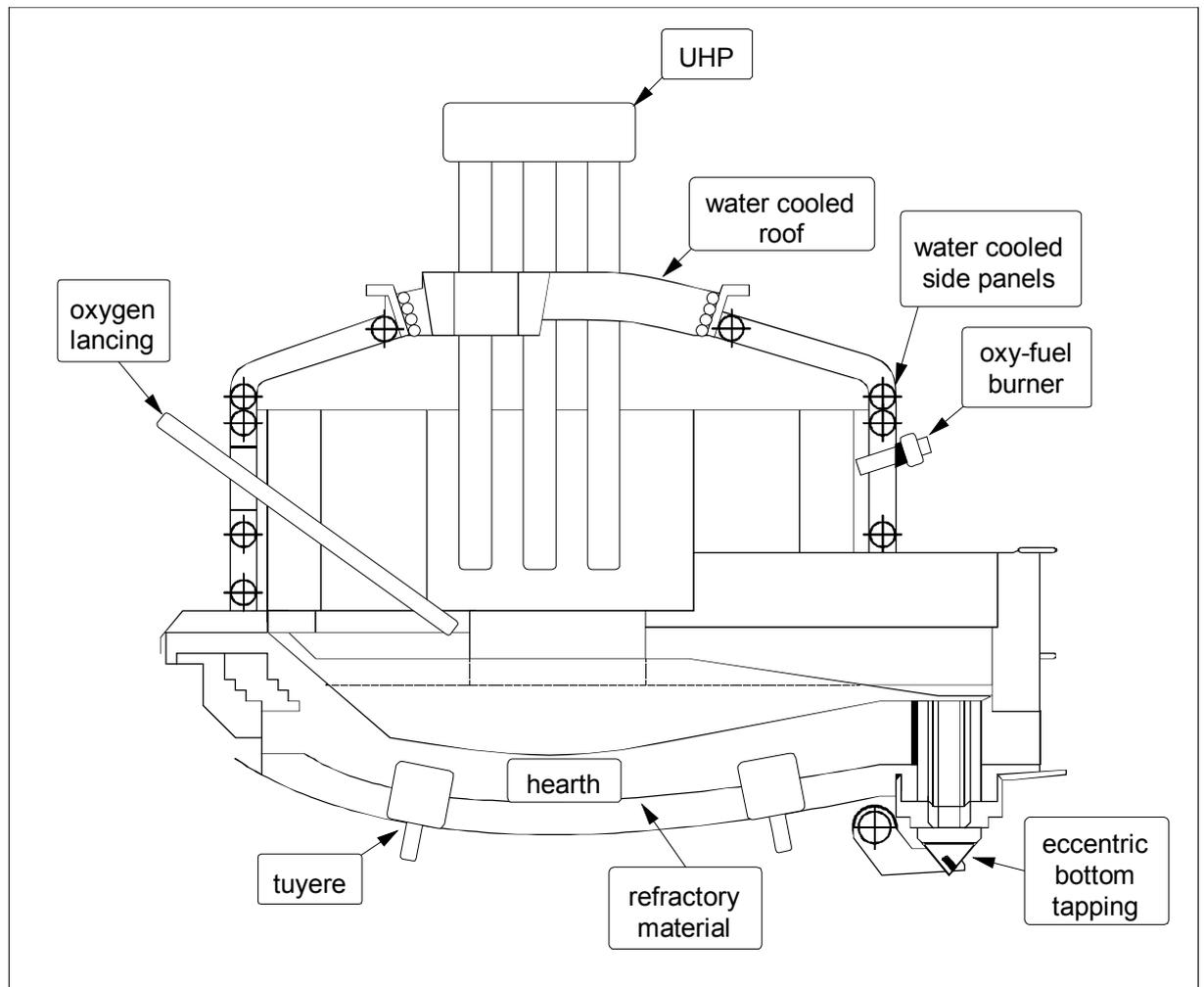


Figure 9.14 : Schematic of an EAF with indication of techniques for optimisation – [D Rentz, 1996]

(Ultra) High power operation:

The efforts to reduce tap-to-tap times led to the installation of more powerful furnace transformers. Decisive features for (Ultra) high power furnaces are installed specific apparent power supply, mean power efficiency (≥ 0.7), and timewise use of the transformer (≥ 0.7). UHP operation may result in a higher productivity, reduced specific electrode consumption, and reduced specific waste gas volume, but also in increased wear of the furnace lining [Heinen, 1997].

Water cooled side walls and roofs:

Within the last two decades, furnace walls and roofs have been lined with water cooled panels, providing the opportunity to save refractory material, to use the (ultra) high power furnace technology, and also to re-use waste heat by the application of measures for energy recovery. However, it has to be checked on a plant by plant basis, if the recovery of energy is economically viable. In principle, two cooling systems can be distinguished. So-called cold or warm cooling draws off power losses by an increase of the cooling water temperature flowing through the pipe coils. Evaporation cooling works by the evaporation of cooling water to draw off radiation heat caused by the electric arc process. To protect water cooled side panels from thermal strain, especially when foaming slag operation (see below) is not possible, a computer controlled regulation of the melt-down process helps to prevent tears in the panels caused by mechanical tension and also saves refractory material [Knoop, 1997].

Oxy-fuel burners and oxygen lancing:

Oxy-fuel burners promote a uniform melting of the scrap. It also partially offsets the effect of maximum demand control on electricity supply. Usually, additional energy input by oxy-fuel burners and oxygen lancing results in a decrease of total energy input required.

Bottom tapping system:

The practice of bottom tapping is widely adopted nowadays, as it makes possible to minimise the amount of oxidic slag (carry over) to the ladle during tapping. It also allows cost savings for the lowering of refractory material needed, for a more rapid tapping, and for reduced energy losses. Furthermore, it simplifies the capturing of fumes. While some older furnaces are still equipped with spouts, usually most of the new electric arc furnaces are equipped with bottom tapping systems.

Foaming slag practice:

Creating a foamy slag within the furnace improves the heat transfer to the charged inputs, and also protects the refractory material inside the furnace. Because of better arc stability and less radiation effects, foaming slag practice leads to reductions in energy consumption, electrode consumption, noise level, and an increase in productivity. It also causes positive effects on several metallurgical reactions (eg. between slag and melt). The density of foaming slag is less than common FeO containing EAF slag (1.15-1.5 t/m³ compared to 2.3 t/m³). For this reason, the volume of slag arising during steelmaking is rising and may require larger slag buckets. After tapping, the slag partly degasses again. Information on adverse impacts of the foamy slag practice on the possibilities to use the slag have not been encountered. It has to be noted, that the use of foaming slag practice for high grade steelmaking is often not possible.

Ladle or secondary metallurgy:

Some production steps need not be carried out in the EAF itself and can be performed more efficiently in other vessels (like desulphurisation, alloying, temperature and chemistry homogenisation). These tasks have been shifted from the EAF to ladles, ladle furnaces, or other vessels nowadays [EPRI, 1992; Heinen, 1997]. The reported benefits of this development are energy savings (net savings of 10-30 kWh/t), a reduction of tap-to-tap times of about 5-20 minutes, increasing the productivity, a better control of steel temperature of the heat delivered to the continuous casting, a possible reduction of electrode consumption (up to 0.1-0.74 kg/t), alloy savings, and a decrease of the emissions from the EAF itself [EPRI, 1992]. A possible drawback of using ladles or other vessels with respect to air pollution control is the increase in the numbers of emission sources, requiring higher investments for air pollution control equipment, as additional fume capturing devices like hoods are needed.

Automation:

Computer control in electric arc furnace plants has become necessary within recent years, as the high throughputs require efficient control systems to manage the material and data flows arising in the raw material selection, EAF, ladle furnace, and continuous caster. Efficient control

systems permit an increase in productivity, a reduction in energy consumption, and also a decrease in dust emissions [Linninger, 1995].

Main achieved emission levels: Mentioned above (description)

Applicability: The described techniques are applicable both to new and existing plants but have to be checked on a plant to plant basis.

Cross-media effects: Oxy-fuel burners increase the off gas flow but on the other hand it decreases the overall energy demand.

Water cooled side walls and roof need additional energy consumption of about 10-20 kWh/t but may be compensated by advantages in the field of plant availability and maintenance. Water cooled side walls and roofs have inter alia provided the opportunity to apply modern technology like HP or UHP furnaces.

Reference plants: Many plants in the EU are equipped with the described techniques and are operated with optimised conditions. Table 9.8 compiles concerned data from nine German EAF operated under optimised conditions.

	EAFF 1	EAFF 2	EAFF 3	EAFF 4	EAFF 5
In operation since	1979	1968/1976	1995	1994	1995
Furnace type	AC UHP	AC UHP	DC UHP furnace	DC UHP furnace	DC UHP furnace
Produced steel grades	Carbon steel	carbon steel	carbon steel	carbon steel	carbon steel
Tapping weight [t]	135	85 each	100	125	120
Nominal apparent power of current transformer [kVA/t]	711	800 each	140	130	120
Raw materials	Scrap	scrap	scrap	scrap	scrap
Cooling system	water cooled side walls and roofs	water cooled side walls and roofs	water cooled side walls and roofs	water cooled side walls and roofs	water cooled side walls and roofs
Tapping system	EBT	OBT	EBT	EBT	EBT
Capacity [t/a]	600000	600000 each	750000	600000	600000
Additional burners	Oxygen lancing (side walls)	fuel burners (side walls and door)	gas burners	oxygen gas burners	oxygen/natural gas burners (7)
Additional fuels	-	-	coal	coal	coal
Emission collection Measures	direct extraction (4 th hole), canopy hood	direct extraction (4 th hole), hood	2 nd hole, hood	2 nd hole, roof hood	2 nd hole, ladle furnace de-dusting, big furnace enclosure
Off gas cleaning system	post-combustion, dry ESP	post-combustion chamber, quenching, fabric filter	post combustion chamber with additional burners quenching (air) fabric filter	post combustion chamber with additional burner quenching (water) fabric filter	post combustion, fabric filter
Energy aspects	Recovery of furnace and waste gas heat (steam generation)	n/a	recovery of waste gas heat	recovery of waste gas heat	water cooled ducts
Secondary metallurgy	ladle furnace desulphurisation	Ladle furnace	ladle furnace vacuum degassing	ladle furnace vacuum degassing	ladle furnace

Legend: n/a = not available; EBT = eccentric bottom tapping; OBT = oval bottom tapping; ESP = electrostatic precipitator

Table 9.8 : Data from 9 German plants operating optimised EAF – [D Rentz, 1997]

Continuation of Table 9.8 :

	EAf 6	EAf 7	EAf 8	EAf 9
In operation since	1978	1981	1995	1982
Furnace type	AC UHP	AC furnace	AC furnace	AC UHP
Produced steel grades	carbon steel	high alloyed steel	high alloyed steel	high alloyed steel
Tapping weight [t]	115	110	100	145
Nominal apparent power of current transformer [MVA]	910	682	570	724
Raw materials	scrap, DRI	scrap	Scrap	scrap
Cooling system	water cooled side walls and roofs	water cooled side walls and roofs	water cooled side Walls and roofs	water cooled side walls and roofs
Tapping system	EBT	EBT	EBT	tapping into runner
Capacity [t/a]	950000	550000 each	400000	600000
Additional burners	oxygen and carbon lancing, gas burners	fuel burners (side walls and door)	oxygen lancing (door)	oxygen lancing (side walls)
Additional fuels	-	-	-	-
Emission collection measures	direct extraction (4 th hole), hood	direct extraction (4 th hole), hood	direct extraction (4 th hole), hood	direct extraction (4 th hole), dog-house
Off gas cleaning system	post-combustion chamber, quenching, fabric filter	dry ESP	fabric filter	post combustion, fabric filters for primary and secondary emissions
Energy aspects	n/a	n/a	n/a	recovery of furnace and waste gas heat (steam generation)
Secondary metallurgy	ladle furnace	ladle furnace vacuum degassing	ladle furnace vacuum degassing	ladle furnace

Driving force for implementation: The high market competition and the need to increase productivity/to reduce costs pushed the introduction of the described techniques.

Operational data and economics: Operational data can be seen from Table 9.8. Economical data are not available.

Reference literature: [Rentz, 1997]

PI.2 Scrap preheating

Description: The recovery of waste heat from off gases is a well-known approach. In seventies about twenty plants have been erected to preheat the scrap in the basket prior to its discharge into the furnace. But all these systems have been taken out of operation, due to technical and emission problems. New furnace concepts with shaft integrated scrap preheating. With the single shaft furnace at least 50 % of the scrap can be preheated [Smith, 1992] whereas the new finger shaft furnace (Figure 9.15) allows the preheating of the total scrap amount [Voss-Spilker, 1996].

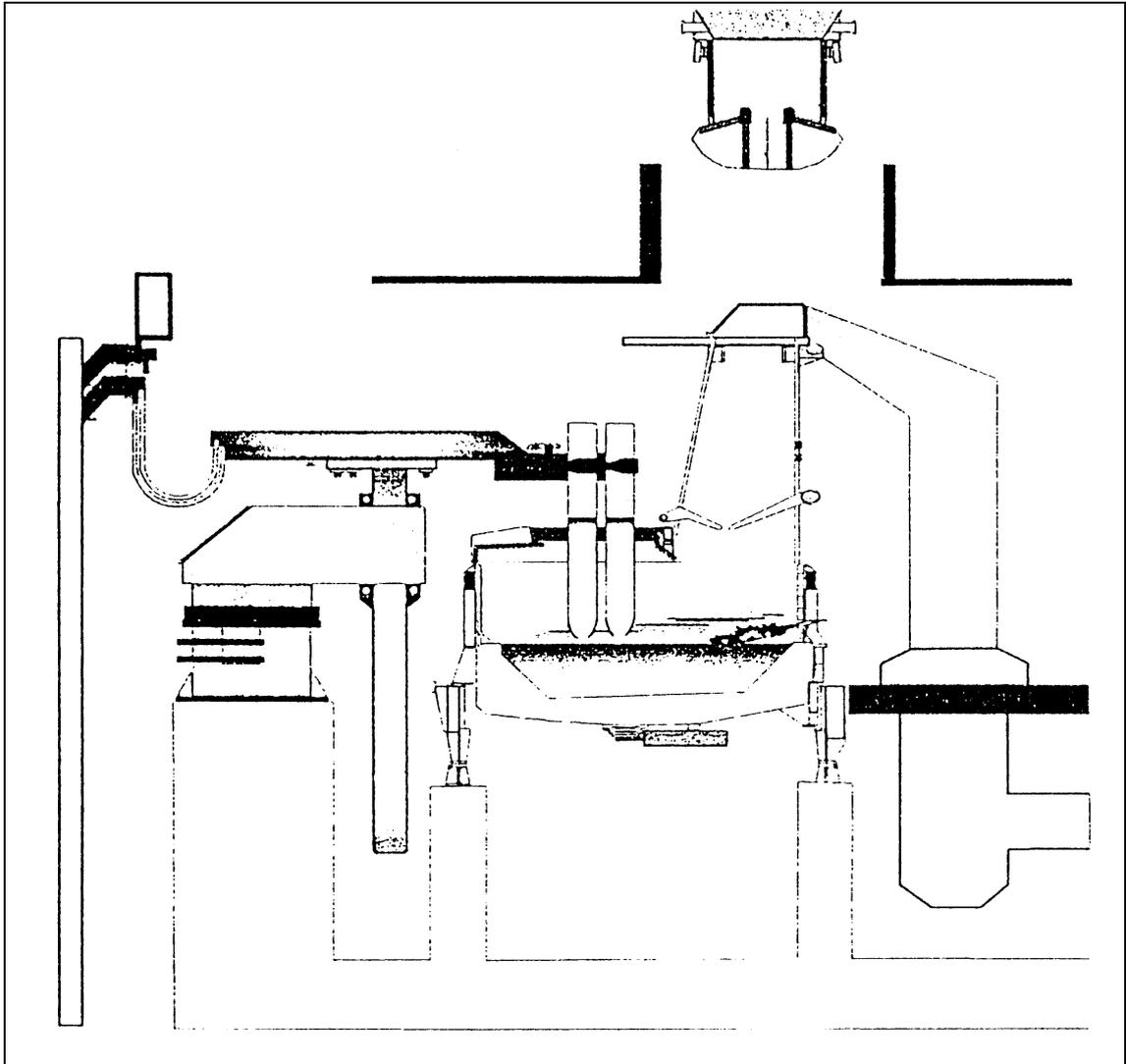


Figure 9.15 : Schematic of an EAF with a shaft equipped with “fingers” in order to retain the scrap (finger shaft furnace) for preheating – [Voss-Spilker, 1996]

With finger shaft EAF tap-to-tap times of about 35 minutes are achieved which is about 10-15 minutes less compared to EAF without efficient scrap preheating. This allows a very short pay back time which is in the order of one year.

Another available process for scrap preheating is the Consteel process [McManus, 1995] (Figure 9.16) but this system is not generally considered as a proven technique.

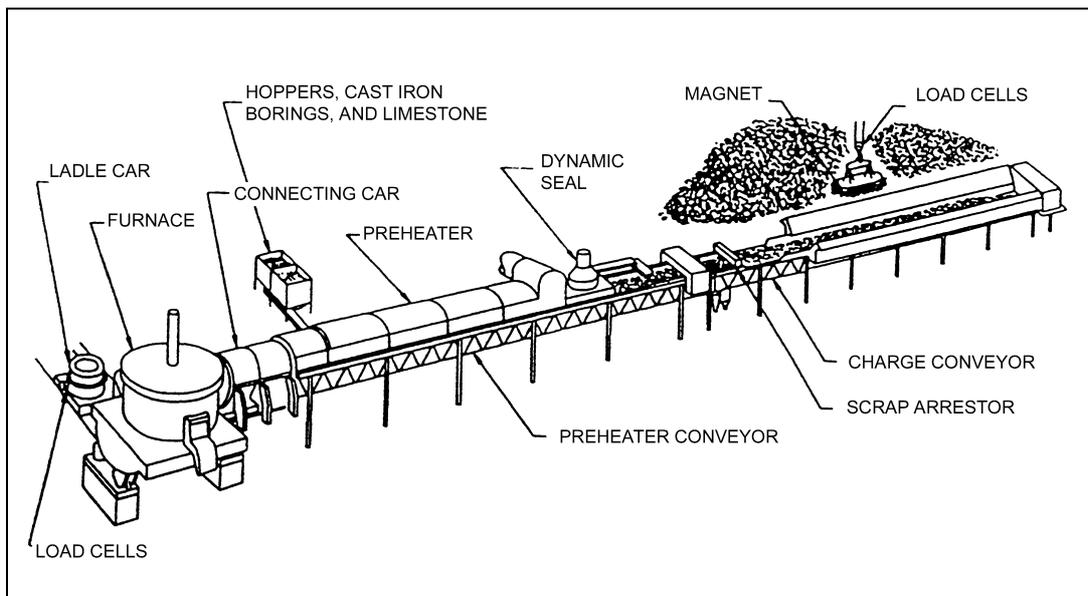


Figure 9.16 : Schematic of the Consteel Process – [Vallomy, 1992]

Main achieved emission levels: With the single shaft furnace up to 70 kWh/t LS of electric power can be saved. Calculated on the basis of primary energy the savings are about three times higher because of the low efficiency of electricity supply. In addition the scrap preheating significantly reduces the tap-to-tap time which means a considerable increase of productivity.

The finger shaft furnace allows energy savings up to 100 kWh/t LS which is about 25% of the overall electricity input. In combination with an advanced off gas treatment (see EP.2) scrap preheating may play a significant role in optimisation of electric arc furnace steelmaking not only related to productivity but also minimise emissions.

As a side effect scrap preheating reduces raw dust emissions about 20% because the off gas has to pass the scrap which acts as a filter. This reduction correlates with an increase of the zinc content in the dust which supports its recycling.

Applicability: Applicable both to new and existing plants. In case of existing installations the local circumstances like space availability or given furnace concept have to be checked on a plant by plant basis.

Cross-media effects: The scrap preheating in a shaft may lead to an increase of organic micropollutants and smell, such as PCDD/F unless adequate thermal treatment of the off-gases is performed. Additional off gas treatment may be necessary which needs additional energy. But in relation to the energy saving by scrap preheating this additional energy consumption may be reasonable and acceptable, especially when taking into account that electric power is generated from thermal energy with a yield of about 35% and for post-combustion natural gas is used.

Reference plants: EAF with a single shaft: Co-Steel Sheerness, UK-Sheerness
 EAF with a finger shaft: Cockerill-Sambre, B-Charleroi; Gerlafingen Stahl AG, CH-Gerlafingen (this furnace has been retrofitted with a finger shaft);
 Twin-shell furnace with integrated preheating in a shaft: ARES, L-Schiffflange; ASW, F-Montereau;
 Nervacero, Spain.

Driving force for implementation: The main driving force is the increase of productivity. In some cases scarp preheating by means of a finger shaft furnace has been installed in combination with advanced off gas treatment.

Operational data and economics: not available

Reference literature: [Voss-Spilker, 1996; Haissig, 1997; Smith, 1992; McManus, 1995]

PI. 3 Closed loop water cooling system

Description: Generally, water is only used in the EAF steelmaking processes in connection with non-contact cooling, and if a wet scrubbing technique for off gas cleaning is used. As wet scrubbing is only applied in few cases, this topic is not further investigated in the following. The most relevant use of water considered here is the water used for the cooling of the elements of the furnace. Additionally, some water may be used for the cooling of waste gas or in the secondary metallurgy section. The water needed with respect to the cooling elements amounts to 5-12 m³/(m²h) [D Rentz, 1997].

Modern plants operate with closed cooling systems in the EAF and secondary metallurgy sections

Main achieved emission levels: No discharge of wastewater.

Applicability: Applicable both to new and existing plants.

Cross-media effects: The closed loop system requires additional energy for water pumping and water re-cooling.

Reference plants: Preussag Stahl AG, D-Peine; BSW, D-Kehl and many other plants in the EU

Driving force for implementation: Legal requirements and limited availability of cooling water.

Operational data and economics: not available

Reference literature: [D Rentz, 1997]

EP.1 Advanced emission collection systems

Description: The primary and secondary emissions to air are of high relevance (see 9.2.2.1). The available abatement techniques should be fed with the raw emissions as complete as possible. Thus the collection of the emissions is important. The combination of 4th hole (in case of three electrodes) respectively of 2nd hole (in case of one electrode) direct extraction with hood systems (or furnace enclosure) or total building evacuation are the most favorite systems.

A 4th, or 2nd hole (see Figure 9.6) should collect practically quantitatively the primary emissions generated during the melting and refining periods. This type of direct extraction technology is state of the art in modern EAF steelmaking for the collection of primary emissions. It can also be applied to secondary metallurgy vessels.

In a hood system (see Figure 9.6), one or more hoods over the furnace indirectly collect fumes escaping from the furnace during charging, melting, slag-off, and tapping steps (up to 90% of primary emissions and also secondary emissions [EC EAF, 1994]). Hood systems are commonly used within the electric arc furnace steel industry. Combined with direct extraction systems, the collection efficiency of primary emissions and also secondary emissions improves up to 98%. Hoods are also installed to collect emissions arising at secondary metallurgy vessels, hoppers and conveyor belts.

Furnace enclosures, also called dog-houses (see Figure 9.6), usually encapsulate the furnace, its swinging roof, and also leave some working space in front of the furnace door. Typically, waste gases are extracted near the top of one of the walls of the enclosure, and makeup air enters through openings in the operating floor [EPRI, 1992]. More complex handling steps, causing time losses and possibly higher investments (e.g. need for additional door opening and closing mechanisms and procedures in order to charge and empty the furnace) are drawbacks of this type of collection technology. Collection rates of dog-houses are similar or usually slightly higher to those of hood-complementary hole combinations. A positive effect of furnace enclosures is a reduction in the noise level, if they are constructed in a suitable manner. Noise abatement at an EAF plant by sound protecting enclosures can reduce the average sound pressure level between 10 and 20 dB(A) [Kuhner, 1996]. Furnace enclosures may also be applied at secondary metallurgy processes [EC EAF, 1994] but it needs a treatment of the shop walls to eliminate reverberation.

Another way to collect secondary emissions from the furnace, as well as preceding and succeeding installations, is a complete enclosure of all plants in one sealed building. It can be regarded, roughly speaking, as just a larger type of furnace enclosure, mainly containing more process steps.

The erection of such buildings and the additionally required large de-dusting installations in order to achieve complete de-dusting impose considerable costs on the operators. For this reason, the costs and benefits need to be weighed up carefully for every special plant before this option is considered. A positive effect of this measure is a reduction in the noise level penetrating to the outside. Usually, the pressure in the enclosing building is below atmospheric pressure to avoid the escaping of fumes through occasional door openings.

Main achieved emission levels: The combination of direct fume extraction and a hood system is often used. This combination achieves a collection of about 98% of the primary emissions. In addition, a significant share of charging and tapping (secondary) emissions can be collected, too, though this depends on the type and the number of hoods [EC EAF, 1994]. A combination of a direct extraction device and a furnace enclosure even achieves collection rates of over 97% up to 100% of the total dust emissions [Heinen, 1997]. Total building evacuation also achieves practically 100% emission collection.

Applicability: Applicable both to new and existing plants.

Cross-media effects: The emission collection systems need energy, especially the fans.

Reference plants: Many plants in Europe have a combination of direct off gas extraction and hoods.

Following German plants are equipped with dog-houses only systems or with a combination of dog-house and hole (direct extraction): Benteler AG, D-Lingen; Krupp Thyssen Nirosta, D-Bochum; Krupp Thyssen Nirosta, D-Krefeld; Mannesmannrohr GmbH, D-Bous/Saar; Moselstahlwerk, D-Trier; Stahlwerke Thüringen GmbH, D-Unterwellenborn

Total building evacuation: ARES, L-Schiffflange; ProfilARBED, L-Differdange and L-Belval.

Driving force for implementation: The main driving force are legal requirements.

Operational data and economics: not available

Reference literature: [Heinen, 1997; EC EAF, 1994; D Rentz, 1997]

EP.2 Efficient post-combustion in combination with advanced off gas treatment

Description: The optimisation of EAF operation (see PI.1), especially the increasing use of oxygen and fuels have increased the amount of chemical energy in the primary off gas (CO and H₂ content) [Evenson, 1996]. In order to use this energy post-combustion trials in electric arc furnace steelmaking were started mid of the eighties and significant progresses had been made. Post combustion in the furnace is developed to use a maximum of chemical energy of the CO in the furnace and to improve the energy balance, but CO and H₂ are never completely oxidised in the furnace; for this reason, it needs post combustion. Post combustion in a combustion chamber aims primarily at the full combustion of CO and H₂ remaining in the off-gas in order to avoid uncontrollable reactions in the gas cleaning equipment. Secondly, this post-combustion, when it is well optimised, reduces the emission of organic compounds. The heat produced by this combustion is generally not recovered unless recovery from cooling water is possible. Today the optimisation of the post combustion chamber can reduce organic micro-pollutants, such as PCB or PCDD/F. Figure 9.17 shows such a plant originally equipped with post combustion chambers. Because of relevant de novo synthesis of PCDD/F the heat exchanger been replaced by a quenching tower for rapid cooling of the off gas.

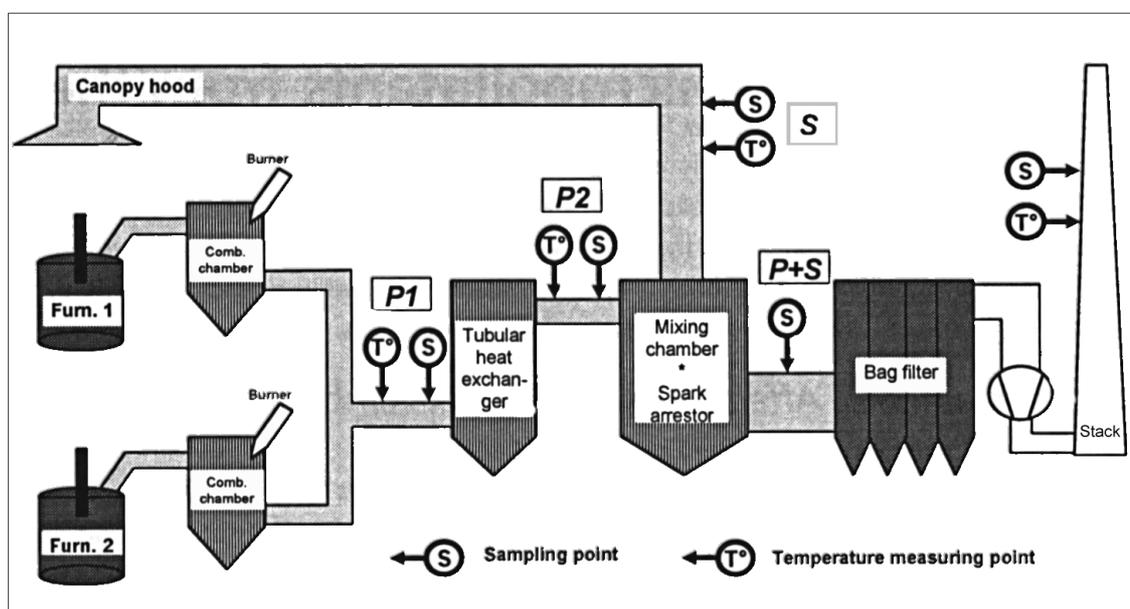


Figure 9.17 : Schematic layout of the treatment of primary off gas from a twin shell EAF – [Werner, 1997];

Because of de novo synthesis of PCDD/F in the tubular heat exchanger, this device has been replaced by a quenching tower for rapid cooling of the off gas

Post-combustion with the additional aim to minimise organic micropollutants needs necessary retention time, turbulence and temperature (3 T's). If a separate combustion chamber can not be introduced, suitable post-combustion can also be achieved in the off gas duct system (Figure 9.18).

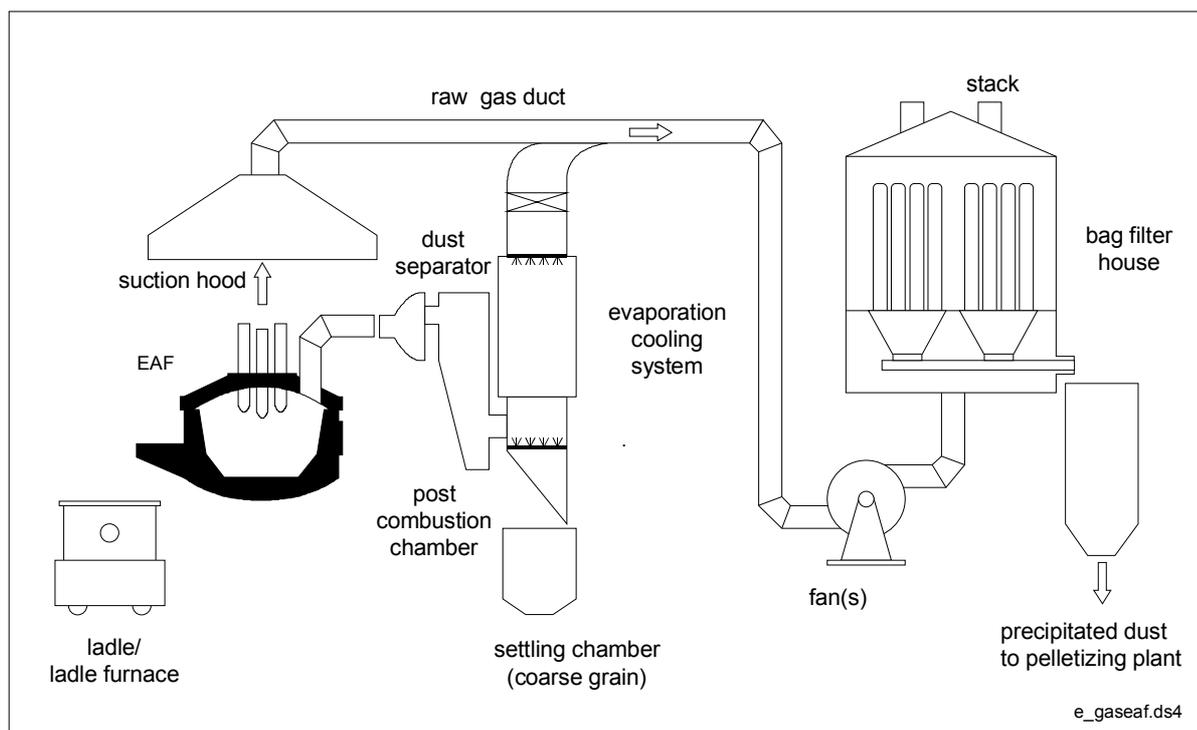


Figure 9.18 : Post-combustion of the primary off gas within the given duct system from an EAF with subsequent rapid cooling – [D-Rentz, 1997]

Recent developments have separate post-combustion chambers with additional burners to achieve the necessary “3 T’s).

To avoid the novo synthesis of PCDD/F, it is necessary to have a fast cooling of the fumes before filtration in a bag filter. In some cases, this is obtained by dilution of the secondary circuit; in other cases, as presented in Figure 9.17, a solution is the water quenching tower.

Main achieved emission levels: With a proper post combustion followed by a rapid cooling (by dilution or water quenching) emission concentration of PCDD/F lower than $0.5 \text{ ng I-TEQ/Nm}^3$ can be achieved (Table 9.9).

Plant	EAF 1		EAF 2			EAF 3		EAF 4			
Features:											
Tapping weight [t]	105		138			85/85		140			
Power supply [MVA]	105		96			57/68		105			
Collection of emissions	4 th hole hood		4 th hole hood			4 th hole hood		4 th hole furnace encapsulation			
Post combustion (PC)	PC chamber (air)		PC in duct			PC chamber (air)		PC in duct			
Waste gas cooling	Injection of water		Water conditioning of waste gas			Spray cooling system (quenching)		Cooling by air cooled heat exchanger			
Off gas cleaning system	Bag filter		Electrostatic precipitator			Bag filter (1 for both)		Two bag filters for primary and secondary de-dusting			
Gas concentrations: *	M 1 **	M 2	M 1	M 2	M 3	M 1	M 2	M 1	M 2	M 3	M 4
Dust in crude gas (p)	3398	14246	4200	12500	3600	-	-	-	-	-	-
Dust in crude gas (s)	148	273	p and s together			-	-	-	-	-	-
Dust in clean gas (p)	0.76	1.05	15	15	18	1.45	1.1	<1	<1	<1	<1
Dust in clean gas (s)	average °		-	-	-	average °		<1	7	3	<1
PCDD/F (p)	-	-	-	-	-	-	-	0.252	0.201	0.240	0.810
PCDD/F (s)	-	-	-	-	-	-	-	0.027	0.010	0.023	0.057
PCDD/F(Mix, p and s)	0.016	0.021	0.01	0.02	0.01	0.13	0.1	0.087	0.061	0.081	0.259
*: crude and clean gas dust concentrations in mg/Nm ³ , PCDD/F concentration in ng I-TEQ/Nm ³											
**: M #: Number of measurement at the plant											
(p): concentrations after de-dusting device for primary off-gases											
(s): concentrations after de-dusting device for secondary off-gases											
°: average of two measurement spots, -: not relevant or no information available											

Table 9.9 : Performance of post-combustion at four German EAF – [D Rentz, 1997, Theobald, 1995]

In separate post-combustion chambers with additional burners PCDD/F emission concentration ≤ 0.1 ng I-TEQ/Nm³ are achievable but there are problems in practice to achieve this level constantly. The reduction of PCDD/F can be considered as a lead parameter. Thus it can be expected that other organic micropollutants are also destroyed.

But it has to be noted that PCDD/F emissions from secondary off gases (which are not combusted but mixed with the primary off gas – see Figure 9.18) could significantly increase the emitted PCDD/F concentration. [Gerlafingen, 1998].

Applicability: In principle post-combustion can be applied both to new and existing plants but in existing ones the local circumstances and possibilities (like available space, given off gas duct system etc.) have to be checked on a plant by plant basis.

Cross-media effects: Post-combustion with additional burners consume considerable quantities of energy (in the order of 30 kWh/t) or prevents heat recovery (see Figure 9.17). The application of post-combustion in combination with efficient scrap preheating (see PI.2) may allow a balanced solution of energy saving and consumption.

Reference plants: ProfilARBED, L-Differdange; BSW, D-Kehl; Gerlafingen Stahl AG, CH-Gerlafingen

Driving force for implementation: The main driving force for the implementation of post-combustion are stringent emission limit values requiring for PCDD/F emission limits $<0.5 \text{ ng I-TEQ/Nm}^3$.

Operational data and economics: The post-combustion unit at ProfilARBED, L-Differdange and at BSW, D-Kehl are operated without significant problems. Investment cost for a quenching tower is about 1.2 millions Ecu₁₉₉₇. More data about economics are not available.

Reference literature: [Karcher, 1996; Werner, 1997, Knapp, 1996]

EP.3 Injection of lignite coke powder for off gas treatment

Description: In order to reduce organic micropollutants in the total off gas (primary and secondary emissions), especially PCDD/F lignite coke powder can be dosed to the duct before the bag filters. The necessary amount is in the order of 100 mg lignite coke powder/Nm³ off gas (see also EP.3 in 4.3). The lignite coke powder is separated in the gas phase in the subsequent bag filters. Attention has to be paid to sparks and principally possible glow fires. The explosion risk has been assessed as to be low.

Main achieved emission levels: Residual PCDD/F emission concentrations of < 0.5 ng I-TEQ/Nm³ are achievable in practice; some measurements show values < 0.1 ng I-TEQ/Nm³.

Applicability: Applicable both to new and existing plants

Cross-media effects: The amount of energy for lignite coke powder dosage is not considerable. The filter dust contains the lignite coke powder and slightly increased PCDD/F amounts but this does not interfere with dust recycling.

Attention has to be paid to carbon content of the dust mixture abated at the bag filter which is about 3% average with local concentrations up to 5% which could be ignitable.

Reference plants: ARES, L-Schiffflange; Gerlafingen Stahl AG, CH-Gerlafingen (the lignite coke dosage went into operation in September 1998 in addition to post-combustion)

Driving force for implementation: The main driving force for the implementation of this technique are stringent emission limit values requiring for PCDD/F emission limits < 0.5 ng I-TEQ/Nm³.

Operational data and economics: Investment for the total off gas flow (primary and secondary off gases) from a EAF plant producing about 1 Mt steel/a is about 300000 Ecu₁₉₉₇.

Reference literature: not available

EP.4 EAF slag recycling

Description: In an EAF the slag amounts to about 100-150 kg per tonne of steel produced (see Table 9.1). EAF slag can be regarded as an artificial rock, similar to natural rock, consisting of iron-oxides (FeO), lime (CaO), silicium-oxide (SiO₂), and other oxides (MgO, Al₂O₃, MnO) (see Table 9.4). EAF slags are characterised by high strength, good weathering resistance, and also high resistance against polishing. They also have properties, that make them suitable for use in hydraulic engineering [Heinen, 1997]. An important criterion for the use of EAF slag in general is the constancy in volume, which depends on the presence of free lime.

Most of the slags from low carbon steel grades are relatively low in free lime (see Table 9.4) and are suitable for various applications like road construction, earthfill, hydraulic engineering. The deciding factors with respect to these uses are environmental acceptability and structural suitability. If the required legal conditions for use in construction are met the EAF slag has to be crushed, screened, and sized for use. Ferrous slag components are separated via magnetic separators. The treated slag is used in various construction purposes, also dependent on the grain size. Figure 9.19 shows a processing scheme for a German plant for slag preparation. In 1994, about 90% of EAF slags generated by the production of non- and medium-alloyed steel in certain EAFs have been used [Heinen, 1997]. Slags arising at high grade steel production are only used to a limited extent, so far. Possible uses may be also in the road construction area, after a preparing treatment.

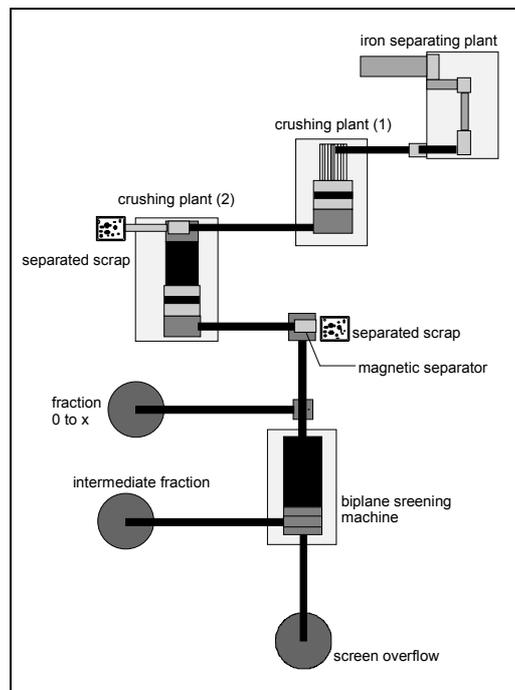


Figure 9.19 : Processing scheme of a plant for slag preparation – [D Rentz, 1997

Options to use the wide spectrum of secondary metallurgy slags are limited. Grain size and constancy of volume are decisive factors for the use of secondary metallurgy slags. They sometimes may be used in the construction area. But a significant share of the arising slags has to be landfilled, as hardly an option for prevention, reduction, or utilisation exists.

Main achieved emission levels: Slag from EAF producing carbon or low alloyed steel can be treated with subsequent reuse in road construction.

Applicability: Applicable both to new and existing plants.

Cross-media effects: The treatment of slags requires energy. Attention has to be paid to alkaline fumes when the slag is containing free CaO (9.2.2.1.3).

Reference plants: BSW, D-Kehl (treatment of slag with subsequent use for construction purposes);

Georgsmarienhütte GmbH, D-Georgsmarienhütte (selling the slag for external preparation with subsequent use for road construction – slag from EAF and secondary metallurgy are mixed);

Preussag Stahl AG, D-Peine (treatment and use in the construction sector)

ARES, L-Schiffange; ProfilARBED, L-Differdange; ProfilARBED, L-Belval (high performance road surfacing, hydraulic engineering and other applications).

Driving force for implementation: The main driving forces are limited space for landfilling and cost aspects like taxes on landfilled wastes.

Operational data and economics: not available

Reference literature: [D Rentz, 1997]

EP.5 EAF dust recycling

Description: Depending on the type of steel produced, about 10-20 kg/t steel of dust are separated from the off gas (see Table 9.1). In case of very low quality scrap up to 25 kg dust/t steel can occur.

Separated dusts obtained by the gas cleaning facilities usually contain a significant share of heavy metals (see Table 9.6). These are toxic and might be leachable, necessitating special care for further processing and possibly the landfill of the dusts.

Generally, there are several options of handling EAF dust, which can be classified roughly into three categories [Kemeny, 1994]:

- Chemical stabilisation or vitrification (can not be considered as appropriate, because other reasonable options do exist)
- Recycling of dusts by returning them to the EAF,
- Hydrometallurgical and pyrometallurgical processes for zinc recovery and recovery or removal of other heavy metals.

These options are desirable to different degrees according to their potential to satisfy the aim of prevention and control of environmental pollution. The use of the iron and heavy metal content of the dust is preferred compared to landfilling.

Recycling of precipitated dusts

Recycling of precipitated EAF dusts for zinc enrichment by returning them to the EAF results in certain impacts on the steelmaking process. On the one hand, recycling decreases the volumetric disposal rate of the dust and increases its zinc content (up to 30-40%) and also the iron content of the dust is returned in EAF process. On the other hand dust recycling possibly reduces furnace efficiency and rises the consumption of electrical energy (appr. 20-30 kWh/t). Technically, the returning of dusts is limited to a certain share of the total dust yield, depending on each steelmaking facility. Also the method of dust addition to the furnace affects the performance of the furnace. To improve performance, some form of pretreatment to agglomerate the dust, like pelletising or briquetting, is usually beneficial, as it reduces the share of dust that is just blown through the furnace. According to figures in literature, the zinc content of the dust and the dust loading increase at the filter can vary, depending on the blow through rate, between 27-32%. [Kemeny, 1994]. For example, a German electric steelmaking plant recycles 75% EAF dust of an original yield of 20-22 kg/t and has finally to take care of about 50% of the dust with an average zinc content of 35%. Generally, the dust is added at the beginning of each melting phase. In principle, the feasibility of EAF dust recycling depends on many factors, that may be dissimilar for different plants.

Zinc recovery and removal of heavy metals

Processes for zinc recovery and recovery or removal of other heavy metals are suitable options for reclaiming valuable resources, that have already been mined and treated, at least once. Pyrometallurgical and hydrometallurgical options exist for the recovery of zinc, in principle. For dusts from production of carbon/low alloyed steel, different techniques exist and are proved like the Waelz process (which is the most important one), the ESINEX process and others [Hoffmann, 1997; Rentz, 1996].

For dusts from production of high alloyed steel also processes for their recycling exist (ScanDust plasma process, B.U.S process [Helgeson, 1995; Kola, 1996]).

Main achieved emission levels: Quantitative reuse of dust can be achieved.

Applicability: Applicable both to new and existing plants.

Cross-media effects: Energy is used for transport and recycling. In case of pelletisation the dust before transport/recycling additional energy is needed for pelletisation and additional dust emissions can occur.

Reference plants: Plants recycling the dust to the EAF: Georgsmarienhütte GmbH, D-Georgsmarienhütte;
Plants for dust recycling to external plants: Many plants in the EU

Driving force for implementation: The main driving forces are limited space for landfilling, stringent standards for landfilling and cost aspects like taxes on landfilled wastes.

Operational data and economics: not available

Reference literature: [D Rentz, 1997; Rentz, 1996; Kemeny, 1994]

9.4 Conclusions

In understanding this chapter and its contents, the attention of the reader is drawn back to the preface of this document and in particular the fifth section of the preface: “How to understand and use this document”. The techniques and associated emission and/or consumption levels, or ranges of levels, presented in this chapter have been assessed through an iterative process involving the following steps:

- identification of the key environmental issues for the sector; for electric arc furnace steelmaking these are dust, organochlorine compounds, energy efficiency and recycling solid wastes;
- examination of the techniques most relevant to address those key issues;
- identification of the best environmental performance levels, on the basis of the available data in the European Union and world-wide;
- examination of the conditions under which these performance levels were achieved; such as costs, cross-media effects, main driving forces involved in implementation of this techniques;
- selection of the best available techniques (BAT) and the associated emission and/or consumption levels for this sector in a general sense all according to Article 2(11) and Annex IV of the Directive.

Expert judgement by the European IPPC Bureau and the relevant Technical Working Group (TWG) has played a key role in each of these steps and in the way in which the information is presented here.

On the basis of this assessment, techniques, and as far as possible emission and consumption levels associated with the use of BAT, are presented in this chapter that are considered to be appropriate to the sector as a whole and in many cases reflect current performance of some installations within the sector. Where emission or consumption levels “associated with best available techniques” are presented, this is to be understood as meaning that those levels represent the environmental performance that could be anticipated as a result of the application, in this sector, of the techniques described, bearing in mind the balance of costs and advantages inherent within the definition of BAT. However, they are neither emission nor consumption limit values and should not be understood as such. In some cases it may be technically possible to achieve better emission or consumption levels but due to the costs involved or cross media considerations, they are not considered to be appropriate as BAT for the sector as a whole. However, such levels may be considered to be justified in more specific cases where there are special driving forces.

The emission and consumption levels associated with the use of BAT have to be seen together with any specified reference conditions (e.g. averaging periods).

The concept of “levels associated with BAT” described above is to be distinguished from the term “achievable level” used elsewhere in this document. Where a level is described as “achievable” using a particular technique or combination of techniques, this should be understood to mean that the level may be expected to be achieved over a substantial period of time in a well maintained and operated installation or process using those techniques.

Where available, data concerning costs have been given together with the description of the techniques presented in the previous chapter. These give a rough indication about the magnitude of costs involved. However, the actual cost of applying a technique will depend strongly on the specific situation regarding, for example, taxes, fees, and the technical characteristics of the installation concerned. It is not possible to evaluate such site-specific

factors fully in this document. In the absence of data concerning costs, conclusions on economic viability of techniques are drawn from observations on existing installations.

It is intended that the general "BAT" in this chapter could be used to judge the current performance of an existing installation or to judge a proposal for a new installation and thereby assist in the determination of appropriate "BAT" based conditions for that installation. It is foreseen that new installations could be designed to perform at or even better than the general "BAT" levels presented here. It is also considered that many existing installations could reasonably be expected, over time, to move towards the general "BAT" levels or do better.

While the BREFs do not set legally binding standards, they are meant to give information for the guidance of industry, Member States and the public on achievable emission and consumption levels when using specified techniques. The appropriate limit values for any specific case will need to be determined taking into account the objectives of the IPPC Directive and the local considerations.

For electric arc furnace steelmaking, the following techniques or combination of techniques are considered as BAT. The order of priority and the selection of the techniques will differ dependent upon the local circumstances. Any other technique or combination of techniques achieving the same or better performance or efficiency can also be considered; such techniques may be under development or an emerging technique or already available but not mentioned/described in this document.

1. Dust collection efficiency
 - With a combination of direct off gas extraction (4th or 2nd hole) and hood systems
or
 - dog-house and hood systems *or*
 - total building evacuation

98% and more collection efficiency of primary and secondary emissions from EAF are achievable.
2. Waste gas de-dusting by application of:
 - Well-designed fabric filter achieving less than 5 mg dust/Nm³ for new plants and less than 15 mg dust/Nm³ for existing plants, both determined as daily mean values.

The minimisation of the dust content correlates with the minimisation of heavy metal emissions except for heavy metals present in the gas phase like mercury.
3. Minimising of organochlorine compounds, especially PCDD/F and PCB emissions, by means of:
 - appropriate post-combustion within the off gas duct system or in a separate post-combustion chamber with subsequent rapid quenching in order to avoid de novo synthesis *and/or*
 - injection of lignite powder into the duct before fabric filters.

Emission concentrations of PCDD/F 0.1 - 0.5 ng I-TEQ/Nm³ are achievable.
4. Scrap preheating (in combination with 3.) in order to recover sensible heat from primary off gas
 - With scrap preheating of part of the scrap about 60 kWh/t can be saved, in case of preheating the total scrap amount up to 100 kWh/t liquid steel can be saved. The applicability of scrap preheating depends on the local circumstances and has to be proved on a plant by plant basis. When applying scrap preheating it has to be taken care of possibly increased emissions of organic pollutants.

5. Minimising solid waste/by-products

For solid wastes, the following techniques are considered BAT in descending order of priority:

- Minimisation of waste generation
- Waste minimisation by recycling of EAF slags and filter dusts; depending on local circumstances filter dust can be recycled to the electric arc furnace in order to achieve a zinc enrichment up to 30%. Filter dust with zinc contents of more than 20% can be used in the non-ferrous metal industry.
- Filter dusts from the production of high alloyed steels can be treated to recover alloying metals.
- For solid wastes, which can not be avoided or recycled, the generated quantity should be minimised. If all minimisation/reuse is hampered, controlled disposal is the only option.

6. Emissions to water

- Closed loop water cooling system for the cooling of furnace devices.
- Wastewater from continuous casting
 - Recycling of cooling water as much as possible
 - Precipitation/sedimentation of suspended solids
 - Removal of oil in shimming tanks or any other effective device.

In principle the techniques according to items 1 - 6 are applicable to new as well as to existing installations considering the mentioned prerequisites and the preface.

9.5 Emerging techniques and future developments

In this paragraph, a number of techniques are mentioned which are not (yet) applied at industrial scale.

Scrap sorting

The emissions of organochlorine compounds, specially PCB can be significantly reduced by minimising the input with the scrap. PCB are mainly contained in small capacitors in several technical devices like washing machines, (hair)driers, cooker hoods, oil burners, fluorescent lamps etc.. The removal of capacitors is a task for the operators of shredder plants. Nevertheless this possibility may be important for the Emissions of EAF. The most important reason that the removal of capacitors is not performed yet seems to be the high costs for it.

New furnace concepts

In recent years a number of new furnace types have been introduced, that might be realised at industrial scale. Such furnaces belonging are presented in the following.

- **Comelt EAF:**

The Comelt furnace is an EAF on a DC basis with side electrodes provided by VAI [Berger, 1995]. In most cases the furnace is featured with four slanted electrodes, resulting in electric energy transmission by four inclined DC arcs. Other features of this concept are integrated shaft scrap preheating, a complete off gas collection in each operating phase and a lowered sound level. The essential advantages, according to the manufacturer, are

- High productivity (tap-to-tap times of less than 45 min),
- Reduction of total energy consumption by integrated scrap preheating (appr. 360 MJ/t compared to conventional EAF),
- Reduction of electrode consumption (approximately 30%),
- Complete off gas collection at all times and a reduction of off gas volume by up to 70%,
- Reduction in maintenance costs due to a simpler plant design,
- Reduced noise level by up to 15 dB(A).

- **State of Development / Realisation:**

A pilot electric arc furnace using the Comelt melting principle has been implemented and investigated by revamping a 50 t plasma primary melting furnace in the former LD steel shop in Linz [Berger, 1995].

- **Contiarc furnace:**

The Contiarc electric arc furnace is a continuously operating annular shaft furnace with a central DC electrode, though in principle it can also be heated by alternating current [Reichelt, 1996]. The shaft, consisting of an outer and an inner vessel, is charged continuously with scrap. By doing so, the scrap is preheated by ascending hot furnace gases (integrated high temperature scrap preheating). This concept with tapping during the melting operation reaches a power-on time of almost 100%. Further advantages according to the furnace providers are:

- Reduced energy losses (720 MJ/t less than with conventional furnace systems),
- Waste gas and dust volumes are considerably reduced (waste gas: 150000 t m³(STP) to 900000 t m³(STP); dust content: up to 40% less for a 100 t/h Contiarc furnace) requiring a lower capacity of the gas cleaning system and also lower electric power consumption (82.3 MJ/t),
- Gas-tight furnace enclosure captures all primary and nearly all secondary emissions,
- Advantages in production costs,
- Reduced electrode consumption (DC furnace: 0.8 kg/t less than AC furnace).

A pilot electric arc furnace has been implemented and investigated at the laboratory of RWTH Aachen; as a next step a demonstration plant is planned [Reichelt, 1996].

10 NEW/ALTERNATIVE IRONMAKING TECHNIQUES

10.1 Introduction

Iron has been made in blast furnaces for more than 500 years. During that time, the blast furnaces have evolved into highly efficient reactors. However, other techniques are now available which present a challenge to the blast furnace route for pig iron production.

Blast furnaces require coke, and coke plants are expensive and have many environmental problems associated with their operation. Thus, it would be beneficial from an economical and environmental point of view to produce iron ore without the use of coke. Nowadays, nearly all blast furnaces reduce their coke consumption significantly by means of reductant injection at tuyeres. However, coke can never be fully replaced in a blast furnace because of its burden supporting function. The minimum blast furnace coke rate is approximately 200 kg/t pig iron.

In order to achieve an efficient operation from an energetic and economic point of view, large blast furnaces are needed. These blast furnaces have a large and constant output. Thus, the capital costs are high and the flexibility is low. Nowadays, there is in some cases a requirement for a more flexible production in smaller production units, in order to meet the requirements of the clients.

There is an increasing production of steel from scrap in electric arc furnaces (EAF). Production of steel from scrap consumes considerably less energy compared to production of steel from iron ores. The problems with the quality of scrap-based steel introduces restraints and the use of direct reduced iron (DRI) as feedstock enlarges the possibilities of the EAF-steelmaking route.

Summarising, the following aspects put pressure on the blast furnace production route of steel:

- Environmental aspects of sinter plants
- Environmental and economical aspects of the coke oven plant;
- Relative inflexibility and scale of the pig iron production;
- Increasing competition by the scrap based- and DRI-EAF steelmaking route.

But the advantages of the BF route as regards recycling capability and economical investment should be recognised.

This has triggered the improved environmental and economical operation of the blast furnace route and the development of alternative routes for ironmaking.

Two main types of alternative ironmaking which can be considered as proven types of alternative ironmaking are following:

1. Direct reduction (DR)
Direct reduction involves the production of solid primary iron from iron ores and a reducing agent (e.g. natural gas). The solid product is called Direct Reduced Iron (DRI) and is mainly applied as feedstock in electric arc furnaces (EAF). The direct reduction process has been commercialised since the 1970's and a variety of processes have been developed.
2. Smelting reduction (SR)
This involves combining iron ore reduction with smelting (cf. blast furnace) in a reactor, without the use of coke. The product is liquid pig iron, which can be treated and refined in the same way as pig iron from the blast furnace. Today, only one variant of SR commercially proven, but a number of variants are in an advanced state of development.

Next to the developments in ironmaking, there is a tendency towards continuous processes instead of batch processes. The shift from ingot casting to continuous casting in the 1980's is a

representative example of this. In future, batch steelmaking (e.g. LD-converter, EAF) will probably be replaced by continuous steelmaking processes. In Figure 10.1 an overview of the previous, the current and the future routes for iron- and steelmaking are given.

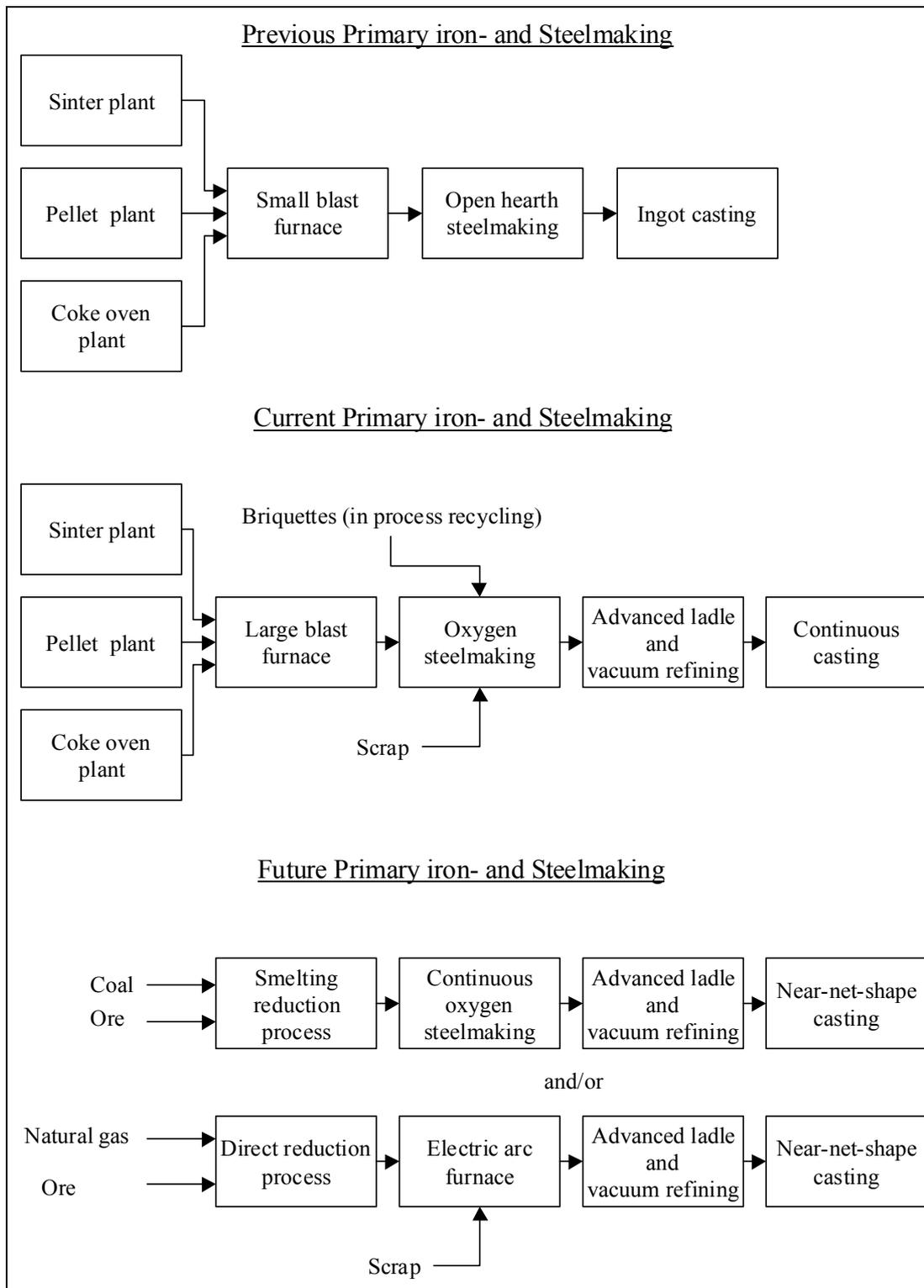


Figure 10.1 : Previous, current and alternative iron- and steelmaking processes in the world – based on [Freuhan, 1993]

10.2 Direct reduction (DR)

10.2.1 General

The concept of direct iron reduction is more than 45 years old, but the first commercial plants were built in the late 1960s. Because the leading direct reduction processes require a cheap source of natural gas, most of the plants are situated in the oil- and gas-rich belt around the equator. To date, direct reduction has not made a significant breakthrough. In 1996/97, approximately 36.5 million tonnes of direct reduced iron were produced. That is 4.4% of the world pig iron production.

Direct reduction involves the reduction of iron ore to metallic iron in the solid state. Thus, process temperatures are less than 1000°C. A solid product called direct reduced iron (DRI), is produced. DRI has a metallisation rate of >92% and a carbon content of < 2%. The direct reduced iron is normally used as feedstock for electric arc furnaces. A drawback of DRI is that it can pose a fire hazard. Therefore, DRI can be melted into briquettes, so called hot briquetted iron (HBI), when the product has to be stored or transported over some distance.

10.2.2 Available processes

Several methods have been used over the past 50 years to produce direct reduced iron. In practice, three major processes operate: MIDREX, HyL (I, II and III) and FIOR. Only recently five new techniques have been developed: FASTMET, IRON CARBIDE, CIRCORED, INMETCO and FINMET.

Approximately 92% of the DRI is produced by using (reformed) natural gas as a fuel. In a limited number of sites, coal is used as a fuel. As feed stock, iron ore pellets and lump ore are used in processes with a shaft furnace (MIDREX, HyL) and fines and concentrates are used in processes with a fluidised bed (CIRCORED, FINMET, IRON CARBIDE) or a rotary hearth furnace (FASTMET, INMETCO).

The shaft furnace has evolved as the reduction reactor for gas-based processes. Two major processes operate: MIDREX (22.9 Mt/a in 1997) and HyL III (6.9 Mt/a). The comparatively small FIOR-unit in Venezuela (0.4 Mt/a) uses a fluidised bed for the reduction of iron ore. HyL I and HyL II use batch reactors to reduce the iron ore, but these processes will almost certainly be replaced by HyL III in due time. In 1995, a FASTMET pilot plant was commissioned [Nagai, 1995]. Two FINMET plants each with a capacity of 2 Mt/a are under construction in Australia and Venezuela. A CIRCORED plant with a capacity of 0.5 Mt/a is under construction in Trinidad.

An alternative to direct reduced iron is iron carbide (Fe_3C). Iron carbide is produced by means of direct reduction also, but the product contains approximately 90 wt.% Fe_3C . The carbon content is relatively high: 6 wt.%, which provides enough energy to reduce electricity consumption in the electric arc furnace. Iron carbide can be used in the same applications as direct reduced iron. The first commercial iron carbide plant, of capacity 300000 metric tonnes per year has been commissioned in 1995 at Trinidad (actual production in 1998 is 150000 t/a).

In Table 10.1, the characteristics of the commercially available types of direct ironmaking are summarised.

Process	MIDREX	HyL III	IRON CARBIDE	FASTMET/ INMETCO	FINMET	CIRCORED
Status	Industrial	Industrial	Industrial	Industrial	Industrial* ¹	Industrial* ¹
Type of reactor	Shaft	Shaft	Fluid bed	Rotary hearth	Fluid bed	Fluid bed
Iron source	Pellet/ Lump ore	Pellet/ Lump ore	Fines: 0.1-1 mm	Fines/ Concentrates	Fines 0.1-12 mm	Fines 0.1-1.0 mm
Type of fuel	Natural gas	Natural gas	Natural gas	Coal/Natural gas	Natural gas	Natural gas
Utilities	-	Steam	Steam	-	Steam	Steam
Peripheral facilities	Reformer	Reformer CO ₂ -removal	Reformer	-	Reformer CO ₂ - Removal	Reformer CO ₂ - Removal
Typical plant capacity (kt/a)	1000	1000	320	450	500	500
Energy Input (GJ/t product)	10.5	11.3	12.6	12.6	12.5	14
Product	DRI/HBI	DRI	Fe ₃ C-powder	DRI/HBI	HBI	HBI
Product metallisation (%)	>92	>92	Fe ₃ C >90%	>92	>92	>92
Product C-content (%)	1-2	1-2	<6.0	<0.2	0.5-1.5	0

*¹ under construction (Status: end of 1998)

Table 10.1 : Characteristics of direct reduction processes – based on [Nagai, 1995]

Use of DRI is appropriate in the following situations:

- When good quality scrap runs short, so that the quality of the steel products deteriorates, and it is necessary to add reduced iron to raise the quality of the raw material.
- In mini-mills built in regions where the delivery of iron sources such as scrap is difficult, or where the construction of an integrated steel plant with blast furnace is not necessary from the viewpoint of the size of the demand, in which case reduced iron can be used as the main raw material [Nagai, 1995].
- In blast furnaces where increased capacity of hot metal output is required.

10.2.3 Environmental aspects of DRI

The main benefit of a direct reduction unit compared to a blast furnace is that the direct reduction unit uses natural gas or coal as a fuel. Therefore, a coke oven plant is no longer needed, significantly reducing the emissions. The impact on the environment of a direct reduction unit itself is very limited. There is little dust emission, which is easy to collect. The water need is low and water can be recycled to a large extent. Furthermore, a methane-based direct reduction unit produces much less CO₂ than a coal based unit.

However, DRI contains some gangue (3-6%) and this leads to an increased power consumption of the electric arc furnace with increasing DRI input [Nagai, 1995]. However, no detailed emission data were available.

10.3 Smelting reduction (SR)

10.3.1 General

In the smelting reduction process, the product is liquid pig iron or (in some cases) liquid steel. More than the direct reduction process, the smelting reduction process can be seen as a direct competitor of the traditional blast furnace. The smelting reduction process has several advantages compared to the blast furnace process, which may in future lead to the adoption of smelting reduction as the main process for pig iron production. The following advantages can be mentioned:

- Smaller units, allowing a more flexible production;
- Few restrictions as to the raw materials used;
- Uses coal as fuel and omits operation of coke oven plant;
- Lower capital costs.

Disadvantages are:

- SR can not utilise fine ores
- Energy requirements and CO₂ emissions greater than BF route.
- Economics very dependent on use of exported energy.

However, economics and possibilities are still largely unknown and currently only one smelting reduction process is commercially proven.

Several smelting reduction processes are in development and only one process is currently operating on a commercial basis: Corex. The process variants differ in the number of reactors, the amount of calorific gas produced, the ore feed (pellet, lump ore or fines). The process variants that are relatively well developed will be treated in more detail in this chapter: Corex, Hismelt, DIOS, AISI-DOE/CCF and ROMELT.

10.3.2 Corex

Description: The Corex process is a two stage process: In the first step, iron ore is reduced to sponge iron in a shaft furnace by means of reducing gas. In the second step, the reduced iron is melted in the melter-gasifier vessel. Reducing gas (CO and H₂) which is used in the reduction shaft is supplied by gasification of coal by means of oxygen, forming a fixed/fluidised bed in the melter-gasifier. The partial combustion of the coal in the melter-gasifier generates the heat to melt the reduced iron. Liquid iron and slag are discharged at the bottom, by a conventional tapping procedure similar to that used in blast furnace operation.

Because of the separation of iron reduction and iron melting/coal gasifying in two steps, a high degree of flexibility is achieved and a wide variety of coals can be used. The process is designed to perform at elevated pressure, up to 5 bar. Charging of coal and iron ore is performed through a lock hopper system.

The reducing gas contains some 65-70% CO, 20-25% H₂ and 2-4% CO₂. After leaving the melter gasifier, the hot gas is mixed with cooling gas to adjust the temperature to appr. 850°C. The gas is then cleaned in hot cyclones and fed into the shaft furnace as a reducing gas. When the gas leaves the shaft furnace, it still has a relatively high calorific value and may be used as an export gas where the opportunity exists. The calorific value of the gas is estimated at 7.5 MJ/Nm³ in case of the use of a typical steam coal (28.5% volatile matter), but other coal types may result in other heating values of the export gas.

Reference plants: Corex is the only commercially proven smelting reduction process. Corex is designed and built by the Austrian company Voest-Alpine Industrieanlagenbau (VAI), A-Linz. The following plants have been commissioned:

- 1989 ISCOR Pretoria Works, South Africa (300000 metric tonnes per year);
- 1996 POSCO Pohang Works, Korea (750000 metric tonnes per year).

Following COREX plants are presently under construction:

- SALDANHA, South Africa (600000 t/a)
- JINDAL, India (2 x 800000 t/a)
- HANBO, Korea (2 x 750000 t/a)

Operational data: At the end of 1989 the first commercial Corex plant started hot metal production at ISCOR's Pretoria Works in South Africa. Due to the economic situation ISCOR decided at the end of 1997 to shut down Pretoria Works and to keep the COREX plant as a standalone iron making unit in operation to supply pig iron to the domestic and international market. Recently also the COREX plant was closed temporarily due to the dramatically changing economic situation in south Africa influenced by the Asian crisis.

At the end of 1995 the second COREX plant started its production at POSCO's Pohang Works in Korea. This plant has a nominal capacity of 700000 tonnes per year. Currently the plant is operated with a mixture of lump ore and pellets. The annual production rate is above nominal capacity.

Main achieved emission levels:

The Corex process uses coal as an energy source. Therefore, emissions from the coke oven are avoided. All the higher hydrocarbons that are liberated from the coal are cracked into CO and H₂ in the melter gasifier. Therefore, no by-products like tar, phenol, BTX, PAH etc. are generated.

The sulphur charged with the coal into the process is to a large extent picked up in the shaft furnace by DRI and calcined additives and is subsequently fed to the melter gasifier. Here, most of the sulphur is transferred to the liquid slag as in the BF route and becomes harmless to the environment. The amount of sulphur discharged from the Corex process by gas and water (2-3% of the total sulphur input) is much lower than from the traditional coke oven/sinter plant/blast furnace route (20-30%). The export gas contains 10-70 ppmv H₂S, depending on the type of coal used and the operational conditions. As oxygen (O₂) instead of air is used for the gasification of char, no significant NO_x and cyanide (CN) formation occurs. The required use of oxygen results in significant additional overall energy demands.

Dust emissions from the Corex plant are significantly less than in the traditional production route. All dust emissions at the coke oven are avoided. The dust content of the export gas is less than 5 mg/Nm³. Most of the dust which is captured in the gas cleaning system is recycled to the process.

Some performance data of the Iscor plant are given in Table 10.2.

	Unit	Using lump ore	Using pellets
Melting capacity	t HM*/hour	45	53
Specific melting capacity	t HM/m ³ per day	3.0	3.4
Coal consumption	kg/t HM	1080	1000
C _{fix} -consumption	kg/t HM	615	570
O ₂ -consumption	Nm ³ /t HM	540	500
Slag quantity	kg/t HM	450	300
Hot metal composition			
Carbon	%	4.5	4.5
Silicon	%	0.3	0.3
Sulphur	%	0.05	0.05
Phosphorous	%	0.15	0.15
Export gas			
Quantity	Nm ³ /t HM	1750	1710
Net calorific value	MJ/t HM	7.5	7.5
Composition			
CO	%	45	45
CO ₂	%	32	32
H ₂	%	16	16
Dust	mg/Nm ³	<5	<5
Emission			
Dust	g/t HM	39-139	39-139
SO ₂	g/t HM	26-333	26-333
NO _x	g/t HM	21-33	21-33
Energy consumption	GJ/t HM	17	17

* HM: hot metal

Table 10.2: Relevant performance data of the Corex plant at Iscor's Pretoria Works, South Africa – based on [Kreulitsch, 1994; Lemperle, 1993]

Cross-media effects: The reduction gas from the melter gasifier is cleaned in cyclones. The dust from these cyclones can be recycled to the melter-gasifier. The top-gas from the shaft furnace and the cooling gas (to cool the reduction gas) are cleaned in scrubbers and thus a sludge is generated. The sludge can be largely recycled into the melter gasifier after granulation or supplied to the cement industry. A small (not quantified) part may be disposed of.

The Corex process has a high specific coal consumption and a relatively large off-gas flow, with a medium high calorific value. The use of this off gas as an energy source largely determines the energetic efficiency of the process. Cooling water is supplied in a closed circuit.

Economics: Reported capital costs are: 195 Ecu₁₉₉₆/metric t hot metal

Reference literature: [Freuhan, 1994; Kreulitsch, 1994; Lemperle, 1993]

10.3.3 Processes under development

The following smelting reduction processes are in an advanced stage of development and are briefly described in this paragraph:

- HIs melt;
- DIOS;
- AISI-DOE/CCF;
- ROMELT;

In Table 10.3, the characteristics of these processes are summarised. Short descriptions of the individual processes will be given in the following pages.

Process	HIs melt (Australia)	DIOS (Japan)	AISI-DOE/CCF (USA/Netherlands)	ROMELT (Russia)
Main components	Vertical smelt Reduction	- Pre-red. fluidised bed - Gas reforming furnace - Smelting red. furnace - Oxygen plant	- Cyclone converter furn. - In-bath melting vessel - Oxygen plant	- Bath smelter
Feedstock	Fines	Fines	Fines	Fines/waste oxides
Fuel	Pulverised coal	Coal fines/granular coal	Coal fines	Coal fines
Metal product	Liquid iron	Liquid iron	Liquid iron	Liquid iron
Oxygen consumption (Nm ³ /t HM)	uses hot blast	500	430-680	750-850
Coal consumption (kg/t HM)	630-700	950	700-750	900-1200
Off gas				
Quantity (Nm ³ /t HM)	1850	2080	n/a	n/a
Cal. Value (MJ/Nm ³)	1.44	3.74	n/a	n/a
Net energy consumption (GJ/t HM)	n/a	n/a	n/a	n/a
Energy output (GJ/t HM)	2.7	7.8	4.0	n/a
Status	Pilot	Pilot	Pilot	Pilot

Legend: HM = hot metal

Table 10.3: Characteristics of the smelting reduction processes under development – based on [Freuhan, 1994; Nagai, 1995]

HIs melt

Description: Ore is ground and pre-heated prior to injection in the vertical smelting reduction vessel. In the smelting reduction vessel, the pre-reduced iron ore is reduced and smelted. Unlike most other direct smelting processes, HIs melt does not require an oxygen lance, but uses a high velocity jet of (oxygen enriched) air. Coal and dusts are injected through bottom tuyeres.

Status: A 14 tonne per hour pilot plant has been built at Kwinana, Western Australia by the HIs melt Corporation, led by CRA (Australia).

Environmental implications: Compared to blast furnace ironmaking, a fuel saving of 10% is predicted. Furthermore, operation of an iron ore pre-treatment plant (pellet plant, sinter plant) and a coke oven plant is no longer necessary. In contrast to the other smelting reduction processes, a hot blast is needed. This will probably influence the NO_x emissions of this process in a negative way.

Reference literature: -

DIOS

Description: The Direct Iron Ore Smelting (DIOS) process consists of three sub-processes: a fluidised bed pre-reduction furnace (PRF) to pre-reduce the iron ore, a gas reforming furnace (GRF) to mix coal powder into the gas, and a smelting reduction furnace (SRF) to further reduce and smelt the iron ore.

Combustion oxygen is injected from the top of the SRF. The generated carbon monoxide (CO) is used to pre-reduce the iron ore in the PRF. Nitrogen is injected in the bottom of the SRF to agitate slag in the furnace.

Status: At NKK's Keihin Works, Japan, a pilot plant is operated since 1994, producing approximately 500 tonnes of iron per day.

Environmental implications: It is expected that the energy consumption of DIOS will be 5-10% lower compared to the blast furnace route. Furthermore, the iron pre-treatment plant (pellet plant, sinter plant) and the coke oven plant are no longer needed.

Reference literature: [Kreulitsch, 1994]

AISI-DOE/CCF

Description: The AISI-DOE project and the CCF project have started out as two separate development projects.

The AISI-DOE project was a co-operative R&D project for direct ironmaking (smelting reduction) by a number of universities and American and Canadian steelmaking companies. The project was co-ordinated by the American Iron and Steel Institute and sponsored by the US Department of Energy. The aim of the project is to produce steel from pre-reduced iron ore and coal in a vertical bath smelter. The development of the vertical bath smelter has formed the most important part of the project.

The Cyclone Converter Furnace (CCF) project is a joint initiative of Hoogovens (NL) and Ilva (I). The most important part of the project is the development of the cyclone reactor. In the cyclone, the iron ore is pre-reduced and melted. The molten mixture falls into the lower part of the vessel where reduction is completed. The fuel consists of granular coal which is injected together with oxygen in the lower part of the vessel.

The high operating temperature of the cyclone reactor and the fact that it can handle a high level of entrained materials from the iron bath make the direct connection of the pre-reduction and final reduction stages possible. Combining the two stages means that the heat transfer efficiency is not critical since there is no interstage cooling. The fact that both pre-reduction and final reduction take place in one vessel marks an important difference between the CCF and the other existing units for bath smelting reduction.

The CCF project had focussed mainly on the development of the cyclone reactor.

In 1995 both parties recognised the possibility to combine their technology. With the combined techniques, a smelting reduction pilot plant can be realised.

Status: The AISI-DOE project has been operated in a number of trials, but no pilot plant has been commissioned. The CCF project has been operated on pilot plant scale, with a capacity of 20 tonnes per hour. With the combined technique a 700000 tonnes per year pilot plant is planned at IJmuiden.

Environmental aspects: Since no coke oven plant, sinter plant or pellet plant is required, a marked reduction of emissions can be achieved. Energy consumption per tonne steel will also be lower. Furthermore, power can be generated from the flue gases which exit the cyclone at about 1800°C.

Reference literature: [Freuhan, 1993; Kreulitsch, 1994; InfoMil, 1997]

ROMELT

Description: The ROMELT process has been under development in Russia for over 10 years. It is similar to other bath smelting processes, but does not use a pre-reducer. The process uses ore or waste oxides. Its coal consumption has been reported to be 900-1200 kg/metric tonne.

Status: A 500-1000 metric tonnes/day pilot plant at Novolipetsk, Russia has produced over 300000 metric tonnes. Detailed plans for a 350000 metric tonnes/year have been made.

Environmental aspects: Since no coke oven plant, sinter plant or pellet plant is required, a significant reduction of emissions compared to conventional primary ironmaking can be expected. Energy consumption per tonne steel will be lower as well.

Reference literature: [Freuhan, 1994; InfoMil, 1997]

10.4 Comparison of conventional blast furnace route with direct reduction and smelting reduction route

There are two proven types of alternative ironmaking – these are Direct Reduction (DR) processes eg MIDREX and Smelting Reduction (SR) processes eg COREX. The primary environmental benefit claimed for DR and SR processes is that they can operate without coke or sinter. This prospect might avoid the necessity for coking plants and sinter machines that potentially have significant environmental impact.

Directly Reduced Iron (DRI) processes have an active, installed production capacity of around 33 million t/a world wide, although this still contributes less than 5% of world raw steel production (1996). This compares with a current EU production in integrated iron and steelworks of around 155 million t/a in 1995. DRI processes have relatively low throughputs compared to the blast furnace and have generally been installed to take advantage of local factors such as very low cost energy and/or iron ore feed.

Dispensing with coking plants avoids emissions to air of dust and VOCs from the ovens and a variety of organic chemicals to air and water from by-products plants. Emissions from refiners processing the residual cokemaking oils and tars from the coke ovens will also be eliminated. In addition the large quantities of water used in the process will be saved. Removing sinter plants reduces releases to atmosphere of metallic/non-metallic dust and gaseous pollutants such as sulphur dioxide. The majority of blast furnaces now have casthouse fume arrestment and bell-less charging systems installed and their environmental performance will therefore be comparable with releases from reduction plants with equivalent systems.

It is important to remember that the traditional ironmaking route provides many recycling and disposal opportunities for ferruginous arisings, filter cakes and oils from down stream steel production that may not be available in many reduction processes. The traditional route also has the ability to use a wide range of feedstocks and reductants of varying quality. In order for the new technology to achieve an equivalent performance to the integrated steelmaking route, means for treating fine ores and other arisings at site would need to be provided.

Emissions from reduction plant are generally low, with particulate releases to air after abatement of the order of 10 mg/Nm^3 . Abatement tends to be based on wet technology leading to an aqueous waste stream, although this may be capable of being addressed by recycling the water or by dry cleaning. If DR or SR processes use iron pellets or sinter then the emissions associated with the processing of these materials must be considered when comparing environmental performances of the various ironmaking routes.

As DR produces no physical change of state or separation of chemical impurities, product quality is wholly dependent on the quality of feedstocks. The DRI produced may not be of equivalent quality to that of iron from blast furnaces if low quality feedstocks are used. For environmental accounting purposes, DRI needs to be in molten form to be directly comparable to blast furnace iron. The additional energy requirements and emissions connected with this physical change of state need to be considered.

Considering smelting reduction processes, large volumes of top gases are produced by COREX and energy efficiency will be poor unless the gases are utilised for power generation or used to produce more sponge iron. Coal consumption and oxygen requirements are higher than the BF route and carbon dioxide emissions are significantly greater. Oxides of nitrogen from gas reforming have to be taken into consideration in both SR and DR operations.

SR processes have still not fully matured and COREX is the only process to be commercially available at the present time. Neither DR nor SR has proven capability at the capacity requirements of modern blast furnaces. To date iron produced by such methods is not directly competitive with iron produced by the blast furnace route, although their low capital cost has made them particularly attractive in areas of the world that do not already have established BF technology but have inexpensive energy supplies.

Current trends to reintroduce coke and sinter to DRI plants, for purposes of process optimisation, may negate much of the environmental benefit originally claimed for these emerging techniques.

In Table 10.4 a comparison of the conventional blast furnace route with the direct reduction and smelting reduction route is given.

Features	Traditional BF Route ^{*1}	Direct Reduction (DR) ^{*2} (eg MIDREX)	Smelting Reduction (SR) ^{*2} (eg COREX)
Scale of Production	Long established and energy and resource efficient with unit plant throughputs of 2Mt/a and greater. Still the principal route for iron making, accounting for 95% of world production.	Gas based processes account for the vast majority of installed DR capacity world-wide, with 63% of that capacity being via the MIDREX route. Such processes have currently a maximum unit plant capacity of 1.3Mt/a DRI as produced is normally used as a replacement for scrap in the electric arc furnace steelmaking route.	SR is still an emerging technology. Only the COREX process has been commercialised. Currently, there is about 1Mt/a of installed operating capacity (two sites). The latest and largest SR unit in operation has a capacity of 700000 t/a.
Feed Stocks	Coal Coking coals required for coke making Coke breeze & anthracite (where used) for sinter plants Coal for BF injection (can be non-coking coal specification) BF Injectants Besides coal, oil (inc. waste oil), natural gas and plastics have all been injected into BF Metallics A wide range of feedstock of variable quality and specification can be used.	Coal (where used - minority of processes) Wide range of solid fuels from anthracite to lignite including charcoal (rotary kilns) Gas Sulphur content of gas must be low to avoid poisoning of reformer catalyst and effecting product quality. Metallics As no physical change of state takes place in the process high quality pellets and lump ore are required.	Coal Non-coking coals, specification requirements more flexible than for BF route Metallics Lump ore, sinter or pellets. Fine ores cannot be used directly as yet. Oxygen Large quantities of oxygen are required for the COREX process (with associated energy implications).
Energy Requirements	Typically around 17-18 GJ/t of liquid iron (less gas, steam and heating credits from carbon in iron).	Typically 10.5 - 14.5 GJ/t solid DRI (gas-based) assuming 100% lump ore operation. (Extra energy required for melting and pellets, if used).	Difficult to quantify as process efficiency is dependent on the credit given for exported power or production of more DRI by gas based DR process.
Product Quality	Stable and of dependable quality.	Product prone to re-oxidation unless passivated or briquetted. Quality highly dependent on feed quality.	Identical to BF iron.

^{*1} coke oven, sinter plant and blast furnace; ^{*2} DR and SR are emerging techniques and complete data concerning environmental effects are not yet available

Table 10.4 : Comparison of the traditional blast furnace route with the direct reduction and smelting routes for ironmaking

Continuation of Table 10.4

Features	Traditional BF Route ^{*1}	Direct Reduction (DR) ^{*2} (eg MIDREX)	Smelting Reduction (SR) ^{*2} (eg COREX)
Environmental Performance	Releases to the environment include dusts, VOC, PAH and a variety of organic chemicals from the coke ovens. Sinter plants release SO ₂ , NO _x , dust, VOC, PCB, PCDD/F and PAH (see table 4-1) while BF discharge dust and SO ₂ from cast houses (see table 6-1). The process route also uses large quantities of water. However, the route provides the recycling of various solid wastes/by-products which would not be available in many DRI processes. The desulphurising capability of the blast furnace also allows higher sulphur containing fuels and reductants to be used in an environmentally friendly manner. BF slag can be used for road construction or pelletised to make slag cement. Both by-products have the environmental advantage that they reduce the demand for primary aggregates.	As most DR processes make use of iron pellets the environmental impact of releases from the pelletisation process should be taken into account. The DRI product typically contains 2-4% gangue requiring further energy for processing and additional environmental releases to be considered. Dust releases are similar to the BF route as raw material fines are screened before processing. There is a need to provide an environmentally satisfactory route for utilisation of fines if DR is to replace traditional ironmaking. NO _x is released at the gas reforming stage. The most successful DR processes use natural gas although coal remains the largest energy source available to man. In terms of sustainable development it may be considered that gas should be reserved for the production of high value products.	In some SR processes large quantities of waste gas require to be utilised. Besides COREX energy requirements and CO ₂ emissions are higher than at the BF route. There is a need to provide and environmentally satisfactory route for utilisation of fines if SR is to replace traditional ironmaking.
Installation Costs (indicative)	1150 Millions EURO for 3.5Mt/a (including cost of sinter plant and coke ovens).	210 Millions EURO for 1.36Mt/a (assuming availability of suitable pellets or lump ore).	240 Millions EURO for 600kt/a (including cost for oxygen plant and assuming lump ore operation)

^{*1} coke oven, sinter plant and blast furnace; ^{*2} DR and SR are emerging techniques and complete data concerning environmental effects are not yet available

11 Conclusions and recommendations

The conclusions and recommendations concern the aspects timing of work, sources of information, best available techniques, level of consensus and future work.

Timing of work

The drawing up of this BREF took nearly two years. The main stations have been:

First TWG meeting (kick-off meeting)	22 – 23 May 1997
First consultation round	Sep/Oct 1997
Second consultation round	Aug/Sep 1998
Second TWG meeting	18 – 20 Nov 1998
Final draft and third consultation round	Jan/Feb 1999
Discussion of final draft at IEF meeting	18 – 19 Feb 1999
Conclusion of IEF to extend the executive summary	29 – 30 Sep 1999

Sources of information

93 comprehensive reports were submitted to the EIPPCB. These reports are very different with respect to kind of information (statistical data, description of certain techniques to reduce input mass streams or emissions by process/production integrated or end-of-pipe-techniques etc.). They have been prepared from different point of views; some focus on single aspects or media, only few cover all environmental aspects. Therefore intensive assessment, checking, questioning, verification of the submitted data was unavoidable.

Two outstanding reports were received, especially the “Dutch Notes on BAT for Production of Primary Iron and Steel” concerning integrated steelworks and, to a minor extent, the German “Report on BAT in the Electric Steelmaking Industry” concerning electric arc furnace steelmaking. Because of the predominant quality of these reports, they have been used to a large extent in this work. The availability of such documents at the beginning of the work on a BREF is essential for the achievable quality. Therefore, the strong recommendation to Member States and industrial/environmental NGOs is to prepare such documents early in such work.

Best available techniques

For the single production steps and plants of integrated steelworks the conclusions have been drawn for each of them:

- sinter plants (chapter 4.4),
- pelletisation plants (chapter 5.4),
- coke oven plants (chapter 6.4),
- blast furnaces (chapter 7.4) and
- basic oxygen steelmaking and casting (chapter 8.4).

For electric arc furnace steelmaking, BAT conclusions are contained in chapter 9.4.

The executive summary includes all these BAT conclusions.

Level of consensus

This BREF enjoys a high level of consensus. No split views had to be noted during TWG and IEF discussions. There is a broad agreement on the document. All partners in the information exchange process consider it as acceptable result. However some necessities for improvements in the future have been identified (see separate item).

Especially regarding the qualification of presented data there is the justified and imperative demand to do so. In this context it was strongly recommended to qualify the data as follows:

“If not stated otherwise, the present emission levels ‘associated with the use of BAT’ in the BAT-chapters are understood as daily averages both in case of emission to air and emission to water”. This recommendation has been made in connection with the statement that the given levels and values do match quite well with the qualification “daily averages”. However, the sources of available data do not contain such a qualification and thus, after the event, such a conclusion does not seem to be possible.

Recommendations for future work

In general the TWG and IEF representing Member States and industrial/environmental NGOs assess this BREF as a good, well-balanced document of high value.

However, some necessities for further improvements have been stressed, especially following aspects:

- development of a methodology to select techniques to consider in the determination of BAT
- development of a methodology for conclusion on BAT
- improvement of the IPPC approach and cross media evaluation
- improvement of qualification of presented emission and consumption data (sampling methods, analysis methods, time intervals, computation methods and reference conditions)
- presentation of more detailed information on aspects concerning energy, noise, materials handling, transport and storage (soil pollution) as well as health and safety.

Especially work on the improvement of qualification of data seems to have highest priority and should therefore be started as soon as possible. It is a fact that comparability of data collected all over the EU (and world) can not always be guaranteed due to different sampling methods, analysis methods, time intervals, computation methods and reference conditions. Consequently there is an imperative and urgent need for harmonisation.

Regarding the whole document a revised issue is recommended to be available in 2005.

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GLOSSARY

General

AISI	American Iron and Steel Institute
AS	Activated Sludge
BF	Blast Furnace
BFG	Blast Furnace Gas
BOD	Biochemical Oxygen Demand
BOF	Basic Oxygen Furnace
BOFgas	Basic Oxygen Furnace gas
BTX	Benzene, Toluene, Xylene
CCM	Continuous Casting Machine
CDQ	Coke Dry Quenching
COD	Chemical Oxygen Demand
COG	Coke Oven Gas
DCI	Direct Carbon Injection
DIOS	Direct Iron Ore Smelting Reduction
DRI	Direct Reduced Iron
EURO	European Currency Unit
EOS	Emission Optimized Sintering
ESP	Electrostatic Precipitator
ETP	Electrostatic Tar Precipitator
FB	Fluidized Bed
FF	Fabric Filter
HBI	Hot Briquetted Iron
MEEP	Moving Electrode Electrostatic Precipitator
MLSS	Mixed Liquor Suspended Solids
PAH	Polyaromatic Hydrocarbons
PCDD/F	Polychlorinated Dibenzo-p-Dioxins/Furans
pre-DN/N	Pre-denitrification/nitrification
PCI	Pulverized Coal Injection
RAC	Regenerative Activated Cokes
SS	Suspended solids
VOC	Volatile organic compounds

Units

bar	bar
Bq	Bequerel
°C	degree Celcius
EURO	European currency
g	gram
J	Joule
hr	hour
I-TEQ	PCDD/F toxicity equivalent
kWh	kiloWattour
l	liter
m	metre
m ²	square metre
m ³	cubic metre (water)
Nm ³	Normalized m ³ (273K, 1013 mbar)
ppm	parts per million
ppmv	parts per million, based on volume
s	second
t	ton (1·10 ⁶ gram)
vol%	Percentage of the volume
W	Watt
y	year
Ω	Ohm
K	Kelvin
a	annum (year)

Prefixes

n	nano	1·10 ⁻⁹
μ	micro	1·10 ⁻⁶
m	milli	1·10 ⁻³
c	centi	1·10 ⁻²
k	kilo	1·10 ³
M	Mega	1·10 ⁶
G	Giga	1·10 ⁹
P	Peta	1·10 ¹²
T	Tera	1·10 ¹⁵

Conversions

2.05 mg NO ₂ /Nm ³	=	1 ppmv NO ₂
2.85 mg SO ₂ /Nm ³	=	1 ppmv SO ₂
1 Watt	=	1 J/s

Elements

Al	Aluminium
As	Arsenicum
Ba	Barium
Be	Beryllium
Ca	Calcium
Cd	Cadmium
Cl	Chlorine
Co	Cobalt
Cr	Chromium
Cu	Copper
Fe	Iron
F	Fluor
Hg	Mercury
K	Potassium
Mg	Magnesium
Mn	Manganese
Na	Sodium
Ni	Nickel
Pb	Lead
Po	Pollonium
Sb	Antimony
Se	Selenium
Sn	Tin
Ti	Titanium
V	Vanadium
W	Wolfram
Zn	Zinc

Compounds

CH ₄	Methane
CN ⁻	Cyanide
CO	Carbon monoxide
CO ₂	Carbon dioxide
CO ₃ ²⁻	Carbonate
C _x H _y	Hydrocarbons
H ₂	Molecular Hydrogen
HCN	Cyanic Acid
HCO ₃ ⁻	Bicarbonate
HCl	Hydrochloric Acid
HF	Hydrofluoric Acid
H ₂ O	Water
H ₂ S	Hydrogen Sulphide
N ₂	Molecular Nitrogen
NH ₃	Ammonia
NH ₄ ⁺	Ammonium
N _{kj}	Kjeldahl-Nitrogen
NO ₂ ⁻	Nitrite
NO ₃ ⁻	Nitrate
NO _x	Nitrogen Oxides
O ₂	Molecular Oxygen
S ⁰	Elemental Sulphur

SCN ⁻	Thiocyanate
SO ₂	Sulphur Dioxide
SO ₃	Sulphur Trioxide
SO ₃ ²⁻	Sulphite
SO ₄ ²⁻	Sulphate

Indications of emissions:

1. Emissions to air
 - Mass of emitted substances related to the volume of waste gas under standard conditions (273K, 1013 mbar), after deduction of water vapour content, expressed in the units [g/Nm³], [mg/Nm³], [µg/Nm³] or [ng/Nm³];
 - Mass of emitted substances related to time, expressed in the units [kg/h], [g/h] or [mg/h];
 - Ratio of mass of emitted substances to the mass of products generated or processed (consumption or emission factors), expressed in the units [kg/t], [g/t], [mg/t] or [µg/t];
2. Emissions to water
 - Mass of emitted substances related to the volume of wastewater, expressed in the units [g/m³], [g/l], [mg/l] or [µg/l].