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Institute for Prospective Technological Studies (Seville)
Technologies for Sustainable Development
European IPPC Bureau

Draft Reference Document on Best Available Techniques for Management of Tailings and Waste-Rock in Mining Activities

Draft September 2002

Edificio Expo-WTC, c/ Inca Garcilaso s/n, E-41092 Seville – Spain
Telephone: direct line (+34-95) 4488-284, switchboard 4488-318. Fax: 4488-426.
Internet: http://eippcb.jrc.es; Email: eippcb@jrc.es

EXECUTIVE SUMMARY

PREFACE

1. Status of this document
This is a working draft.
To date the status of this draft and the future final document has not been decided upon within the European Commission.

1.1. Background
The basis for this document is the Communication from the European Commission COM(2000) 664 final (hereafter: the Communication). As outlined under Section 5.5 of this Communication, core extraction activities are not covered by Council Directive 96/61/EC of September 1996 concerning integrated pollution prevention and control (IPPC Directive)[1]. However, activities of the kind undertaken at the Baia Mare site (production of metal by leaching of gold) are already inside the scope of the IPPC Directive. Paragraph 2.5 (b) of Annex I lists "installations for the production of non-ferrous crude metals from ore, concentrates or secondary raw materials by metallurgical, chemical or electrolytic processes".

The Communication further recognises that the IPPC Directive does not cover *all* sites in the European Union, and in fact it does not cover *most* sites, where tailings management facilities are used.

Section 6 of the Communication proposes a follow-up action plan, which includes three key actions:

- amendment of Council Directive 96/82/EC of 9 December 1996 on the control of major-accident hazards involving dangerous substances (Seveso II Directive)[2]
- an initiative on the management of waste from the extractive industry (Mine and Quarry Waste Directive (MQWD))[3]
- a BAT reference document.

Although not strictly a part of a series of reference documents pursuant to Article 16(2) of the IPPC Directive this document is seeking to follow the same principles of BAT as the IPPC Directive in accordance with the current draft of the Mine and Quarry Waste Directive. It has to be seen in context with the other two initiatives.

2. The definition of BAT
Even though most installations included in this document, as mentioned above, are not covered by the IPPC Directive, it is the intention of this document to follow the IPPC principle. Hence, in order to help the reader understand the 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 IPPC 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. This document aims at introducing this approach to the management of tailings and waste-rock in mining activities. 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 that operators should take all appropriate preventative measures against pollution, in

1 OJ N° L 257 of 10 October 1996.
2 OJ N° L 129 of 18.May 1976, p. 23.
3 Working documents for the elaboration of a Proposal for a Directive on mine and quarry waste are available on internet at http://europa.eu.int/comm/environment/waste/mining.htm

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 IPPC 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."

"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.

3. Objective of this Document

Under Section 6.3 the Communication says that the BREF should deal with techniques to

- reduce everyday pollution and
- prevent or mitigate accidents.

Furthermore it states that a BAT reference document on the management of tailings and waste-rock in mining activities will contribute to the knowledge about the measures that are available to prevent similar accidents (e.g. to Baia Mare) in the future. With this information at their disposal, the licensing authorities would be in a position to require that, in the European Union, installations using tailings management facilities meet high environmental standards while retaining economic and technical viability of the sector.

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 permitting procedure foreseen by the draft MQWD.

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

Chapters 1 and 2 provide general information on the industrial sector concerned and on the industrial processes used within the sector. Chapter 3 provides data and information concerning applied techniques and current emission and consumption levels, reflecting the situation in existing installations at the time of writing.

Chapter 4 describes in more detail the emission and risk 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 permits, for example new, existing, large or small installations. Techniques that are generally seen as obsolete are not included.

Chapter 5 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 a appropriate permit conditions. It should be stressed, however, that this document does not propose emission limit values. The techniques and levels presented in Chapter 5 will 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:

Edificio Expo-WTC, c/ Inca Garcilaso, s/n, E-41092 Seville, Spain
Telephone: +34 95 4488 284
Fax: +34 95 4488 426
e-mail: eippcb@jrc.es
Internet: http://eippcb.jrc.es

Best Available Techniques Reference Document on Management of Tailings and Waste-Rock in Mining Activities

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SCOPE

The basis for this work is the Communication on the safe operation of mining activities. One of the follow-up measures suggested in this Communication is the compilation of a BAT reference document. Under Section 6.3 the Communication says to “prevent similar (to Aznalcóllar or Baia Mare) accidents in the future” and that “the processing of certain mining minerals and residues could be included.”

Against this background a stakeholder technical working group (TWG) decided on the following scope of the work:

Horizontal Scope
The mining, processing and tailings management associated with gas and liquid mining (oil and salt from brine) will not be covered in this work, because the processes are very much different from the processing of dry ores and the tailings issue is very different to the other sectors to be covered. However, metals leaching will be covered.

The underlying theme is that this work will cover mineral processing, tailings and waste-rock management of ores that have the potential for a strong environmental impact or that can be considered as examples of “good practice”. The intention here is to raise awareness of best practice across all activities in this sector.

On this basis all metals mined and/or processed in the European Union (EU15) and Candidate Countries will be covered. These are:

- Aluminium
- Cadmium
- Chromium
- Copper
- Gold
- Iron
- Lead
- Manganese
- Mercury
- Nickel
- Silver
- Tin
- Tungsten
- Zinc.

These metals will be covered no matter what mineral processing methods are used (e.g. mechanical methods such as flotation, chemical (or hydrometallurgical) methods such as leaching, etc.).

Some delegations recommended restricting the scope to metals mining tailings dams. However, the group decided, following the above-mentioned theme to also include selected industrial minerals and coal.

In order to keep the work within a reasonable time frame not all industrial minerals can be covered. Here a selection was made based on two factors: significant production within EU15 and Candidate Countries and the generation of tailings that could have a high environmental impact if not handled properly. Also some further minerals will be addressed if the management of their tailings and waste-rock can be considered examples of “good practice” that may be applicable to other minerals.

Against this background the TWG decided to include the following industrial minerals:

- Barytes
- Borate
- Feldspar (if recovered by flotation)
- Fluorspar
- Kaolin (if recovered by flotation)
- Limestone (if processed)
- Phosphate
- Potash
- Strontianite
- Talc.

It was pointed out by the TWG that there are only tailings resulting from the processing of Feldspar and Kaolin if they are recovered by flotation.

Coal will only be included if it is processed and there are tailings produced (thereby following the above-mentioned theme). Generally, this means that hard coal (or rock coal, black coal) will be covered, whereas lignite (or brown coal), which is usually not processed, is not covered. However, should there be exceptions, they will be considered case by case.

In the kick-off meeting for this document it was stated that oil shale is being processed in Estonia and large amounts of tailings are being managed. Therefore, it was decided to include this in the document.

The issue of abandoned sites for the management of tailings and waste-rock will not be addressed in this work. However, some examples of recently closed sites will be discussed.

Vertical Scope
For all minerals defined in the horizontal scope (and only for these) the document will:

- look at waste-rock management
- include topsoil and overburden if they are used in the management of tailings
- include mineral processing as relevant to tailings management
- focus on tailings management, e.g. in ponds/dams, heaps or as backfill.

The figure below illustrates the vertical scope. Shaded boxes indicate process steps covered by the document.

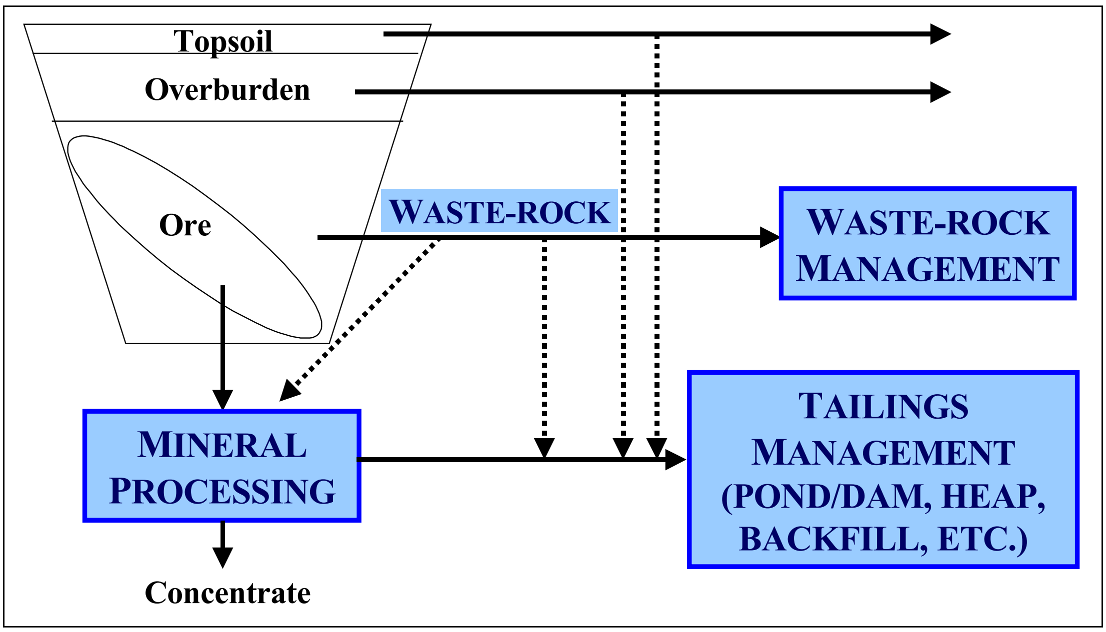

Illustration of vertical scope

In this document

“mine production” means, in the case of metals, the amount of metal in the concentrate after production, in all other cases, unless stated otherwise, the amount of concentrate by weight after mineral processing;

“Europe” means current EU Member States (EU15) and Candidate Countries.

“TMF” means “tailings management facility”, which can be a pond/dam system, backfill, tailings heap or any other way of managing tailings.

1 GENERAL INFORMATION

1.1 Industry overview

Mining is one of the oldest industries. This industry has a significant historical background all over Europe. Archaeological investigations at the Los Frailes mine in southern Spain discovered the body of a worker with a copper collar dated 1500 ad. Mining has been undertaken by many civilisations and has in many areas been a source of wealth. A good example in more recent times is the importance of coal mining (together with other “heavy industries”) in Germany for the “Wirtschaftswunder” after World War II.

In the last decades mining has moved away from smaller underground operations to larger bulk mining in open pits. This has resulted in larger amounts of residues resulting from these operations, because the often unwanted topsoil and overburden have to be removed to gain access to the ore. In many cases the amount of overburden and waste-rock that have to be transported is many times more than the tonnage of ore that is extracted. This development has also led to the fact that mining has become a more capital intensive business, where it can take many years before the invested money is “returned” through the sold product, typically the concentrates.

The purpose of mining is to make money by selling marketable products. These can be, for example, metalliferous minerals or metals, coal, or minerals that can be used in the chemical sector or for construction purposes. At any rate, the management of the residues produced, topsoil, overburden, and of special concern in this document, tailings and waste-rock typically present an undesired financial burden for any operator. Typically the mine and the mineral processing plant are designed to extract as much marketable product(s) as possible. The residue management is then designed as a consequence of these process steps.

The metal and coal mining sectors are struggling within Europe, mainly because the deposits can no longer compete on an international level. Hence the ability for these sectors to invest in non-productive expenditures such as tailings and waste-rock management is constrained. Despite the reduced mine production in these areas, consumption is steadily increasing. Therefore imports into Europe are also on the rise.

In contrast to the decline in the “traditional” mining sectors of metal and coal, the industrial mineral sector has been expanding steadily on a European scale.

The following sections try to give an overview of the sectors metal, potash, coal and oil shale mining and the tailings and waste-rock management associated with these mining activities

1.1.1 Metals

This sector will be divided and discussed in different sub-sectors:

- aluminium
- base metals (cadmium, copper, lead, nickel, tin, zinc)
- chromium
- iron
- manganese
- mercury
- precious metals (gold, silver)
- tungsten.

The following table shows that for most of these metalliferous ores, European production is small compared to overall world production.

Commodity	% of world
Iron	3 %
Bauxite	3 %
Cadmium	16 %
Chromium	12 %
Copper	7 %
Lead	11 %
Manganese	0.5 %
Mercury	17 %
Nickel	2 %
Tin	1 %
Tungsten	11 %
Zinc	12 %
Gold	1 %
Silver	10 %

Table 1.1: Production of metal concentrates (metal concentration in concentrate) within EU15 and Candidate Countries as percent of world metal concentrate production in 1999

In Europe, ore deposits containing metals in viable concentrations have been progressively depleted and few indigenous resources remain. Most concentrates are thus imported into Europe from a variety of sources worldwide.

Metalliferous minerals are finely disseminated within the ore. To liberate the desired mineral the ore has to be reduced in size to a fine powder, so that the metalliferous minerals can be recovered from the ore via froth flotation. Since flotation is a "wet" process the tailings from metal mining are typically in the form of a slurry and are managed in tailings ponds. If the metal(s) are mined in an open pit large amounts of waste-rock have to be handled on heaps or dumps.

Most metals are mined as sulphidic minerals. No matter what method of mineral processing is used some of these metal-sulphide complexes will be included in the tailings. These ores also contain pyrite, an iron sulphide, which is often not recovered and also will be part of the tailings and the waste-rock. If air and water have access to the tailings or the waste-rock acids can be formed, that can have a high environmental impact. This phenomenon is called "acid rock drainage (ARD)" and is explained in detail in Section 2.4.2.2.

1.1.1.1 Aluminium

1.1.1.1.1 Industry overview

Bauxite is a naturally occurring, heterogeneous material composed primarily of one or more aluminum hydroxide minerals, plus various mixtures of silica, iron oxide, titanium oxide, aluminosilicate, and other impurities in minor or trace amounts. The principal aluminum hydroxide minerals found in varying proportions with bauxites are gibbsite ($Al(OH)_3$) and the polymorphs boehmite ($AlO(OH)$) and diaspore ($HAlO_2$). The bulk of world bauxite production (approximately 85 %) is used as feed for the manufacture of alumina via a wet chemical caustic leach method commonly known as the Bayer process. Depending on the grade of the Bauxite ore 2 - 3 tonnes of Bauxite yield 1 tonne of alumina.

The refining of Bauxite to alumina, called "alumina refining" is part of the scope of this document. The smelting is discussed in the BREF on non-ferrous metals.

The Bauxite is in most cases imported from Australia or Guinea. The products of alumina refineries are calcined alumina and in some cases aluminium hydrate. The alumina is usually shipped to smelters [33, **EURALLUMINA**, 2002].

Looking at the aluminium demand, which directly determines the alumina demand, the worldwide tendency after a long period of continuous increase is showing a standstill position. The annual production of metal aluminium at 21 Million tonnes and correspondingly the alumina metallurgical grade around 44 Million tonnes. [33, **EURALLUMINA**, 2002].

1.1.1.1.2 Structure of the industry

There are six sites in Europe that mine Bauxite, which all together produced 2.2 Mt in 2001 [70, EAA, 2002]. However, there are ten sites that refine imported and/or mined Bauxite (see Table 1.2).

Please provide information on where the six mines are and what their production is.

Country	Plant	Production (tonnes)
France	Picheney, Gardanne	600000
Germany	Aluminium Oxid, Stade	820000
Greece	Aluminium de Greece, Distomon	710000
Ireland	Aughinish Alumina, Aughinish	1550000
Italy	Eurallumina	990000
Spain	Alcoa Inespal, San Ciprian	1200000
UK	British Alcan, Burntisland	100000
Hungary	Ajka	300000
Romania	Tulcea Oradea	330000 200000
TOTAL:		6800000

Table 1.2: Alumina refineries in Europe alumina production year 1999
[34, EAA, 2002]

The dominant Bauxite producer worldwide is Australia, with about 50 Mt in 1999. Other producers are New Guinea, Brazil, Jamaica, China and India.

The European alumina production of 6.8 Mt represents 13 percent of the world alumina production. Typically the Bauxite is refined near the mines in order to minimise the transport costs. Only high-grade Bauxite is shipped to refineries over long distances.

1.1.1.1.3 Economics of the industry

Most of the alumina is sold under long term contracts, with prices fixed at 11 to 13 % of the metal price fixed for aluminium by the London Metal Exchange (LME). After a period at USD 1500 per tonne, the Al price has now dropped due to recession in US and Japan. At present Al is priced at USD 1300 per tonne, which is expected to remain unchanged for the next two years. Hence, the corresponding alumina price hardly reaches USD 160 per tonne [33, **EURALLUMINA**, 2002].

The alumina operating cost of the EU producers ranges between USD 160 and 200 per tonne, which is higher than in most other countries [33, **EURALLUMINA**, 2002].

1.1.1.1.4 Characterisation and management of tailings and waste-rock

The tailings from the refining are a reddish slurry called "red mud" and a coarser fraction called "sand". They have an elevated pH and contain several metal complexes. Of the EU15 refineries some use thickened tailings management of these caustic tailings, some discharge into the Mediterranean and one site manages the red mud in a pond after neutralising the mud with seawater [33, **EURALLUMINA**, 2002].

1.1.1.2 Base Metals (Cadmium, Copper, Lead, Nickel, Tin, Zinc)

Please provide list of producers and their production for each metal.

1.1.1.2.1 Industry overview

Currently base metal prices are low and in many cases the mineral deposits are relatively complex from a processing point of view. These two factors combined with the high labour costs in Europe have led to some temporary and some final closures of mines.

Base metals can often be found in one mineral deposit. They are often separated in the mineral processing phase by selective flotation.

There is a big imbalance between the European mine production and the European consumption of these metals. A good example is lead where in the year 1999 the European consumption was close to 2 million tonnes which is about 6 times the amount of lead produced from European mines (350000 t) in the same year.

<u>Cadmium</u>
There are only a few cadmium minerals, such as Greenockite (CdS) or Otavite ($CdCO_3$ and as CdO). None of these minerals are of industrial importance. The chemical element cadmium (Cd) can replace zinc (Zn) in the Sphalerite mineral. Hence Cadmium is often found in the zinc-concentrate after mineral processing. In addition lead and copper ores may contain small amounts of cadmium [35, EIPPCB, 2000]. The cadmium is recovered from the concentrates in the subsequent smelting.

<u>Copper</u>
Copper is usually found in nature in association with sulphur. Pure copper metal is generally produced from a multistage process, beginning with the mining and concentrating of low-grade ores containing copper sulphide minerals, and followed by smelting and electrolytic refining to produce a pure copper cathode. Worldwide an increasing share of copper is produced from acid leaching of oxidised ores. Because of its properties, singularly or in combination, its high ductility, malleability, and thermal and electrical conductivity, and its resistance to corrosion, copper has become a major industrial metal, ranking third after iron and aluminium in terms of quantities consumed [36, USGS, 2002].

The most common copper minerals are:

- Sulphides
    - Chalcopyrite ($CuFeS_2$)
    - Chalcocite (Cu_2S)
    - Covellite (CuS)
    - Bornite Cu_5FeS_4

The yield of chalcopyrite is rather low in terms of atoms per molecule. It is only 25 %, compared to other copper minerals such as chalcocite – 67 %; cuprite – 67 %; covellite – 50 % or bornite – 50 %. However the large quantities and widespread distribution of chalcopyrite make it the leading source of copper. Chalcopyrite is a common mineral and is found in almost all sulphide deposits.

- Oxides
    - Cuprite (Cu_2O)

Cuprite has been a major ore of copper and is still mined in many places around the world. Of all the copper ores, except for native copper, cuprite gives the greatest yield of copper per molecule since there is only one oxygen atom to every two copper atoms [37, Mineralgallery, 2002].

Sulphide minerals are usually recovered using flotation. Oxides are leached.

<u>Lead</u>
Lead is found in pure sulphide ores or nowadays more in mixed ores where it is associated with zinc and small amounts of silver and copper. There have been major changes in the pattern of lead use over the years. The battery industry creates up to 70 % of the demand and is fairly stable but other uses for lead, which include pigments and compounds, protection against radiation, rolled and extruded products for the building industry, cable sheathing, shots and gasoline additives, are in decline.

The most important lead mineral for the mining industry is galena (PbS), which can contain up to 1 % Silver.

<u>Nickel</u>
Nickel (Ni) is a transition element that exhibits a mixture of ferrous and non-ferrous metal properties. It is both a siderophile (associates with iron) and a chalcophile (associates with sulphur). The bulk of the nickel mined comes from two types of ore deposits:

- laterites, where the principal ore minerals are nickeliferous limonite ((Fe,Ni)O(OH)) and garnierite (a hydrous nickel silicate), or
- magmatic sulphide deposits, where the principal ore mineral is pentlandite ($(Ni,Fe)_9S_8$).

The ionic radius of divalent nickel is close to that of divalent iron and magnesium, allowing the three elements to substitute for one another in the crystal lattices of some silicates and oxides. Nickel sulphide deposits are generally associated with iron- and magnesium-rich rocks called ultramafics and can be found in both volcanic and plutonic settings. Many of the sulphide deposits occur at great depth. Laterites are formed by the weathering of ultramafic rocks and are a near-surface phenomenon. Most of the nickel on Earth is believed to be concentrated in the planet's core [36, USGS, 2002].

Nickel is used in a wide variety of products. Most primary nickel is used in alloys; the most important of which is stainless steel. Other uses include electroplating, foundries, catalysts, batteries, coinage, and other miscellaneous applications [35, EIPPCB, 2000].

<u>Tin</u>
Nearly every continent has an important tin-mining country, Tin is a relatively scarce element, with an abundance in the earth's crust of about 2 ppm, compared with 94 ppm for zinc, 63 ppm for copper, and 12 ppm for lead. Most of the world's tin is produced from placer deposits; at least one-half comes from South-east Asia. The only mineral of commercial importance as a source of tin is cassiterite (SnO_2), although small quantities of tin are recovered from complex sulphides such as stanite, cylindrite, frankeite, canfieldite, and teallite.
[36, USGS, 2002].

<u>Zinc</u>
Sphalerite (zinc iron sulphide, ZnS) is and has been the principal ore mineral in the world. Zinc, in tonnage produced, stands fourth among all metals in world production—being exceeded only by iron, aluminium, and copper. Zinc uses range from metal products to rubber and medicines.

1.1.1.2.2 Structure of the industry

Cadmium

The world production in 1999 was about 16500 tonnes of Cadmium in the concentrate of which 14.5 % (2400 tonnes) were produced from European mines. The following chart shows the producers in Europe.

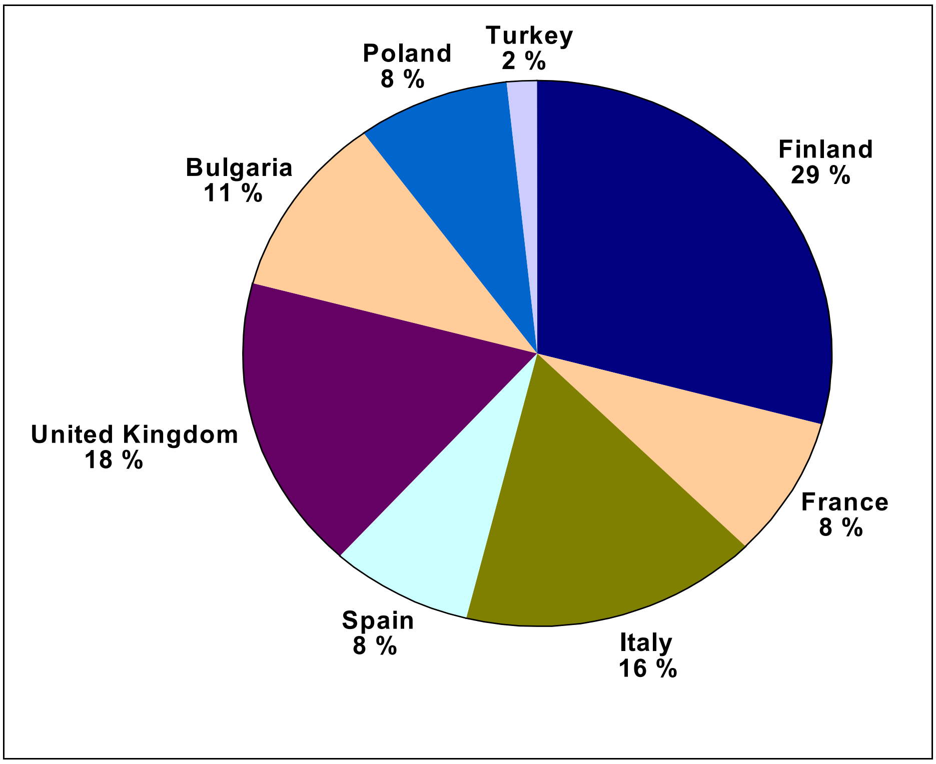

Figure 1.1: Cadmium mine production in EU15 and Candidate Countries

Copper

The world production of copper in 1999 was 12.4 million tonnes. European mine production was 890000 tonnes, which represents 7.2 % of the world production. The following figure shows the main producers in Europe.

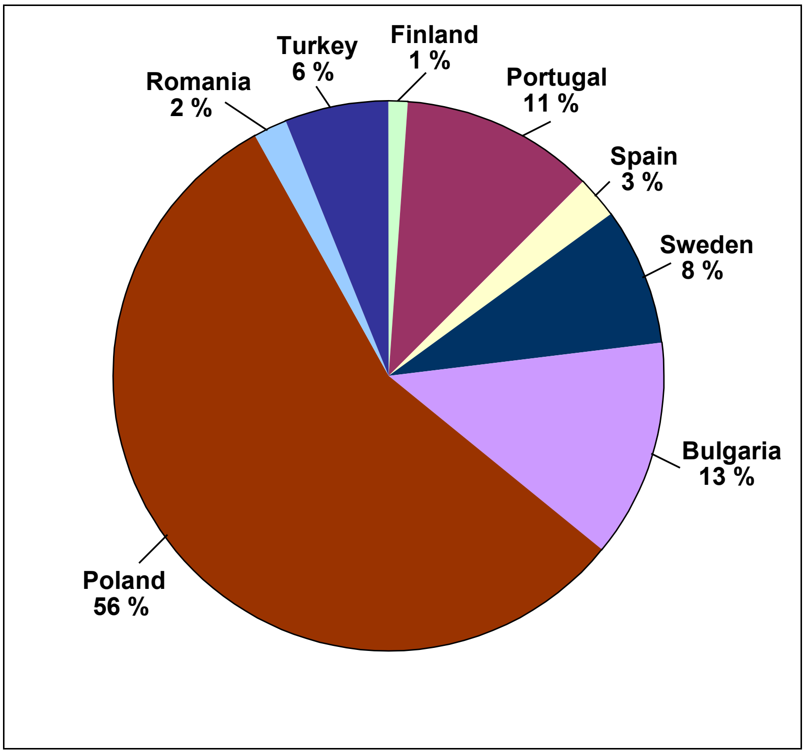

Figure 1.2: Copper mine production in EU15 and Candidate Countries

<u>Lead</u>

Lead world mine production in 1999 was 3.3 million tonnes, about one tenth of which (about 350000 tonnes) came from the European mines. The following figure shows the main producers in Europe.

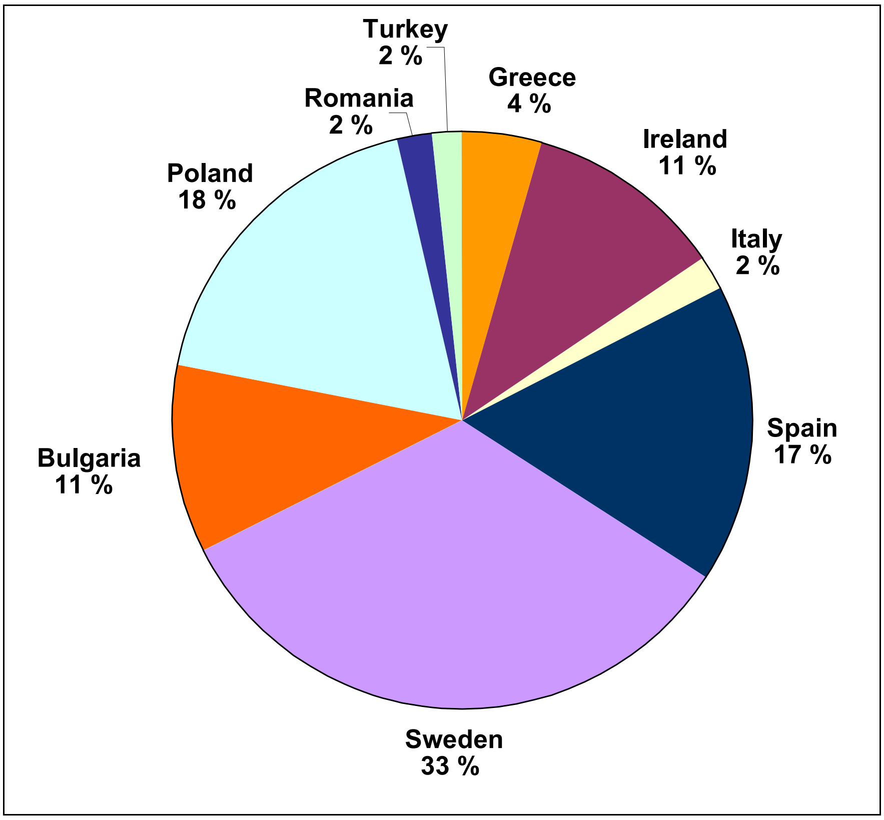

Figure 1.3: Lead mine production in EU15 and Candidate Countries

Nickel
Europe only produced only 1.4 % of the world mine production in 1999 (About 1.1 million tonnes). The following chart shows the most important producers in the world.

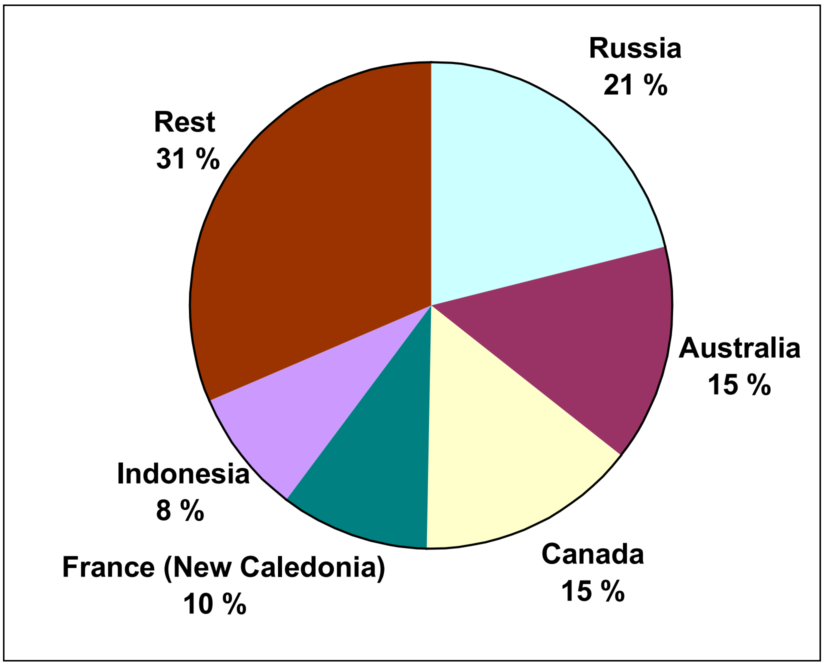

Figure 1.4: World nickel mine production in 2001

There are only 2 producers in Europe: Greece with 13462 tonnes and Finland with 1021 tonnes in 1999. However since New Caledonia is part of France, this should also be considered part of European production, which would mean that European production provides more than 11 % of the world production.

Tin
The world tin production in 1999 was about 230000 tonnes. To this Europe contributed 1 %. The only European producers are Portugal (2163 tonnes) and the UK (100 tonnes).

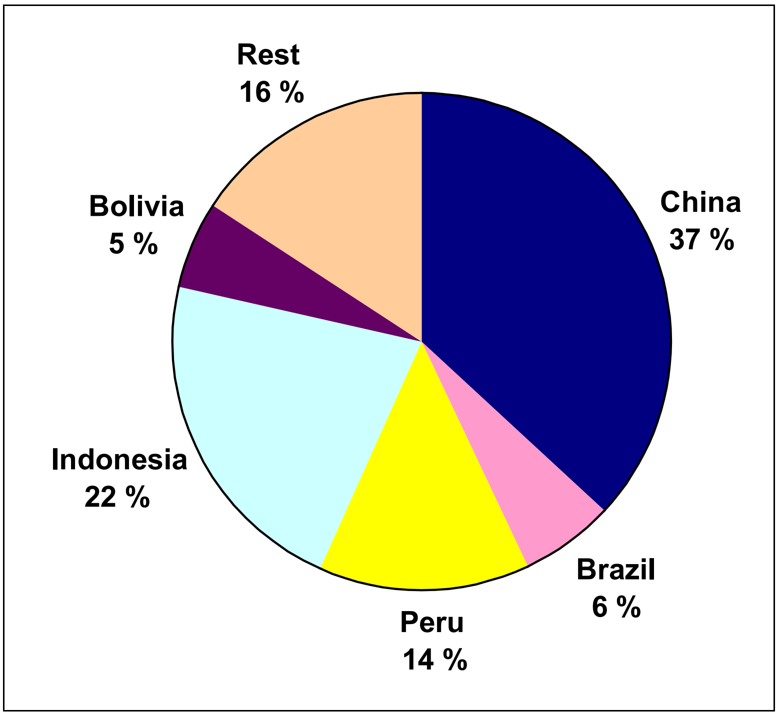

Figure 1.5: World tin mine production in 1999

As can be seen from Figure 1.5 China is by far the biggest producer of tin. This country also has the largest tin reserves.

Zinc

Europe accounted for 11.8 % of the mine world production of about 7.5 million tonnes in 1999. The following chart displays the major European zinc producers.

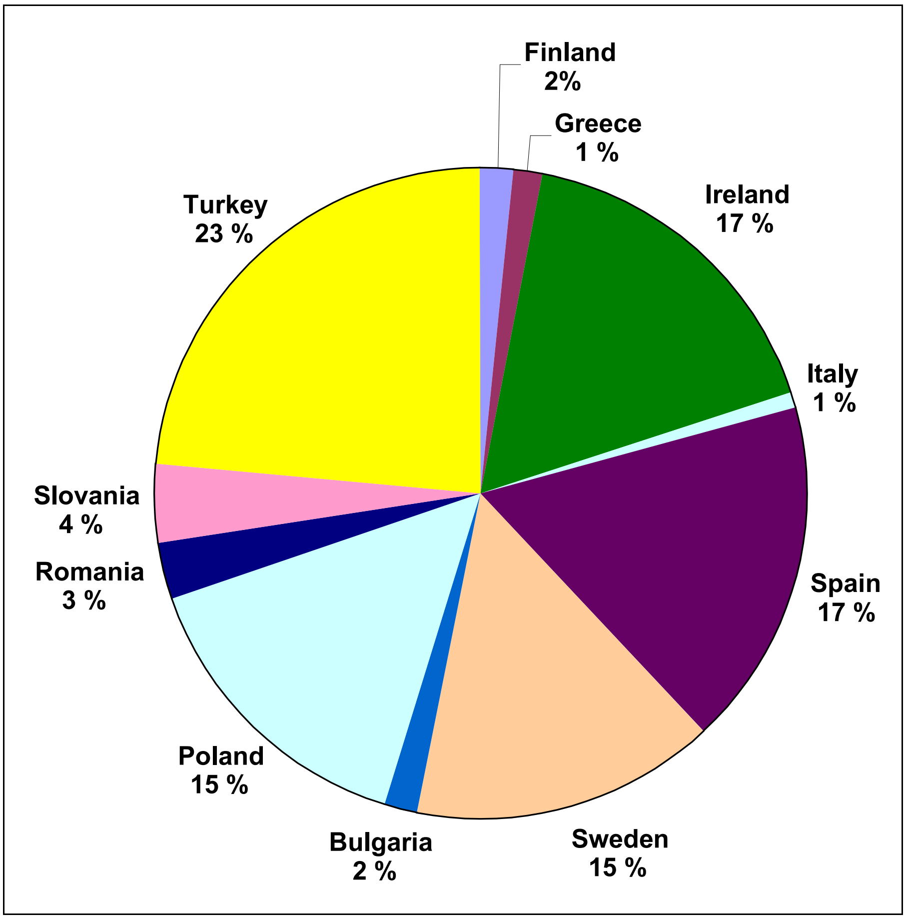

Figure 1.6: Zinc mine production in EU15 and Candidate Countries in 1999

1.1.1.2.3 Economics of the industry

Tin

Tin prices continued to decline in 2001. Industry observers attributed lower prices to an oversupply of tin in the market [36, USGS, 2002].

World tin consumption was also believed to have declined somewhat during the year.

Please provide further information for this section.

1.1.1.2.4 Characterisation and management of tailings and waste-rock

Tailings from base metal mining activities can be characterised as follows:

- a slurry of 20 - 40 % solids by weight
- containing metals
- containing sulphides
- large amounts.

The slurried tailings are managed in ponds. In some underground mines the coarse tailings are used as backfill.

The sulphide in tailings and waste-rock can oxidise when water and air have access and an acidic leachate can be generated, a phenomenon called acid rock drainage (ARD). Therefore, besides the physical stability of the tailings ponds and dams the chemical stability of acid generating tailings during operation and after the mine closure is a major issue.

The waste-rock from these activities can also have a high environmental impact if it has a net acid generating potential. Waste-rock is stacked on heaps.

1.1.1.3 Chromium

1.1.1.3.1 Industry overview

The use of chromium (Cr) to produce stainless steel and non-ferrous alloys are two of its more important applications. Chromite ($FeCr_2O_4$) is the most important ore of chromium from which it derives its name.

Chromite forms in deep ultramafic magmas and is one of the first minerals to crystallise. It is because of this fact that chromite is found in some concentrated ore bodies. While the magma is slowly cooling inside the earth's crust, chromite crystals form and because of their density, fall to the bottom and are concentrated there.

1.1.1.3.2 Structure of the industry

In Europe two countries produce significant amounts of chromium ore. One is Finland (about 250000 tonnes in 1999) from the Avesta Polarit Chrome Oy Kemi Mine, and the other is Turkey (about 430000 tonnes in 1999). Turkey is the fourth largest Chromium producer in the world. Greece produced 1000 tonnes in 1999. The European mine production represents about 12 % of the world production (5.8 million tonnes in 1999). The three major producers are South Africa, India and Kazakhstan

The concentrate from the Kemi mine is shipped directly to a stainless steel smelter owned by the same company.

1.1.1.3.3 Economics of the industry

No data has been supplied for this section. Please provide information.

1.1.1.3.4 Characterisation and management of tailings and waste-rock

The slurried tailings are managed in ponds. Currently at the Kemi site the waste-rock is managed on heaps. In the future the operation will turn from an open pit to underground mining, which will almost eliminate the production of waste-rock. Some waste-rock will then be used as backfill.

1.1.1.4 Iron

1.1.1.4.1 Industry overview

Iron ore is a mineral substance which, when heated in the presence of a reducant, will yield metalliferous iron (Fe). It almost always consists of iron oxides, the primary forms of which are magnetite (Fe_3O_4) and hematite (Fe_2O_3). However, iron ore is also mined, in the case of the "Steirischer Erzberg" as Sideroplesit, a Fe-Mg-Ca Carbonate [55, Iron group, 2002].

Iron ore is the source of primary iron for the world's iron and steel industries. It is therefore essential for the production of steel, which in turn is essential to maintain a strong industrial base. Almost all (98 %) iron ore is used in steelmaking [36, USGS, 2002].

1.1.1.4.2 Structure of the industry

From well before 1900, virtually all iron ore has been used to make steel, so the iron ore production continues to be linked to those of the steel industry. In the beginning of the 20th century the USA was the world's largest iron ore producer, accounting for about 60 % of the total yearly world output of approximately 45 million tonnes. By the end of the century the world iron ore production has grown to more than one billion tonnes per year.

In 2000, China was the largest producer in gross weight of ore produced, but because its ore was of such low grade, the country's output ranked well below Australia's and Brazil's output, with 171 and 200 million tonnes respectively. Iron ore is mined in about 50 countries. The seven largest of these producing countries account for about three-quarters of total world production, which was about 560 million tonnes in 1999. Australia and Brazil together dominate the world's iron ore exports, each having about one-third of total exports The European iron ore mining industry is of little significance on a world scale, only generating 3 % of the yearly world production.

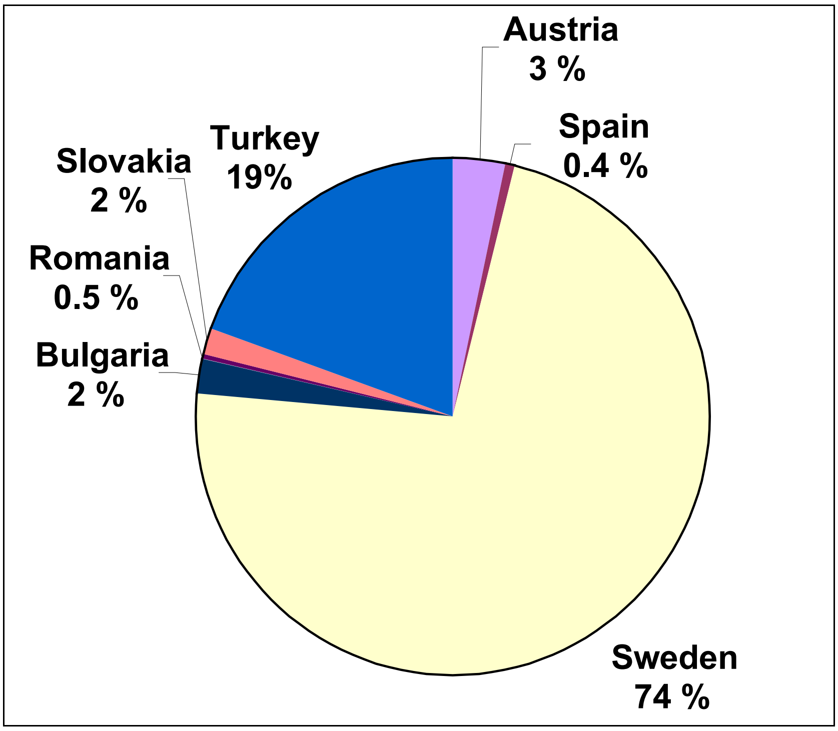

Figure 1.7: Iron mine production in EU15 and Candidate Countries in 1999

The biggest iron ore producing company in the world is CVRD of Brazil. The sales of the group reached a new record of 143.6 million tonnes in 2001. The London-based Rio Tinto group produced 115.8 million tonnes and shipped 110.6 million tonnes. Corresponding figures for the Australian/South African BHP Billiton, was 82.6 million tonnes and 84.5 million tonnes respectively. At present the "big three" control approximately 70 % the iron ore market.

Iron ore production in Western Europe is mainly concentrated in Sweden, as production of iron ore in the "minette" regions of France/Luxembourg ceased in the first half of 1990s, as did the mining in Spain. There are still small scale operations for domestic use in Turkey, Austria and

Norway, the latter also producing some for export. In Eastern Europe, the Slovak Republic, Bulgaria and Romania are represented in the statistics of iron ore producers.

Of the merchant iron ore products, 490 Mt in 2000, pellets accounted for about 90 Mt. The rest consisted of coarse ores (approximately 70 million tonnes) and fines. Iron ore fines are used as feed to blast furnaces, after sintering or pelletising processes. Pellets are split up into two types, blast furnaces use and feed for the expanding DRI/HBI-industry.
[49, Iron group, 2002]

1.1.1.4.3 Economics of the industry

The end of the 20th century saw a wave of consolidation in the iron ore industry as producers strove to reduce production costs and become more competitive. The period of consolidations is thought to have come close to an end, though there are still some potential for acquisitions and mergers. [49, Iron group, 2002]

1.1.1.4.4 Characterisation and management of tailings and waste-rock

In the case of iron ore mining in Europe, this metal is only mined in the form of oxides and the ores do not contain any sulphide minerals. Hence ARD is not a concern. Typically a coarse tailings fraction is generated which is managed on heaps. The fines are discharged into tailings ponds.

1.1.1.5 Manganese

1.1.1.5.1 Industry overview

Steelmaking accounts for most of the manganese (Mn) demand [36, USGS, 2002].

Pyrolusite (MnO_2) is the most common manganese mineral and is an important ore. The mining term "wad" is used to indicate ores that are a mixture of several manganese oxides, such as pyrolusite, psilomelane and others that are difficult to distinguish. Pyrolusite is an oxidation product of weathered manganese minerals and also forms from stagnant shallow marine and freshwater bog and swamp deposits. Minerals such as rhodochrosite, rhodonite and hausmannite are often replaced by pyrolusite [37, Mineralgallery, 2002].

In some cases manganeseis the prime product (e.g. Hotazel mine in South Africa or Nikopol mine in the Ukraine). Mangenese can also be associated with other minerals (e.g. iron-carbonates), which has a positive effect that in the steel production less manganese needs to be added [38, Weber, 2002, 12.4.]

1.1.1.5.2 Structure of the industry

The European mine production of 43344 tonnes in 1999 represents 0.5 % of the world production in the same year. The following figures show the European and the largest international producers.

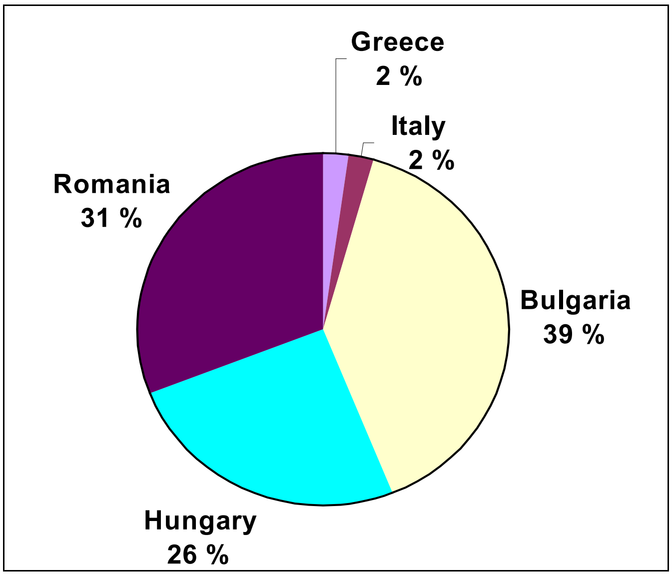

Figure 1.8: Manganese mine production in EU15 and Candidate Countries in 1999

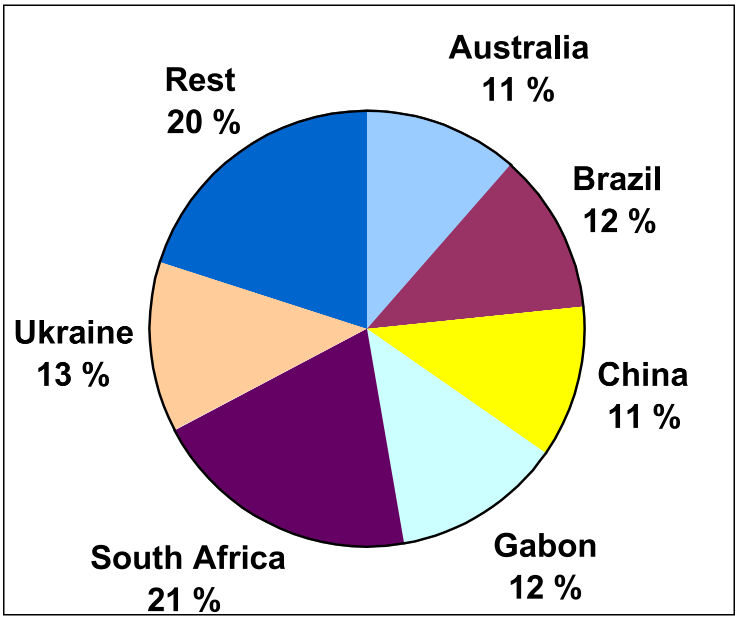

Figure 1.9: World manganese mine production in 1999

1.1.1.5.3 Economics of the industry

No data has been supplied for this section. Please provide information.

1.1.1.5.4 Characterisation and management of tailings and waste-rock

No data has been supplied for this section. Please provide information.

1.1.1.6 Mercury

1.1.1.6.1 Industry overview

Cinnabar (HgS) is the main ore of mercury. Some mines used by the Romans are still being mined today. Cinnabar shares the same symmetry class with quartz, but the two form different crystal habits [37, Mineralgallery, 2002].

Mercury is the only common metal that is liquid at room temperature. It occurs either as native metal or in cinnabar, corderoite, livingstonite, and other minerals [36, USGS, 2002].

Describe current developments is EU regarding Mercury.

1.1.1.6.2 Structure of the industry

There is only on mine in Europe that mines Mercury. the Almaden mine in Spain. However, this mine in now closed and chances are it will stay out of production. Other mines, mining other sulphides sometimes produce Mercury as a “side-product”. One example is the Pyhäsalmi Mine Oy, which produces Cu-, Zn-, Pyrite concentrates that include Cd, Hg, Au, Ag.

Other Mercury producers are … *No data has been supplied for this section. Please provide information.*

In 1999 European production represented 17.4 % of the world production of about 1700 t.

1.1.1.6.3 Economics of the industry

No data has been supplied for this section. Please provide information.

1.1.1.6.4 Characterisation and management of tailings and waste-rock

Since the tailings contain sulphides, the generation of ARD is certainly an issue with Hg. Older Hg mines, waste-rock heaps and tailings management facilities will cause problems. ARD and seepage of heavy metals can be expected for many years. However HgS is not water soluble and should therefore remain stable in the tailings and waste-rock.

1.1.1.7 Precious Metals (Gold, Silver)

1.1.1.7.1 Industry overview

Most of the gold and silver that are produced go into the manufacture of jewellery but, due to their properties such as high electrical conductivity and resistance to corrosion, they are also used as industrial metals.

Of an estimated 140000 tonnes of all gold ever mined, about 15 % is thought to have been lost, used in dissipative industrial uses, or otherwise unrecoverable or unaccounted for. Of the remaining 120000 tonnes, an estimated 33000 tonnes are official stocks held by central banks and about 87000 tonnes is privately held as coin, bullion, and jewellery [36, USGS, 2002].

In some cases gold and silver are directly turned into crude metal at the on-site mineral processing plant as doré containing typically 75 % gold and 25 % silver. In other cases, gold and silver are parts of other metal concentrates and are recovered in the smelting process [36, USGS, 2002].

There are very few true gold ores, besides native gold, because it forms a major part of only a few rare minerals. It is found as little more than a trace in a few others or it is alloyed to a small extent with other metals such as silver. A few of the minerals that bear gold in their respective formulas are in a subclass of sulphides called the tellurides. The element gold seems to have an affinity for tellurium and this is one of the few elements that gold can bond with easily. In fact only a few rare tellurides are found without gold. A few of the tellurides are nagyagite, calaverite, sylvanite and krennerite. These are all minor ores of gold but their contributions to the supply of gold pales next to native gold's own contribution. Occasionally these minerals are associated with native gold [37, Mineralgallery, 2002].

1.1.1.7.2 Structure of the industry

Of the about 2.5 million kg of gold mine production worldwide in 1999, Europe produced only 0.8 %. In the case of Silver European production represented a good 10 % of a world production of 17.3 Million kg in the same year.

The following two graphs show the main producers of gold and silver in Europe.

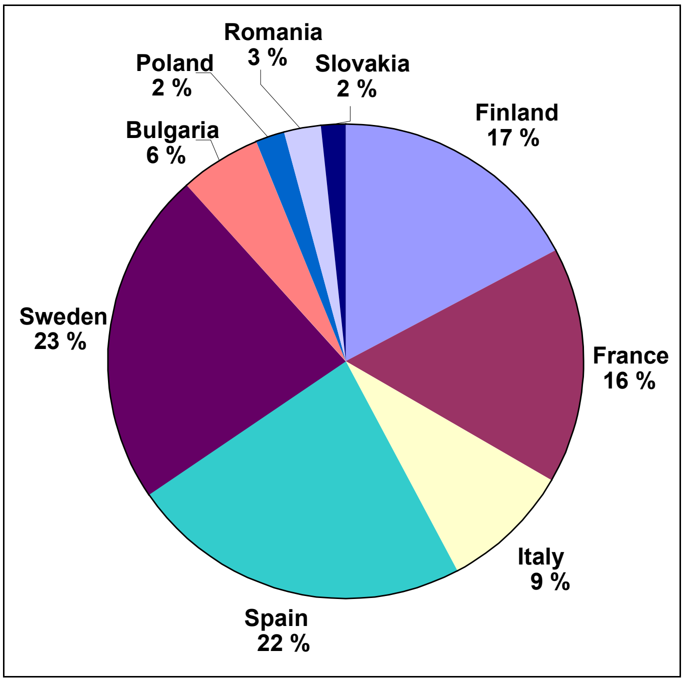

Figure 1.10: Gold mine production in EU15 and Candidate Countries in 1999

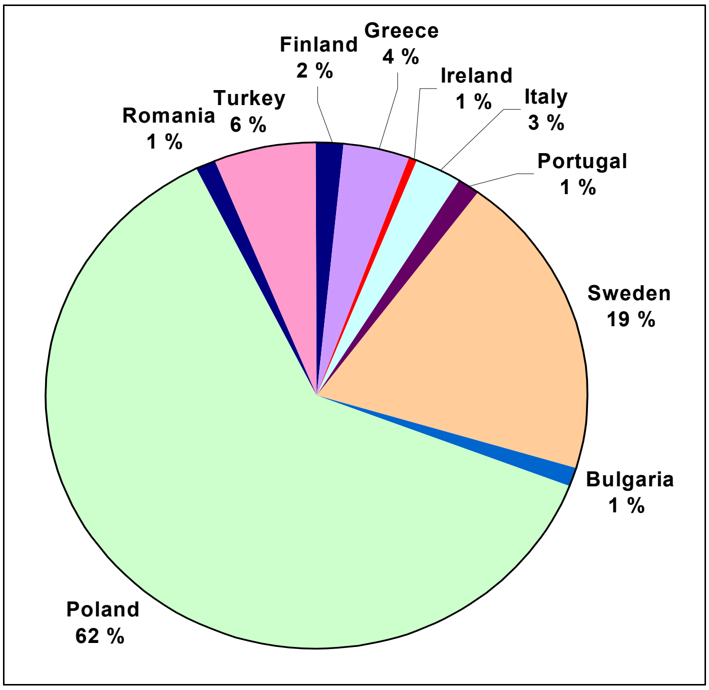

Figure 1.11: Silver mine production in EU15 and Candidate Countries in 1999

Country	Plant
Finland	Outokumpu Mining Oy Orivesi Mine
France	Salsigne
Italy	Sardinia
Spain	Rio Narcea Gold Mines S.A. Filón Sur, SA (Grupo OAK -Suiza)
Sweden	Boliden
Bulgaria	*No data has been supplied. Please provide information.*
Poland	*No data has been supplied. Please provide information*
Romania	*No data has been supplied. Please provide information*
Slovakia	*No data has been supplied. Please provide information*
Turkey	Normandy Madencilik AS Newmont Izmir-Bergama-Ovacik

Table 1.3: List of European gold producers

A new gold mine in Sweden is currently in the permitting stage.

The following figure shows the world gold mine production in 2001.

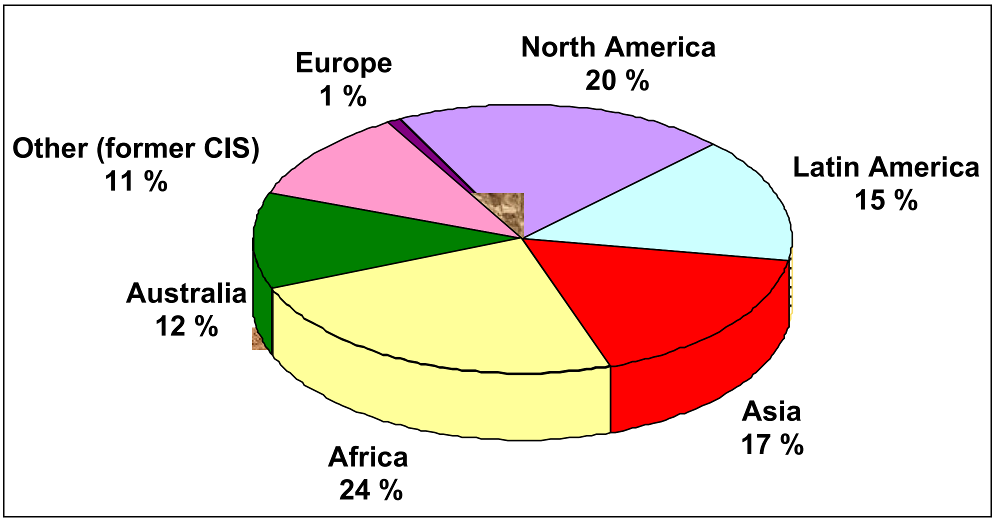

Figure 1.12: World gold mine production in 2001

The use of cyanide to leach gold has been a much discussed issue in recent years. Baia Mare accident especially brought special attention to this technique. Currently, there are about 875 gold and silver operations in the world, of which about 460 utilise cyanide [26, Mudder, ?]. Figure 1.13 shows the world distribution of mines using cyanidation.

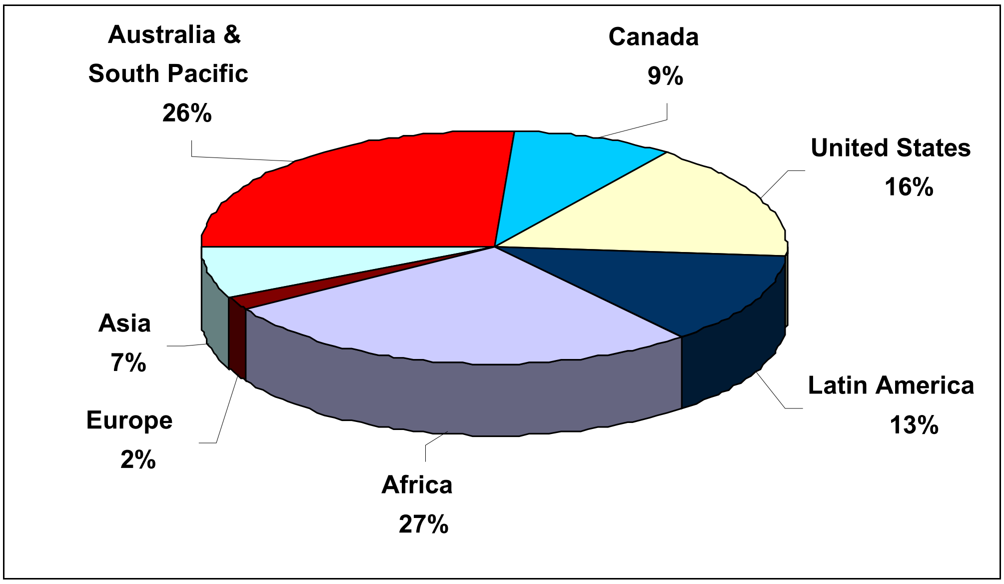

Figure 1.13: World distribution of mines using cyanidation
[26, Mudder, ?]

1.1.1.7.3 Economics of the industry

During the first 9 months of 2001, the Engelhard Corporation's daily price of gold ranged from a low of about USD 257 per troy ounce in April to a high of almost USD 294 in September. For most of the year, this price range was below USD 270. The traditional role of gold as a store of value was able to lift the price of gold out of its low trading range when terrorists attacked the United States in September 2001. In 2001, the Swiss National Bank continued selling 1300 tonnes of gold (one-half of its reserves), and the United Kingdom government completed its drive to sell 415 tonnes of gold from British gold reserves. Concerns about the true position of central bank gold sales, prospects for more consolidations within the gold mining sector, and a lack of renewed investor interest in gold kept gold prices depressed until the middle of September 2001.

1.1.1.7.4 Characterisation and management of tailings and waste-rock

Gold is a very valuable natural resource. Therefore it is still worth mining if the ore grade is in the grams/tonne-range. This results in large amounts of tailings being produced in gold mining relative to the amount of gold produced. At a gold grade of 5 g/t, 200000 tonnes of ore have to be mined to produce 1 tonne of gold.

Coarser gold particles can be recovered using gravity separation. However the finer gold particles can often only be recovered by leaching the ore with a cyanide solution. Due to the high toxicity of cyanide special attention has to be given to the tailings management where this process is applied.

There is research ongoing with the aim of replacing cyanidation with less hazardous techniques. Also new techniques to destroy cyanide in the tailings or to recycle cyanide from the tailings to the process are investigated.

Gold mining tailings are usually in the form of a fine slurry that is managed in ponds. All sites within the EU15 destroy the CN in the tailings. Both chemical and physical stability of tailings management facilities are of high importance, since the tailings can also have an ARD potential.

1.1.1.8 Tungsten

1.1.1.8.1 Industry overview

Wolframite ((Fe, Mn)WO_4, Iron manganese tungstate) is actually a series between two minerals; Huebnerite and Ferberite. Huebnerite is the Manganese rich end member while ferberite is the iron rich end member. Wolframite is the name of the series and the name applied to indistinguishable specimens and specimens intermediate between the two end members. Most specimens found in nature fall within the 20 - 80 % range of the series and these are termed wolframites. Only if they are more pure than 80 % manganese are they called huebnerite and conversely if they are 80 % iron they are called ferberite. Scheelite ($CaWO_4$, Calcium Tungstate) is an important ore of tungsten which is a strategically important metal. Scheelite is named after the discoverer of tungsten, K. W. Scheele. Although most of the worldwide production of tungsten comes from the mineral wolframite, scheelite is especially abundant in the US and provides the United States with most of its supply [37, Mineralgallery, 2002].

1.1.1.8.2 Structure of the industry

In 1999, a total of 3215 tonnes of Tungsten was produced in Europe, 2015 tonnes in Austria and 1200 tonnes in Portugal. European production accounted for 11.5 % of the world production in 1999.

The average worldwide consumption of tungsten is 40000 t (W) per year. The main producers are China (>80 %), Canada, the reformed CIS, Austria, Portugal, Bolivia and Peru [52, Tungsten group, 2002].

1.1.1.8.3 Economics of the industry

Due to low prices, many mines throughout the world have had to close during the last 2 decades [52, Tungsten group, 2002].

No further data has been supplied.. Please provide information

1.1.1.8.4 Characterisation and management of tailings and waste-rock

The slurried tailings are managed in ponds. Typically there are no sulphides contained within the ore, hence ARD is not an issue.

1.1.2 Industrial minerals

This sector will be divided and discussed in different sub-sectors:

- Barytes
- Borates
- Feldspar
- Fluorspar
- Kaolin
- Limestone
- Phosphate
- Strontianite
- Talc.

The following table shows that for most of these minerals, European production, other than in the case of metalliferous minerals, presents a major fraction of the world production.

Commodity	% of world
Barytes	11 %
Borates	38 %
Feldspar	64 %
Fluorspar	5 %
Kaolin	31 %
Limestone	*No data has been supplied.. Please provide information*
Phosphate	1 %
Strontianite	*No data has been supplied.. Please provide information*
Talc	17 %

Table 1.4: Production of industrial minerals within EU15 and Candidate Countries as a percent of world metal concentrate production in 1999

Industrial minerals are recovered in many different ways. Some are sold as mined without being processed. In other cases all sorts of mineral processing methods have to be applied to achieve a highly concentrated product. Hence the amounts and characteristics of tailings and waste-rock vary significantly. In general these operations are small compared to most metal mines and the grade of the mineral is higher. Therefore in most cases the amounts of waste-rock and tailings are also smaller. Acid rock drainage is typically not an issue in the industrial minerals sector.

1.1.2.1 Barytes

1.1.2.1.1 Industry overview

Barytes is the naturally occurring mineral form of barium sulphate ($BaSO_4$). It is a relatively low-value industrial mineral. Filler applications command higher prices after more intense mineral processing; and there are also premiums for colour – whiteness and brightness [29, Barytes, 2002].

1.1.2.1.2 Structure of the industry

EU15 consumption of barytes is estimated to be around 700000 tonnes, with EU15 mined production around 340000 tonnes and the balance being imported, mainly from China but also from Morocco and India [29, Barytes, 2002].

The following graphs shows the main producing countries in current member states and Candidate Countries. The total annual production of both is about 715000 tonnes.

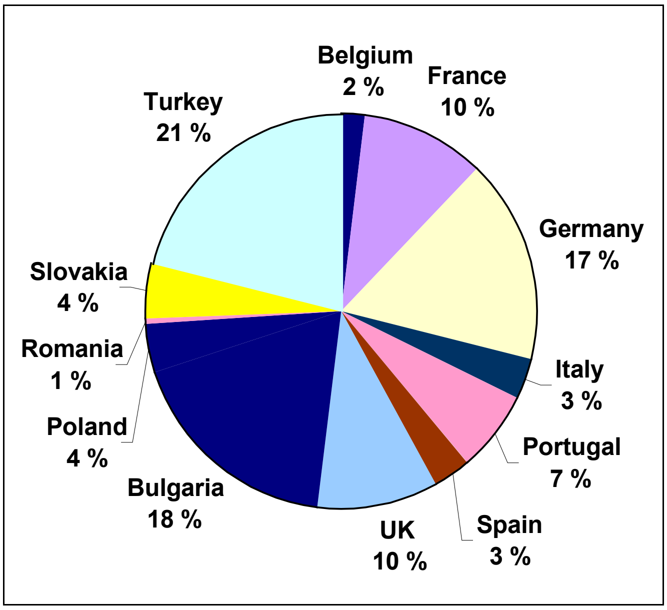

Figure 1.14: Barytes mine production in EU15 and Candidate Countries)

Of the total 6.4 million tonnes production, the US consumed some 2.7 million tonne and EU 15 an estimated 0.7 million tonnes. The following figure shows the main producers in the world.

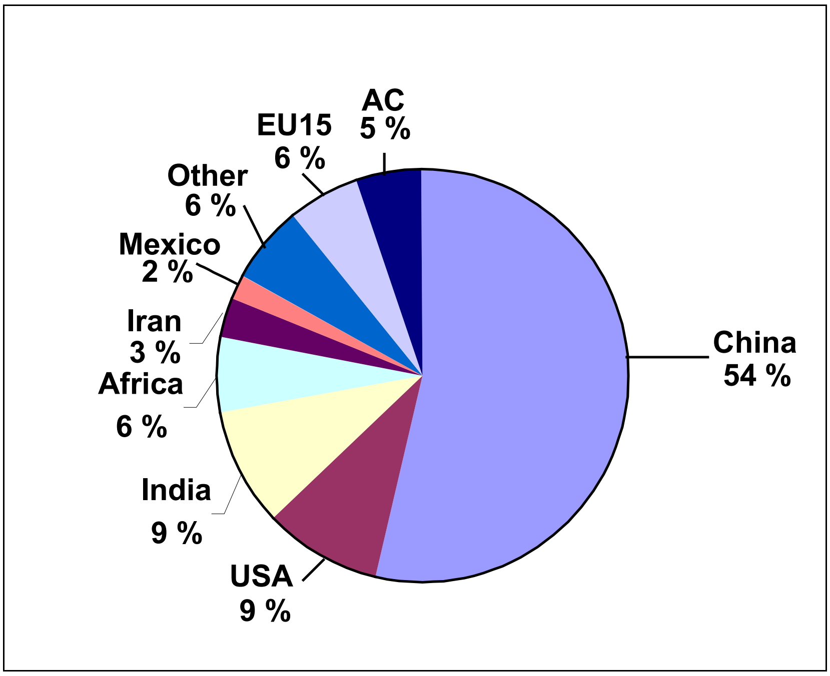

Figure 1.15: Barytes production in the world (production figures) in 2000

The following production sites within the EU15 were reported to this work:

Germany:
Sachtleben Bergbau Services GmbH. - Wolfach
- Dreislar

Deutsche Baryt-Industrie Dr Rudolf Alberti GmbH & Co KG - Bad Lauterberg

France:
Solvay Barium Strontian GmbH - Barytine de Chaillac - Chaillac

United Kingdom:
M-I Drilling Fluids UK Ltd - Foss Mine - Aberfeldy, Scotland

Viaton Industries Ltd - Closehouse Mine - Middleton-in-Teesdale, England

Spain:
Minerales y Productos Derivados SA - Vera, Almeria

[29, Barytes, 2002]

Portugal:
Comital, Companhia Mineira de Talcos, Lda - Alto da Caroceira

[28, INR, 2002]

Furthermore Barytes is mined and processed in Belgium, Greece and Italy [30, Weber, 2001].

1.1.2.1.3 Economics of the industry

Quoted prices (*Industrial Minerals* magazine) for oil-well crushed lump are around EUR 55 - 60/tonne rising to EUR 100/tonne for ground material. Mined production output in Europe has remained steady for several years; and provides direct employment for over 400 people and directlycontributes over EUR 50 million to GDP [29, Barytes, 2002].

1.1.2.1.4 Characterisation and management of tailings and waste-rock

The average grade of ores mined in the EU-15 is around 50 % [29, Barytes, 2002]. This indicates that to produce 715000 tonnes of Barytes about 1400000 tonnes of ore has to be mined. Some of this ore has been sold as other mineral products [29, Barytes, 2002].

Only a small percentage of the tailings produced within the EU15 are discarded as slurry in ponds. Typically coarse tailings are sold as aggregates. Finer tailings are mostly dewatered and also sold or used as backfill in the mine.

1.1.2.2 Borates

1.1.2.2.1 Industry overview

Borates are naturally-occurring minerals containing boron, the fifth element on the Periodic Table. Trace amounts exist in rock, soil and water. The element boron does not occur in nature but traces of its salts are present almost everywhere in rocks, soil and water. Nevertheless, borate minerals are comparatively rare and large deposits exist in only a few places in the earth's crust (Turkey, USA, China, Russia, and South America).

The oldest form of boron known is the mineral salt called tincal (sodium tetraborate decahydrate, or simply borax). Other boron-containing minerals that occur naturally and are mined commercially include colemanite (calcium borate), hydroboracite (calcium magnesium borate), kernite (another sodium borate), and ulexite (sodium calcium borate).

Borates are used in hundreds of products and processes.
[92, EBA, 2002]

Currently a Canadian company is developing plans for a borate project in Piskanja, Serbia [36, USGS, 2002].

A Turkish producer is building a pyrite burning sulphuric acid plant at Bandirma to supply the acid for boric acid plant at Bandirma and Emet [36, USGS, 2002].

1.1.2.2.2 Structure of the industry

The only European borate producer is Turkey with an annual production of about 1.2 Mt from nine operations (7 open pits, 2 underground mines). This represents almost 40 % of the world production. Other significant producers of borates are the US and Russia (1 Mt each) [36, USGS, 2002].

1.1.2.2.3 Economics of the industry

No data has been supplied.. Please provide information.

1.1.2.2.4 Characterisation and management of tailings and waste-rock

No data has been supplied.. Please provide information.

1.1.2.3 Feldspar

1.1.2.3.1 Industry overview

Feldspar is by far the most abundant group of minerals in the earth's crust, forming about 60 % of terrestrial rocks. Feldspar minerals are essential components in igneous, metamorphic and sedimentary rocks, to such an extent that the classification of a number of rocks is based on feldspar content. The crystalline structure of feldspar consists of an infinite network of SiO_2 octahedron and AlO_4 tetrahedron. They usually crystallise in the monoclinic or triclinic system.

The mineralogical composition of most feldspars can be expressed in terms of the ternary system Orthoclase ($KAlSi_3O_8$), Albite ($NaAlSi_3O_8$) and Anorthite ($CaAl_2Si_2O_8$). The minerals whose composition is between Albite and Anorthite are known as plagioclase feldspars, while those comprised between Albite and Orthoclase are called alkali feldspars. This latter category is of particular interest in terms of the industrial use of feldspars.

More than 70 % of the feldspars produced in the EU are used in the ceramic industry, with most of the rest being used in glass production. In the manufacture of ceramics, feldspar is the second most important ingredient after clay, acting functionally as a flux.
[39, IMA, 2002].

1.1.2.3.2 Structure of the industry

In 2000, a total of 6 million tonnes of Feldspar were produced in Europe which is almost one quarter (22.4 %) of the total world production. The following chart shows the major European producers.

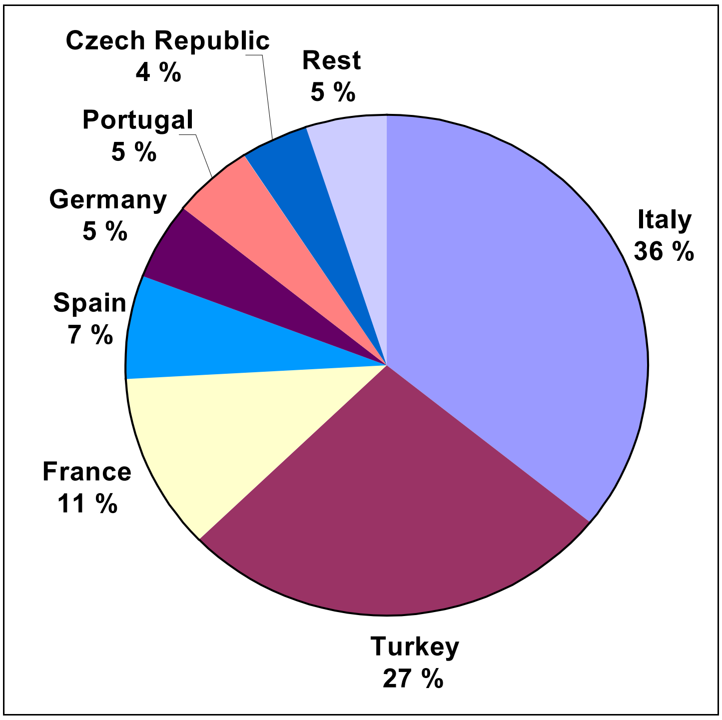

Figure 1.16: Feldspar mine production in EU15 and Candidate Countries

Minor producers (<100000 tonnes/yr) are Finland, Greece, Sweden, UK, Hungary, Poland and Romania.

The following table lists the European production sites.

Country	Location	Location	Company
	Processing plant	Quarry/mine	
Finland	Kimito	Kimito	SP Minerals OY
France	St-Paul de Fenouillet	St Paul de Fenouiilet	DAM
France	Montebras	Montebras	DAM
France	Etang-sur-Arroux	Etang-sur-Arroux	DAM
Germany	Hirschau	Hirschau	AKW-Kick
Germany	Schnaittenbach	Schnaittenbach	AKW-Kick
Germany	Tükishühle-Sarre	Türkishühle-Sarre	DAM
Italy	Giustino	Giustino	Maffei
Italy	Orani		Maffei
Italy	Ottana		Maffei
Italy	Trento		Maffei
	Darzo		Maffei
	Gallese		Maffei
Portugal	Seixoso	Seixoso	Unizel-Minerais
Portugal	Fraguicas	Fraguicas	Unizel-Minerais
Portugal	Viseu	Baloca (Almendra)	FELMICA
Portugal	Mangualde	Real (Penalva do castelo)	FELMICA
Portugal		Vila Seca (Mangualde)	FELMICA
Portugal		Lagares (Queiriga)	FELMICA
Portugal		Alvarroes (Gonçalo)	FELMICA
Spain	Navas de Oros	Navas de Oro	Compania de Rio Piron
Spain	Carrascal del Rio	Carrascal del Rio	INCUSA
Sweden	Forshammar	Forshammar	North Cape Minerals

Candidate Countries producers to be added!!

Table 1.5: Feldspar producers in Europe
[39, IMA, 2002]

Feldspars are either selectively mined or processed by optical, flotation and/or electrostatic separation, in order to remove the accessory minerals (e.g. quartz, mica, rutile, etc.) present in the ore. The feldspar may then undergo a grinding step to adapt the particle-size to the intended use. For a number of uses, it is perfectly acceptable, and even advantageous, that the product retains some accessory minerals, e.g. quartz, while at the other extreme some applications require extremely pure and fine-ground grades. Basically, the two properties which make feldspars useful for downstream industries are their alkali and alumina content.

The flotation process is only used by AKW, INCUSA, and SP Minerals. The feldspar recovered by flotation only represents about 10 % of the European feldspar production The flotation process is essential to get a high quality grade (low iron content and high alumina content) required for some specific and important applications (e.g. TV/computer screens). For instance, although the Italian producer Maffei is the biggest producer in Europe, the three above-mentioned companies supply the Italian market with these high quality grade products.

1.1.2.3.3 Economics of the industry

No data has been supplied. Please provide information

1.1.2.3.4 Characterisation and management of tailings and waste-rock

Beside the tailings heap made of coarse sand, gravel and stones, there are tailings ponds for the fine tailings.

1.1.2.4 Fluorspar

1.1.2.4.1 Industry overview

Fluorspar is the industrial name of a mineral, fluorite, which is natural calcium fluoride with the chemical formula CaF_2. It is extracted from mines (underground and open pits), with natural concentrations between 20 and 90 %. Ore and concentrated marketable products have the same name, i.e. Fluorspar.

The chemical element F is not really rare in the earth's crust (at 0.07 % it si the.13th most abundant element by weight), but naturally occurring concentrations are scarce. The elements fluorine (F) and calcium (Ca) are strongly bound in CaF_2 and this molecule is very stable.
[43, Sogerem, 2002]

1.1.2.4.2 Structure of the industry

Worldwide production is between four and five million tonnes per year. The main producing countries are China (2.5 million tonnes), Mexico (0.5 million tonnes), the EU15 (0.4 million tonnes), South Africa (0.3 million tonnes). Around 20 countries have declared a substantial production in 2000 [43, Sogerem, 2002]. The European producers are displayed in the following figure.

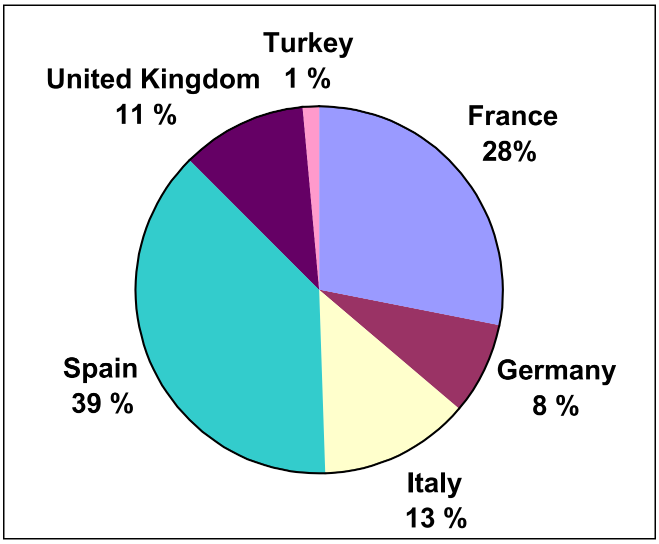

Figure 1.17: Fluorspar mine production in EU15 and Candidate Countries (1999)

The European producers are listed in the following table

Country	Location	Location	Company
	Processing plant	**Quarry/mine**	
France			Sogerem
Italy	Cagliari, South Sardinia	South East Sardinia	Nuova Mineraria Silius
Table to be completed			

Table 1.6: European fluorspar producers

The Sardinian mine produces fluorspar and a lead sulphide concentrate. The average rate of production per year is 45000 tonne of 97 % CaF_2 and 5000 tonnes of 67 % PbS. Silius Mine is the only operating mine in Europe for Fluorspar and Lead Sulphide. The fluorspar product is sold to a nearby chemical plant and the lead sulphide is sold to a smelting plant located in South West Sardinia [44, Italy, 2002].

1.1.2.4.3 Economics of the industry

In the case of the Sardinian fluorspar/lead sulphide mine the average value of products are EUR 120 per tonne for Fluorspar and USD 190 per tonne for Lead Sulphide [44, Italy, 2002].

1.1.2.4.4 Characterisation and management of tailings and waste-rock

No data has been supplied. Please provide information

1.1.2.5 Kaolin

1.1.2.5.1 Industry overview

The word kaolin derives from the Chinese "Kao-ling" (High Crest), the name of a hill in Central China near where this substance was mined for use in ceramics. This is also the origin of the name "China Clay". Since those early days, the use of kaolin has widened to paper, rubber, paints and plastics manufacture [40, IMA, 2002].

Clay minerals are divided into four major groups. One of these is the kaolinite group. This group has three members (kaolinite, dickite and nacrite) and a formula of $Al_2Si_2O_5(OH)_4$. The different minerals are polymorphs, meaning that they have the same chemistry but different structures. The general structure of the kaolinite group is composed of silicate sheets (Si_2O_5) bonded to aluminum oxide/hydroxide layers ($Al_2(OH)_4$) called gibbsite layers. The silicate and gibbsite layers are tightly bonded together with only weak bonding existing between the layers [37, Mineralgallery, 2002].

1.1.2.5.2 Structure of the industry

In 1999, European Kaolin production was about 5 million tonnes, about 20 % of the world production in the same year. The biggest European producers are listed in the following chart.

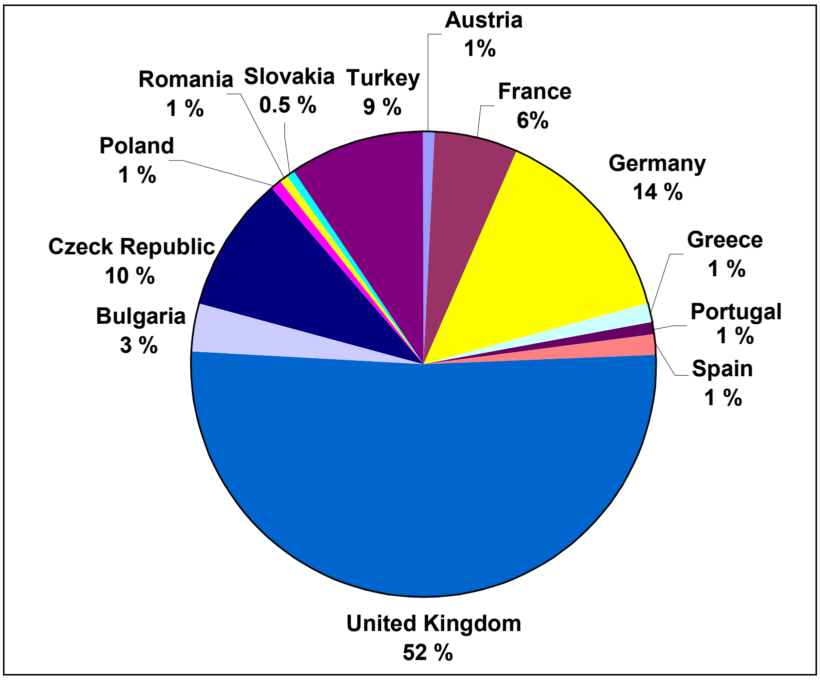

Figure 1.18: Kaolin mine production in EU15 and Candidate Countries

The following table lists the European production sites.

Country	Location	Location	Company
	Processing plant	Quarry/mine	
Czech Republic	Horni Briza	Horni Briza	Keramika Horni Briza
Czech Republic	Kaznejov	Kaznejov	Keramika Horni Briza
Czech Republic	Bozicany	Bozicany	Sedlecky Kaolin
Czech Republic	Sadov	Sadov	Sedlecky Kaolin
Czech Republic	Unanov	Unanov	Sedlecky Kaolin
Czech Republic	Hlubany	Hlubany	WBB MINERALS
France	Ploemeur	Ploemeur	DAM
France	Echassières	Echassières	DAM
France	Berrien	St Brieuc	DAM
France	Loqueffret		DAM
France	Quessoy	Quessoy	SOKA
Germany	Hirschau	Hirschau	Gebruder Dorfner
Germany	Hirschau,	Hirschau,	AKW-Group
Germany	Schnaittenbach	Schnaittenbach	AKW-Group
Germany	Camimau,	Canimau,	AKW-Group
Germany	Kemmiltz	Kemmiltz	AKW-Group
Germany	Seilitz (Meissen)	Seilitz (Meissen)	WBB MINERALS
Spain	Higueruelas	La Yesa	WBB MINERALS
Spain	La Coruna	La Coruna	Caolines de Vimianzo
Portugal	Mosteiros	Braçais	DAM
Portugal	Braçais		DAM
Poland	Surmin	Surmin	AKW-Group
United Kingdom	St Austell	St Austell	Imerys
United Kingdom	Bodmin Moor	Bodmin Moor	Imerys
United Kingdom	Dartmoor	Dartmoor	Imerys
United Kingdom	Tregonning Hill	Tregonning Hill	Imerys
United Kingdom	Land's End	Land's End	Imerys
United Kingdom	Cornwall	Cornwall (5 open pits)	Goonvean
United Kingdom	Cornwall	Cornwall	WBBMINERALS

Candidate Countries producers to be added!!

Table 1.7: Feldspar producers in Europe
[39, IMA, 2002]

1.1.2.5.3 Economics of the industry

No data has been supplied. Please provide information

1.1.2.5.4 Characterisation and management of tailings and waste-rock

This is almost identical to the feldspar section. Beside the heap of coarse sand, gravel and stones, there are also tailings ponds for the fine tailings.

1.1.2.6 Limestone

1.1.2.6.1 Industry overview

Limestone is used in three different ways: as an aggregate, as calcium carbonate and in the cement and lime industry. The aggregate path does not produce any tailings, hence only the two latter routes are of interest to this document.

The calcium carbonate industry operates withdeposits for which the grade is usually higher than 96 %. Therefore, there is usually no need for further mineral processing steps. In Europe, only 7 plants need to use flotation to separate calcium carbonate from unwanted minerals (mainly graphite and mica). These seven plants account for less than 5 % of the European calcium carbonate production. Five of these plants do not have tailings ponds, since they use dewatering devices (e.g. thickening & filter press).

From the mineralogical point of view, calcium carbonate falls into three structurally different groups: the calcite and the aragonite groups (both $CaCO_3$), and the dolomite group ($CaMg(CO_3)_2$).

These minerals constitute rocks, of which chalk, limestone, marble, and travertine are the most important ones.
[42, IMA, 2002]

1.1.2.6.2 Structure of the industry

No data has been supplied. Please provide information

1.1.2.6.3 Economics of the industry

No data has been supplied. Please provide information

1.1.2.6.4 Characterisation and management of tailings and waste-rock

No data has been supplied. Please provide information

1.1.2.7 Phosphate

No data has been supplied. Please provide information

1.1.2.7.1 Industry overview

1.1.2.7.2 Structure of the industry

1.1.2.7.3 Economics of the industry

1.1.2.7.4 Characterisation and management of tailings and waste-rock

1.1.2.8 Strontianite

No data has been supplied. Please provide information

1.1.2.8.1 Industry overview

1.1.2.8.2 Structure of the industry

1.1.2.8.3 Economics of the industry

1.1.2.8.4 Characterisation and management of tailings and waste-rock

1.1.2.9 Talc

No data has been supplied. Please provide information

1.1.2.9.1 Industry overview

1.1.2.9.2 Structure of the industry

1.1.2.9.3 Economics of the industry

1.1.2.9.4 Characterisation and management of tailings and waste-rock

1.1.3 Potash

Even though potash is an industrial mineral, the TWG members at the kick-off meeting decided, that due to the different techniques in mineral processing and tailings management this mineral would be treated in a separate section.

1.1.3.1 Industry overview

The main products used as fertilisers (with the nutrients potassium, sulphur and magnesium) are potassium chloride (MOP), potassium sulphate (SOP) and kieserite. They are produced with different values of K_2O (40 - 62 %) and in a fine, standard or coarse grade. Potassium sulphate and sulphates of potash-magnesia are non-chloride potash fertilisers.

Please provide short definition of K_2O.

1.1.3.2 Structure of the industry

About one fifth of the potash world production comes from European mines in France, Germany, Spain and the UK.

The European mine production in 1999 was just over 5 Mt K_2O. The following figure shows the production percentages by country.

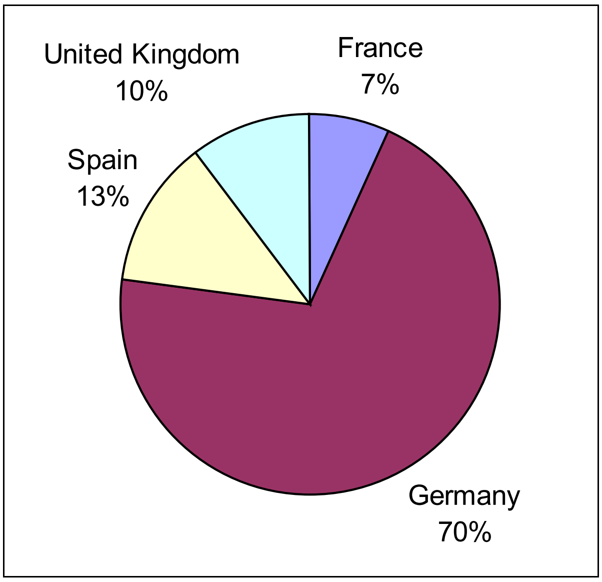

Figure 1.19: Potash mine production (K_2O) in EU15 and Candidate Countries

World potash production is dominated by Canada, Russia and Germany, which together account for about 76 % of world production. Potassium chloride (KCl), commonly referred to as muriate of potash (MOP), is the most common and least expensive source of potash. Potassium chloride accounts for about 95 % of world potash production.
[19, K&S, 2002]

1.1.3.3 Economics of the industry

The world potash industry has experienced unstable conditions since the late 1980s (just prior to the economic collapse of the Eastern block countries). Up to that point, average industry operating rates were exhibiting a slow but steady upward trend that ended abruptly in 1988. The average world operation rate, which had steadily increased to 83 % in 1988, declined steadily to only 56 % of previous levels. During this period, apparent world consumption, which is equal to world production by definition, declined from 31 million tonnes of K_2O to only 21 million tonnes.

World potash demand in 2000 was approximately 26 million tonnes of potassium oxide (K_2O) or 42 million tonnes of product (KCl and K_2SO_4). Compared with this, the manufacturing capacity was approximately 37 million tonnes of potassium oxide (K_2O) or 59 million tonnes of products. Therefore, a considerable overcapacity exists worldwide.

The economic situation, particularly in developed countries, greatly influences the extent and regional distribution of exports. Both the quantity exported and its distribution among consumers are greatly affected by the state of their agriculture, by the demand for (or availability) of convertible currency in the exporting or importing country and fluctuations in currency exchange rates. Transport costs for potash fertilisers have a considerable bearing on total cost to the consumer, therefore logistical considerations also influence the direction and size of exports or imports.

1.1.3.4 Characterisation and management of tailings and waste-rock

Five methods are used in Europe for managing the tailings, these are:

- storing solid tailings on tailings heaps
- backfilling solid tailings into mined out rooms of underground works
- discharging solid and liquid tailings into the ocean/sea (marine tailings management)
- discharging liquid tailings into deep wells
- discharging liquid tailings into natural flowing waters (rivers).

Potash tailings can be described as table salt (sodium chloride) with a few percent of other salts (e.g. chlorides and sulphates of potassium, magnesium and calcium) and insoluble materials like clay and anhydrite.

Tailings heaps themselves generate saline solutions when atmospheric precipitation dissolves salt from tailings material.

1.1.4 Coal

1.1.4.1 Industry overview

The term "coal" includes the high and medium-ranking "A" coals (sub-bituminous coals) as defined in the "International codification system of coal" of the UN Economic Commission for Europe.

The production of coal mining in Europe has been declining for decades due to the deep-lying and relatively slim deposits called seams. Coal mining in Members States and Candidate Countries will keep closing. Therefore, the opening of new mines is not foreseeable at the moment. Except for Spain, coal is usually mined underground.

1.1.4.2 Structure of the industry

The total coal production in EU15 and Candidate Countries in 1999 was 220.7 Mt. The following chart shows the main producers. Note that countries producing less than 1 million tonnes are not included in this graph, these include Hungary, 740000 tonnes, and Bulgaria 17200 tonnes.

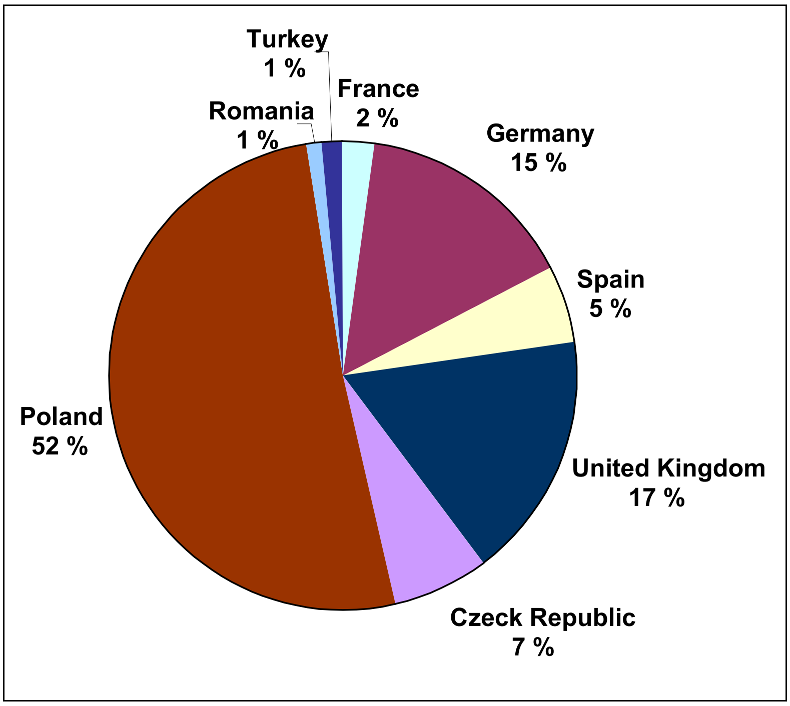

Figure 1.20: Coal mine production in EU15 and Candidate Countries

Total coal production within EU15 and Candidate Countries represents 3.6 % of the world coal production of about 3700 million tonnes.

In Germany, by the end of 2000 there were 12 mines left in production.

Hard coal in the Czech Republic mainly occurs in the Upper Silesian Basin. Of the resources within this region, about 15 % is in the Czech Republic, and the balance is in Poland. Mining production of hard coal in the Czech republic reached 17 Mt in 2000.
[83, Kribek, 2002]

1.1.4.3 Economics of the industry

Worldwide coal is mostly mined in open pits from seams that are tens of metres thick. In Europe significantly thinner seams (maximum several metres) have to be mined underground several hundred or even over thousand metres deep. Hence, in many cases European production costs are several times the world average. These mines are still in production because they receive subsides.

1.1.4.4 Characterisation and management of tailings and waste-rock

Tailings from coal mining are the coarse tailings, which are managed on heaps, and the flotation slurries, which are transported into ponds. The ponds can be small settling basins, which need to be dug out periodically. In other cases coal tailings ponds can cover tens of hectares and being contained by tailings dams. Coal tailings often contain pyrite and flotation reagents.

Efforts have been made to use coal tailings as construction materials. Due to their low permeability dried flotation fines can also be used as liners for landfills.

1.1.5 Oil shale

1.1.5.1 Industry overview

This mineral/ore is only mined in Estonia, with the exception of one mine in Germany where oil shale is mined for incineration in rotary kilns for klinker production. In Estonia oil shale is mainly used for power generation [10, Estland, 2001].

1.1.5.2 Structure of the industry

Oil shale is the only mineral resource mined in Estonia on a large scale. Oil shale is an organic and sedimentary rock formed about 450 million years ago. The seam mined is 2.8 - 3.8 m thick and has a practically horizontal bedding, up to a depth of 70 m. The average caloric value is 8 MJ/kg and contains 1.7 % S.

From the end of 2001 there will be two underground and two surface mining units for oil shale in operation.

In 2000, power plants based on oil shale produced 7765 GWh of electric energy, which is 93 % of the total production in Estonia. [10, Estland, 2001]

1.1.5.3 Economics of the industry

5500 workers in the oil shale mining industry [10, Estland, 2001]

No further data has been supplied.. Please provide information.

1.1.5.4 Characterisation and management of tailings and waste-rock

No data has been supplied.. Please provide information.

1.2 Key environmental issues

The sectors and ores, minerals or mineral resources covered by this document are very diverse in many aspects, e.g. environmental impact, size and duration of the activity, characteristics of tailings and waste-rock. However, the TWG identified certain key issues that are relevant for all of them.

The management of tailings and waste-rock is one part of the entire mining operation, which also includes the actual extraction and the mineral processing stage. However, not only do these other parts of the operation influence the management of tailings and waste-rock - in reality the methods of mining and mineral processing usually determine the management of waste-rock and not vice versa.

Tailings and waste-rock management sites go through certain phases from design to after care. It is essential to manage these facilities in a way that makes most sense in all phases of that life-cycle. For example, it may be that for one specific site, wet management of tailings may make most sense in the operational phase. However, if for the after-closure phase a dry tailings management facility (herafter TMF) was shown to be the most useful solution, this may result in dry management of these tailings being applied even in the operational phase.

Another important issue to consider is the adaptation to changes in reality. An example here may be that fact that after 10 years of operation the sulphide content in the waste-rock increases to a level such that acid drainage will be an issue in the longer term unless this is taken care of during the operational phase by maybe mixing this waste-rock with other waste-rock containing buffering minerals or by separately depositing material with ARD potential in an adequate way. In this example one would have to project findings during operation to steps much further down the "life-cycle road" and act accordingly to achieve the best overall environmental and economic benefit.

In many parts of the world this way of thinking has been lacking in the mining industry. The Iberian Pyrite Belt in Spain and Portugal is an example where a lack of knowledge has over a long time created significant problems with ARD generation and consequently environmental effects. In such cases the environment and the tax payer will have to pay a high price to remediate the damage done in the past. Modern mining has to be more progressive and must predict all future effects of today's decisions and act accordingly.

Historically mining activities have generally neither been planned nor operated according to the above described principles. The reason for this may have been the lack of knowledge, unawareness or simply insufficient legislation. Today knowledge has evolved, the effects are better well known, as are also the required management practises possible in order to avoid more bad examples in the future. These advances need to be borne in mind in the permitting process where it is also necessary to secure financial guarantees for the restoration of the mine site in the event that the operator vanishes unexpectedly.

1.2.1 Site location

Mining is a unique sector in so much as geology determines the location of the mine. This is a major difference to other industries. An ore has to be mined where the deposit is. Of course the choice of mining method and the exact location of shafts and other infrastructure still has to be made.

The degrees of freedom in design increases the further downstream one goes in the process. The location for the extraction itself is predetermined as mentioned above. However, the mineral processing may be done somewhere completely different. Typically, this process step is undertaken as close to the actual mine site as possible, due to the often low grade of the ore, which implies that the ore cannot cover high transport costs. In the case of Bauxite however, the ore is often processed many thousand of kilometres away from the mine as the processing of Bauxite into aluminium is very energy demanding and the transport cost of the ore can be recovered by lower energy costs for the processing (some pre-refining is often done at the site though).

For the management of tailings and waste-rock the degrees of freedom again increase, but as for the mineral processing it is generally preferable to limit or reduce the transportation cost. However, in many cases tailings are pumped or trucked many kilometres to an appropriate site for deposition.

When it comes to the selection of a tailings and/or waste-rock management site many other factors have to be considered, such as:

- preferable use of existing geographic formations (e.g. existing pits or slopes)
- need to respect the hydrogeological setting of the surrounding area (groundwater)
- adaptation of facility to surrounding area (e.g. dust, noise, odour control if village nearby)
- meteorology (e.g. rainfall data)
- geotechnical and geological background (e.g. foundation conditions, seismic risk data)
- natural and cultural environment
- relationship of tailings facility to underground operation
- topography of long-term construction
- land take
- proximity to surface water.

Proximity to surface water is often a diverse issue. On the one hand, if discharge to surface water is required it is preferable to have the river "next door". On the other hand, would this surface water act as the ideal transport medium of tailings in the case of an accidental release, if so it is probably better to maintain a safe distance from these surface waters.

In general a balance has to be maintained between the proximity of the tailings or waste-rock management site to the mineral processing site for economical reasons and the other factors as listed above. In reality often the site investigation results in several "candidate locations". The actual decision is then achieved in the permitting process as a compromise between operator, permitters and public concerns.

1.2.2 Material characterisation including prediction of long-term behaviour

The only way of determining the long-term behaviour of tailings and waste-rock is to characterise them properly. This may sound trivial, but has often been neglected in the past. Too often has the focus been on the saleable concentrate, which generates revenue and not on the residue. However, one should not forget the negative economic effect that improper tailings and waste-rock management can incur.

From an environmental point of view, the main difference between the mineral in the original deposit and the same mineral, less as much as possible of the desired mineral, in the tailings and waste-rock is the potentially **increased bioavailability**. This means that through the treatment of the ore (mainly comminution) the constituents of the tailings and waste-rock are more accessible. The following two examples may further explain this phenomenon:

A hard rock sulphidic metal orebody may be impermeable to water and not accessible to atmospheric oxygen. The finely ground tailings of this ore being discarded in a pond are much more accessible to water and oxygen. The surface area of accessible sulphides is increased by orders of magnitude through the size reduction. This implies that, if not managed properly, the rate of weathering and thereby the mobilisation of weathering products may be significantly increased.

Another example may be a potash ore. These ores consist of potash minerals and rock salt. The deposits are protected from water by impermeable layers (typically of clay and gypsum). The tailings of this same ore consist mainly of rock salt (> 90 %) and are typically piled up on heaps. This salt is accessible for precipitation and washed-off over a long period of time. Thereby the dissolution rate of the salt may be increased significantly.

The mineral processing of the ore may also change the **chemical characteristics** of the processed mineral and hence the tailings and waste-rock.

Characteristics that have to be investigated are, e.g.:

- chemical composition, including the change of chemistry through mineral processing
- leaching behaviour
- physical stability
- behaviour under pressure
- erosion stability
- settling behaviour
- hard pan behaviour (e.g., crust formation on top of tailings).

Proper material characterisation is the basis for any further planning of the management of tailings and waste-rock. Only if this "homework" is done properly can the most appropriate management measures be applied.

General issues around long-term issues are discussed in Section 2.4.2. Applied measures are shown in Section 3.1.5.

Each mining operation will have an irreversible impact on the environment. To qualify this impact baseline studies are carried out as a point of reference. Baseline studies are described in more details in Sections 2.5.1 and 3.1.2.1.

1.2.3 Environmentally relevant parameters

The environmentally relevant parameters of tailings and waste-rock management facilities can be subdivided into two categories: operational and accidental. Both have to be taken into consideration.

Operational parameters are, e.g.:

- Overall management of water and reagents
  Consumption and treatment and/or recycling of
  - reagents (e.g. flotation reagents, Cyanide, flocculants) and
  - water

  prior to discharge into tailings facility or surface water

- Emissions to air
  - dust, including heavy metals
  - noise
  - odour.

- Emissions to water
  - includes process water, suspended solids, reagents, heat from process water, gathering of surface waters, dissolved metals and other elements and compounds.

  It is also possible to distinguish between:
  - emissions to surface water
  - emissions to ground water.

- Soil contamination
  - heavy metals
  - reagents (e.g. CN, Xanthates, Cl)
  - others.

In the case of an accidental situation
- flooding
- suffocation
- crushing and destruction
- cut-off of infrastructure
- poisoning
- etc.

It should be noted that emissions to land are a highly site-specific issue and that there are very few default emission scenarios currently available to characterise these emissions.

The same factors need to be considered for accidental releases. The underlying question that has to be considered is “what if ?”. This means several scenarios have to be considered and based on the possible impact emergency or contingency response plans have to be developed and, this is the essential part, known and understood by the staff.

The two main causes of TMF incidents have been found to be
- lack of control of water balance
- a general lack of understanding of the features that control safe operations.

[9, ICOLD, 2001, p. 6]

1.2.4 Site rehabilitation and aftercare

When an operation comes to an end the site has to be prepared for its subsequent use. Usually these plans are part of the permitting of the site and would therefore have undergone regular updating depending on changes in the operation and the negotiations with the permitters and other stakeholders. In some cases the goal is to leave as little a footprint as possible whereas in other cases a complete change of landscape may be the goal. The concept “design for closure” implies that the closure of the site is already taken into account in the feasibility study of a new mine site and continuously prioritised during the life cycle of the mine. In any case, negative environmental impacts have to be kept to a minimum.

Some sites can be handed over to the subsequent user after they are covered and revegetated. In other cases after care will have to be undertaken for long periods of time, sometimes even in perpetuity.

It is impossible to restore a site to its original condition. However, the operator, the authorities and the involved stakeholders have to agree on the successive use and it is usually the operators responsibility to prepare the site for this. The question of responsibility for the site and liability for
- maintenance
- planned or unplanned treatment
- or failures

after the site has been handed over to its successive use is often discussed.

2 COMMON PROCESSES, TECHNIQUES AND EQUIPMENT

This chapter is aiming at providing background information to the non-experts in the management of tailings and waste-rock. Together with the specific glossary this chapter should allow the reader to understand the subsequent chapters.

2.1 Mining techniques

The extraction of an ore, (a process called mining), the subsequent mineral processing and the management of tailings and waste-rock are in most cases considered to be one single operation. Even though this document does not cover the ore extraction, mineral processing techniques and tailings and waste-rock management highly depend on the mining technique. Hence it is important to have an understanding of the most important mining methods.

For the mining of solids there are two basic mining concepts: (1) Open pit and (2) underground mining. The choice between these two alternatives depends on many factors, such as:

- value of the desired mineral(s)
- grade of the ore
- size, form and depth of the orebody
- *please provide further factors.*

Often the uppermost part of an orebody is mined in an open pit, but in time with increasing depth the removal of overburden makes this mining method uneconomical, so deeper parts are mined underground (see figure below).

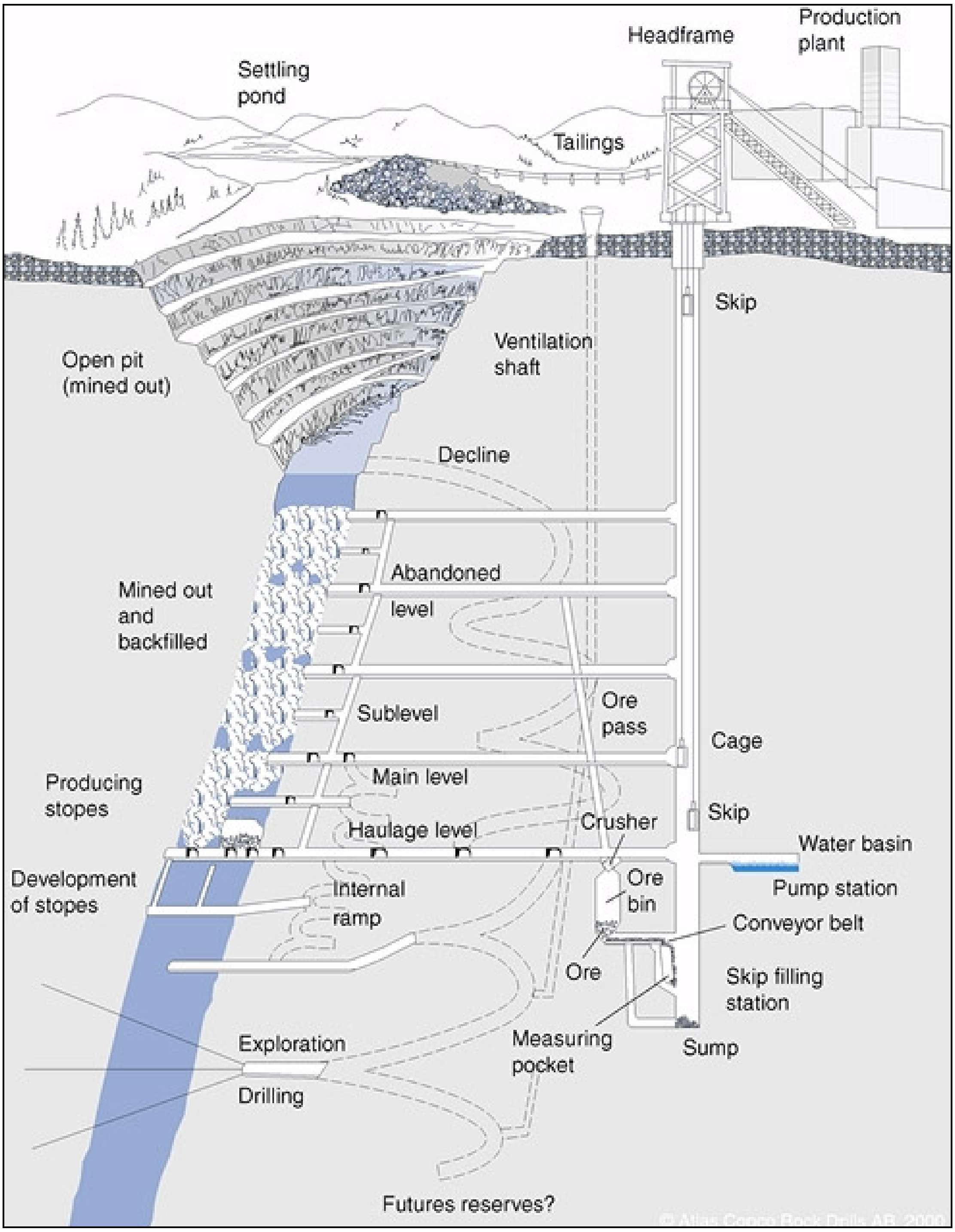

Figure 2.1: Transition from open pit to underground mining
[93, Atlas Copco, 2002]

If open pit is the chosen mining method, it will in most cases result in larger amounts of waste-rock. This is indicated in the following two figures.

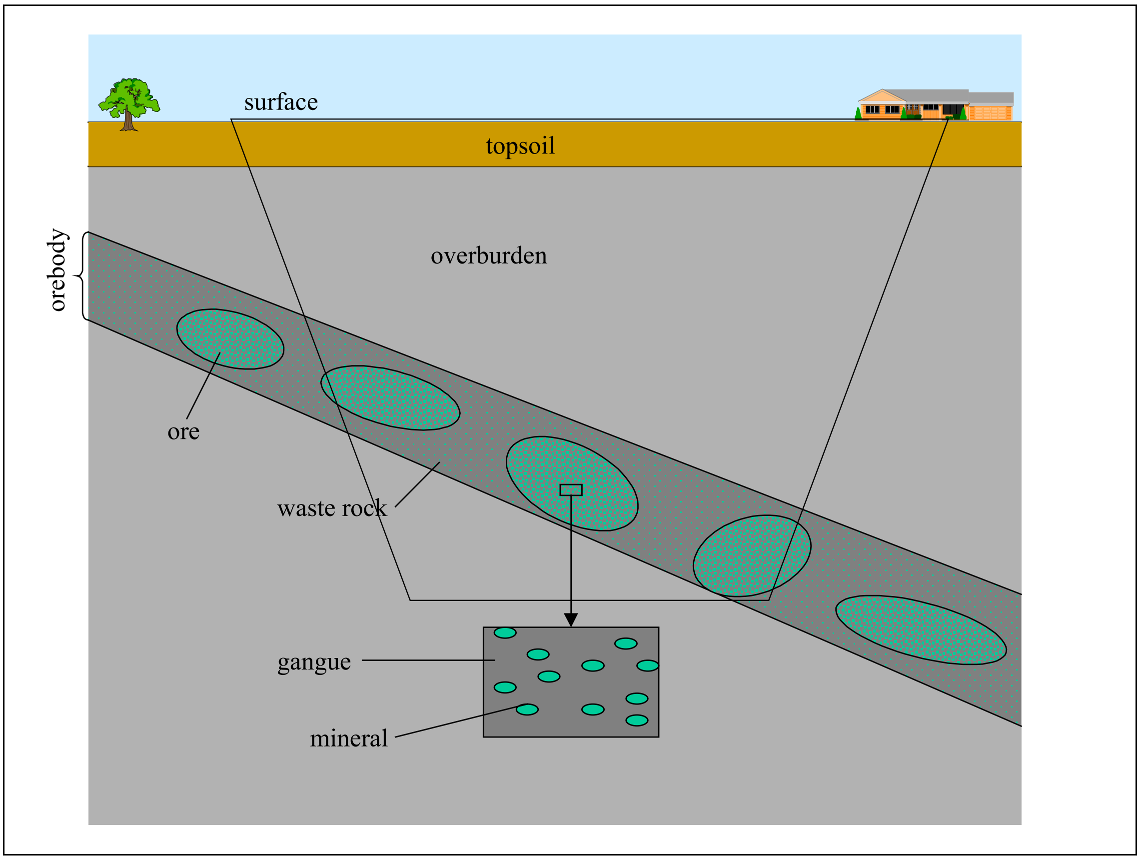

Figure 2.2: Schematic drawing of an open pit

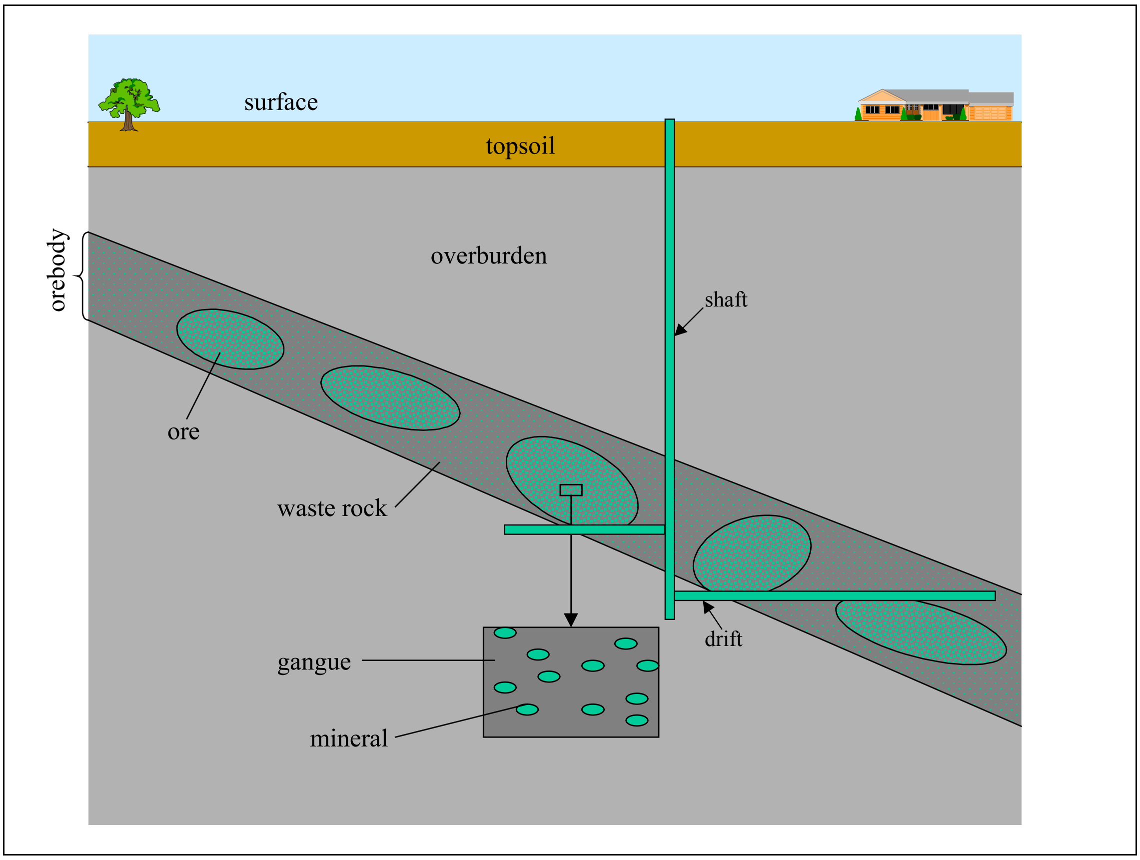

Figure 2.3: Schematic drawing of an underground mine

In the example shown in the above figures the amounts of topsoil, overburden and waste-rock that will have to be moved using the open pit technique are greater than with the case of underground mining. In the latter case a shaft and drifts are constructed from which the ore can be mined more selectively, meaning areas of waste-rock and/or low grade ore can either be left out or can easily be shifted within the mine.

In the case of underground mining it is also possible to backfill mined out areas. This is difficult to realise in an open pit operation while it is still being mined, unless the backfill material can be moved to another pit.

2.1.1 Types of ore bodies

The type of ore body has a big influence on the choice of mining method. The following types of ore bodies are known:

- seam-type ore body
- vein-type ore body
- massive-type ore body (e.g. massive sulphides with high variations of grade within ore body)
- disseminated-type ore body (e.g. copper porphyries).

Often the disseminated type has a “cap” of weathered sulphides (hence oxides) on top of a disseminated-type ore body. The ore within this weathered cap is called “gossum”.

2.1.2 Underground mining methods

There are many different ways of exploiting an ore body using underground mining methods. The most commonly used underground mining methods are:

Mining method	Application
Longwall mining	Flat, thin seam type ore bodies, soft rock
Room and pillar mining	Inclined massive-type ore bodies
Sublevel stoping	Steep, large ore bodies (massive- or disseminated-type)
Cut and fill mining	Steep, firm ore bodies, selectivity, mechanisation (seam-, vein-, massive-, disseminated-type)
Sublevel and block caving	Steep, large or massive ore bodies, extensive development effort (mostly massive-, disseminated-type)

Table 2.1: Most important underground mining methods and their areas of application [47, Hustrulid, 1982]

These methods have been often described (e.g. AIME/SME Underground mining methods handbook, http://sg01.atlascopco.com). The basic objective in selecting a method to mine a particular ore body is to design an ore extraction system that is most suitable under the existing circumstances. This means aiming for maximum profit from the operation. This decision is based upon both technical and non-technical factors (e.g. high productivity, complete extraction of the ore, safe working conditions).

A reduction of tailings and waste-rock can be achieved by using the most selective mining method, so that only undiluted ore is fed to the mineral processing plant and the amount of waste-rock that has to be handled is minimised. Feeding diluted ore to the mineral processing plant results in a decrease of recovery and therefore larger amounts of the desired mineral lost in the tailings.

The large stopes that are created in sublevel stoping makes this an ideal mining method to be combined with backfilling, because it is easy to dump solid or slurried tailings into the large openings. The usually much smaller remaining voids in longwall, room and pillar, and cut and fill mining lead to increased costs for backfilling. Backfilling may still be applied in these cases if the ore has a high value and backfilling allows a higher extraction rate, because safety pillars can be mined after the previous voids are backfilled. If caving is applied, backfilling is not possible, because the voids are immediately filled with material falling in.

The internet site http://sg01.atlascopco.com provides more detailed information about these techniques together with visual illustrations.

2.2 Mineralogy

This section has been developed without contributions from the TWG. Further input is welcome.

Mineralogy is set by nature and determines in many ways the subsequent recovery of desired minerals and the tailings and waste-rock management. The mineralogy often changes within an ore body and hence during the life of a mine. Sometimes these changes are well known and can be planned for, sometimes they occur unexpectedly. Some examples are listed below:

- oxides on top and sulphides in deeper lying parts of the orebody, which require completely different mineral processing and tailings management methods
- the ore changes from a copper ore to a zinc ore
- the ore changes from a magnetite to a hematite type iron ore (Malmberget).

Mineralogy has a big influence on the mining technique and the mine sequencing. For example in the case of gold mining the gossum is mined because it is more easily accessible and naturally enriched and easier to recover. The deeper lying sulphides have to be oxidised before they can be recovered, which makes the process less profitable. In the case of copper it is also easier to recover the oxide section, which can easily be leached using sulphuric acid, whereas the sulphides have to be recovered using flotation.

The sulphide content, which is determined by mineralogy, influences the tailings and waste-rock management, because of the acid generating potential (see Section 2.4.2.2).

Having a good knowledge of mineralogy can lead to

- environmentally sound management (e.g. separate management of acid-generating and non-acid-generating tailings or waste-rock)
- reduced need of end-of-pipe treatment (e.g. lime treatment of acidified seepage water from a TMF)
- more possibilities to utilise tailings and/or waste-rock as aggregates.

2.3 Mineral processing

This section is still under development. The basic structure will be as follows.

2.3.1 Equipment

2.3.1.1 Comminution

2.3.1.1.1 Crushing

- jaw crushers
- gyratory crushers
- cone crushers
- roll crushers
- impact crushers.

2.3.1.1.2 Grinding

- tumbling mills
  - rod mills
  - ball mills
  - autogenous mills
  - semi-autogenous mills
  - centrifugal mills.

2.3.1.2 Screening

2.3.1.2.1 Stationary screens

2.3.1.2.2 Moving screens

- revolving screens
- vibrating screens
- flop-flow screens.

2.3.1.3 Classification

2.3.1.3.1 Hydraulic classifiers

2.3.1.3.2 Hydrocyclones

2.3.1.3.3 Mechanical classifiers

2.3.1.4 Gravity concentration

2.3.1.4.1 Dense medium separation

- gravitational vessels
- centrifugal separators.

2.3.1.4.2 Jigging

2.3.1.4.3 Shaking tables

2.3.1.4.4 Spirals

2.3.1.4.5 Cones

2.3.1.4.6 Centrifugal mineral processing plants

2.3.1.4.7 Flotation

2.3.1.4.8 Flotation reagents

2.3.1.4.9 Flotation machines

2.3.1.5 Magnetic separation

2.3.1.6 Electrostatic separation

2.3.1.7 Sorting

2.3.1.8 Leaching

2.3.1.9 Thickening

2.3.1.10 Filtering

2.3.1.11 Drying

2.3.2 Techniques and processes

heap/in situ/tank leaching, flotation,

gravity separation: dense-medium, vertical currents (jig), streaming currents (table, spiral)

typical flowsheets for metal processing, ind. minerals (if possible), coal and variations\
products are concentrate and tailings

Please provide more information on what should be included here.

2.3.2.1 Alumina refining

Alumina is a white granular material, a little less coarse than table salt, and is properly called aluminium oxide. Aluminium does not occur as a metal, but must first be refined from bauxite in its oxide form. The Bayer refining process used by alumina refineries worldwide involves four steps - digestion, clarification, precipitation and calcination.

The digestion (dissolution) of Aluminium hydrate ($Al_2O_3 \cdot 3H_2O$) out of the bauxite is carried out under pressure in high temperature (250 ºC) sodium hydroxide. The insolubles, sand and red mud, are separated by filtration, washed and deposited in the TMF. The aluminium hydrate is precipitated as a white slurry and dried (calcined) to produce alumina (Al_2O_3), the white powder product. Six to four tonnes of bauxite are needed to produce two tonnes of alumina and subsequently one tonne of aluminium [22, Aughinish,].

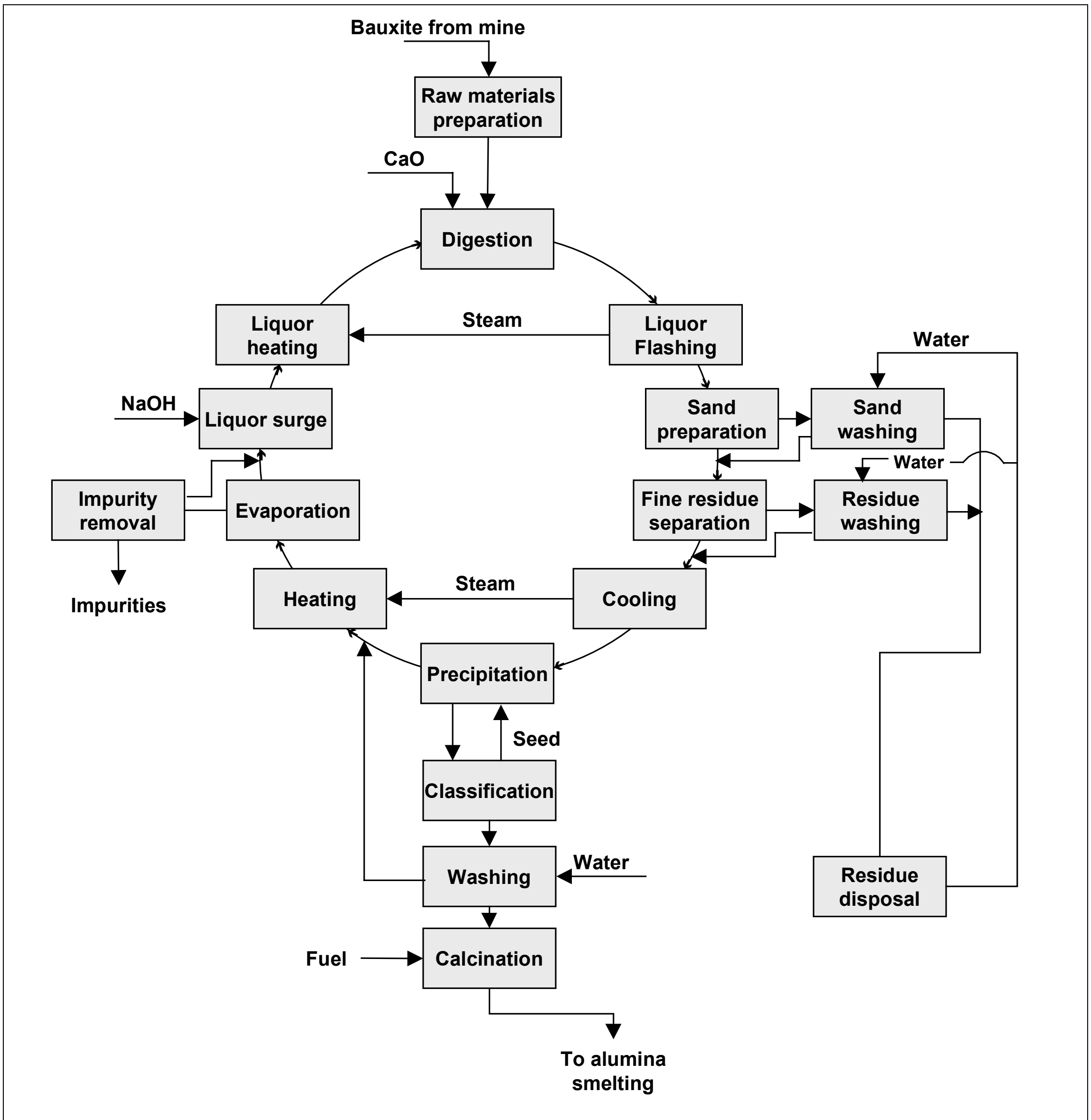

Figure 2.4: Flowsheet of Bayer-process

This process is normally carried out close to the mine site but there are sites in Europe where bauxite is converted to alumina at the same site as an aluminium smelter or at stand-alone alumina refineries.

The raw material for this process is Bauxite. It is washed, crushed and ground before it is dissolved in caustic soda (sodium hydroxide). The dissolution occurs at high pressure and temperature. The solution contains sodium aluminate and undissolved Bauxite (containing TiO_2, Fe_2O_3, Al_2O_3, SiO_2, Na_2O, CaO and sometimes traces of other metals such as zinc, nickel and vanadium). The settled solids, called “red mud” are removed. Hence, in accordance with the definitions used for this document, red mud are the tailings from the Bayer process. The amount of red mud generated per tonne of alumina produced depends on the grade of the ore, i.e. the Bauxite. Typically this number can vary from 0.3 to 2.5 tonnes. The resulting red mud contains dissolved sodium aluminate and a mixture of metal oxides.

Tailings also result from the washing stage.

Although the basic process is standard across the industry there are variations in the equipment used, in particular in the digesters and calciners. These variations mainly affect the energy used in the process.

Alumina is then precipitated out of the sodium aluminate solution. To achieve thisthe aluminate solution is cooled and seeded with alumina to crystallise hydrated alumina and the whole solution is then calcined at 1100 °C (to drive of the chemically bound crystalline water). The caustic soda is recycled and the resulting alumina, a white powder, is sent away, typically for smelting. More information about alumina refinery is available at http://www.world-aluminium.org/production/refining/.

2.3.2.2 Gold leaching with cyanide

Strictly speaking leaching is less a typical mineral processing technique than a hydro-metallurgical process. However, in the case of gold leaching it is applied to run-of-mine ore or is integrated into the other mineral processing steps (e.g. after comminution and gravity separation). Therefore leaching is generally considered part of mineral processing. Although other minerals may be leached and lixiviants other than cyanide are used (e.g. salt is leached or dissolved with water, copper may be leached with sulphuric acid), due to the high toxicity of cyanide and the public concern about its use in the mining sector, this chapter will focus on the use of cyanide in the leaching of gold. It should be noted that cyanide may also be used in the flotation of sulphides as a depressant for Pyrite (FeS_2).

Sampling and analytical methods can be found in Annex 1. *This will be taken from the CN code*

The following sections on the use of cyanide for leaching of gold are taken from the "International cyanide management code for the manufacture, transport and use of cyanide in the production of gold" (www.cyanidecode.org), unless otherwise mentioned.

2.3.2.2.1 Cyanide chemistry

This section provides a brief overview of the chemistry of cyanide. Cyanide chemistry is complex, and those seeking more detailed information should consult the list of reference materials found at www.cyanidecode.org.

Cyanide Species

The term cyanide refers to a singularly charged anion consisting of one carbon atom and one nitrogen atom joined with a triple bond, CN. The most toxic form of cyanide is free cyanide, which includes the cyanide anion itself and hydrogen cyanide, HCN, either in the gaseous or aqueous phase. At a pH of 9.3 - 9.5, CN and HCN are in equilibrium, with equal amounts of each present. At a pH of 11, over 99 % of the cyanide remains in solution as CN, while at pH 7, over 99 % of the cyanide will exist as HCN. Although HCN is highly soluble in water, its solubility decreases with increased temperature and under highly saline conditions. Both HCN gas and liquid are colourless and have the odour of bitter almonds, although not all individuals can detect the odour.

Cyanide is very reactive, forming simple salts with alkali earth cations and ionic complexes of varying strengths with numerous metals. The stability of these salts is dependent on the cation and on pH. The salts of sodium, potassium and calcium are highly soluble in water, and since they readily dissolve to form free cyanide, they are quite toxic themselves. Operations typically receive cyanide as solid or dissolved NaCN or $Ca(CN)_2$. Weak or moderately stable complexes such as those of cadmium, copper and zinc are classified as weak-acid dissociable (WAD), with equal concentrations of the complex and of its component metal and cyanide ions existing at a pH of approximately 4.0. Although metal-cyanide complexes themselves are less toxic than free cyanide, their dissociation releases free cyanide. Even in the neutral pH range of most surface water, WAD metal-cyanide complexes are sufficiently soluble so as to be environmentally significant.

The differing stabilities of various cyanide salts and complexes under varying pH conditions results in different potential environmental impacts and interactions with regard to their acute or chronic effects, attenuation and re-release. Cyanide forms complexes with gold, mercury, cobalt and iron that are very stable under mildly acidic conditions. However, both ferro- and ferricyanides release free cyanide when exposed to direct ultraviolet light in the presence of water.

Cyanide-metal species also form complexes with alkali or metalliferous cations, such as potassium ferricyanide ($K_3Fe(CN)_6$) or copper ferricyanide $Cu_3(Fe(CN)_6)_2$. The solubility of these complexes varies with the metal cyanide and the cation. Nearly all alkali salts of iron cyanates are very soluble, and if one of these double salts does dissociate to the cation and the cyanide-metal complex, the complex itself may then further dissociate to produce free cyanide. Heavy metal salts of iron cyanides form insoluble precipitates.

The cyanide ion also combines with sulphur to form thiocyanate, SCN. Thiocyanate dissociates under weakly acidic conditions, but is typically not considered to be a WAD species because it has similar complexing properties as cyanide itself. Thiocyanate is chemically and biologically oxidised to carbonate, sulphate and ammonia.

The oxidation of cyanide, either by natural processes or from treatment of effluents containing cyanide, can produce cyanate, OCN. Cyanate is less toxic than HCN, and readily hydrolyses to ammonia and carbon dioxide.

Cyanidation
The process of extracting gold from ore with cyanide is called cyanidation. The reaction, known as Elsner's equation, is:

$$4\ Au + 8\ NaCN + O_2 + 2\ H_2O \Leftrightarrow 4\ NaAu(CN)_2^- + 4\ NaOH$$

Although the affinity of cyanide for gold is such that it is extracted preferentially, cyanide will also form complexes with other metals from the ore, including copper, iron and zinc. The formation of strongly bound complexes such as those with iron and copper will tie up cyanide that would otherwise be available to dissolve gold.

Copper cyanides are moderately stable, and their formation can be a cause of both operational and environmental concerns. High copper concentrations in the ore increase costs and lower recovery efficiencies by requiring higher cyanide application rates to compensate for the reagent that complexes with copper rather than gold. The process water or tailings from such an operation can have significantly higher cyanide concentrations than would otherwise be present in the absence of copper.

Cyanidation is also adversely affected by the presence of free sulphur or sulphide minerals in the ore. Cyanide will preferentially leach sulphide minerals, and will react with sulphur to produce thiocyanate. These reactions will also enhance the oxidation of reduced sulphur species, lowering the solution pH and volatising HCN.

2.3.2.2.2 Use of cyanide in the gold industry

Gold typically occurs at very low concentrations in ores - less than 10 g/t or 0.001 %. At these concentrations the use of hydrometallurgical extraction processes, i.e. based on aqueous chemistry, are the only economically viable methods of extracting the gold from the ore.

Typical hydro-metallurgical gold recovery involves a leaching step during which the gold is dissolved in an aqueous medium, followed by separation of the gold bearing solution from the residues or adsorption of the gold onto activated carbon and final gold recovery either by precipitation or elution and electro-winning (see Figure 2.5).

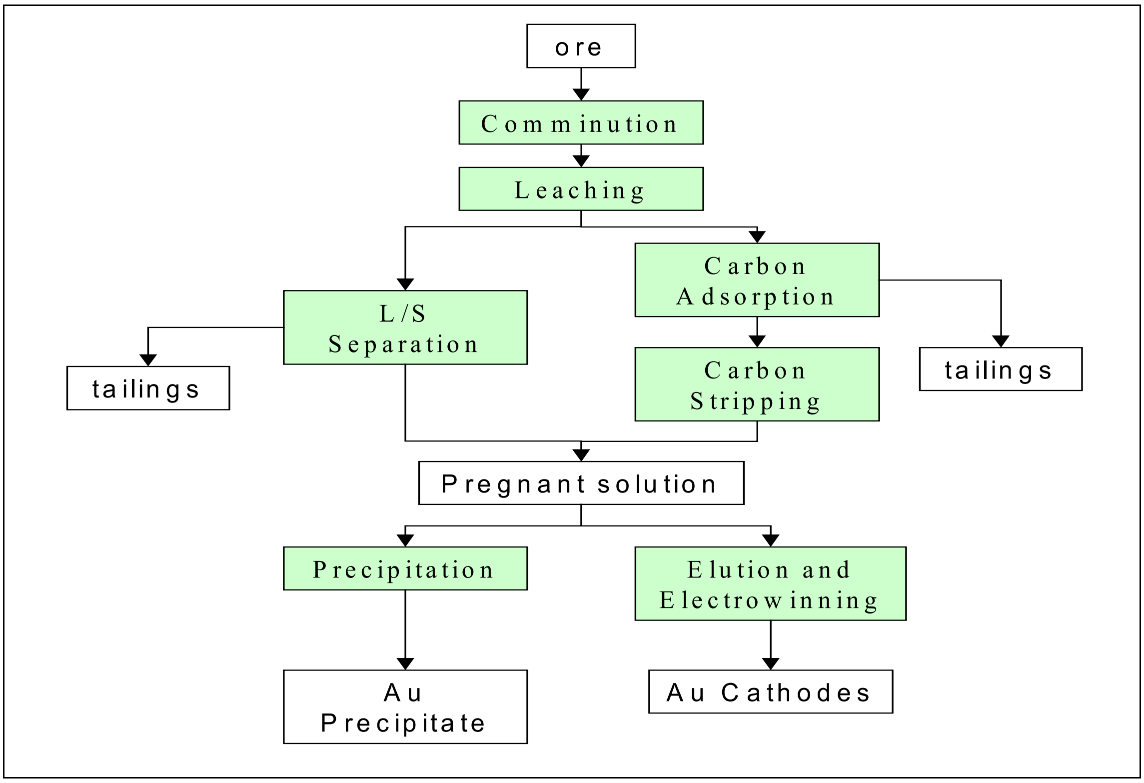

Figure 2.5: Principal of gold recovery by leaching

Often a gravity separation circuit is incorporated into this process after comminution to recover the sufficiently coarse gold particles (>30 µm) prior to leaching. The use of gravity separation in the field of gold recovery is rapidly advancing into ever smaller particles sizes (see Chapter 6).

Gold is one of the noble metals and as such is not soluble in water. A complexant, such as cyanide, which stabilises the gold species in solution, and an oxidant such as oxygen are required to dissolve gold. The amount of cyanide in solution required for dissolution may be as low as 350 mg/l or 0.035 % (as 100 % NaCN).

Alternative complexing agents for gold, such as chloride, bromide, thiourea, and thiosulphate form less stable complexes and thus require more aggressive conditions to dissolve the gold. These reagents are often more expensive to use and/or also present risks to health and the environment. This explains the dominance of cyanide as the primary reagent for the leaching of gold from ores.

<u>Manufacture, transport and storage of cyanide</u>
Approximately 1.4 million tonnes of hydrogen cyanide are produced annually worldwide, with only about 13 % being used to produce cyanide reagents for gold processing.

Cyanide is manufactured and distributed for use in the gold mining industries in a variety of physical and chemical forms, including solid briquettes, flake cyanide or liquid cyanide. Sodium cyanide is supplied as either briquettes or liquid, while calcium cyanide is supplied in flake form and also in liquid form. The strength of bulk cyanide reagents varies from 98 % for sodium cyanide briquettes, 44 – 50 % for flake calcium cyanide, 28 – 33 % for liquid sodium cyanide and 15 – 18 % for liquid calcium cyanide. The product strength is quoted on a molar basis as either sodium or calcium cyanide.

The form of cyanide reagent used is typically dictated by availability, means of transport, distance from manufacture source and cost. Large operations close to manufacturing facilities typically prefer liquid cyanide, while smaller and more remote operators typically use solid forms of cyanide, due to the risk of transporting liquids over long distances and the associated cost.

Where liquid cyanide is used, it is transported to the mine by tanker truck or rail car and is off-loaded to a storage tank. The truck or rail car may have a single or double walled tank and the location and design of the discharge equipment varies (e.g., top or bottom of tank).

Solid briquette or flake cyanide is transported to the mine in drums, plastic bags, boxes and returnable bins. Depending on how the reagent is packaged, the mine will have designed and constructed the necessary equipment to safely dissolve the solid cyanide in a high-pH solution. The pH value of cyanide solutions must be maintained above pH 10 to avoid the volatilisation of hydrogen cyanide (HCN) gas. The resulting cyanide solution is then pumped to a storage tank prior to introduction into the process.

The inventory of bulk cyanide reagent kept at the mine is dictated by the requirement to maintain continuous operations and to limit the frequency of off-loading events, which are regarded as safety critical events.

The cyanide solution is fed from the storage tank into the metallurgical process stream in proportion to the dry mass of solids in the process stream. The feed rate of cyanide is controlled so as to maintain an optimum cyanide level in the process as demanded by the metallurgy of the ore being treated.

Although the forms in which cyanide is manufactured, transported to the mine and stored for use varies, once introduced into the process, the technologies used for gold recovery are the same.

<u>Ore Preparation</u>
The aim of ore preparation is to present the ore to the lixiviant (the aqueous cyanide solution) in a form that will ensure the optimal economic recovery of the gold. The first step in ore preparation is crushing and grinding, which reduces the particle size of the ore and liberates the gold for recovery.

Ore that contains free gold may not yield a sufficiently high recovery by means of cyanide leaching only, and may be subject to a gravity recovery process where the free gold is recovered before the remainder of the gold is subject to cyanide leaching.

Gold bearing ores that contain gold associated with sulphide or carbonaceous minerals require additional treatment, besides size reduction, prior to gold recovery. Gold recovery from sulphide ore is poor because the cyanide preferentially leaches the sulphide minerals rather than the gold, and cyanide is consumed by the formation of thiocyanate. These ores are subject to a concentration process such as flotation, followed by a secondary process to oxidise the sulphides thus limiting their interaction with the cyanide during the gold leach. Carbonaceous minerals adsorb the gold after it has been dissolved. This is prevented by oxidising the ore prior to leaching. The leaching process may also be modified to counter this effect by the addition of activated carbon to preferentially adsorb the gold.

<u>Leaching with Aqueous Cyanide Solutions</u>
Gold is leached in aqueous cyanide by oxidising it with an oxidant such as dissolved oxygen and complexing with cyanide to form the gold-cyanide complex. This complex is very stable and the cyanide required is only slightly in excess of the stochiometric requirement. However, in practice the amount of cyanide used in leach solutions is dictated by the presence of other cyanide consumers and the need to increase the rate of leaching to acceptable levels.

Typical cyanide concentrations used in practice range from 300 to 500 mg/l (0.03 to 0.05 % as NaCN) depending on the mineralogy of the ore. The gold is recovered by means of either heap leaching or agitated pulp leaching.

With heap leaching, the ore or agglomerated fine ore is stacked in heaps on a pad lined with an impermeable membrane. The term "dump leaching" is sometimes applied to heap leaching of uncrushed ore. Cyanide solution is introduced to the heap by sprinklers or a drip irrigation system, the solution percolates through the heap leaching the gold from the ore. The gold bearing solution is collected on the impermeable membrane and channelled to storage facilities for further processing. Heap leaching is attractive due to the low capital cost involved, but is a slow process and the gold extraction efficiency is also relatively low.

In a conventional milling and agitated leaching circuit, the ore is milled in semi-autogenous, ball or rod mills to the consistency of sand or powder. The milled ore is conveyed as a slurry to a series of leach tanks. The slurry is agitated in the leach tanks, either mechanically or by means of air injection, to increase the contact of cyanide and oxygen with the gold and enhance the efficiency of the leach process. As mentioned earlier, the cyanide dissolves gold from the ore and forms a stable gold-cyanide complex.

The pH of the slurry is raised to pH 10 - 11 using lime, at the head of the leach circuit to ensure that when cyanide is added, hydrogen cyanide gas is not generated and the cyanide is kept in solution to dissolve the gold. The slurry may also be subject to other preconditioning such as pre-oxidation at the head of the circuit before cyanide is added.

Where oxygen instead of air is used as the oxidant, it has the advantage of increasing the leach rate and also decreasing the cyanide consumption due to the inactivation of some of the cyanide consuming species present in the slurry.

Where carbon is used to recover the dissolved gold, highly activated carbon is introduced into the process, either directly into the leach tanks (referred to as carbon-in-leach - CIL) or in separate tanks after leaching (referred to as carbon-in-pulp - CIP). The activated carbon adsorbs the dissolved gold from the solution component of the leach slurry thereby concentrating it onto a smaller mass of solids. The carbon is then separated from the slurry by screening and subjected to further treatment to recover the adsorbed gold, as described below.

Where carbon is not used to adsorb the dissolved gold in the leach slurry, the gold bearing solution must be separated from the solids component of the slurry utilising filtration or thickening units. The resultant solution, referred to as pregnant solution is subjected to further treatment (other than by carbon absorption) to recover the dissolved gold as discussed under gold recovery.

The material from which the gold has been removed by adsorption or liquids/solids separation is referred to as tailings. The tailings are either dewatered to recover the water and residual cyanide reagent, treated to either neutralise or recover cyanide, or sent directly to the TMF (see section on tailings management).

Recovery of dissolved gold

The gold is recovered from the solution by using cementation on zinc powder (so-called Merrill-Crowe process) or by first concentrating the gold using adsorption on activated carbon, followed by elution and either cementation with zinc or electro-winning. For efficient cementation a clear solution is required that is typically prepared by filtration or counter-current decantation. These are capital-intensive processes and have been superseded by processes using adsorption of the dissolved gold onto activated carbon. Adsorption is achieved by contacting the activated carbon with the agitated pulp. This can be done while the gold is still being leached with the carbon-in-leach or CIL-process, or following leaching with the carbon-in-pulp or CIP-process. Activated carbon in contact with a gold containing pulp can typically recover more than 99.5 % of the gold in the solution in 8 to 24 hours. The loaded carbon is then separated from the pulp using screens that are air or hydro-dynamically swept to prevent blinding by the near-sized carbon particles. This separation of ore particles (typically < 100 µm) from the coarser carbon particles (> 500 µm) is a lot less capital intensive than the filtration needed when using the Merrill-Crowe technique).

The fine barren ore, the tailings, is then either thickened to separate the cyanide containing solution for recovery or destruction of the cyanide, or sent directly to the TMF from which the cyanide containing solution is often recycled to the leach plant.

The gold adsorbed on the activated carbon is recovered from the carbon by elution, typically with a hot caustic aqueous cyanide solution. The carbon is then regenerated and returned to the adsorption circuit while the gold is recovered from the eluate using either zinc cementation or electro-winning. This gold concentrate is then calcined, if it contains significant amounts of base metals, or directly smelted and refined to gold bullion that typically contains about 70 – 90 % gold. The bullion is then further refined to either 99.99 % or 99.999 % fineness using chlorination, smelting and electro-refining. Recently developed processes utilise solvent extraction to produce high purity gold directly from activated carbon eluates, or following intensive leaching of gravity concentrates.

2.3.2.2.3 Process operation and the environment

The following are sources of cyanide emissions to the environment:

- CN to air as HCN
- seepage from tailings ponds
- tailings pond discharges necessary to manage overall water balance.

It is part of normal operation to attempt to optimise process economics. This may coincide with the objective to minimise cyanide impact on the environment and cyanide consumption. Process economics are sensitive to the amount of cyanide consumed in the process. Increased cyanide addition may have a “double-barelled” effect, meaning the operating costs increase through the extra amounts of cyanide that have to be purchased and because of the higher amounts of cyanides that will have to be destroyed or recycled prior to effluent discharge. Cyanide classified as “consumed” from a process point of view may still be active from an environmental perspective, e.g. copper cyanide complexes [24, BC CN guide, 1992].

The following operation strategies may be applied to minimise cyanide addition:

- taking steps to reduce consumption of cyanide by other components such as copper minerals, pyrrhotite, etc.
- attempting to retain cyanide in the circuit rather than discharging it to the tailings pond. This may be achieved by washing tailings, where practical
- strict control of water additions to the circuit to reduce the need to discharge solution in order to maintain a water balance. In arid climates no-discharge facilities are possible
- close monitoring of cyanide concentration in the process and in the tailings in order to keep cyanide addition to a minimum. Some sites have installed on-line analysis systems. These instruments can be coupled with automatic reagent dosing instrumentation
- improving aeration in the leach and/or adding oxygen or other oxidants to achieve the maximum rate of dissolution
- pre-aeration of the slurried ore before cyanidation to oxidise cyanide consuming constituents, which can then be thickened and removed from of the process.

[24, BC CN guide, 1992]

2.3.2.2.4 Human health and environmental effects of cyanide

Cyanide is produced in the human body and exhaled in extremely low concentrations with every breath. Cyanide is produced by over 1000 plant species including sorghum, bamboo and cassava. However, cyanide can be highly toxic to people and wildlife at low concentrations. For more detailed information on cyanide's effects on human health and the environment, see the bibliography included on the Cyanide Code website, www.cyanidecode.org or the British Columbia "Technical guide for the environmental management of cyanide in mining."

2.4 Tailings and waste-rock management

2.4.1 Types of tailings and waste-rock management facilities

There are many ways of managing tailings. The most common ones are:

- tailings ponds or dams
- dry-stacking of tailings
- tailings heaps
- backfilling of tailings
- discarding of tailings into surface water (e.g. sea, lake, river) or ground water.

Waste-rock is either managed on heaps or is sometimes dumped on existing hill sides.

The ways in which these different techniques are applied will be discussed in this section.

2.4.1.1 Tailings dams

Tailings dams are surface structures in which slurried tailings are managed. This type of TMF is typically used for tailings from wet processing, like flotation. A slurry is a mix of solids and water. Typically the slurries fed to tailings dams consist of 20 – 40 % solids by weight.

Figure 2.6 shows a cross-sectional view of a tailings dam and illustrates the water cycle of this type of TMF.

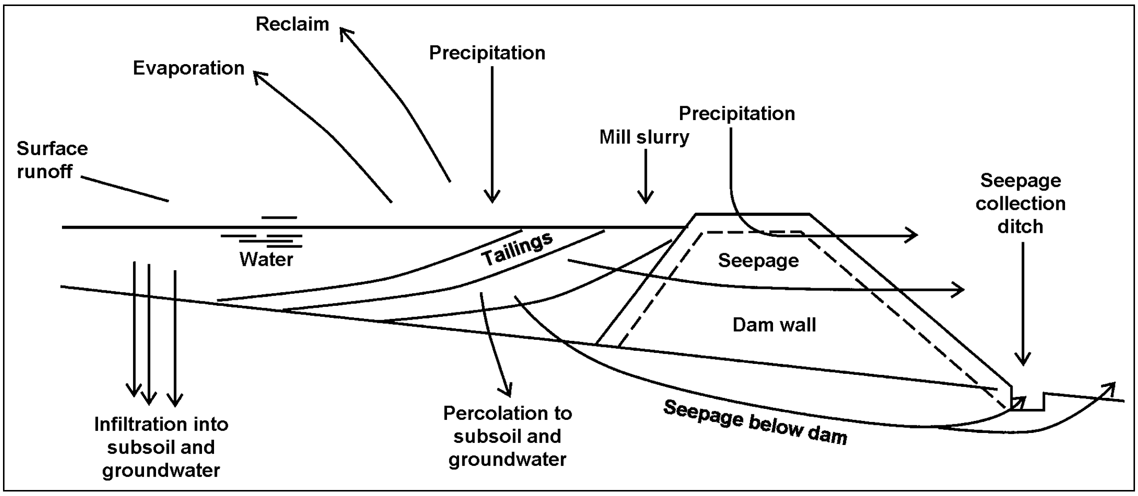

Figure 2.6: Dam water cycle
[11, EPA, 1995]

The following section on tailings dams is mostly gathered from ICOLD Bulletin 106 [8, ICOLD, 1996]. Other references are mentioned where appropriate.

The vast majority of tailings are managed on land. This entails the selection of a tract of land on which the tailings are stored for an extended period while the tailings are being generated by the mineral processing plant and, unless reclaimed for further treatment, for an indefinite period thereafter. The deposit must be secure against physical damage from outflow and must not pollute the surrounding area, neighbouring water courses, the groundwater, nor the atmosphere.

Since the tailings are conveyed as slurry from the plant and may remain as a suspension, or be capable of reverting to a fluid, the deposited mass requires confinement to the extent necessary to prevent the flow of the material out of the designated area. In most tailings dams the solids settle out of the slurry after discharge and it is thus composed of settled solids and supernatant water, which may be supplemented by natural run-off or direct precipitation forming a pond. The supernatant fluid may be returned to the processing plant for re-use, stored in the impoundment for future use, for removal by evaporation or discharged into surface water courses, often after undergoing treatment.

The basic arrangements of tailings dams may be classified as:

- existing pit
- valley site
- off valley site
- on flat land.

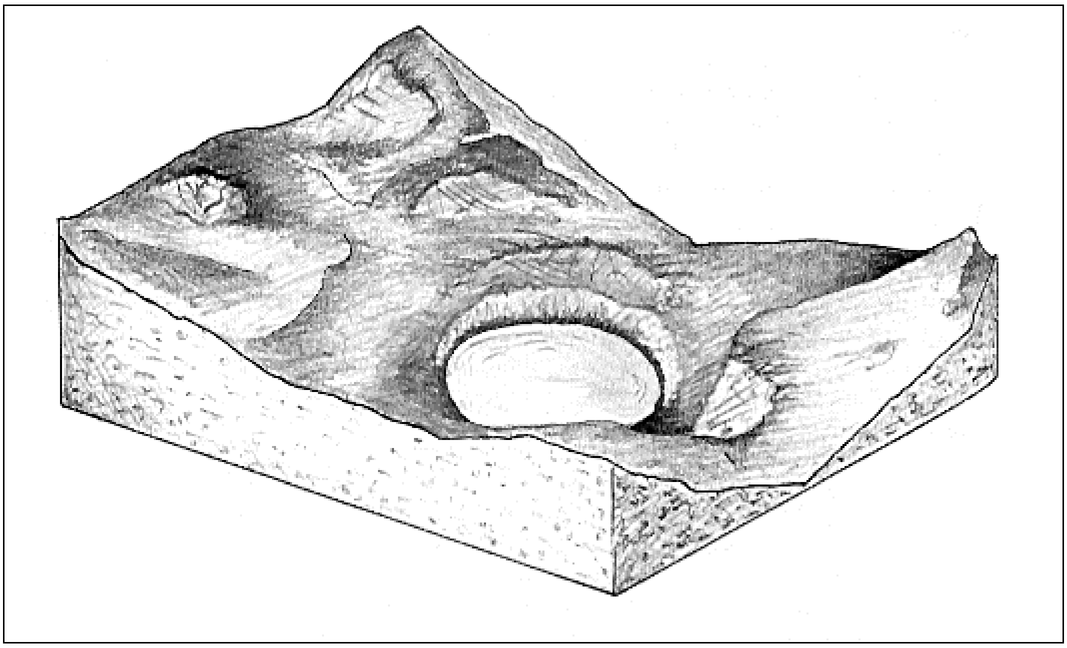

Figure 2.7: Illustration of tailings pond in an existing pit
[8, ICOLD, 1996]

The following gives an actual example of this type of TMF.

Figure 2.8: Picture of existing pit tailings pond

The following two pictures illustrate a valley site and an off-valley site tailings pond.

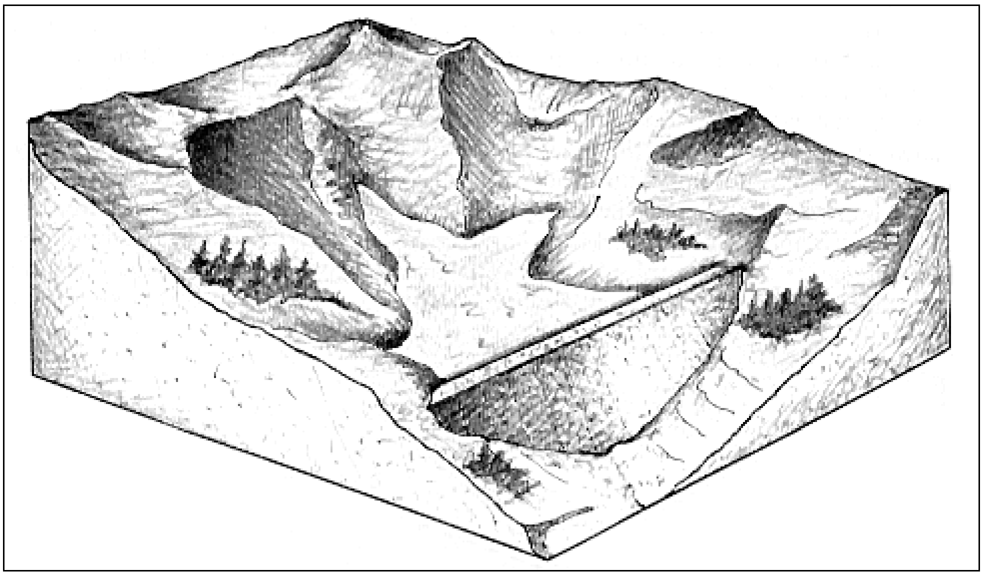

Figure 2.9: Illustration of tailings pond on a valley site
[8, ICOLD, 1996]

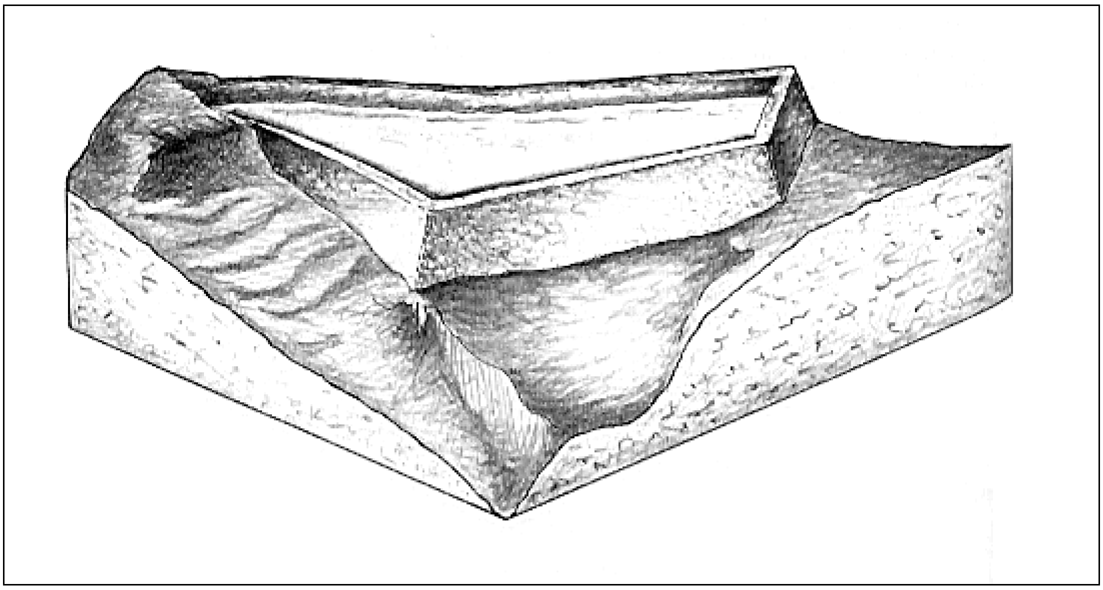

Figure 2.10: Illustration of an off-valley tailings pond
[8, ICOLD, 1996]

If a tailings pond is built on flat land it is often referred to as a paddock. The following picture gives an impression of paddocks used in South African gold mining operations.

Figure 2.11: Tailings pond on flat land
(Courtesy of AngloGold, South African Division)

For each tailings impoundment the following activities need to be considered:

1. tailings delivery from the mineral processing plant to the tailings dam
2. dams to confine the tailings
3. diversion systems for natural run-off around or through the dam
4. deposition of the tailings within the dam
5. evacuation of excess supernatant water
6. protection of the surrounding area from environmental impacts
7. instrumentation and monitoring systems to enable surveillance of the dam.

2.4.1.1.1 Delivery systems for slurried tailings

Slurry transport from the plant to the TMF is most often undertaken by pipeline. In some cases open channel conveyance may be used, as it is cheaper. The pipeline is seldom buried. Occasionally the slurried tailings are transported from the mineral processing site to the TMF by trucks.

2.4.1.1.2 Confining dams

The construction material and methods used in forming the dam vary widely to accommodate the particular needs of the selected site, the availability of materials and the financial and operating policies of the entire operation. The dam types may be classified as follows:

- water-retention type dams
  - conventional dam
  - staged conventional dam
  - staged dam with upstream impermeable zone.

- progressively built tailings dams
  - dam with tailings impermeable zone
  - dams with tailings in structural zone
  - upstream construction using beach or paddock.

These types will be briefly discussed below.

<u>Conventional dam</u>
This type of dam is completely built before tailings are discharged at this site. Hence, tailings can not be used to build the dam. Conventional dams are constructed where the confinement is to be effected for both tailings and free water during the whole period, from the start of tailings management to the end of the particular site selected.

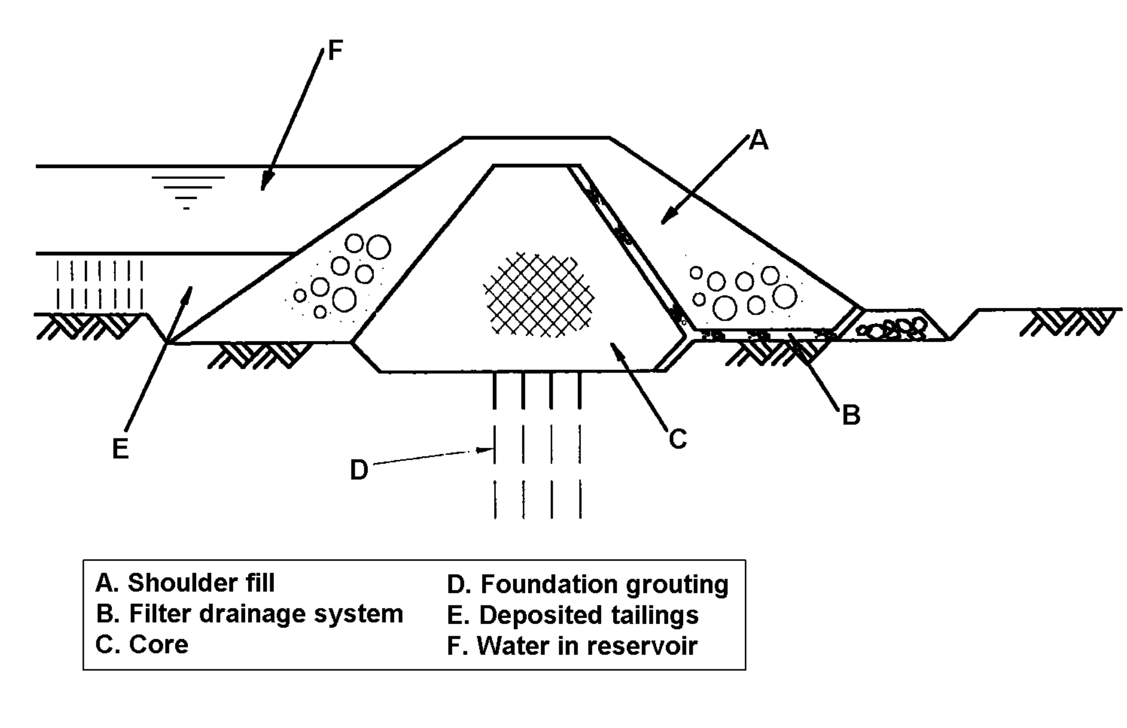

Figure 2.12: Conventional dam
[8, ICOLD, 1996]

The purpose of the shoulder fill is to increase the overall dam strength, but also to protect the core from erosion (wind on downstream site, water inside the dam) and wave action from the supernatant water

A conventional central core section is illustrated in the above figure but the range of options is wide and similar to that for dams designed to confine water alone. The dam must be capable of
- controlling the passage of water
- supporting the loads imposed by the tailings and water in the impoundment
- transmitting the seepage water effectively and without the passage of solids (filtration system).

This type of dam is applied where
- the tailings are not suitable for dam construction
- the impoundment is required for the storage of water, usually on a seasonal basis, for plant or other use
- the tailings management site is in a remote and inaccessible location
- retention of the tailings water is needed over an extended period for the degradation of a toxic element (e.g. cyanide)
- the natural inflow into the impoundment is large or subject to high variations and water storage is needed for its control.

Advantages:
- dam is built with supervised construction in a relatively short time span
- minimal supervision of the dam during operation
- protection against pollution by water and wind erosion.

Disadvantages:
- need for high capital expenditure before the facility is operational
- all construction materials have to be imported.

Staged conventional dam

This is similar to a conventional dam but has reduced the initial capital cost by staging the construction so that the costs are spread more evenly over the period of deposition.

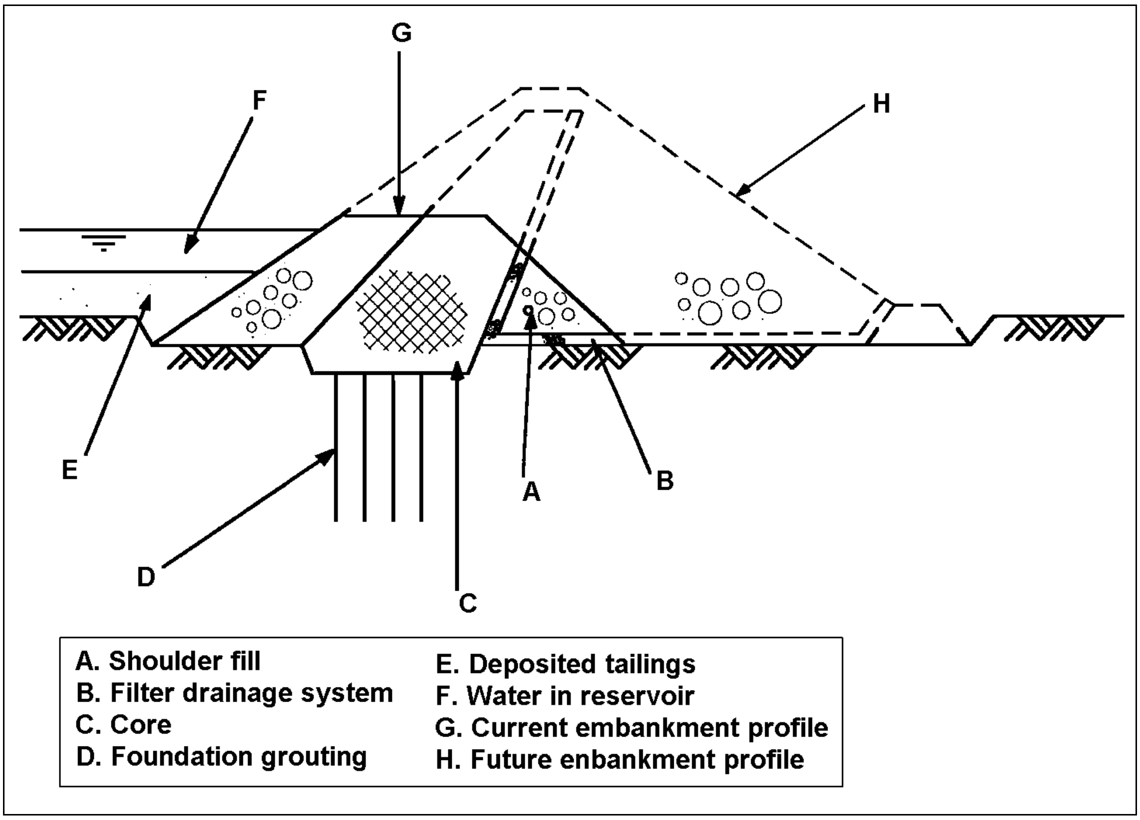

Figure 2.13: Staged conventional dam
[8, ICOLD, 1996]

Staged dam with upstream core

If the deposited tailings lie close to or above the level of the supernatant water in the impoundment, the low permeability core zone of the dam may be located on its upstream face. This is possible because the core is protected against erosion and wave action by the tailings.

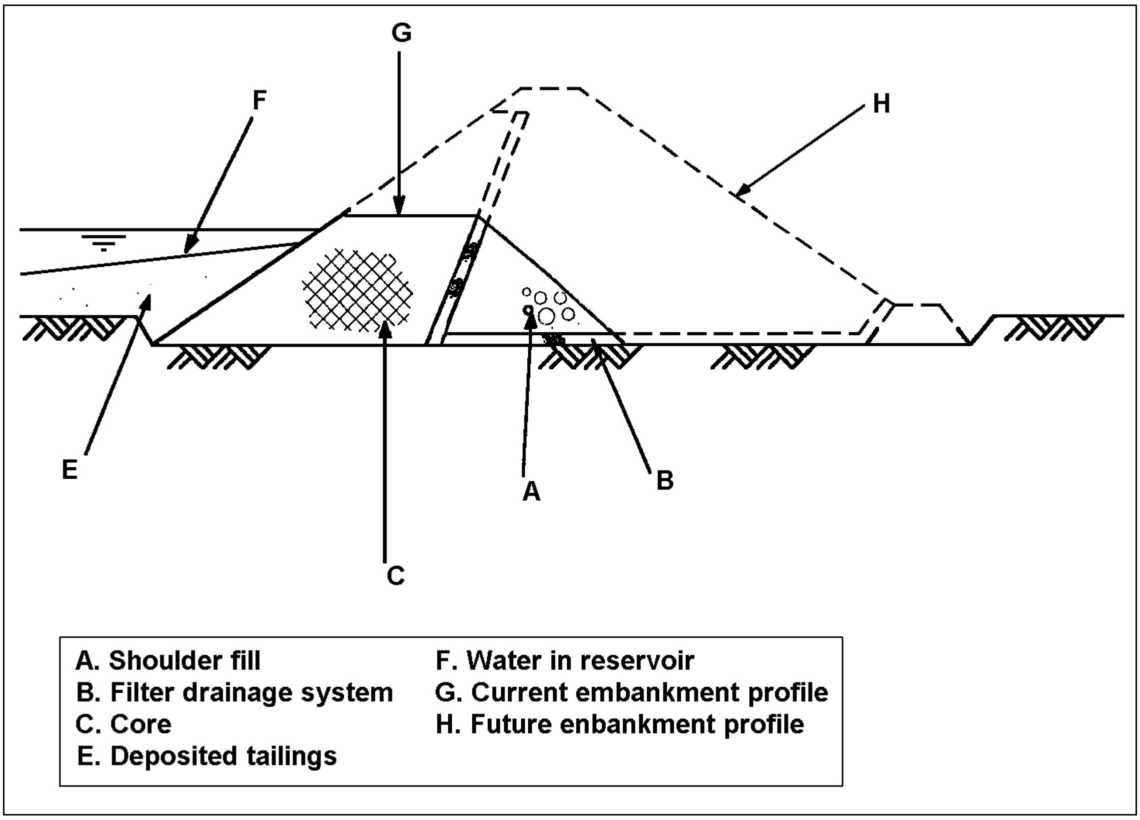

Figure 2.14: Staged dam with upstream low permeability zone
[8, ICOLD, 1996]

<u>Dam with tailings low permeability core zone</u>
Where all or part of the tailings deposition into the pond occurs from the dam a beach of tailings may be formed. It is then possible for the tailings beach alone to provide the less permeable zone of the system.

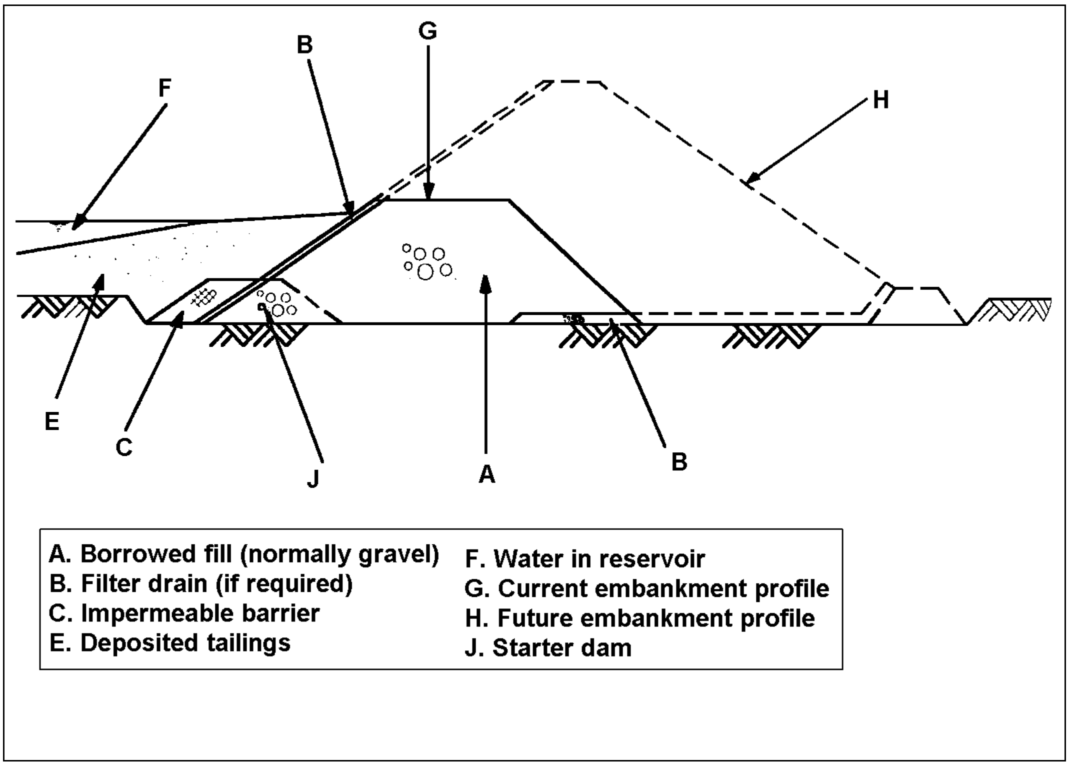

Figure 2.15: Dam with tailings low permeability core zone
[8, ICOLD, 1996]

This arrangement is only possible where the inflow of water will not allow the impoundment water level to rise above the uppermost level of the beach and against the more pervious dam material. Therefore, continuos monitoring is required for this kind of scheme.

For this arrangement it is necessary to build and impermeable barrier (C) into the starter dam, until the beach has developed far enough away from the dam itself.

Dam with tailings in structural zone
In this arrangement tailings are not only used as a water barrier but also as construction material of the dam. In this case typically the coarser hydrocyclone underflow is for the structural zone and the finer hydrocyclone is discharged into the pond forming the beach.

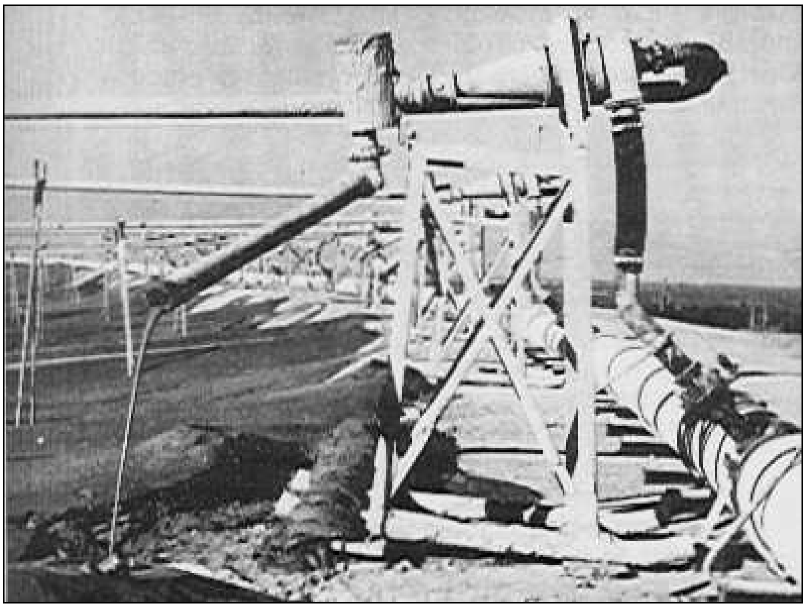

Figure 2.16: Row of hydrocyclones on the crest of a dam

For further information on hydrocyclones please refer to Section 2.3.1.

There are three main approaches when considering the progressive construction of a dam. These are:

- upstream method
- downstream method
- centreline method.

These methods allow for staged construction of the dam, which minimises start-up capital cost. The following figure illustrates these methods.

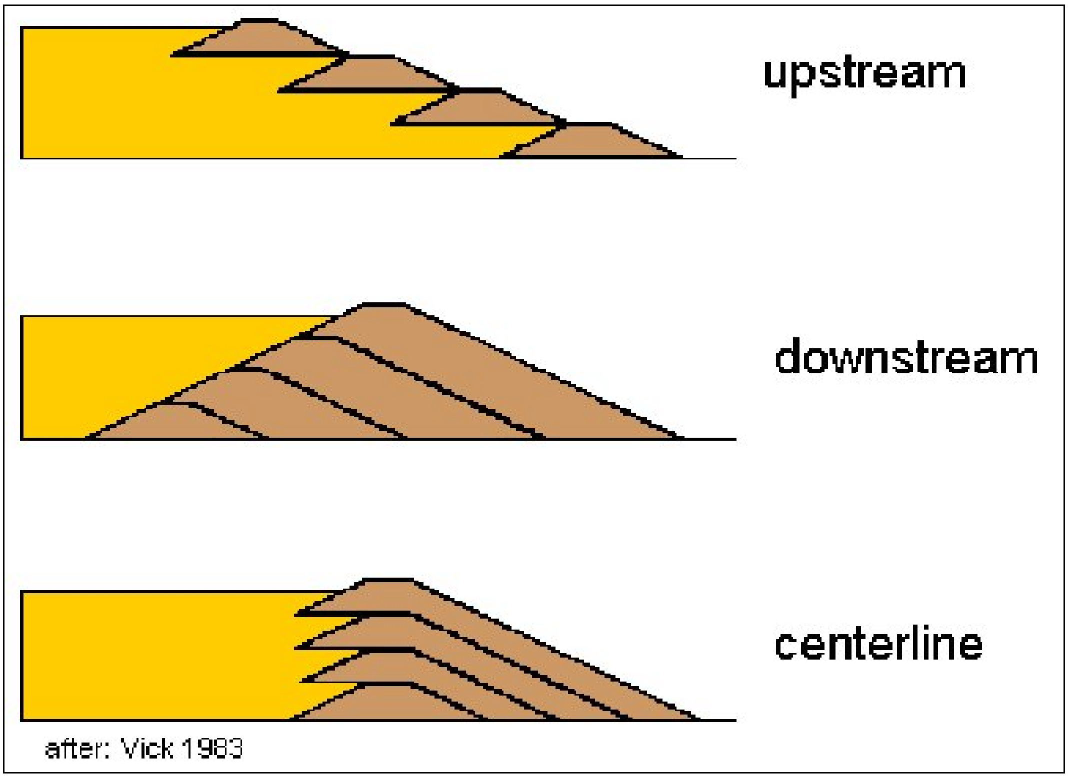

Figure 2.17: Types of sequentially raised tailings dams
[11, EPA, 1995]

Upstream method using cycloned tailings
This method is very economical in the use of the coarser fraction of the tailings since only a thin outer zone of this material will result

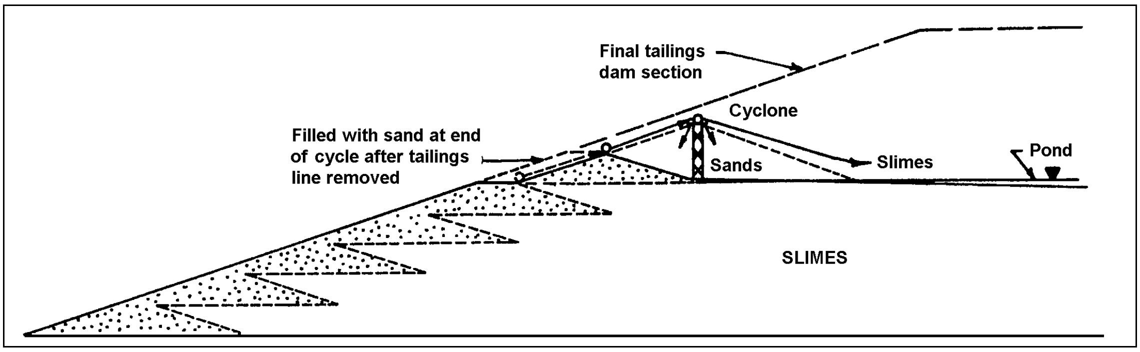

Figure 2.18: Upstream method using cycloned tailings
[11, EPA, 1995]

The following picture shows a dam built using the upstream method. The dam itself consist of borrowed rock-fill.

Figure 2.19: Dams raised using the upstream method at the Aughinish site

The main disadvantage of this method has in the past been the physical stability on the dam and its susceptibility to liquefaction. Care must be taken in the design in order to control the phreatic surface, which can be achieved by correct drainage. Also, the exposed tailings, used to build the dam should not have ARD potential.

The finer tailings discharged as cyclone overflow are often intrinsically weaker than the coarser fraction and less dense since it is either deposited under water or may not be able to drain, consolidate or desiccate. Their less permeable nature may also result in a high phreatic surface indicating the development of pore pressures. The material can form a weak zone with respect to downstream slope stability and the upstream method is consequently only used where it can be demonstrated that the finer material is rendering strong enough by drainage and/or desiccation to provide adequate support to the slope. Stability is a major concern in areas of high seismicity.

Downstream method

The coarse fraction of the tailings, separated by the cyclone, may be used to form the complete structural portion of the dam or a large part of it. The size of cyclone is selected such that a bank of them acting in parallel can deal with the tailings throughput. With the tailings delivery line and the bank of cyclone offtakes located initially on the crest of the starter dam, the underflow is discharged downstream to form the dam, and the overflow is discharged into the impoundment, as illustrated in Figure 2.20.

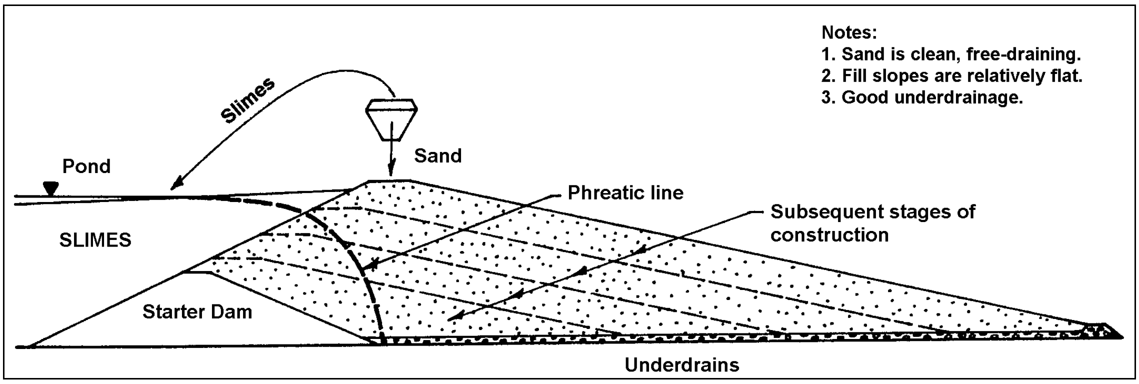

Figure 2.20: Downstream construction of a dam using cyclones
[11, EPA, 1995]

This method is called the downstream method because as the dam height rises, the crest moves downstream.

Centreline method

The downstream method of construction entails the use of a considerable volume of coarse tailings for the dam, and an area of land under the "footprint" of the dam. Where the proportion of the coarse tailings separated out by cycloning is insufficient to permit the dam to keep ahead of the rise of the impoundment level, the tailings zone may need to be supplemented by a zone of borrowed material. As an alternative to this option the upstream portion of the dam may be composed of the beach of deposited tailings. This is possible because the upstream face of the dam is progressively supported by the rise of tailings. The resultant structure is illustrated in Figure 2.21 and the method is generally termed the centreline method.

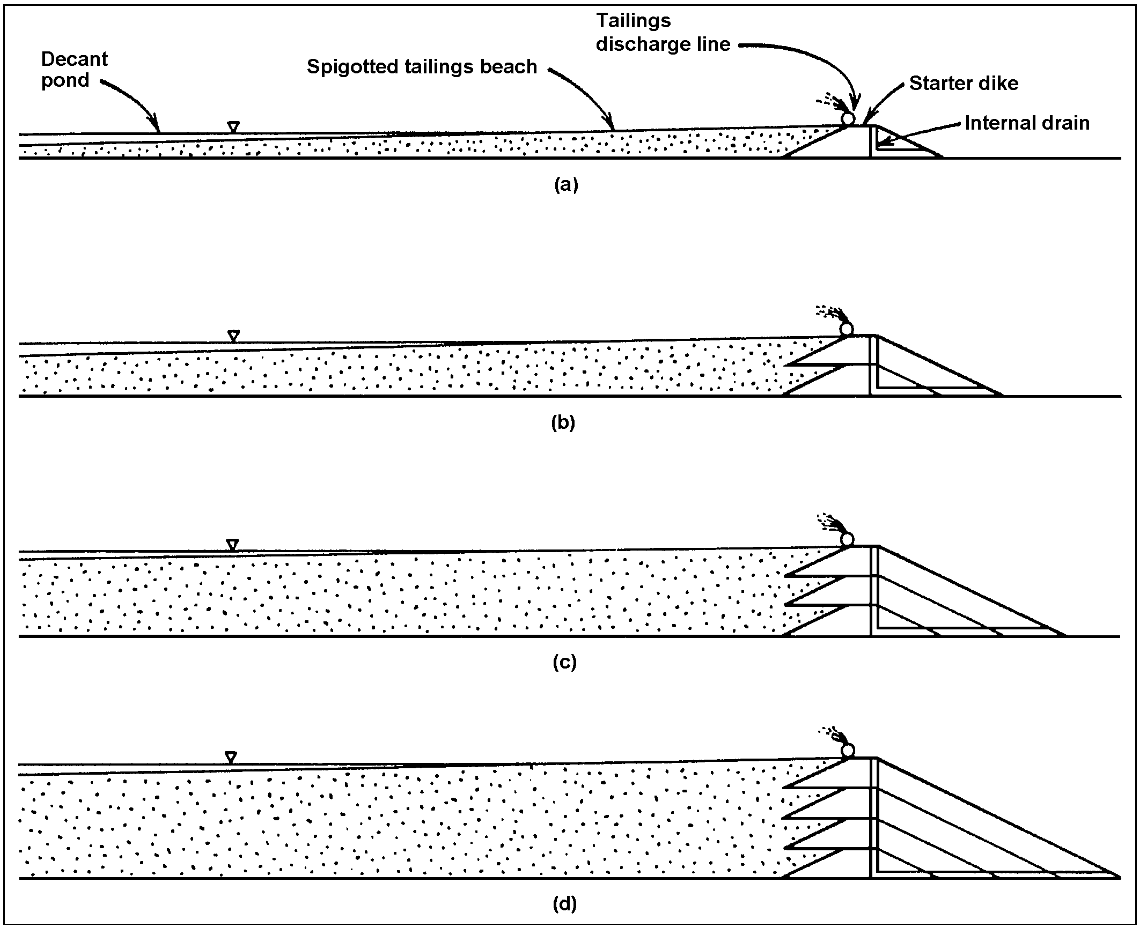

Figure 2.21: Centreline method
[11, EPA, 1995]

Upstream construction using beach or paddock

This traditional tailings dam construction method uses the beach instead of a cyclone to size-sort the tailings. This method makes maximum use of the tailings itself for confinement, may provide the cheapest system of tailings management. The system relies on the formation of a satisfactory beach by control of the deployment of the discharge arrangements and by control of the length of time material is discharged from each point.

2.4.1.1.3 Embankment stability

The stability of dam slopes depends on factors such as

- the geometry of the cross-section
- the phreatic surface
- the strength parameters of the materials in the dams and its foundations.

The safety factor of a slope is defined as the ratio of available shear strength to the shear stress required for equilibrium.
[75, Minorco Lisheen/Ivernia West, 1995]

Please provide further information to this section.

2.4.1.1.4 Diversion of natural run-off

Diversion of natural run-off may be required

- to maintain the necessary freeboard
- to avoid contamination of the natural run-off with process liquids or chemicals
- to reduce the volume of water in those impoundments relying on evaporation to remove excess water rather than treatment and discharge.

Three standard methods of diversion are employed, the choice generally being related to site topography and expected flow rates:

- channels above and around the dam
- conduits underneath the dam
- tunnels through the flank of the dam.

The diversion system is critical to the safety of a tailings dam. Failure of any part can lead to the impoundment receiving floods of which it was not designed and possibly overtopping with total failure of the dam. The engineering of diversion structures has thus to be given a high priority in planning the facility.

2.4.1.1.5 Deposition in the impoundment

<u>Wet deposition</u>
In some applications, particularly where conventional dams are employed, the management of tailings into the impoundment can take the form of a **single-pointed open-ended discharge**. In other cases a more controlled deposition method may be desirable. This may incorporate **line or perimeter discharges** or the use of **cyclones** [21, Ritcey, 1989]. In the case of progressively built tailings dams the discharge arrangements are dictated by the dam construction method selected.

The increase of density of deposited material is accelerated by the action of drainage and evaporation. Therefore storage efficiency can be increased by **deposition taking place on a beach**.

<u>Thickened deposition</u>
Since the angle of deposition increases with the solids content of the tailings the **thickening of tailings** is becoming a more and more popular option. Usually tailings would be pumped into the tailings pond with 25 to 50 % solids. Thickened tailings have a solids content of over 60 %. This enables the storage efficiency in terms of the storage volume to dam height to be substantially increased.

In some cases it is necessary for all the tailings to be deposited under water (e.g. tailings with ARD potential). This is referred to as **sub-aqueous** deposition.

In the case of very fine tailings **special techniques** may be employed, such us the addition of coarser particles or flocculants.

2.4.1.1.6 Removal of supernatant fluid

The aim throughout the development of the impoundment is usually to keep the pool of supernatant water as low and as small as possible as a means of risk management. However, this needs to be balanced against several other objectives, e.g. tailings need a certain amount of time to settle within the pond. Also, in some cases the water has to remain in the dam for a certain period of time in order to allow deterioration of the process chemicals. Water saturation of the tailings may also be required to avoid dusting.

A good balance between the need to keep the pool low and the contradicting requirements to leave a certain amount of water in the pond may be a clarification pond. This allows the settling of the fines slimes and deterioration of process chemicals, whilst the water level in the actual dam, containing the settled tailings, can be kept to a minimum.

The main requirement for successful removal of the water is the provision of an outlet arrangement, the effective level of which can be adjusted throughout the progressively increasing impoundment level, or of a pump, which can perform a similar function. The removed water is either returned to the mineral processing plant and/or, usually after treatment, discharged into natural water courses.

The outlet structure, or “decanting system” as it is normally termed, is usually composed of two elements:

- an extendible intake, and
- a conduit to convey the discharge away from the dam.

The intake may take the form of a vertical tower, or a sloping chute founded usually in natural ground on a flank of the impoundment and occasionally on the upstream face of the dam.

The following figures show the three basic alternatives:

- decant tower
- decant chute
- pumped decant.

Other options are

- drained pond or
- overflow systems
    - within the dam
    - around the dam.

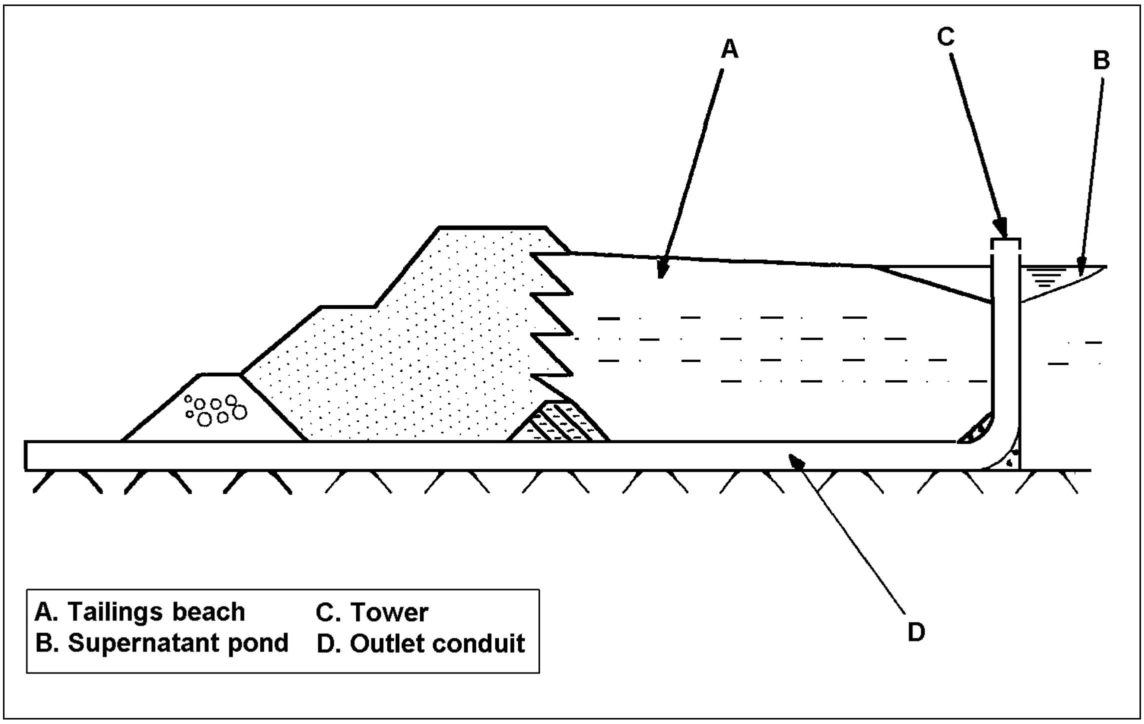

Figure 2.22: Tower decanting system
[8, ICOLD, 1996]

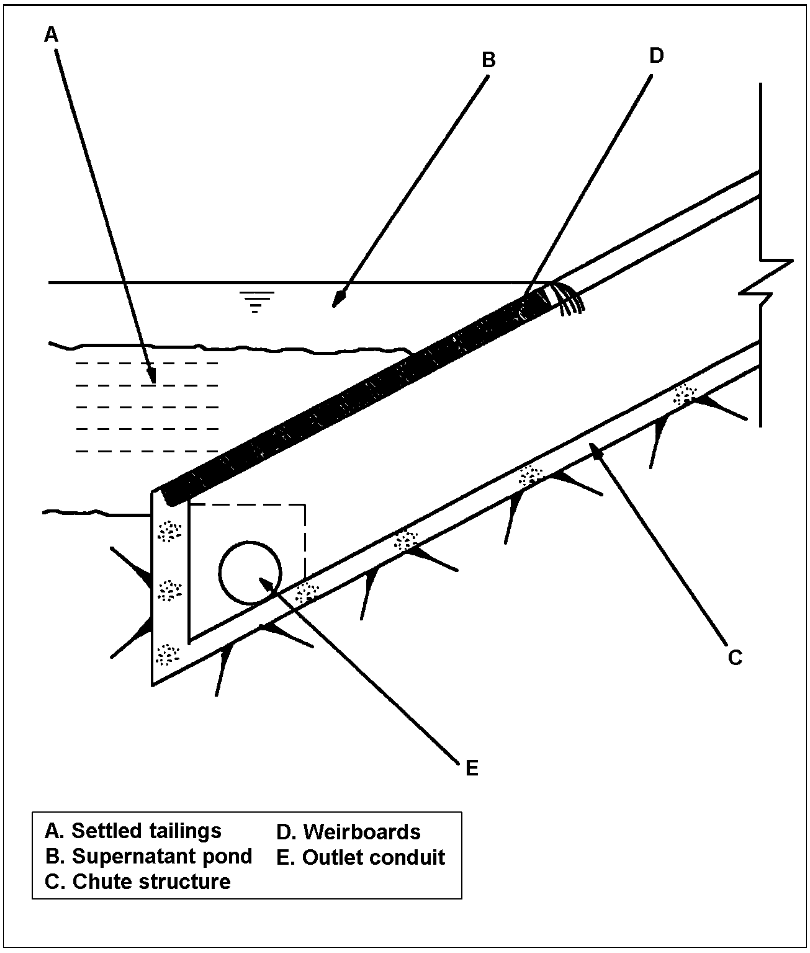

Figure 2.23: Chute decanting system
[8, ICOLD, 1996]

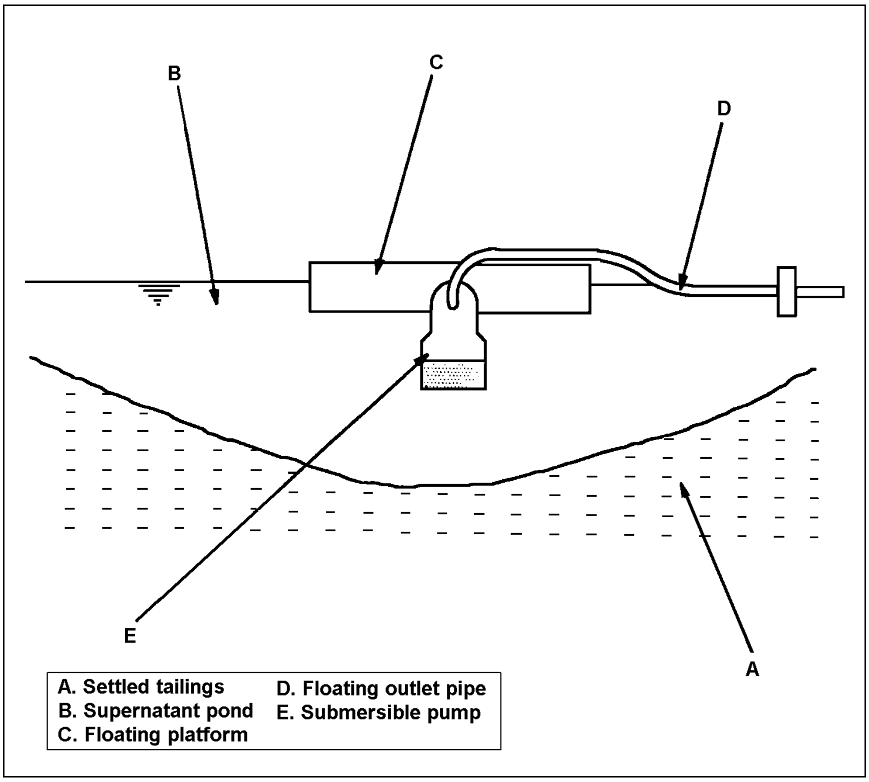

Figure 2.24: Pump barge
[8, ICOLD, 1996]

2.4.1.1.7 Protection of the surrounding area from environmental impacts

The main causes of pollution from tailings management facilities are

- the dispersal of **solid particles** of tailings to the surrounds
- the seepage or flow of polluted **water** into the soil and groundwater or into natural water courses and
- the discharge of **gases** into the atmosphere.

Particular attention should be given to the geochemistry of the materials to be stored in order to estimate possible emissions from the TMF during the planning stage of a mine/mineral processing site.

The following table lists means of dispersion of **solid** tailings from the tailings dam and some prevention options.

Solids may be dispersed by:	Prevention:
water erosion to the surfaces of the impoundment	▪ covering the **sloping surfaces** of the impoundment with a protective layer such as gravel, a soil and grass cover, geofabric and grass cover or some form of synthetic coating ▪ impregnation of the **surface layer of the tailings** with a chemical which will repel water or result in particle binding such as a silica compound, cement, bitumen or bentonite ▪ the **chemical properties of the tailings**, such as those containing sulphides, may assist particle binding.
wind erosion to the surfaces of the impoundment ▪ crest of dam ▪ slopes of impoundment ▪ surface of the beaches	▪ dam crest and slopes may be treated as for water erosion ▪ surface may need wind breaks, water spraying, application of binding material (i.e. spraying with bituminous emulsion) (ICOLD 106), surface mulch (Aus EPA) ▪ in extreme cases tailings may have to be deposited under water ▪ surface vegetation, either floating or on inactive areas ▪ frequent change of discharge points around perimeter to achieve constantly wetted surface (AUS EPA).
removal with the decanted water	operational control
outflow ▪ as a result of failure of the decanting system or of the slopes of the impoundment.	adequate design of these structures

Table 2.2: Dispersion of solid tailings from tailings dams and prevention options

The escape of process **water** from the pond is a problem to be overcome. The water may emanate from any of the following sources:

- discharge through the decanting system
- seepage through the dam and
- seepage from the impoundment into the ground and groundwater.

If the supernatant water in the impoundment can not be discharged directly into the natural water courses, it will be necessary either to arrange the deposition such that all supernatant water is either returned to the plant or, in arid, hot climates, evaporated. The decant water may be stored by a reclaim dam downstream of the impoundment and, in some cases, treated before discharge into the natural water course.

Seepage capture by pumping is an option of control for emissions into groundwater, provided that it is recognised that there may be an ongoing commitment after the tailings impoundment is closed. The necessity for pumping after closure should be reviewed in the rehabilitation and closure plan.

A prerequisite for the design of seepage control systems is a thorough understanding of the hydrogeological background of the site. This would normally involve the installation and monitoring of piezometers to determine directions of flow, hydraulic gradients and aquifer characteristics. On consideration of such data decisions can be made on the appropriate measures.

Two types of control measure could be considered, namely

- seepage barriers and
- return systems.

Seepage barriers serve to prevent seepage and include cut-off trenches, slurry walls and grout curtains.

Please provide definitions for: include cut-off trenches, slurry walls and grout curtains

However, possible disadvantages of these measures in connection with the stability of the tailings dam must be considered in each case.

In some cases it may be more appropriate to install return systems instead of seepage barriers. Return systems collect, rather than impede, seepage flow, thereby enabling seepage water to be retained for treatment or then discarded in a manner which will not damage the environment. The return system could consist of collector ditches and wells.

The advantages and limitations of seepage control measures are given in the following table.

Seepage control measures	Type	Advantages	Limitations
Seepage	Cut-off trench	Inexpensive; installations can be well controlled	Not practical for saturated barrier foundations; effective only for shallow pervious layers
	Slurry walls	Low-permeability barrier can be constructed	High cost; not well suited for steep terrain or bouldery ground; impervious lower boundary required
	Grout curtains	Barrier can be constructed to great depths; not affected by site topography	High cost; limited effectiveness due to the permeability of grouted zone; cement grouting practical for only coarse soils of wide rock joints
Return systems	Collector ditches	Inexpensive; suitable for any type of dam	Effective for shallow pervious layers, but still beneficial in other cases
	Collector wells	Greater depth possible; useful as a remedial measure	Expensive; effectiveness depends on local aquifer characteristics.

Table 2.3: Summary of seepage control measure

The use of liners is another means of reducing seepage (see Section 3.1.6). All liner systems have a leakage rate which depends on

- the magnitude of the hydraulic head above the liner
- the thickness and effectiveness of the liner material
- the length of time the hydraulic head is applied to the liner.

It is important to be aware of the hydrogeological background and the geochemical features of the tailings to be managed [11, EPA, 1995].

The use of liners is an often debated subject. The advantage of using them is the possibly high reduction of seepage. However, critics say that it is not possible to predict how long the liner will function properly for. The alternative is to handle seepage from the start.

Gas emissions from the tailings are normally limited by the deposition of the tailings under water and the retention at all times of water over the deposited material. With careful control other management methods, such as the sub-aqueous method, may be programmed to keep gas emissions within acceptable limits.

Gas which may have dissolved in the water, can be liberated when the water is discarded through decant systems, but measurements have shown that amounts are usually small. Tailings from gold leaching may generate cyanide gas (HCN), released from the supernatant water. However, before cyanide bearing solutions enter into the pond the cyanide content should be reduced as much as possible (see Section 2.3.2.2)

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2.4.1.1.8 Instrumentation and monitoring systems to enable surveillance of the dam

The tailings dam design should include a monitoring system for assessing ongoing stability (chemical and physical) of the dam structures.

The following table indicates the types of measurements that are usually performed and the instrumentation used [7, ICOLD, 1996]

Measurement	Instrumentation
Water level	Level
Seepage discharge through ▪ the dam itself ▪ the foundation ▪ the abutments.	Weirs or containers
Position of phreatic surface	Piezometer (typically open standpipe)
Pore pressure	Piezometer or Bourdon tube pressure gauge
Movement of dam crest and tailings	Geodetic datum points on beach (completed dam) and crest of the dam, aerial photography, GPS
Seismicity	Strong motion accelerographs
Dynamic pore pressure and liquefaction	Vibrating wire piezometers
Soil mechanics	Penetrometers for density and shear strength
Tailings placement procedures	Shear strength, compressibility, consolidation, grain size and density samples, width of the non-submerged beach as indication of phreatic surface via aerial or satellite photography

Table 2.4: Measurements and instrumentation required for tailings dams monitoring [7, ICOLD, 1996]

2.4.1.2 Thickened tailings

Thickened tailings management uses mechanical equipment to dewater tailings to about 50 – 70 % solids. The tailings are then spread in layers over the storage area to further dewater through a combination of drainage and evaporation [11, EPA, 1995]. The main benefit of this technique is the fact the dam is not as high as in the case of a conventional slurry TMF. On top of that the tailings are less mobile, which is beneficial in the event of a tailings dam burst.

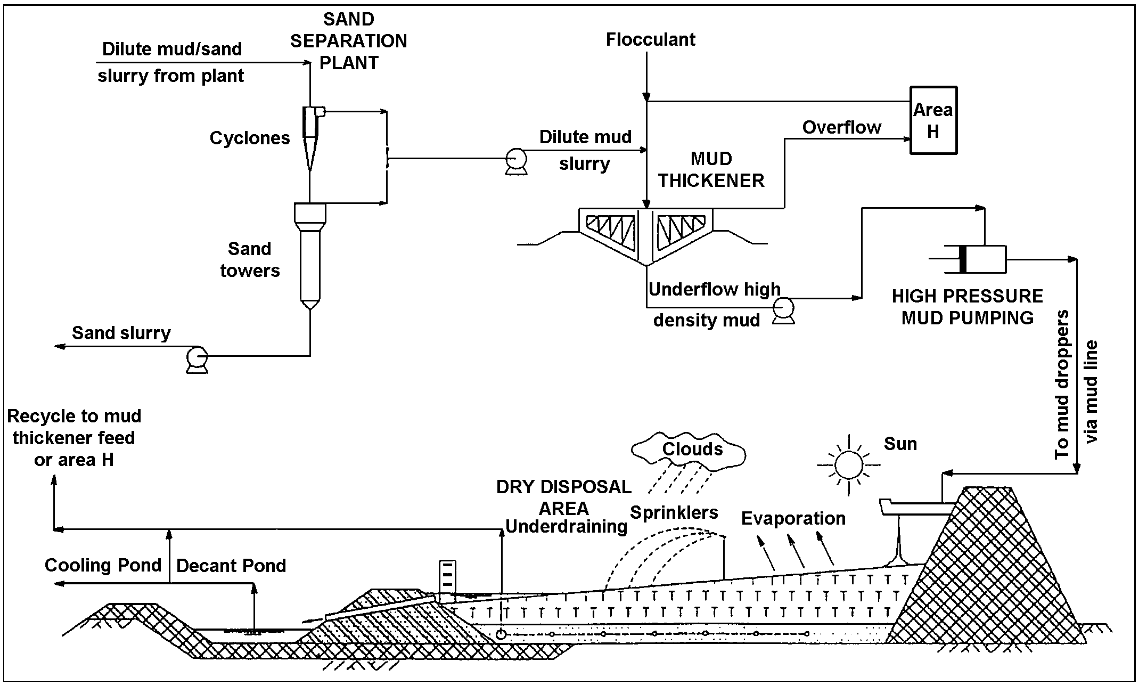

Figure 2.25: Schematic drawing of thickened tailings management operation
[11, EPA, 1995]

Other advantages and disadvantages are:

Advantages:
▪ low initial capital investment required
▪ cost of maintenance and closure reduced
▪ storage capacity is greater for the same height of perimeter dam
▪ susceptibility to liquefaction is low, giving higher earthquake resistance
▪ decant system, with its problems, is eliminated
▪ reduced seepage to surrounding terrain results
▪ most water separated at mineral processing plant, hence the need to recycle water from pond is eliminated [77, Robinsky, 2000]
▪ easy closure and rehabilitation
Disadvantages:
▪ transport of thickened tailings may be difficult and expensive; effectively done by thickening facility at management site.

Table 2.5: Thickened tailings deposition advantages and disadvantages
Source: [21, Ritcey, 1989] unless otherwise mentioned

In addition to being a discharge method this method has been recommended to cover existing conventional tailings ponds [21, Ritcey, 1989].

The thickened tailings method may be of particular advantage under the following conditions:

- flat topography, allowing development of a wide conical deposit with flat slopes
- where the construction of a conventional dam may be costly because of site conditions
- where the tailings are so fine that no coarse fraction is available.

[21, Ritcey, 1989]

One publication claims that thickened tailings are also advisable for tailings with acid generating potential. This is justified by the fact that the fines in the homogenous mix of thickened tailings provide high capillary suction that maintains the tailings in a saturated state, thereby inhibiting acid generation [77, Robinsky, 2000]. However this is often disputed and it is hard to understand how these saturated tailings are stable over a long period of time.

The following picture shows a comparison of a thickened tailings system and a conventional tailings pond in the same setting.

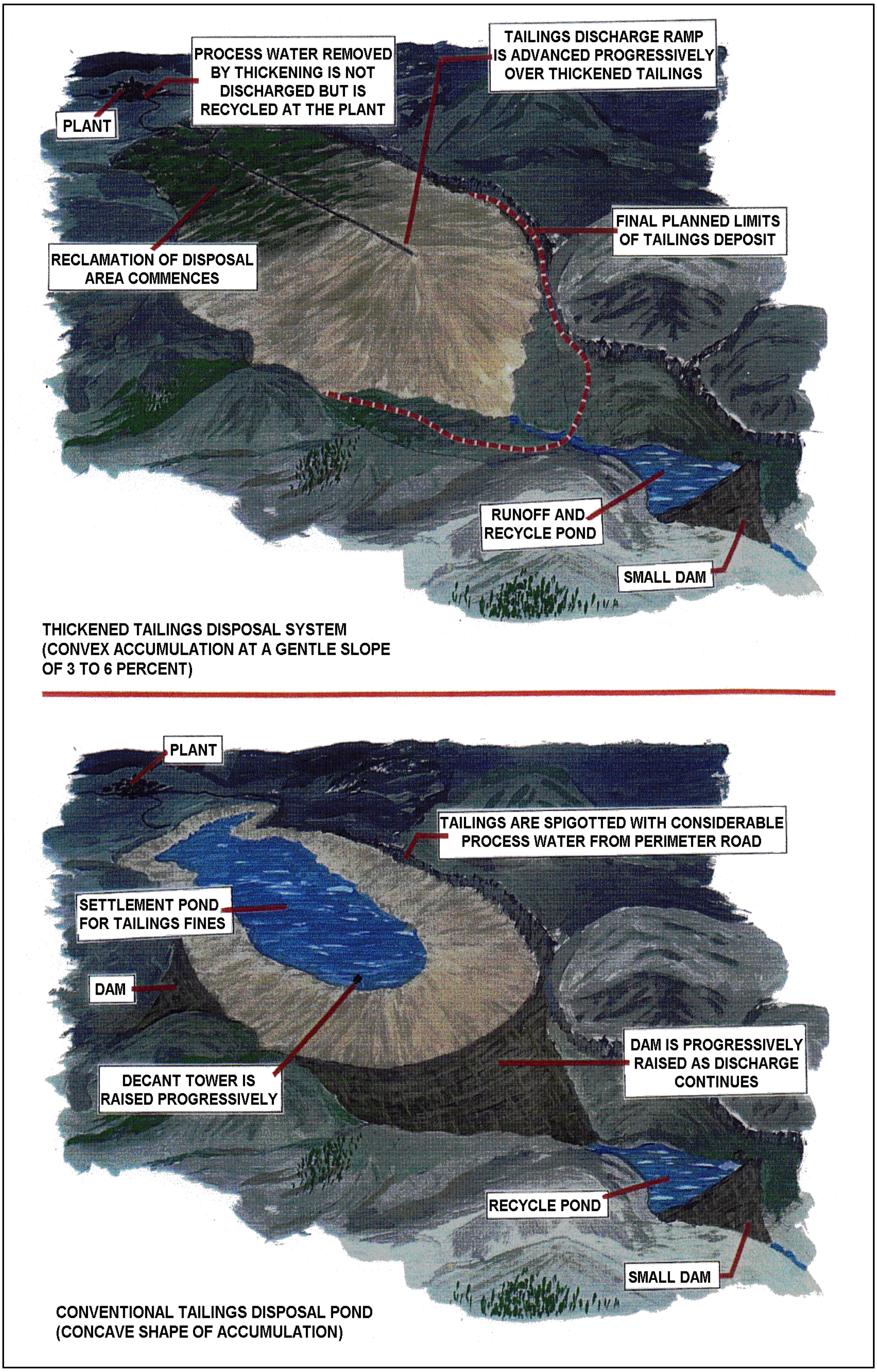

Figure 2.26: Comparison of thickened tailings system and conventional tailings pond in same geological setting
[77, Robinsky, 2000]

2.4.1.3 Tailings and waste-rock heaps

The coarse tailings in iron, potash and coal mining are often managed on heaps. Large amounts of waste-rock are managed in most metal mines using the open pit mining method.

Delivery is carried out by a conveyor belt or trucks. The heaps are surveyed to monitor for stability of the structure. Surface run-off is collected and treated, if necessary, prior to discharge or diverted into the tailings ponds. Geotechnically, the coarse tailings and waste-rock are usually stable. The coarseness of the material and truck dumping stabilise the material during deposition. The stability of the supporting strata has also to be considered in the design and operation of heaps.

Dust emissions from heaps can be quite significant. In the case of dumping from conveyor belts the operation may have to be interrupted in windy conditions. If the tailings or waste-rock are transported by trucks the transport paths may have to be sprayed in dry periods. Progressive reclamation, if possible, helps prevent erosion and dusting.

Please provide further information for this section, especially on construction methods, stability and monitoring.

2.4.1.4 Backfilling

Backfill is the reinsertion of materials in extracted part(s) of the ore body. Materials used for backfilling can be waste-rock or tailings from the mineral processing plant. In most cases backfill is used to refill mined-out areas for in order to

- assure ground stability,
- prevent or reduce underground and surface subsidence
- provide roof support so that further parts of the ore body can be extracted and to increase safety,
- provide an alternative to surface disposal.
- to improve ventilation.

Slurried tailings are sometimes used in underground mines or abandoned pits or portions of active pits as backfill. Besides the benefits for the mining operation itself (see list above), this decreases the aboveground surface disturbance. Due to the increase in volume by size reduction a maximum of about 50 % of the tonnage extracted can be backfilled. This means that in cases where the ore grade is only in the grams-per-tonne range it will not be possible to backfill all the tailings. Hence a surface TMF as well as backfilling may be necessary in these cases. However, in cases where the content of the desired

There are 4 types of mine backfill

- dry backfill
- cemented backfill
- hydraulic backfill
- paste backfill.

[94, Mining Life, 2002]

<u>Dry backfill:</u>
Dry backfill generally consists of unclassified surface sand, waste-rock, smelter slag and is transported underground by dropping down a small shaft (or raise) from the surface directly into a stope or to a level where it is hauled to a stope with loaders or trucks. Despite its name it usually contains some adsorbed surface moisture.

This type of backfill is suitable for mechanised cut and fill or other methods where structural backfill is not required.
[94, Mining Life, 2002]

Cemented backfill

Cemented backfill generally consist of waste-rock or coarse tailings mixed with a cement slurry to improve the bond strength between the rock fragments. Methods of placement involve mixing the rock and cement slurry in a hopper before placing in stopes, or percolating a slurry over the rock after it has been placed. The waste-rock or tailings can be classified or unclassified. Cemented backfill contains a mixture of coarse aggregate (<150 mm) and fine aggregate (<10 mm fraction). Cement slurry concentration is often around 55 % by weight (1.2:1 water/cement ratio).

Cemented backfill is applied for longhole open stoping, undercut and fill, and other methods where a structural fill is required.
[94, Mining Life, 2002]

Hydraulic backfill

Hydraulic backfill can consist either of classified slurried tailings or naturally occurring sand deposits mined on the surface. Hydraulic backfill is prepared by dewatering the mineral processing tailings stream to a pulp density of approximately 65 - 70 % solids and then passing it through hydrocyclones to remove the "slimes" and retain the sand fraction for backfill. Fines are removed to improve the drainage capacity of the backfill, leading to an improved stability. The backfill mixture is hydraulically pumped from the surface through a network of pipes and boreholes to the stope. Sand obtained from surface borrow pits will be screened prior to use in a backfill plant to remove oversize particles that could plug the backfill line. Hydraulic backfill can be cemented or uncemented.

Hydraulic backfill has permeability coefficients in the range of $1x10^{-7}$ m/s to $1x10^{-4}$ m/s corresponding to a medium silt to coarse sand. Hydraulic placement of backfill results in a loose fill structure with a void ratio of about 0.70.

In practice, an apparent cohesion often develops in uncemented backfill which increases the shear strength of the backfill. Often a vertical face of 3 - 4 m can be maintained under some mining conditions. Nearby blast vibrations can also act to densify the fill and increase its shear strength. To overcome the lack of true cohesion in the backfill, cement and other binders are added. Note that backfill strength decreases with water content and the water content needed to transport hydraulic backfill is far in excess of what is required for cement hydration. Hence, mine operators are moving towards less water in the fill to decrease cement and binder consumption. Flow velocities in excess of 2 m/s are required to maintain a homogeneous dispersion of the fill components in the slurry.
[94, Mining Life, 2002]

Paste Backfill

Paste backfill is a high density backfill (>70 % solids depending on the density of the solids). In order to pump material at this density, a component of fines is required. As a general rule, the fines content (<20 µm) must be a minimum of 15 % by weight.

Paste backfill is pumped by piston type pumps of the same type used to pump concrete. Whole mineral processing tailings can often be used to make paste backfill. The final product has a lower void ratio so the backfill is denser. Many mines are moving towards paste backfill because a lower cement content is necessary to gain equivalent strengths to withstand roof pressure when compared to conventional hydraulic back fill.
[94, Mining Life, 2002]

The environmental benefit of using backfill is that tailings with an acid-generating potential will remain water saturated once the mining operation has ceased and the groundwater level has risen above the backfill. However, it has to be ensured that the groundwater will not be contaminated by other constituents (e.g. reagents).

2.4.1.5 Others

2.4.1.5.1 Deep see tailings management

In mining areas where tailings are likely to generate acids deep lake, deep sea or submarine tailings management is sometimes an acceptable method. Deep see tailings management, either confined or unconfined, reduces engineering requirements, provides chemical stability and reduces the footprint on land. Often submarine tailings management is considered risky because of the inability to predict, control or rectify the spread of contaminants throughout the environment. Another concern is that too little is known about the subaqueous environment. Therefore an impact assessment is difficult to undertake.

2.4.1.5.2 River tailings management

This practice is applied for water soluble materials (e.g. salt). Some potash mines discharge the rock salt (NaCl) tailings into rivers to a level that the river water still meets drinking water limits. Non-soluble tailings are not discharged into running surface waters for the obvious reason that the settled tailings will alter the river bed.

2.4.1.5.3 Lined cells

At the **Las Cruces Project** the proposed deposition method of the tailings is dry deposition in impermeable cells. It is proposed that the cells are constructed as blocks of 100 x 100 m with a height of 25 m. The deposited tailings are proposed to be continuously covered by clay. The final encapsulation will be done using a multi-layer cover, utilising the clay (marl) extracted in the uncovering of the orebody. The cells are proposed to be constructed with an impermeable base, constructed by various layers of clay, synthetic liner and drainage layers. A system for the capturing of drainage will be installed and the drainage treated for re-use in the process or discharge.
[67, Base metals group, 2002]

2.4.2 Closure, rehabilitation and aftercare of facility

A mine together with the mineral processing plant and the tailings and waste-rock facilities is usually in operation for no more than a few decades. The mine voids (not part of the scope of this work), tailings and waste-rock however, may remain long after the cessation of the mining activity. Therefore special attention has to be given to the proper closure, rehabilitation and aftercare of these facilities.

In many cases the tailings and waste-rock do not contain any substances that are harmful to the environment. In these cases in the closure phase the operator will ensure that the water is drained from the tailings pond to ensure physical stability, the dams flattened to allow access for machinery. Ponds and heaps are then prepared for subsequent use, which in most cases means covering the ponds and/or heaps with soil and vegetating them. In some cases these facilities may be used again, e.g. in the case of potash mining the tailings heaps consist of more than 90 % salt (NaCl), which may be an economic resource in the future when purer salt deposits are depleted. In other cases the mineral processing techniques may develop in a way that more minerals can be extracted profitably. So, keeping tailings accessible for the future may be a desirable objective.

If tailings and waste-rock facilities contain substances that can be hazardous to the environment, other measures may have to be taken. These measures are aimed at stability of the tailings and waste-rock facilities whilst minimising future monitoring.

Generally, the major issues to be considered for the reclamation and closure of tailings and waste-rock management facilities include the long term

(a) physical stability of constructions
(b) chemical stability of tailings and waste-rock and
(c) land use.

The TMF areas of a mine site should be stable under extreme events such as floods, earthquakes and perpetual disruptive forces including wind and water erosion, such that they do not impose a hazard to public health and safety or the environment [12, K. Adam,].

If tailings and/or waste-rock contain sulphide minerals they may create an acid discharge. Even though acid rock drainage (ARD) is a phenomenon that may occur during operation it is the time after the closure of the facility when ARD becomes a problem. While in operation tailings impoundments are usually saturated and the voids are filled with water. Therefore chemical oxidation is limited during operation. It is at the closure phase of an operation, when usually the water level within the tailings drops and air enters the voids when pyrite oxidation can create a problem.

In the following two sections first general long-term safety issues of TMFs will be discussed before the specific issue of acid rock drainage will be investigated.

2.4.2.1 Long-term safety

2.4.2.1.1 Dams

To be elaborated from ICOLD Bulletin 103 [6, ICOLD, 1996]. Further contributions are welcome.

2.4.2.1.2 Heaps

Please provide information for this section.

2.4.2.2 Acid Rock Drainage (ARD)

The past two decades have brought widespread awareness of a naturally occurring environmental problem in mining known as "acid rock drainage" or ARD. Though difficult to reliably predict and quantify, ARD is associated with sulphide ore bodies mined for Pb, Zn, Cu, Au, and other minerals including coal. While ARD can be generated from sulphide-bearing pit walls, and underground workings, only tailings and waste-rock are considered here [13, Vick,].

The key issues that are the root of these environmental problems are:

- tailings and/or waste-rock often contain metal sulphides
- sulphides oxidise when exposed to oxygen and water
- sulphide oxidation creates an acidic metal-laden leachate
- leachate generation over long periods of time.

This section is intended to give an overview of the ARD problem. For a more complete and scientifically correct description of all relevant issues regarding ARD generation a large amount of recently published literature is available. Recently published state-of-the-art reports for research purposes, with substantial amounts of literature references included, are available free on the internet (www.mimi.kiruna.se) on: Sulphide oxidation (Herbert, 1998); Predictive modelling (Destouni et al., 1998); Prevention and control of pollution from tailings and waste-rockproducts (Elander et al., 1998); Laboratory studies of key processes (Herbert et al., 1998); Field studies and characterisation (Öhlander et al., 1998); and Biogeochemical modelling (Salmon, 1999).

The above-mentioned references are only included to give examples. A significant number of publications are the result of research initiatives that are currently being undertaken, or that have been undertaken during the last 15 - 20 years, within large research programs like MEND, Post-MEND, AFR, MiMi, MIRO, INAP, PYRAMID and ERMITE. The most active countries within the area have so far been Canada, Australia, the United States, Sweden, Norway and the UK.

Unless otherwise mentioned the following information is from [20, Eriksson, 2002].

2.4.2.2.1 ARD basics

When sulphide minerals come into contact with water and oxygen they start to oxidise. This is a slow heat generating process (kinetically controlled exothermal process) which is promoted by:

- high oxygen concentration
- high temperature
- low pH
- bacterial activity.

The overall reaction rate for a specified quantity of sulphides is also dependant on other parameters such us, for example, the type of sulphides and the particle size, which also governs the exposed surface area. When the sulphides oxidise they produce sulphate, hydrogen ions and dissolved metals [20, Eriksson, 2002].

Tailings and waste-rock consist of the different natural minerals found in the mined rock. In the unmined rock, often situated deep below the ground level, the reactive minerals are protected from oxidation. In oxygen-free environments, such as in deep groundwater, the sulphide minerals are thermodynamically stable and have low chemical solubility. Deep groundwater in mineralised areas therefore often have low metal content. However, when excavated and brought to the surface the exposure to atmospheric oxygen starts a series of bio-geo-chemical processes that may lead to production of acid mine drainage. Hence, it is not the content of metal sulphides in itself that is the main concern, but the combined effects of the metal sulphide content and the exposure to atmospheric oxygen. The effect of exposure increases with decreasing grain size and therefore increased surface area. Hence the sulphides in the finely ground tailings are more prone to oxidation [14, Höglund, 2001].

Tailings and waste-rock are normally composed of a number of minerals, of which the sulphides only constitute one part, if present at all. Therefore, if sulphide oxidation occurs in mining waste, the acid produced may be consumed by acid consuming reactions in varying degrees, depending on the acid consuming minerals available. If carbonates are present in the mining waste, pH is normally maintained as neutral, the dissolved metals precipitate and thus are not transported to the surrounding environment to any significant degree. Other acid consuming minerals are, e.g., alumino-silicates. The dissolution of alumino-silicates is kinetically controlled and can normally not maintain a neutral pH in the drainage.

The interaction between the acid producing sulphide oxidation and the acid consuming dissolution of buffering minerals determines the pH in the pore water and drainage, which in turn influences the mobility of metals. If the readily available buffering minerals are consumed, the pH may drop and ARD (Acid Rock Drainage) will then be formed.

The release of ARD to surface and groundwater deteriorates the water quality and may cause a number of impacts, such as depletion of alkalinity, acidification, bioaccumulation of metals, accumulation of metals in sediments, effects on habitats, elimination of sensitive species and unstable ecosystems.

2.4.2.2.2 Sulphide oxidation (acid generation)

Sulphide minerals extracted from the bedrock have been formed under strongly reducing conditions resulting in sulphur being present in its lowest oxidation states. The most commonly occurring sulphides are iron sulphides (pyrite FeS_2(s) and pyrrotite FeS(s)). These iron sulphides often coexist with other sulphides of higher economic value such as (chalcopyrite $FeCuS_2$(s); galena PbS (s); sphalerite ZnS(s)) or with sulphides of very little economic value such as arsenopyrite ($FeAsS_2$(s)). In unaltered bedrock the overlying overburden and groundwater minimise the contact with oxygen which almost eliminate the oxidation of the sulphides. However, when the sulphides become exposed to an oxidising and humid atmosphere, e.g., by mining activity, they start to oxidise (weather, dissolve etc). This process is commonly exemplified by pyrite (FeS_2(s)) oxidation by oxygen and water as

$$FeS_{2(s)} + 7/2O_2 + H_2O \rightarrow Fe^{2+} + 2SO_4^{2-} + 2H^+ \quad (1)$$

Sulphide oxidation, which is a slow kinetically controlled exothermal process, can also take place with other oxidants such as ferric iron, Fe^{3+} as

$$FeS_{2(s)} + 14Fe^{3+} + 8H_2O \rightarrow 15Fe^{2+} + 2SO_4^{2-} + 16H^+ \quad (2)$$

Oxidation of sulphides, mainly pyrite, and the processes that influence the oxidation rate of the sulphides have been studied in detail during the last decades. Of the various factors that influence the sulphide oxidation rate the availability of oxygen has been found to be the most important. To sustain a continuous sulphide oxidation, oxygen has to be supplied from the surrounding atmosphere. This is true for sulphide oxidation with oxygen (eq.1) as well as indirectly for sulphide oxidation with ferric iron (eq.2), since oxygen is required for the oxidation of ferrous to ferric iron according to

$$Fe^{2+} + 1/4O_2 + H^+ \rightarrow Fe^{3+} + 1/2H_2O \quad (3)$$

Ferric iron may contribute to sulphide oxidation (eq.2) or it may hydrolyse and precipitate as ferric oxyhydroxide (dominant at pH>3.5) according to

$$Fe^{3+} + 2H_2O \leftrightarrow FeOOH_{(s)} + 3H^+ \quad (4)$$

It is also stated that there are indications that cycling of iron through the ferrous and ferric oxidation states may potentially be a key process in anaerobic tailings and waste-rock management facilities. Field studies, however, indicate that the overall sulphide oxidation rate is dramatically reduced by applying oxygen diffusion barriers. Bio-geochemical modelling results calibrated to field data from a covered tailings deposit do not indicate that pyrite oxidation by ferric iron plays any significant role in the remediated deposit.

As described above, many factors have been found to influence the sulphide oxidation rate such as e.g., bacterial activity, pH, Eh (oxygen concentration), temperature and galvanic processes between different sulphides. This has to a large extent been investigated and numerical expressions (rate laws) have been developed for pyrite oxidation under various conditions. These rate laws are available in the literature. However, under natural conditions, as e.g., in a tailings or waste-rock facility, these various factors are codependant and influenced by other factors such as e.g., the available surface area for oxidation determined by the grain size distribution, mineralogy, hydrology, availability of buffering minerals etc, as will be described in the following sections.

2.4.2.2.3 Dissolution of buffering minerals (acid consumption)

If readily available buffering minerals (carbonates) are present in the tailings or waste-rock acid produced by the oxidation of sulphide minerals (eq.1 and 2) and the precipitation of iron oxyhydroxide (eq. 4) they will be consumed by the dissolution of the buffering minerals, here exemplified by the dissolution of calcite

$$CaCO_{3(s)} + 2H^+ \rightarrow Ca^{2+} + CO_{2(g)} + H_2O \quad (5)$$

Dissolution of calcite is a fast reaction in comparison to pyrite oxidation and is therefore normally assumed to be in equilibrium (i.e., the acid's consumed at the same rate as it is produced). If there are not enough readily available buffering minerals present, or if they are depleted over time, the pH in the drainage may drop and the solubility of dissolved metals will increase. This is what is normally called ARD.

Acid is also consumed by the dissolution of other buffering minerals, such as alumino-silicates, but normally at a slow rate, that cannot keep up with the acid production from sulphide weathering, as the dissolution of the alumino-silicates is kinetically controlled. Acid consumption by dissolution of alumino-silicates is exemplified below by dissolution of K-feldspar, muscovite and biotite.

$$KAlSi_3O_{8(s)} + H^+ + 9/2\ H_2O \rightarrow K^+ + 2H_4SiO_4 + 1/2Al_2Si_2O_5(OH)_{4(s)} \quad (6)$$

$$KAl_2(AlSi_3O_{10})(OH)_{2(s)} + H^+ + 3/2\ H_2O \rightarrow K^+ + 3/2Al_2Si_2O_5(OH)_{4(s)} \quad (7)$$

$$KMg_{1.5}Fe_{1.5}AlSi_3O_{10}(OH)_{2(s)} + 7H^+ + 1/2\ H_2O \rightarrow$$
$$K^+ + 1.5\ Mg^{2+} + 1.5Fe^{2+} + 2H_4SiO_4 + 1/2Al_2Si_2O_5(OH)_{4(s)} \quad (8)$$

2.4.2.2.4 Weathering at the field scale

ARD may be produced where sulphide minerals are exposed to the atmosphere (oxygen and water) and there is not enough readily-available buffering minerals present, as illustrated by equations. 1 to 5. In mining this could be in, e.g., waste-rock deposits, marginal ore deposits, temporary storage piles for ore, tailings deposits, pit walls, underground workings or heap leach piles. Historically sulphide-containing material has also been used for construction purposes at mine sites, e.g. in the construction of roads, dam constructions and industrial yards. However, regardless of where ARD production occurs, the fundamental processes behind the generation of ARD are the same..

Figure 2.27 schematically shows some of the most important geochemical and physical processes and their interaction and contribution to the generation of ARD and the possible release of heavy metals from mining waste. As can be concluded from the figure, the ARD and metal release will depend primarily on the sulphide oxidation rate, the potential immobilisation/remobilisation reactions along the flow path and the water flow. However, the sulphide oxidation rate is dependant on redox conditions (oxygen concentration), pH, and microbial activity. The pH is in turn determined by the sulphide oxidation rate and buffering reactions (carbonate dissolution and silicate weathering). Furthermore, the potentially metal retaining immobilisation reactions that can occur along the flow path are dependant on pH, redox conditions and the sulphide oxidation rate.

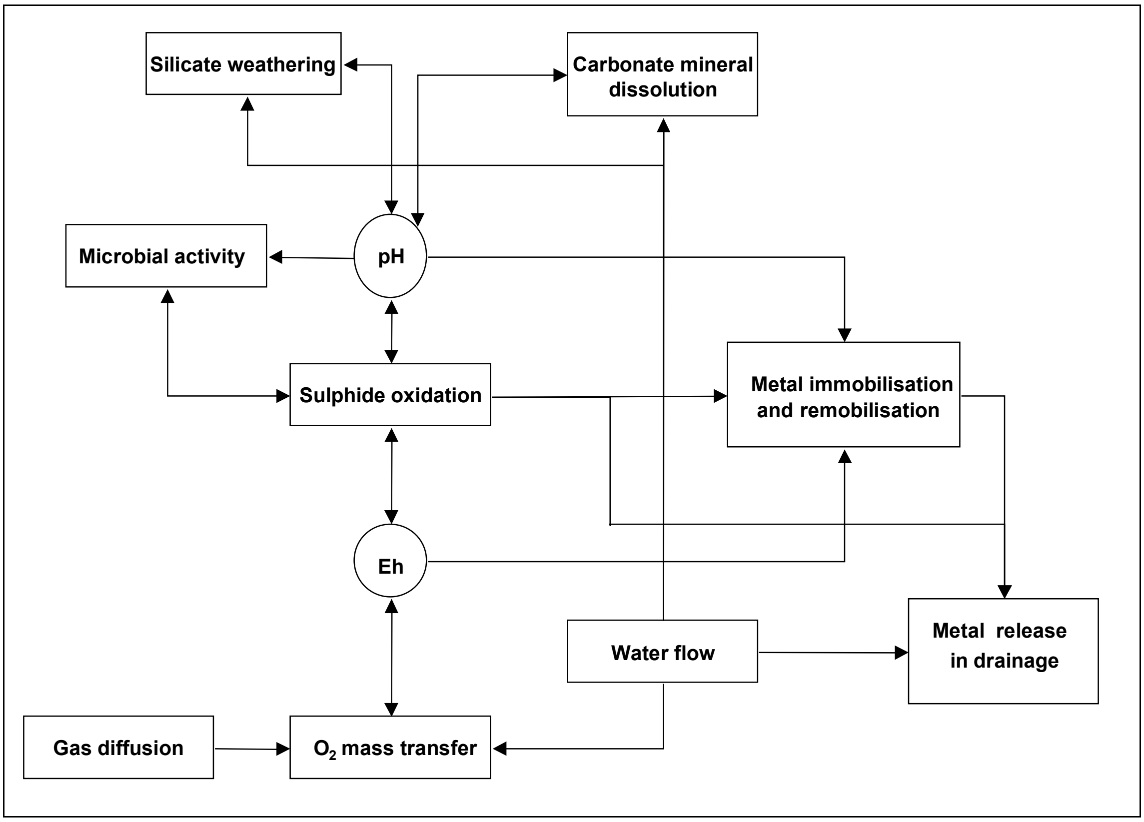

Figure 2.27: Schematic illustration of some of the most important geo-chemical and physical processes and their interaction and contribution to the possible release of heavy metals from mining waste.

At the field-scale not only are the temporary variations of material characteristics important for the evolution of the drainage water quality but the spatial variations will also be a factor to take into account. The resulting drainage characteristics depend on a number of additional parameters, such as infiltration rate, evaporation rate, oxygen profile in the deposit, height of the deposit, and the construction of the deposit. Heterogeneities in the material characteristics, such as varying mineralogy and degree of compaction, are other parameters that may affect the drainage water quality. Due to the normally long residence time of the infiltrating water in the deposit the influence of various immobilisation reactions (precipitation and adsorption) can also be significant. The interaction between the tailings and/or waste-rock and the atmosphere is illustrated schematically in the following figure.

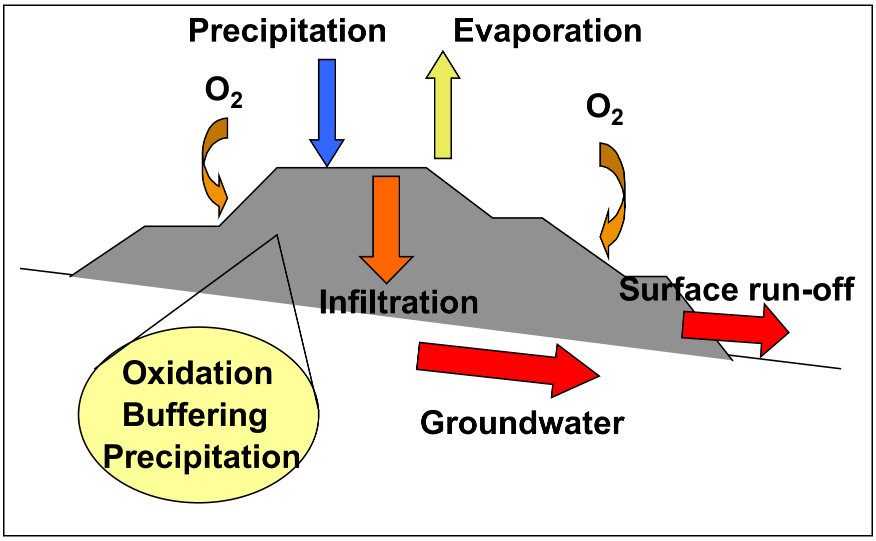

Figure 2.28: Schematic illustration of the drainage water generation as a function of the interaction between the tailings or waste-rock in the facility and the atmosphere.

2.4.2.2.5 ARD management

The management of potentially ARD generating tailings or waste-rock normally follows a risk based approach. In the risk assessment the accurate characterisation and understanding of the material is of critical importance. The management process is a cyclic process that is originally done in the planning phase of the mine but renewed and re-evaluated continuously through the mine life. The assessment process always covers the "cradle to grave" concept, i.e., any preferred option with respect to management of tailings and waste-rock during the operational phase of the operation should also include an acceptable closure strategy. Initial material characterisation is done in the planing stage of the mine, however, the initial characterisation results are continuously followed-up and confirmed by material characterisation during the operational phase of the mine.

<u>Tailings and waste-rock characterisation</u>
Characterisation of the material is important in order to avoid unacceptable environmental impacts and/or significant costs for controlling ARD. Using the characterisation as a basis for the evaluation of various management options, the right management decision can be taken and a tailings and waste-rock management plan developed whereby environmental impacts and costs can be minimised. Several guidelines for the characterisation and management of potentially ARD generating tailings and waste-rock have been developed and published, e.g.:

- British Columbia AMD Task Force, 1989, Steffen Robertson Kristen Inc., Acid Rock Drainage Technical Guide, Vol. 1
- Environment Australia,1997. Best Practice Environmental Management in Mining: Managing Sulphidic Mine Wastes and Acid Drainage. Technical report series
- Morin K.A. and Hutt N.M., 1997. Environmental Geochemistry of Minesite Drainage: Practical theory and case studies. MDAG Publishing, Vancouver, Canada.

Various characterisation methods (static, kinetic and modelling, characterisation in field) are used in the characterisation and management of potentially ARD generating mining waste. Figure 2.29 gives an example of a schematic decision tree for the management of potentially ARD generating mining waste. Several alternative methods for the risk assessment of ARD have been described.

Picture will be inserted later

Figure 2.29: Decision tree for the management of potentially ARD generating tailings and waste-rock or other mining residues

Depending on the questions to be answered and the characteristics of the material various degrees of detail are needed in the material characterisation. Some commonly used characterisation methods are listed in Table 2.6, and the complete methods are given in Annex 2. The various characterisation methods are normally divided into four groups: material analysis, static tests, kinetic tests and field investigations. Normally a selection of the various characterisation methods are used in combination with computer modelling in order to reach a good understanding of the material characteristics and for predicting drainage water qualities for various management options.

Material analysis, e.g., whole rock analysis or mineralogical analysis, gives the composition and the total content of analysed elements of the material. These analyses are necessary for long-term predictions (turn-over-time) of loads from deposited material.

Static tests, often combined with the results from the material analysis, give the total capacities of acid production and acid consumption (readily available acid consumption capacity). With a certain degree of probability, static tests can predict if the material can or will produce ARD. These tests do not give any information about how fast the various minerals in the material reacts (reaction rate). However there are some materials where this type of test cannot, with sufficient degree of accuracy, answer if the material will produce ARD or not.

Kinetic tests study how the material reacts over time. The leachate from a specified amount of tailings or waste-rock under specified conditions (temperature, oxygen concentration, infiltration rate, humidity etc) is analysed periodically from which reaction rates for the various minerals can be calculated.

Weathering rates calculated from kinetic tests can be combined with the material analysis and the turn-over-times of various minerals can be calculated. Using this information, predictions about the evolution of the drainage water quality over time can be made. Note: these results are not directly applicable to field conditions! By combining the tests and calculations the following questions can be answered with a certain degree of confidence. *Will this material produce ARD?* and if so *When?* By combining the derived information, assessments can be made on: *What is the potential load over time for the deposited material?* This is normally done using some modelling tool. The scale dependency between the laboratory scale and the field scale has to be taken into account when predicting loads and concentrations in the field.

Field studies are normally done to follow up the laboratory and modelling results. If the deposit already exists the drainage water characteristics can be directly analysed and the change in time followed. Field studies give the result of the integrated behaviour between all the influencing factors. Field studies are important management tools and important for the further development of operational and closure plans. Field studies and the feedback and comparison to laboratory tests and predictions are also important for the development of better prediction techniques.

The following table lists the above mentioned tests.

Test	Group of tests	Result
analysis	material analysis	composition
ABA fizz test	static tests	capacity
batch humidity cell column	kinetic tests	reaction rate
field monitoring	field studies	integrated behaviour

Table 2.6: Commonly used methods for characterisation of tailings and waste-rock with regard to its potential ARD production
[20, Eriksson, 2002]

2.4.2.2.6 Principal prevention and control options

There is a number of prevention and control options developed for potentially ARD generating mining waste, applicable for the operational as well as the closure phases of the mine life. Primarily three types of prevention and control measures can be distinguished (e.g., Elander et al., 1998):

- prevention of the generation of ARD
- control of contaminant migration; and
- collection and treatment of contaminated drainage.

The basis for any preventive measure is the characterisation of the tailings and waste-rock and a comprehensive management plan which identifies and minimises the amount of tailings and waste-rock that requires special attention. Many of the preventive methods focus on minimising the sulphide oxidation rate and thereby the primary mobilisation of weathering products. This can be accomplished through minimising the oxygen transport to the sulphides through applying an oxygen transport barrier (cover). The covers are normally variations on two basic concepts: “water covers” (flooding) or “dry covers”. A third type, “oxygen consuming” covers have also been developed and applied. Other preventive methods aim at removing sulphide minerals from the tailings or waste-rock (depyritisation), adding buffering minerals, minimising the bacterial activity or minimising the mineral surface area available for weathering. Oxidation of the sulphide minerals can be minimised during operation by e.g., underwater management of tailings.

Prevention and control method	Used principle
Water cover and underwater (sub-aqueous) discharge	Uses a free water cover as an oxygen diffusion barrier. Oxygen diffusion is 10^4 times less in water than in air.
Dry cover	Uses a low permeable layer with high water content as an oxygen diffusion barrier.
Oxygen consuming cover	Uses a low permeable layer with high water content as an oxygen diffusion barrier. In addition, the low permeable layer has a high content of organic matter which, when it degrades consumes oxygen and thereby further reduces oxygen transport to the underlying sulphides.
Raised groundwater level	Maintains the underlying sulphide material constantly below the groundwater table by retaining water by increased infiltration, reduced evaporation, increased flow resistance and capillary forces.
Blending	Adds tailings and waste-rockwith high buffering capacity to potentially ARD producing material thereby pH can be maintained at a neutral level.
Addition of buffering minerals (liming)	Adds buffering capacity to potentially ARD producing material thereby pH can be maintained at a neutral level.
Selective material handling	Selective management of various tailings or waste-rock fractions determined by their composition and properties, e.g. separation of material with ARD generating potential for separate handling.
Depyritisation	Separation of pyrite from the tailings and separate discharge of the pyrite (e.g. under water)

Table 2.7: Examples of ARD prevention methods and the principle on which their function is based

Reference sites are discussed in Section 3.1.5.

When the weathering reactions cannot be prevented (e.g., might be the case during the operational stage of the mine life), the migration of ARD needs to be controlled. Methods that focus on minimising the transport of weathering products from the deposit into the environment include, e.g., selective placement of ARD generating waste, diversion of unaffected surface water, collection of affected surface water, control of groundwater flow, prevention/minimisation of infiltration into the deposit often obtained by simple covers.

During the operational phase of a mine or where the minimisation of the sulphide oxidation rate is not readily obtainable, it might become necessary to collect and treat the drainage before it reaches the environment. This treatment could be done either through passive treatment (e.g., wetlands or anoxic limestone drains) or through active treatment in a water treatment plant (straight liming, HDS-process, etc). At closure, it may be necessary to treat the drainage even after a cover has been put in place, until the impact of releasing the resulting drainage to the environment can be regarded as acceptable.

The three principal prevention and control methodologies mentioned are listed in order of preference. If the reactions releasing the contaminants are prevented there is no risk of the contaminants entering the environment. Where the reactions cannot be prevented, control of contaminant migration should be implemented. If neither of these control measures result in a sufficient control of the ARD generation, it is necessary to collect and treat the drainage. Clearly, the least risk of environmental impact exists if the contaminants are never mobilised by sulphide weathering. The greatest risk results when there is a continuous and long-term need to operate and maintain collection and treatment systems. Often a combination of a number of the above-mentioned methods are applied to achieve an acceptable remedial result. The best closure results are obtained if the site is planned for closure right at the outset of the operation (cradle to grave philosophy.

Various guidelines for mine closure planning have recently been developed (e.g., MIRO, 1999. A technical framework for mine closure planning. Mineral Industry Research Organisation. Technical Review Series No. 20.). The following figure presents one of the decision trees available in the literature for closure design of a potentially ARD generating tailings and waste-rockdeposit.

Picture will be inserted later

Figure 2.30: Example of decision tree for closure of a potentially ARD generating tailings and waste-rockdeposit
[20, Eriksson, 2002]

Apart from their chemical and biological complexity, another feature of these reactions is their longevity. This makes ARD problems even more difficult than those associated with cyanide, which is chemically unstable and does not persist in the environment. Mines worked by the Romans in Spain and the Vikings in Scandinavia continue to generate ARD today, and once exposed to oxygen ARD reactions on mineral surfaces will continue for thousands of years. Even sulphide-bearing tailings isolated from oxygen will remain available for initiation of these reactions indefinitely should they be exposed at some future time.

For existing tailings deposits and other sources already generating ARD, there is little to be done except damage control. Here, the low-pH effluent needs to be collected and treated by lime addition to precipitate and remove the metals. But not only is the cost of doing so over essentially indefinite periods of time objectionable, the ever-growing volumes of precipitate sludge might eventually approach even that of the tailings, introducing further concerns for its own long-term disposal, chemical stability, and future metal release [13, Vick,].

2.4.2.2.7 Dry closure

"Dry" closure is a conventional cap-and-cover solution common to other waste materials. Following termination of mining and cessation of active tailings deposition, ponded water is removed from the surface of the tailings deposit and the surface allowed to dry, although much of the finer-grained tailings remain soft and saturated. A low-permeability cover is then constructed over the surface and graded to enhance run-off, sometimes incorporating pervious layers for drainage, monitoring, or capillary breaks. In principle, such a cover achieves two purposes, (1) it restricts oxygen from the surface tailings and oxygen diffusion into void spaces, reducing reaction rates and therefore ARD generation(2) thereby, the cover acts to prevent ponding and reduce infiltration of surface water, thereby restricting transport of reaction products. In practice, however, for a variety of reasons these effects can be hard to assure and may be only partially realised. Moreover, suitable cover materials may not be locally available, and the cost and difficulty of earthwork operations on the soft tailings surface can be considerable [13, Vick,].

2.4.2.2.8 Wet closure

Permanent submergence provides an alternative closure strategy. Elegant in simplicity and minimal in cost, the entire surface of the tailings deposit is flooded with water and maintained in this condition indefinitely. The tailings voids remain permanently water-filled and thus unavailable to oxygen, while a water depth of two to three metres sufficiently reduces dissolved oxygen at the submerged tailings surface. Both laboratory and field demonstrations, notably from the Canadian MEND programme, have shown submergence to be virtually 100 percent effective from a geochemical point of view: the water "smothers" the oxidation reactions in the same way as when sprayed on a fire. Thus, permanent submergence has come to be viewed by some as the preferred closure strategy for tailings ARD control [13, Vick,]

2.4.2.2.9 "Wet" vs "dry" cover

A "wet" cover is not possible if the catchment area is too small to guarantee a permanent water surface covering the area. In this case a "dry" till cover must be arranged in order to reduce infiltration and diffusion and to prevent water and oxygen to reach the tailings. A general method to design a cover is to arrange a number of layers consisting of different soil types as clay, silt, sand and gravel. How effective the cover is depends on the content of moisture in the covering layers. The total thickness of the cover layers normally range between 0.3 - 3.0 m and the permeability for the sealing cover ranges between 1 x 10^{-7} – 1 x 10^{-9} m/s.

Before the tailings pond can be covered it has to be dewatered so the sand can consolidate. The consolidation can take a long time depending on the properties of the sand. Consequently it might be necessary to apply a dust control cover on the tailings to prevent dusting during the consolidation phase. To prevent gathering of water it might be necessary to construct by-pass ditches and reshape the surface of the pond. Ideally the surface should decline 0.5 - 1.0 % towards the edges of the pond.
[66, Base metals group, 2002]

2.5 Techniques to evaluate and reduce environmental impact of tailings and waste-rock management

2.5.1 Baseline Studies

These studies have become a standard procedure and are often a legal requirement. They are necessary as a point of reference in order to quantify the impact of an operation.

A baseline study is usually established as part of the Environmental Impact Assessment. This study is necessary to inform all parties involved in the permitting process on what the conditions are before a new site goes into operation.

This baseline investigation identifies the range of resources potentially at risk from a site and shows data describing these resources. It therefore provides measures from which the environmental impacts of a proposed development can be predicted and a database against which future changes in environmental quality can be judged [25, Lisheen, 1995]. A well performed baseline study also provides valuable data for the further design, layout and planning of the site.

Typical aspects to be included in a baseline study are:

- archaeology and local history
- socio-economic features
- health
- infrastructure
- traffic
- climatology
- air quality
- geology
- landscape
- ecology

- noise
- vibrations
- soils and soil suitability
- soil and herbage sampling
- crop and animal production
- soil moisture
- veterinary
- mineral status of garden soils and vegetables
- hydrogeology
- groundwater quality
- surface water quality
- surface water hydrology
- fisheries, fish population and spawning
- surface water flora and macroinvertebrate fauna.

[25, Lisheen, 1995]

2.5.2 Typical emissions

Tailings and waste-rock facilities can have an environmental impact via emissions to

- air
- water
- land.

Emissions to air can be dust, odour and noise. Usually the latter two are of less concern unless the tailings or waste-rock are transported with trucks and there is housing nearby. Dust can consist of materials such as quartz or any other components found in rocks and minerals including metals.

Emissions to water can include

- reagents from mineral processing, such as
    - cyanide
    - xanthates
    - acids or bases resulting in high or low pH
    - solid or dissolved metals or metalliferous compounds
    - *… Please provide further information*

Emissions to land can occur via settled dust or via percolation of liquids from tailings and /or waste-rock management facilities into the ground.

2.5.3 Environmental impact of emissions

Effluents and dust emitted from tailings and waste-rock management facility, controlled or uncontrolled, can be toxic in varying degrees to man, animal and plants. The effluents can be highly acidic or alkaline, can contain dissolved metals, soluble and entrained insoluble complex organic constituents from mineral processing, but also natural occurring organic substances such as humic and long-chain carboxylic acids from the mining operations. The substances in the emissions, together with pH, dissolved oxygen, temperature and hardness can be important aspects in the toxicity to the receiving environment.

Certain reagents, such as cyanides, frothers and xanthates require retention time, oxidation (air, bacteria, sunlight) and, in the case of xanthates temperatures above 30 ºC to decompose. Therefore the planning of the mineral processing circuit and the TMF must consider the

environmental impacts of these substances and the necessity for extra ponding or treatment to provide for certain reagent decomposition.
[21, Ritcey, 1989]

The environmental impact of emissions to watercourses always depend on many factors such as concentration, pH, temperature, water hardness etc. However, Ritcey [21, Ritcey, 1989] and many other sources provide tables listing, e.g.

- maximum and minimum pH levels for various aquatic life form
- ammonia toxicity data
- acute toxicity data for various flotation agents
- toxicity of specific chemicals
- toxicity data for flocculent and coagulants.

These tables can give an impression of the potential impact of certain reagents, but, as mentioned above the whole picture has to be taken into consideration.

The following table shows effects on humans, animals and plants of some metals. Values applicable to water intended for human consumption as set by the Drinking Water Directive, Council Directive 98/83/EC of 3 November 1998 on the quality of water intended for human consumption[4], are included to which e.g. a tailings pond effluent may be compared. This is not to say that the effluent may be used for that purpose, but these values are useful as a baseline.

Metal	Drinking water threshold level	Effect
Arsenic (As)	10 µg/l	Highly poisonous and possibly carcinogenic in humans. Arsenic poisoning can range from chronic to severe and may be cumulative and lethal
Cadmium (Cd)	5.0 µg/l	Cadmium is concentrated in tissue and humans can be poisoned by contaminated food, especially fish. Cd may be linked to renal arterial hypertension and can cause violent nausea. Cd accumulates in liver and kidney tissue. It depresses growth of some crops and is accumulated in plant tissue
Chromium (Cr)	50 µg/l	Cr^{+6} is toxic to humans and can induce skin sensitisations. Human tolerance of Cr^{+3} has not been determined
Lead (Pb)	10 µg/l	A cumulative body poison in humans and live-stock. Humans may suffer acute or chronic toxicity. Young children are especially susceptible
Mercury (Hg)	1.0 µg/l	Hg is biologically magnified accumulating in the brain, liver and kidneys of animals. Hg poisoning may be acute or chronic
Copper (Cu)	2.0 µg/l	Small amounts considered non-toxic and necessary for human metabolism. Large doses may induce vomiting or liver damage. Toxic to fish and aquatic life at low levels
Iron (Fe)	-	Essentially non-toxic but causes taste problems in water
Manganese (Mn)	-	Affects water taste and may stain laundry. Toxic to animals at high concentrations
Zinc (Zn)	-	Zinc is necessary and beneficial, but may affect water taste at high levels. Toxic to some plants and fish

Table 2.8: Effects of some metals on humans, animals and plants and limit values for drinking water
[53, Vick, 1990] and Council Directive 98/83/EC

4 OJ N° L 330 of 5 October 1998.

2.5.4 Techniques to reduce impact

In general it can be said that mining companies have become aware that safe and environmental sound management of tailings and waste-rock are an essential part of a mining operation. As a worst case the collapse of a tailings dam can result in bankruptcy of a mining company.

The best way of managing emissions it to avoid them. E.g. to avoid dusting from tailings ponds the surface is usually kept wet or heaps are progressively revegetated with the added benefit that erosion is reduced.

Efforts are also being made to reduce the amount of reagents added. This provide both economical and environmental benefits. In many cases the ore feed is constantly monitored for its chemical composition which automatically adjust the reagent addition to optimum values.

Selective management of waste-rock is a technique that has been developed in recent years. After investigation of the acid generation potential of the waste-rock the acid generating and the non-acid generating waste-rock can be managed accordingly. Selective management of pyrite containing tailings is also under investigation.

Tailings dams used to be constructed of the course tailings fraction. This is can still be a very appropriate way of retaining the tailings slurry. However, the qualities of the ore can change and the processing method can change and therefore the characteristics of the tailings may change. Hence quality management is a tricky issue over the entire life span of an operation. Therefore there is a trend to construct the initial starter dam, but often also the raises with borrow material, whose quality can be more easily monitored during the construction of the dam.

Backfilling of tailings underground is a way of reducing the footprint of the TMF on the surface. An added benefit is that because of the extra roof support more ore can be extracted from an underground mine. It is, however, important to carefully analyse whether backfill is a suitable option, since this practice can lead to ground water contamination.

There are two basic approaches practised in the management of seepage into the ground. The first approach is trying to completely seal the ground by using liners of clay or a synthetic membrane or a combination of both. An alternative method is to allow seepage into the ground and measuring the groundwater quality and, if necessary lifting and treating the water. Liners are becoming more popular. However, critics quote the “bathtub effect” meaning that the liners holds back the liquids for a certain amount of time but will then overflow at a certain point.

The following sub-sections discuss techniques to treat acidic and alkaline effluents as well as some techniques to manage some often discussed mining-typical components in effluents, namely cyanide, xanthate and arsenic.

2.5.4.1 Effluent treatment

2.5.4.1.1 Acid waters

Water treatment methods used to eliminate or reduce acidity and heavy metals precipitation from impacted waters can be grouped into two types: active and passive treatment.

Active treatment involves neutralising acid-polluted waters with alkaline chemicals. However, the chemicals can be expensive and the treatment facility is expensive to construct and operate. **Passive** treatment involves the construction of a treatment system that employs naturally occurring chemical and biological reactions that aid acid rock drainage treatment and require little maintenance. Passive control measures include anoxic drains, limestone rock channels, alkaline recharge of groundwater, and diversion of drainage through man-made wetlands or other settling structures.

Active Treatment - Chemicals

- Limestone (calcium carbonate)
  Advantages of using limestone include low cost, ease of use, and formation of a dense, easily handled, sludge. Disadvantages include slow reaction time, loss in efficiency of the system because of coating of the limestone particles with iron precipitates, difficulty intreating ARD with a high ferrous-ferric ratio, and ineffectiveness in removing manganese. A typical flowsheet of an acid water treatment plant is show in Figure 2.31.

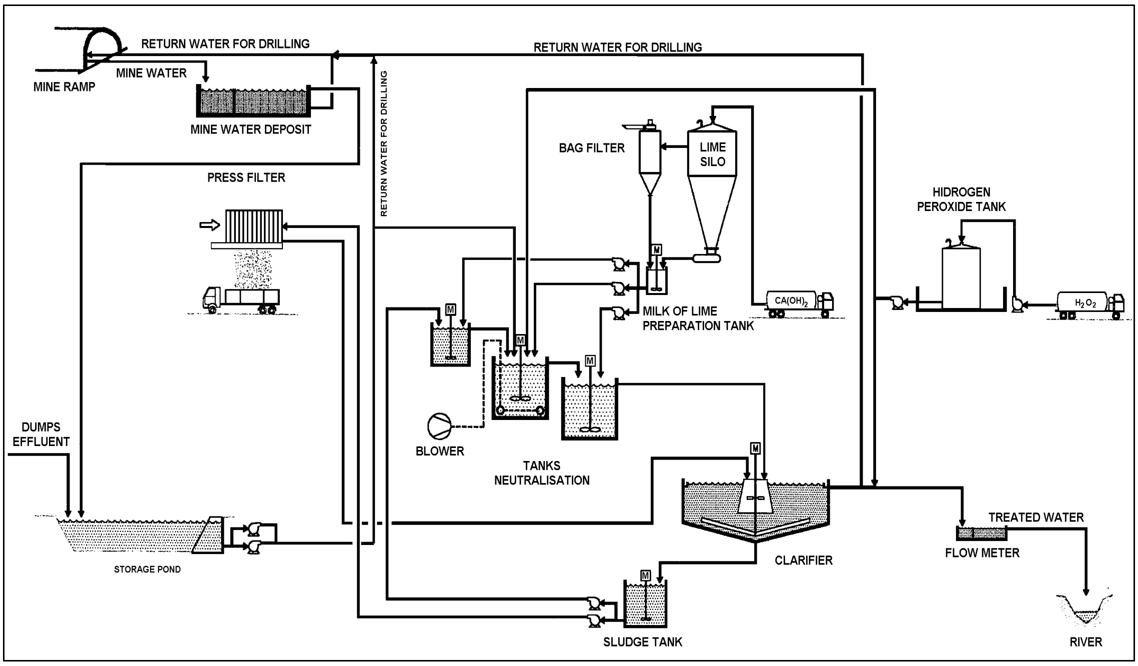

Figure 2.31: Typical flowsheet of a water treatment plant for low pH process water (from Almagrera)

- Hydrated lime (calcium hydroxide)
  Hydrated lime is normally the neutralising agent of choice by the coal mining industry because it is easy and safe to use, effective, and relatively inexpensive. The major disadvantages are the voluminous sludge that is produced (when compared to limestone) and high initial costs incurred because of the size of the treatment plant.

- Soda ash (sodium carbonate)
  Soda ash briquettes are especially effective for treating small ARD flows in remote areas. Major disadvantages are higher reagent cost (relative to limestone) and poor settling properties of the sludge.

- Caustic soda (sodium hydroxide)
  Caustic soda is especially effective for treating low flows in remote locations and for treating ARD having a high manganese content. Major disadvantages are its high cost, dangers involved with handling the chemical, poor sludge properties, and freezing problems in cold weather.

- Ammonia
  Anhydrous ammonia is effective in treating ARD having a high ferrous iron and/or manganese content. Ammonia costs less than caustic soda and has many of the same advantages. However, ammonia is difficult and dangerous to use, and can affect biological conditions downstream from the mining operation. The possible off-site impacts are toxicity to fish and other aquatic life forms, eutrophication and nitrification. Fish species generally exhibit low tolerance to unionised ammonia and toxicity levels can be affected by pH, temperature, dissolved oxygen and other factors. Ammonia use is not allowed in all areas and, where permitted, additional monitoring is required.

Passive treatment

- Constructed wetlands
  Constructed wetlands utilise soil- and water-borne microbes associated with wetland plants to remove dissolved metals from rock drainage. Initial design and construction costs may be significant, ranging into tens of thousands of dollars. Unlike chemical treatment, however, wetlands are passive systems requiring little or no continuing maintenance. This is a relatively new treatment method with many specific mechanisms and maintenance requirements not yet fully understood. Optimum sizing and configuration criteria are still under study. Seasonal variations in metal removal efficiency have been noted, with lesser amounts removed in cold weather. Wetlands are generally more effective in removing iron than manganese. The greatest utility of wetlands appears to be in the treatment of small flows in the order of tens of litres per minute.

- Open limestone channels/anoxic limestone drains
  Most simply constructed passive treatment method. Open ditches filled with limestone (anoxic drains are covered). Dissolution of limestone adds alkalinity and raises pH Coating of limestone, by iron and aluminum precipitates, affect the performance of this treatment method.

- Diversion wells
  Acidic water is diverted to a "receptacle" or "well" containing crushed limestone. Iron precipitate coating is prevented by turbulence of the flow through the well. Needs periodic replenishment of limestone.

[85, EPA, 2002]

2.5.4.1.2 Alkaline waters

Please provide information

2.5.4.2 Cyanide

Cyanide rapidly decomposes in the sun. Hence in on a worldwide level the cyanide discharged into the tailings pond is not treated. In Europe, however, it is common to destroy cyanide prior to discharge into the pond. Other methods of recycling cyanide are investigated at the moment. Methods for treating cyanide are discussed in more detail in Section 3.2.3.7.3.

2.5.4.3 Xanthates

“Xanthate” is commonly used as a collective name for a number of commercially available organic compounds such as: K-ethylxanthate; Na-isopropylxanthate; Na-isobutylxanthate; and K-amylxanthate. Xanthates are commonly used in the flotation of sulphide ores as a collector. Xanthates are organic compounds that are naturally degraded in the environment. Degradation is faster at lower pH. The half-life of sodium ethyl xanthate at pH 7 at 25 °C is reportedly about 260 hours, increasing to over 500 hours in the pH range 8 to 11 [88, NICNAS, 2002].
A typical xanthate concentration in the flotation of sulphide ores is about 50 - 500 g/tonne of ore.

The potential environmental effect of xanthates has been studied with regard to various aspects over the years. The identified potential effects are:

- acute toxicity – the LC_{50} for bacteria (microtox technique) and algae (phytoplankton: monoraphidium) ranges from 25 - 650 µg/l (Bertills et al, 1988). The acute toxicity on fish (Atlantic salmon: salmo salar) is considerably lower with LC_{50} in the range of 11000 to 65000 µg/l
- increased metal toxicity when simultaneously exposed to xanthates (both algae and fish). An increased toxicity for Pb and Cd has been observed, however, the toxicity of Cu is reduced considerably by xanthates
- increased accumulation of mainly Pb in fish liver.

[87, Bertills, 1998]

Xanthates in the tailings are not monitored, but most would be expected to be retained in the concentrate froth. Xanthate residues in the ore concentrate are expected to decompose during drying or smelting [88, NICNAS, 2002].

Assuming a treatment rate of 500 g sodium ethyl xanthate per tonne of feed, 1 % loss to tailings and 30 % solids content in the tailings slurry, the concentration of sodium ethyl xanthate in the slurry will be in the order of 5 g in 3.3 tonnes, or about 1.5 ppm. These predictions are consistent with measured values in the range of 0.2 to 1.2 mg/l. Concentrations of sodium ethyl xanthate likely to be found in the tailings slurry may be toxic to aquatic fauna. Such tailings streams should therefore not be discharged to waterways. In well managed mining operations, tailings from mineral processing are excluded from waterways through retention in tailings dams, where any xanthates that they may contain decompose. Tailings in themselves can have severe detrimental impact on stream ecology. When suitable precautions are taken to avoid entry of tailings to waterways, the environmental risk of sodium ethyl xanthate can be described as minimal in view of the low environmental exposure and limited persistence.
[88, NICNAS, 2002]

2.5.4.4 Arsenic

Trace metals are effectively removed from mining effluent by the addition of ferric salts. Through precipitation, arsenic is removed as either calcium or ferric arsenate. Arsenites can also be precipitated, but they are generally more soluble and less stable then arsenates. Arsenite-containing effluent is generally oxidised prior to precipitation to ensure that the arsenate predominates. Process water from the processing of arsenic bearing ores may contain varying amounts of arsenic (III) and (V) oxyanions, arsenites and arsenate. The presence of such metal ions as copper, lead, nickel, and zinc limit the solubility of arsenic because of the formation of sparingly soluble metal arsenates.

The stability and solubility of these arsenates depends on the ratio of iron to arsenic. The larger the ratio, the more insoluble and stable the precipitate. Thus, where ferric arsenate is relatively soluble, the basic arsenates with an iron-to-arsenic molar ratio of eight or more are orders of magnitude less soluble in the pH range of approximately 2 to 8. Dissolved arsenic concentrations of 0.5 mg/l or less can be obtained by precipitation with ferric iron.

The precipitation of insoluble ferric arsenates is very likely accompanied by co-precipitation of other metals such as selenium; that involves interactions between the various metals species and the ferric hydroxide precipitate. This makes ferric a very effective scavenger for the removal of trace contaminants. Thus, arsenic and many other elements such as antimony and molybdenum can be reduced to levels of less than 0.5 mg/l by contact with ferric hydroxide. The process normally involves the addition of a soluble ferric salt to the process water followed by the addition of sufficient base to induce the formation of insoluble ferric hydroxide. In many situations, the process water contains adequate iron, thus only the addition of a base is required to induce the precipitation of ferric hydroxide.
[78, Ron Tenny, 2001]

3 APPLIED PROCESSES AND TECHNIQUES

Chapter 3 can be subdivided in two parts. Part one (Section 3.1) describes international practice. The information used comes mostly from international guidelines and from a "management framework for mine waste" developed by Euromines [45, Euromines, 2002].

The other part comprises Sections 3.2 onwards where current practice in Europe is described. Metals, industrial minerals, potash and coal are discussed in different sections, as was decided by the TWG at the kick-off meeting. Within each of these sections the

- mineralogy and mining techniques
- mineral processing
- tailings management
- waste-rock management
- current emissions and consumption levels
- costs.

are described for all the commodities (e.g. iron, base metals, aluminium, etc. in the "metals" section). Another option would have been to handle each commodity separately. However with the current approach it is easier to compare e.g. the costs for tailings and waste-rock management for all industrial minerals.

3.1 Current international practice

A lot of work has been done to ensure safe management of tailings and waste-rock. Countries with major mining industries (e.g. Australia, USA, Canada) have been engaged in ensuring good practice in this area. Many companies operating in Europe are "global players" and the way they manage their tailings and waste-rock is based on their worldwide experience.

The purpose of this section is to describe current practice on an international scale. This is useful in order to be able to the techniques applied in Europe.

3.1.1 General principles

If the total operation (mine, mineral processing plant, tailings and waste-rock management facilities) is designed concurrent with the tailings and waste-rock, taking into consideration the various chemical, physical and biological interactions due to the influence of the mining and the processing, then the tailings and waste-rock management environmental problems and costs can be reduced [21, Ritcey, 1989]. Also, the management of tailings and waste-rock, including water management, is usually an integral part of the entire life-cycle of on operation, as fundamental as the extraction itself [45, Euromines, 2002].

Typically **risk assessment** is applied in order to ensure the use of techniques, which are most appropriate to the specific circumstances in environmental, safety, technical and engineering terms [45, Euromines, 2002].

Good management of tailings and waste-rock includes evaluation of alternatives for

- Minimising the volume of tailings and waste-rock generated in the first place
    - choice of mining method (open pit/underground, different underground mining methods)
    - …

- Maximising opportunities for alternative use of tailings and waste-rock
    - use as aggregate or for the restoration of other mine sites
    - backfilling
    - …
- Conditioning of tailings and waste-rock within the process to minimise any environmental or safety hazard
    - de-pyritisation
    - addition of buffering material
    - …

The tailings and waste-rock that cannot be avoided (due to accessibility to the ore body, safety reasons etc.) and that are not suitable for alternative use (due to physical and chemical properties, transport costs, lack of market) require a suitable management strategy, which aims at assuring.

- safe, stable and effective management of tailings and waste-rock with minimised risk for accidental discharges into the environment in the short, medium and long term
- minimisation of quantity and toxicity of any contaminated release/seepage from the management facility
- progressive reduction of risk with time.

If more than one type of tailings and waste-rock are generated it is desirable to deposit them according to type. This facilitates any future recovery of deposited materials for alternative use or re-processing and allows for the appropriate closure decisions to be made with respect to tailings and waste-rock type [45, Euromines, 2002].

3.1.2 Design phase

In this section examples are listed for considerations to be made in the design stage of a TMF. Unless otherwise mentioned, this information is taken from the “Canadian guide to the management of tailings facilities” [18, Canada, 1998] and oral contributions from TWG members.

3.1.2.1 Environmental baseline

The following is a summary of considerations for collecting and collating environmental baseline information for use in site selection, design and operation. This same baseline information is important for the development of closure plans and environmental monitoring programs. More comprehensive lists may be found in specific environmental assessment guidelines.

- Existing resources and use

Existing resources and land uses within the tailings facility area and within the greater potential impact area are identified.

- Land and water use:
    - current and historical uses, including recreation, parks, aboriginal traditional use and land claims, human habitation, drinking water sources, archaeological considerations, mining, logging, farming, hunting and fishing.
- Land Tenure:
    - establishment of the right to acquire the necessary land for a TMF
    - identification of land ownership and mineral rights, which may include mining claims; land-use permits; easements, including those for power lines and transportation corridors; state-owned land; and aboriginal land claims.

- Baseline scientific data

Compilation of baseline environmental scientific data relevant to the tailings project area.

  - Physical
    - climate (e.g. temperature, wind, precipitation, evaporation, return period floods, precipitation and runoff, air quality)
    - water (e.g. hydrology, watershed delineation and flow patterns, stream flow, lake bathymetry, hydrogeology (groundwater) characteristics, surface water and sediment quality
    - land forms
    - geology and geochemistry (e.g. surficial deposits (type, location, density, permeability), stratigraphy, geomorphology, mineral and petroleum resources, background elemental content)
    - topography (e.g. regional and detailed topographic maps, stereo aerial photography, satellite imagery)
    - soils (e.g. soils sampling and characterisation)
    - natural hazards (landslides, avalanches, seismic events, flood potential, frost action.

  - Biological
    - ecosystem identification
    - terrestrial survey (e.g. flora, natural pastures, fauna, endangered and threatened species, migratory species)
    - aquatic survey (benthos, macro-invertebrates, fish, aquatic plants).

- Baseline socio-economic data

Compilation of baseline socio-economic data relevant to the tailings project area, including historical background, population, regional economy (e.g. health, education, culture, demography). Identification of socio-economic issues which might arise for the tailings project.

3.1.2.2 Contents of a baseline study

The following sub-sections provide an example of a baseline study for a tailings pond recently performed in Europe [25, Lisheen, 1995].

3.1.2.2.1 Archaeology and local history

This section of the baseline study investigates whether archaeological findings can be expected based on historical information. It gives answers to the questions whether any important findings may be inhibited or even promoted by a new development. From the perspective of the operator repeated archaeological findings can significantly slow down the development of a site. The public may be concerned about destruction of archaeologically significant sites.

3.1.2.2.2 Socio-economic

The level of employment is considered and trends for the future are discussed briefly. Major sources of employment are listed. Hence predictions can be made of the prosperity of the investigated area for the future.

3.1.2.2.3 Health

The typical lifestyle (e.g. eating habits) in the region is examined, mortality rates are listed and compared to “average” conditions (e.g. country/world average) and the possible reasons for abnormal findings discussed.

3.1.2.2.4 Infrastructure

This section describes the road, railway, shipping and airway situation. Furthermore the access to water and electricity is described. This section may also mention the waste collection in the area.

3.1.2.2.5 Traffic

Local traffic situation is quantified. Traffic flow compared to other areas or country average may be investigated.

3.1.2.2.6 Climatology

Data like annual rainfall, prevailing winds (strength and predominant direction), humidity and air and soil temperatures are presented. If useful these figures may be compared to other regions.

3.1.2.2.7 Air quality

The results of a baseline sampling programme are presented here. The levels measured are shown and the origins determined. The values measured include total dust and metals.

3.1.2.2.8 Geology

This part describes the geology of the mineral deposit and the nearby area. It usually includes:

- deposit depth
- strata dip
- mineral assemblage
- dimensions of the deposit
- mineable resources
- description of topsoil, overburden, bed and waste-rock.

3.1.2.2.9 Landscape

The countryside of the study area is described here. Will the site be in the mountains or on flat pastureland? Are there many trees and/or hedges? The visual impact of the new development may be mentioned in this context.

3.1.2.2.10 Ecology

This section describes, e.g.:

- the soil of the area
- woodlands of interest
- species in the habitats studied
- diversity of herbs and woods
- plant species
- diversity of birds and mammals.

3.1.2.2.11 Noise

Day and night time noise levels, measured for the study are often listed as 12 hour averages.

3.1.2.2.12 Soils and soil suitability

The overall quality of soils must be investigated in the area possibly effected by the development and compared to other areas. The field survey includes soil characteristics, quality and suitability for grassland and crop and stock production.

3.1.2.2.13 Soil and herbage sampling

This section investigates the soil fertility status of the area. It includes the measurements of metal (i.e. magnesium, copper, molybenum, manganese, cobalt, zinc, lead, cadmium) and other elements such as phosphorus, nitrogen, potassium, calcium, sulphur, iodine, selenium. These values are compared to other areas and anomalies are analysed. Special attention is dedicated to establish baseline levels of any constituents that may be altered by a future mining operation.

3.1.2.2.14 Crop and animal production

Surrounding farms are examined for the productivity of their field crop and grassland. Their types numbers and conditions of livestock are examined at the same time.

3.1.2.2.15 Soil moisture

The purpose of this part of the study is to address the concern that dewatering of a mine may adversely affect the growth of crops and other aboveground vegetation including scrubs and trees. To achieve this a survey on the movement of water in soils and the possible relationship between the depth of the watertable and the soil water balance may be carried out.

3.1.2.2.16 Veterinary

Within an appropriate survey area herds are surveyed for metals and other important chemical elements in blood silage and milk. Also a 12 month animal health survey may be part of this section.

3.1.2.2.17 Hydrogeology

All factors influencing groundwater flow should be mentioned here, including aquifer/aquitard systems, faults and fault zones as well as any other geological features influencing groundwater flow. The existence of hydraulic barriers and hydraulic conduits should be discussed. Other issues mentioned in this section may be groundwater levels, transmissivity (product of permeability and aquifer thickness and storage capacity).

3.1.2.2.18 Groundwater quality

This part of the study analyses the groundwater chemistry. The water is typically sampled in wells and piezometers. If the sampled water is contaminated, it are tried to identify the sources (i.e. other industrial activities, farming etc.).

3.1.2.2.19 Surface water quality

The results from a baseline surface water sampling program are presented here. The sampling points should be selected to provide a baseline for what part of the catchment area may potentially be effected by discharges from the proposed development.

Typically the overall quality of the water is discussed as well as the levels of organic pollution, nutrient and metal levels. Possible sources of contamination are identified.

3.1.2.2.20 Surface water hydrology

In order to determine the assimilation capacity of the receiving waters, flow data of all surface waters potentially influenced by the project is required. “Knowledge of the surface water hydrological characteristics is also important in establishing the recharge and discharge relationship between the rivers/streams and the groundwater system.” [25, Lisheen, 1995]

3.1.2.2.21 Fisheries, fish population and spawning

A fish stock assessment in representative sections of the main watercourses within the survey area is part of this section. This assessment includes tissue analysis and density measurements of the existing species. In addition the measures like the mean redd count (Redds are the nests that mated adult salmon build in the gravel, where they deposit and cover their eggs. A redd count is the number of such nests counted in the river at the end of the spawning season. The number of redds is a good indication of the health of a salmon run) can provided for each of the watercourses as a means of investigating the spawning activities in these rivers.

3.1.2.2.22 Surface water flora and macroinvertebrate fauna

Selected plant and macroinvertebrate species may be utilised as indicators of water quality. To investigate these aquatic ecological surveys and water chemistry surveys are carried out. This part of the study should list the flora and macroinvertebrate fauna encountered and the implications of their existence and/or lack of existence.

3.1.2.3 Characterisation of tailings

The following characterisations of ore, waste-rock (if used for dam construction or managed within the same TMF) and tailings are used for the design of a TMF

- ore and waste-rock characterisation
    - reserves
    - mineralogy
    - chemical properties
    - physical and engineering properties
    - acid generating potential
    - leachable contaminants
    - ore and changes of ore qualities during mine life
    - low-grade ore and mine rock quantity and schedule.

- tailings characterisation

A general description of physical and chemical characteristics, such as:
    - daily/annual throughput and total quantity
    - size distribution
    - solid or slurried tailings, pulp density (%solids)
    - density of solids
    - stability/plasticity
    - liquid phase chemistry
    - acid generating potential
    - geochemical characteristics (metal content, leaching behaviour)
    - pore water
    - consolidation behaviour.

- mineral processing characteristics
    - reagents used, their concentrations and quantities
    - water recirculation requirements
    - mineral processing plant treatment processes (e.g. cyanide destruction)
    - other inflows to tailings pond
    - pipes and appurtenances
    - potential for pit and/or underground backfilling
    - ratio of management of tailings on surface to backfill.

[18, Canada, 1998]

3.1.2.4 TMF studies and plans

The following is a summary of studies and plans which are developed in the design of a TMF to an adequate level of detail relevant to each stage (conceptual, preliminary and detailed design) and then maintained throughout operation and closure:

- site selection documentation
- environmental assessment
- risk assessment
- emergency preparedness plan
- deposition plan
- water balance and management plan and
- decommissioning and closure plan.

The plan contents listed represent minima. There may be additional aspects which have to be included.
[18, Canada, 1998]

3.1.2.4.1 Site selection

The operator selects a preferred site and prepares a documented rationale for selection, including discussion of alternate sites studied and rejected. Furthermore public perception issues related to the project (internal and external stakeholder requirements) have to be identified.

- Environmental considerations
    - effluent treatment requirements
    - emissions to surface water
    - emissions to groundwater (hydrogeological containment)
    - historical use of the receiving watershed
    - background environmental conditions
    - impact on vegetation, wildlife and aquatic life
    - natural flora and fauna
    - archaeological considerations
    - potential emissions to air
    - aesthetic considerations
    - conceptual water balance.

- Planning considerations
    - accessibility (road construction)
    - distance from the mineral processing plant
    - relative elevation from the mineral processing plant
    - distance from habitation and areas of human activity
    - topography
    - existing land and resource use
    - property ownership and mineral rights
    - aboriginal land claims

  - transportation corridors, power lines, etc.
  - watershed and surface area considerations
  - volumetric capacity
  - pond volume/storage capacity ratio
  - geology, including potential ore bodies
  - construction material availability
  - conflict with mining activity
  - dam foundation conditions
  - basin foundation conditions
  - downstream hazards
  - hydrology
  - groundwater, contaminant seepage
  - potential impact area
  - human and environmental risk
  - water management scheme and preliminary water balance
  - operational plan
  - deposition plan
  - preliminary containment and water management structures
  - preliminary cost estimate based on preliminary considerations
  - conceptual risk assessment.

- Decommissioning/reclamation considerations:
  - flood routing requirements
  - revegetation potential
  - long-term physical and chemical stability
  - ease of establishing permanent drainage
  - reduction and/or control of acid drainage and other contaminants
  - dust control
  - long-term maintenance, monitoring and treatment requirements.

- Development, operating and closure cost considerations:
  - capital cost
  - cost of tailings transport
  - tailings facility operating and maintenance costs
  - closure costs
  - cost per tonne of ore processed.

3.1.2.4.2 Environmental assessment

In order to obtain stakeholder and regulatory acceptance for siting a new TMF, it is often necessary to conduct an environmental impact assessment (EIA, often a legal requirement).

The environmental impact assessment process requires integration of knowledge about the project as it is being designed, the natural and social environments in which the project is situated, and community and stakeholder concerns. At the environmental impact assessment stage, tailings facilities are usually components of a larger, integrated project. The following is a summary of some significant aspects related to tailings, which are addressed in an environmental impact assessment:

- environmental baseline
- mineral processing plant tailings aspects
- tailings and waste-rock facility site selection, with a clearly documented rationale for the selected site
- conceptual tailings and waste-rock facility design

The environmental assessment addresses the projected impacts of the tailings and waste-rock facility on the environment, including:

- physical impacts
- physiography
- climate
- air quality
- noise
- hydrology
- hydrogeology
- water quality
- biological impacts
- aquatic life
- vegetation
- wildlife
- archaeological impacts
- socio-economic impacts
- land-use impacts.

3.1.2.4.3 Risk assessment

Risk assessment addresses what could go wrong (i.e. hazards or failure modes) with a facility and its associated plans and procedures; what are the probabilities of failure; and what are the consequences of failure. Risk assessment provides a basis for the development of risk management, including communication, contingency, mitigation and emergency response plans.

Risk has to be assessed (and managed) through each phase of the life cycle of the TMF. However, the intensity of assessment varies at different stages, depending on the objectives of the review, complexity of the issue and the extent of information available.

<u>Scope and Purpose of Assessment</u>
At this stage all stakeholders in the risk assessment are identified.

<u>Risk Assessment Team</u>
An experienced, multi-disciplinary risk assessment team is required to determine potential failure modes, probabilities and consequences of failure. The team typically includes the TMF designer, construction contractor, operators, environmental and management staff and, in cases of detailed assessments, a risk assessment specialist. Consequence evaluation involves environmental staff and specialists including, in some cases, health experts and cost engineers. Involving tailings operating staff is critical for a risk assessment of an existing tailings facility in order to incorporate their knowledge and experience of the facility.

<u>Evaluation Criteria</u>
Criteria have are developed to guide the evaluation of findings and establish levels of acceptable or unacceptable risk. High probability, high consequence failure modes are obviously of concern, but low probability, high consequence modes may also require examination. Potential human health and safety, environmental impact or business (e.g. downtime, reputation, property damage) consequences are considered.

<u>Methodology</u>
Risk assessment can be qualitative (subjective ratings of probability, consequence and overall risk) or quantitative (numeric values of probability and Euro values for consequences). A simple qualitative assessment is appropriate to evaluate a number of potential TMF sites whereas a detailed quantitative assessment is more appropriate for a proposed major modification to an existing facility.

Commonly practised methodologies for risk assessment include

- process/system checklists
- system design models
- safety reviews
- relative ranking
- preliminary hazard analysis
- "what-if" analysis
- hazard and operability (HAZOP) studies
- failure modes, effects (and criticality) analysis - FMEA, FMECA
- probabilistic simulation analysis
- fault-tree analysis
- event-tree analysis
- cause-consequence analysis and human error analysis.

Potential triggers and failure modes

Reservoir overtopping - landslide into reservoir generates a wave which overtops the dam, wave action overtops dam; perimeter bypass system fails and water enters reservoir, exceeding capacity of spillway or storage, or an external stream diversion failed and water entered reservoir; pond allowed to reach crest of dam; discharge from top end of pond to save dam height; blocked outlet structures; precipitation exceeds storage capacity; water balance not maintained.

Dam instability (upstream or downstream) –seepage causes piping and removes dam material (i.e. filter failure); seepage raises pore pressures and causes shallow instability; seepage raises pore pressures and causes deep instability; seismic liquefaction of dams; seismic deformation of dams; seismic liquefaction of tailings leads to erosion; seismic liquefaction of tailings applies horizontal thrust to dam; non-seismic liquefaction of dam due to straining or increased pore pressures; seepage failure raises pore pressures and triggers a slide; construction pore pressures rise and slope moves; saturation of uncompacted fill either by first fill or rain or snow encapsulated in dam fill melts, dam settles, overtops; uncontrolled toe erosion retrogresses; dam face erodes due to uncontrolled precipitation or snow melt.

Foundation instability - Karst collapses beneath dam; collapse due to mine subsidence allows tailings to escape into mine or void; sliding on weak soil or liner interface; compression of weak soils leads to cracking of dam; permafrost degrades; construction pore pressures rise and foundations move; seepage through a poor membrane or pervious soils into groundwater system, bypassing seepage recovery systems; seismic liquefaction of foundations; seismic deformation of foundations; non-seismic liquefaction of foundations.

Structural failures - piping around a culvert or decant pipe; reclaim tower fails; pumps fail due to loss of power; pipeline or conduit fails; landslide blocks spillway; ice blocks spillway.

Power failure.

Probability of Failure

The probability of failure for each potential failure mode based on past experience with facility, experience with similar facilities, engineering analysis and professional judgement is estimated.

Consequences of Failure

Consequences of failure for each potential failure mode are estimated, including consideration of impacts on health and safety of workers, contractors and general public; environmental impacts including consideration of assimilative capacity and environmental sensitivity of site; and business impacts.

Link to Operations

Identification of operating, maintenance, inspection and incident response (e.g. unusual occurrences) procedures which can reduce risk; and parameters and operating characteristics to be measured, monitored and documented to provide early warning of potential failure.

Reporting
Results of risk assessments is presented and summarised in a clear manner for both operating and management personnel. It is essential that this information be well understood by all relevant staff.

3.1.2.4.4 Emergency preparedness plan

It is of prime importance to be ready for emergencies and to have appropriate contingency and emergency preparedness plans in place. Emergency preparedness includes preparation both for on-site incidents and for incidents having off-site implications, including dam breach. Contingency and emergency preparedness plans should be reviewed on a periodic basis, tested, and widely distributed within an organisation and to potentially affected external stakeholders.

The site's emergency preparedness plan usually integrates the tailings facility aspects into the overall site emergency preparedness plan and includes, but is not limited to, the following:

- identification of planning co-ordinator, team and organisational structure
- identification of emergency organisation, roles and responsibilities
- identification of requirements of legislation, codes of practice, notification and reporting obligations
- identification of available resources
- mutual aid agreements
- public relations plan
- telephone lists
- establishment of communication system for notifications and for post-notification purposes
- risk analysis for on-site and off-site effects
- inundation study, maps and tables for both physical and environmental releases (including facility failure)
- basis for activation of emergency plan and emergency decision making
- training of personnel
- investigation and evaluation of incidents and accidents
- restoration of safe operating conditions.

3.1.2.4.5 Deposition plan

A tailings deposition plan is developed for the expected mine life. Deposition plans can allow for the staging of TMF lifts and raises over the life of the mine to accommodate long-term storage of tailings solids, maintain adequate solids storage capacity, and allow adequate polishing of supernatant water during operation of the mine.

Appropriate consideration for expanded requirements and/or capacity should be considered in the plan. Deposition plan development requires tailings quantity and density, water content and production information estimated from the process/mineral processing plant water balance, including provisions for estimating uncertainty and contingencies. The basic parameters are validated and updated on a periodic or regular basis.

3.1.2.4.6 Water management plan

The water issue is considered in conjunction with the mine, so that an integrated water management is achieved. A water management plan considers and/or develops site-specific standards, targets, operational or contingency plans and procedures (as appropriate) for all of the following:

- community expectations
- statutory requirements
- risk management
- monitoring of hydrological process
- operational monitoring
- emergency monitoring
- water supply
- soil erosion
- water quality
- computer models
- performance indicators, and
- training and research.

[97, Environment Australia, 2002]

Hydrology:
Hydrology data, including the delineation of tailings site catchment area(s) and all potential water sources, both natural and process, are used in the development of a water/contaminant balance and design of tailings facility components. Design parameters are established and documented, then actual experience has to be monitored to identify variances, validate projections and anticipate potential problems.

Design Flood:
The appropriate probable maximum flood (PMF) is identified, with reference to current design standards and in consultation with regulatory agencies. Design flood considerations should be consistently applied through all stages of the life cycle. Storage requirements, operating freeboard and spillway design are based on the hydrology of the watershed.

Water Balance:
A water balance study is performed. Specification of requirements for ongoing data collection for the mineral processing plant and TMF water balance calibration purposes is necessary.

Surface Water Management Plan:
Completion of a water management plan detailing appropriate designs and strategies, where required, for

- seepage collection
- reclaim/pump-back systems
- treatment/discharge systems, including all water conveyance systems
- water retention and discharge strategy, including operating parameters.

Emissions Balance and Release:
The emissions balance provides estimates of emissions to land, air and groundwater. A plan is developed to minimise emissions.

Effluent Criteria:
Development of effluent criteria for the TMF, with reference to regulatory requirements and operating licences and permits, including

- dissolved and suspended matter
- suspended solids
- effluent quality
- periods of discharge
- bacterial and biological levels
- toxicity.

[18, Canada, 1998]

3.1.2.4.7 Decommissioning and closure plan

Closure plans and performance criteria are developed in the early stages of facility design, and then verified and updated periodically through the operating life of the facility in preparation for decommissioning and closure. Closure is usually covered by regulations, and the following are general considerations applicable to development of closure plans. In some circumstances closure has to be followed by long-term aftercare. This requires similar plans and controls as for closure.

Elements of a Closure Plan:

- determination of background data, including
    - history of site
    - infrastructure
    - process flow controls
    - system operations
    - mineralogy
    - topography
- hydrology/water management
- hydrogeology
- soil capability
- revegetation
- impact assessment;
- long-term maintenance
- geotechnics
- chemistry and geochemistry
- monitoring program
- communications
- financial assurance
- stakeholder consultation
- potential end land use; and closure technology (i.e. “dry” or “wet” cover, flooded, wetlands, perpetual treatment, vegetative cover).

Aspects of TMF stability for closure:
Closure plans require a thorough re-assessment of the facility and its stability under closure conditions. All aspects of the facility and physical and chemical stability are reviewed. In particular, the actual performance of the facility in service, including

- deformation
- seepage
- foundation and sidewalls.

are checked against design projections as well as against projected post-closure conditions. Design loads might be different after decommissioning and closure.

Structural monitoring and inspections are continued for all facilities until they are decommissioned and thereafter as appropriate. Identification and delineation of any requirements for continuing inspection and/or monitoring of remaining structures after closure is necessary.

Action plans are prepared to deal with shortcomings in closure quality and/or difficulties in complying with closure specifications. Examination of the consequences of closure of the facilities on emergency preparedness procedures, and updating these plans as appropriate, is also desirable. Continuing availability of design, construction and operating records after closure for structures remaining in place has to be ensured.

3.1.2.5 TMF and appurtenant structures design

The following considerations relate to TMF and containment basin design. The list may not apply to all sites or all situations. It is up to the operator and the permitting authority to decide which aspects apply. Site-specific conditions may require the use of additional criteria.

3.1.2.5.1 Design input considerations

Information relating to the pond site is compiled from literature survey and field/laboratory investigation programs.

Hydrology and Hydrogeology:
- hydrological and hydrogeology studies
- water balance, water quality
- design flood
- freeboard requirements
- drought design (i.e. water cover requirement)
- catchment runoff and diversion arrangements
- deposition plan
- erosion management plan.

Foundations, Geology and Geotechnical Engineering:
- geomorphology
- regional and local geology, faults
- stratigraphy
- bedrock and soil characteristics
- geotechnical information, including
  - compressibility
  - shear strength
  - angle of friction
  - grain size
  - density
  - plasticity
  - fractures
  - liquefaction potential
  - permeability
  - erosion potential
  - hydraulic fracture.

Construction materials:
The availability of naturally occurring construction materials is assessed as well as the engineering characteristics of these potential borrow materials, tailings, grout/concrete or other potential liner material (both natural and synthetic), such as

- grain size
- density
- volume
- shear strength
- permeability
- acid generating potential
- chemical reactivity (acid generating potential, reaction with pond water, thiosalt generating potential)
- wind and water erosion potential.

Potential detrimental effects of tailings and/or process water on construction materials are determined. Environmental impacts, stability and rehabilitation requirements for the use of any construction materials are considered at this stage.

Topography:
Regional and topographical mapping and air photos.

Special Environmental Considerations:
Seismic risk; seismic attenuation of foundation strata and construction materials; liquefaction potential of foundation strata and construction materials; climatic conditions, including

- extreme values to be expected
- wind and wave actions
- permafrost effects
- frost.

Seepage:
Maximum allowable seepage objectives for environmental and structural requirements is determined. Requirements for pervious vs. impervious materials and construction methods are identified and a seepage management plan is developed.

3.1.2.5.2 Design elements

Required design parameters:
- facility classification (if existing under local jurisdiction)
- stability
- earthquake criteria
- factors of safety
- design permeabilities
- acid rock drainage
- wildlife
- dust
- closure considerations.

These parameters are outlined in the following paragraphs.

Stability:
Stability of the foundation, facility and appurtenances under conditions covering construction, operations and closure; and under static and dynamic conditions, including consideration of wave, frost/ice action and rapid drawdown (in case of pond) must be analysed. Density and compaction targets are established.

Foundation Preparation:
The requirements for preparation of the TMF foundations prior to construction are determined, including consideration of

- vegetation removal, including merchantable timber
- excavation of organic soils
- cut-off walls
- groundwater control and containment
- bedrock cleaning and slush grouting
- high-pressure grouting
- diversion wells
- diversion channels
- dewatering requirements
- stability
- constructability
- other special construction requirements.

Seepage Analysis and Management:
The requirement for seepage control are assessed, including into groundwater, consideration of water chemistry and acid generating potential. Implementation of appropriate measures, such as

- filter design
- cut-off wall
- grout curtain
- ditching
- low permeability core
- interception wells.

are planned for.

Appurtenances:
The following options are designed, as required:
- spillways
- towers
- pipelines (e.g. vacuum breakers, secondary containment)
- maximum flood-handling requirements
- gates and valves
- siphons
- pumps
- natural hazards handling requirements (e.g. debris, beavers, rabbits, ice blockage).

TMF design:
- type of facility (e.g. heap, dam (type of dam))
- design philosophy
- criteria for major elements.

TMF construction plan:
A plan for executing the initial TMF construction and subsequent lifts, including sequencing and requirements for stability monitoring are developed. A construction methodology, schedule and anticipated costs are established. Potential environmental impacts due to construction of the proposed design are determined.

TMF monitoring systems:
- piezometers
- inclinometers
- settlements gauges
- seepage flow monitoring
- temperature (permafrost, frost penetration, heating)
- surveillance methods.

Failure Mode Analysis:
Potential TMF failure modes are analysed during construction, during operation, in its final condition and after closure.

Design for Closure:
Design elements are co-ordinated with planning for eventual decommissioning and closure.

3.1.2.6 Control and monitoring

Quality assurance/quality control (QA/QC) plan:
Construction drawings and as-built construction records should be maintained and available throughout construction, operation and closure phases in an orderly and secure fashion, including revisions to construction drawings; test results; meeting minutes; construction photographs; monitoring notes are maintained.

Construction Control:
Typical components of a construction management system include:
- planning and scheduling
- survey control (layout, as-built records)
- grouting monitoring
- foundation preparation monitoring
- material quality control
- compaction control
- instrumentation monitoring and data synthesis
- record keeping
- construction safety
- construction environmental criteria.

Dust control:
Minimise dust releases from the tailings facility. This may include keeping the tailings wet and/or using short- or long-term chemical or organic covers.

Inspection of tailings management facilities:
- performance monitoring - visual inspection – with high frequency
- groundwater pressure (pore water pressure)
- seepage
- deformation (settlement and stability)
- weather influence
- seismic events (after the fact)
- special inspection programs after major events (earthquakes, hurricanes, spring breakup, floods).

- Indicators of instability:
    - soft zones and boils along the toe
    - dirty sediment in seepage
    - increased seepage rates
    - new areas of seepage
    - longitudinal and transverse cracking
    - settlement.

- Areas requiring special attention:
    - spillways
    - decant structures
    - drain and pressure relief wells
    - concrete structures
    - pipes and conduits through dams
    - rip rap areas
    - siphons
    - weirs
    - trees and animal holes.

Stability monitoring program plans:
- location of control stations
- schedule (control period and inspection)
- type of monitoring (visual inspections, measures and parameters)
- appropriate level of instrumentation (e.g. piezometers) with clearly identified purpose
- inspection methods, data compilation and
- evaluation
- persons responsible for monitoring
- data storage and reporting systems
- criteria to assess monitoring program.

Water Quality Plan

- Hydrology
  - severe storm events and drought events
  - necessary information and parameters for water management activities
  - criteria to manage water levels within safe limits,including any required daily or seasonal water level control.

- Water control
  - safe water management must be ensured within the confines of the system
  - damage to all structures must be prevented/controlled/repaired
  - reviews and revision as required after changes in design or methods, during and after construction program, when the pond level exceeds specified critical elevations, after major storm or spring melt events must be performed.

- Perimeter seepage
  - evaluate potential for seepage from the tailings area
  - define levels and characteristics of acceptable seepage
  - prepare action plans to deal with deviations from design seepage
  - measure performance including control of seepage within design rates
  - monitoring and controls to ensure that systems are performing as per design.

Tailings Deposition Plan:
Efficient use of the tailings capacity and effective closure of the facility is ensured. Long- and short-term scheduling of TMF lifts and raises are provided. At pre-set intervals a schedule for deposition of the tailings and a filling curve (volume/elevation/graph) are validated against actual field conditions.

3.1.3 Construction phase

For some mining tailings and waste-rock facilities the distinction between construction and operational phases are not so clear, because often construction continues or reoccurs during operation (e.g raising of the dam). Construction of the facility is documented and follows the construction plan established in the design phase. “As built” documentation is provided highlighting any changes occurred compared to the construction plan.

In the construction of the facility and for the future:

- ‘as built’ drawings and ‘actual’ procedure records are maintained, highlighting any variances from the original design and if necessary revisiting the design criteria
- construction is supervised by independent qualified engineering/geo-technical specialist
- records of results of test work (e.g. compaction) carried out for and during construction are properly maintained.

[45, Euromines, 2002]

3.1.4 Operational phase

The two main causes of TMF incidents have been found to be

- lack of control of water balance
- a general lack of understanding of the features that control safe operations.

[9, ICOLD, 2001, p. 6]

This indicates that successful operational management is the key factor in running a safe TMF.

Geotechnical engineering has advanced far enough to design sound and safe dams. It is the management of TMF that makes the difference between a smooth operation or a possible disaster.

The following actions can be taken to avoid these incidents:

- monitoring of phreatic surface with properly sited placed piezometers and open tube standpipes
- foreseeing provisions for diverting water and tailings discharge away from an impoundment in event of difficulties
- providing alternative discharge, possibly into another impoundment
- providing second decant facilities and/or standby pump barges for emergencies
- measuring ground movements with deep inclometers and having a knowledge of pore pressure conditions
- providing adequate drainage
- maintaining records of design and updates/changes in design
- educating and training staff.

[9, ICOLD, 2001]

and furthermore

- providing continuity in the engineering of the dam
- and in some cases independent audits of the dam with a “sign-off” by the third-party auditor.

The operation of the management facility follows the tailings and waste-rock management plan, the operational instructions and the monitoring plan for the facility. Any deviations from these plans are documented and evaluated. Monitoring data is evaluated on a regular basis and followed up were necessary. Internal and external reviews (audits) are performed in some cases.

The following is considered good practice:

- the production of tailings and waste-rock receives the same level of management attention as the production of saleable product
- effective operational control and monitoring is maintained
- there are systems for keeping records of tailings and waste-rock production quantities and characteristics
- accountabilities and responsibilities for tailings and waste-rock management are clearly defined with appropriately qualified personnel
- management facilities are routinely inspected by a qualified professional engineer experienced in tailings and waste-rock management and signed off to confirm that all significant risks have been identified and are adequately managed in the continued operation of the facilities
- operating instructions are prepared in the language of the operators and followed. These instructions include all the monitoring requirements
- operating records such as rise in levels, tonnes contained, seepage quantities, water consumption (maybe meteorological data) etc. are stored and properly maintained
- operating conditions which occur beyond the boundaries identified by the design are immediately reported to the designer or checked by a qualified technical person
- appropriate training to operational personnel is provided including incipient fault diagnosis
- special attention is given to the follow-up of the water management plan
- effective mechanisms for reporting of faults are established and maintained
- effective emergency response plans are maintained and further developed.

[45, Euromines, 2002]

3.1.4.1 OSM manuals

Several operators use dam safety manuals These dam safety manuals are known as OSM-manuals (operation, supervision and maintenance) [50, Au group, 2002]. An example of such an OSM manual covers the following:

- dam safety organisation
- emergency preparedness plan
- classification according to consequences by dam failure
- dam construction
- hydrology
- environment
- operation
- supervision
- permits
- reports.

[50, Au group, 2002]

3.1.4.1.1 Dam safety organisation

The dam safety organisation consists of one dam safety manager appointed at each site. To support these managers there is also one dam safety co-ordinator who specialises in tailings dams and works full-time on dam safety. For operation, supervision and maintenance, the manager has people in his own organisation, often the same staff responsible for the environmental sampling and supervising the tailings storage facilities.

3.1.4.1.2 Emergency preparedness plan, EPP

For each tailings storage facility there is an EPP in case of an accident related to the tailings pond. The EPP includes lists of who to inform within the operation and the authorities. Consultants and contractors who are familiar with the site are also listed in case support is needed within short notice. The EPP also includes examples of what to do and what measures to take in various possible situations. In general, the manager and co-ordinator are always consulted and involved in all major decisions and measures taken regarding the dams. The manager is the person who has to make the final decisions of what to do in every situation.

3.1.4.1.3 Risk management of tailings facilities

In some cases the tailings dams are classified according to the consequences of a possible dam failure (and not on the probability of a failure). In one example, according to the possible consequences there are four different classes; 1A, 1B, 2 and 3 according to the tables below. The table is split into two classes, with classification of risks for humans separated from the risk for property, infrastructure and environment.

Class	Consequences
1A	Obvious risk for people’s lives.
1B	Non-negligible risk for people’s lives or serious injury.

Table 3.1: Classification with regards to loss of lives or serious injury

Class	Consequences
1A	Obvious risk of: ▪ serious damage on important infrastructure, important structures or significant harm to the environment and ▪ serious economical damage.
1B	Considerable risk of: ▪ serious damage on important infrastructure, important structures or significant harm to the environment and ▪ serious economical damage.
2	Non-negligible risk of: ▪ considerable damage to infrastructure, important structures, harm to the environment or third parties property.
3	Negligible risk for: ▪ considerable damage to infrastructure, important structures, harm to the environment or third parties property.

Table 3.2: Classification with regard to damage to infrastructure, environment and property

The classification forms the basis for operation and supervision. It sets the limits for the freeboard required and the spillway capacity, i.e. the safety margin from the maximum water level up to the crest of the dam and the maximum discharge capacity respectively. At the moment, when the classification system is new, and the limits between different classes are not yet strictly defined, the classification of all dams is being reviewed. More data is also being collected and investigated to verify likely consequences.

3.1.4.1.4 Dam construction

Each tailings impoundment and its dams are described in detail. From the starter dam to present height, a full description is recorded of the type of construction and material used, the name of the contractor, any problems that occurred during construction, the type of spillway, the volume of tailings and water being deposited, etc. In this way, at any time, all information about the tailings dam relevant for dam safety should easily be found.

3.1.4.1.5 Hydrology

The requirement is that every dam must have a minimum free board, a maximum wave height allowance and minimum spillway capacity. This means all dams classified as 1A or 1B are designed for a spillway capacity to take a one in a 100-year storm, excluding any allowance for water storage. These dams should also be designed for a “class 1 flow” (which should roughly correspond to a one in a 10000-year storm) allowing storage of water to a safe level. Dams classified as 2 are designed for the one in a 100-year storm and class 3 does not have any specific requirements.

3.1.4.1.6 Environment

For each tailings impoundment and mine there is an environmental monitoring program, which includes sampling, evaluation and reporting to the authorities.

3.1.4.1.7 Operation

Proper operation of the tailings impoundment is essential for reliable operation and a high level of dam safety. Detailed up-to-date instructions are given of the way the tailings impoundment is operated to meet design requirements, respond to tailings properties, and fulfil the demand for process water and climatic conditions. Everybody working on the plant and on the tailings facility is to be familiar with these instructions. Education is therefore stressed as an essential requirement.

3.1.4.1.8 Monitoring

Supervision and correct operation of the tailings impoundment are probably the most important requirements to obtain a high level of dam safety. Supervision requires suitable instrumentation, which in turn requires competent staff for installation to evaluate the results yielded and to draw the correct conclusions on the basis of the monitoring results.

Regular monitoring is carried out basically at four different levels, following a stage-wise approach starting with daily inspections, ending with profound safety audits carried out with long intervals:

1) Site inspections
2) Supervision
3) Yearly inspection/audit
4) Survey

Site inspections are made at different intervals for each tailings dam, varying from three times a day to several days a week. It is normally staff from the plant or staff doing the environmental sampling that carry out the daily inspections.

Supervision is carried out monthly or at least once every three months by the manager or an appointed person.

A yearly inspection is carried out by the co-ordinator, or an external specialist. The inspector will audit all events and measures at the dams since the last inspection and will issue a report. The yearly inspection will also include a full review of the OSM-manual.

A complete survey is be carried out at intervals between 10 to 20 years. The survey includes a full investigation of archive material and inspections, and also includes an inspection on site and a review of the OSM-manual. The result are a report stating the status of the tailings impoundment and its embankments.

3.1.4.1.9 Permits

All permits given for each tailings dam are compiled to make it easy to check on how operations are meeting the given permits.

3.1.4.1.10 Reports

All reports relevant for dam safety are in one place and easy to find when necessary.

3.1.4.1.11 Additional information regarding dam safety

After completion of the dam safety manuals, a lot of effort has to be made to implement the OSM-manuals on site and educating staff working on the dams. In one example, as a first step all manuals were presented on site, then a four-hour introduction course was held for all staff and other people at each plant involved with the dams. The next step was a three to four-day program including theory, practical training, review of present conditions (labour availability and physical resources), with time for adequate discussions. Implementation of OSM-manuals and education of staff is an ongoing project, connected with the yearly inspection. The result of the inspection is presented to all relevant staff and further education can be linked to this.

Section 3.1.4.1 all from [50, Au group, 2002]

3.1.5 Closure and aftercare phase

Usually the closure of tailings and waste-rock management facilities occurs simultaneously with the closure of a mine. Therefore an integrated closure and aftercare plan is developed and carried out. However, this section focuses on sites within the scope of this work. Where necessary or useful interfaces with the overall closure plans are mentioned. It is standard practice that successive reclamation activities that have been performed during the operational phase of the mine life are evaluated before the final closure of the site. The following issues are included in the previous phases, but are reconsidered against the "as built" situation at the site and closure plans adjusted accordingly:

- closure costs are included in the assessment of alternatives
- closure plans adopt a risk assessment approach
- closure plans are maintained throughout the active life of the facility and are routinely updated taking into account modifications to the design and during operation
- facilities are designed to facilitate premature close out if necessary
- aftercare design should minimise the need for active management
- the closure plan developed in the planning stage is be reviewed and up-dated with certain frequency during the design and operational phase of the mine life.

[45, Euromines, 2002]

It should be noted that the OSM manuals, mentioned in the previous section are also applied throughout the closure and aftercare phase.

3.1.5.1 Long-term closure objectives

The reference [100, Eriksson, 2002] used in this section is mainly based on the MIRO (1998) guidelines "A TECHNICAL FRAMEWORK FOR MINE CLOSURE PLANNING" and the MiMi (1998) State-of-the-art-report on "Prevention and control of pollution from tailings and waste-rockproducts". Both of these documents are recommended to interested people as they give a good overview of the subject and many good ideas.
This contribution has been submitted late. Hence it was not possible to fully edit the text. It is recognised that this section overlaps partially with Section 2.4.2. Comments on how to combine these two sections are welcome.

The following table summarises the fundamental criteria for closure processes from initial planning through to actual implementation.

Issue	Closure Criteria
Physical stability	All remaining anthropogenic structures are physically stable.
Chemical stability	Physical structures remaining after closure are chemically stable.
Biological stability	The biological environment is restored to a natural, balanced ecosystem typical of the area, or is left in such a state so as to encourage and enable the natural rehabilitation and/or reintroduction of a biologically diverse, stable environment.
Hydrological and hydrogeological environment	Closure prevents any physical or chemical pollutants from entering and subsequently degrading the downstream environment - including surface and ground waters.
Geographical and climatic influences	Closure is appropriate to the demands and specifications of the location of the site in terms of climatic (e.g. rainfall, storm events, seasonal extremes) and geographic factors (e.g. proximity to human habitations, topography, accessibility of the mine).
Local sensitivities and opportunities	Closure optimises the opportunities for restoring the land and upgrade of land use is considered whenever appropriate and/or economically feasible.
Land use	Rehabilitation is such that the ultimate land use is optimised and is compatible with the surrounding area and the requirements of the local community.
Financial assurance	Closure has adequate and appropriate financial assurance to ensure implementation of the mine closure plan.
Socio-economic considerations	Consideration must be taken of opportunities for local communities whose livelihoods may depend on the employment and economic fallout from the mining activities. Adequate measures are made to ensure that the socio-economic implications of closure are maximised.

Table 3.3: Summary of criteria for closure
[100, Eriksson, 2002]

3.1.5.1.1 Chemical stability

After closure tailings and waste-rock management sites and the structures within them must be chemically stable. This means, for example, that the consequences of any chemical changes or conditions leading to leaching of metals, salts or organic compounds should not endanger public health and safety nor result in deterioration in environmental resources. In practice, aspects such as the short and long terms effects of changes in tailings geochemistry, the seepage from tailings impoundments, waste-rock dumps and underground backfill, or the surface waters draining from the site must be examined. Where contaminated discharges are predicted in advance, appropriate mitigatory measures, (e.g. settlement or passive treatment using wetlands), must be employed to alleviate or eliminate such discharges if these are likely to cause adverse environmental effects. Monitoring is aimed at demonstrating that there are no adverse effects, (e.g. concentrations that exceed statutory limits), from the waters, soils and air surrounding the closed site.
[100, Eriksson, 2002]

In the case of **sulphide tailings** the most evident closure objective is to maintain chemical stability of the tailings by preventing release of oxidation products to the surrounding environment, whether this is accomplished by preventing oxidation reactions from occurring, preventing the transport of these products beyond the site boundaries, or both. Natural processes can strongly influence how this objective is achieved. For example, measures to restrict infiltration into the deposit may be preferred over those such as low-permeability bottom **liners** with accompanying hydraulic gradients that promote contaminant transport (the so-called “bathtub” effect). Biological processes may also play a role, since organisms have adapted over millions of years to overcome the kinds of conditions engineered measures seek to impose. Studies found that the most significant pathway for contaminant release was through uptake by the deep-rooted and moisture-seeking artemisia tridentata (sagebrush) that introduced these contaminants into the food chain by grazing animals. But biological processes have also been exploited to advantage through the use of “vegetative covers” which encourage the establishment of self-sustaining grasses and forbs on inert cover materials that reduce infiltration by enhancing evapo-transpiration [13, Vick,].

The following figure shows some typical covers for TMF.

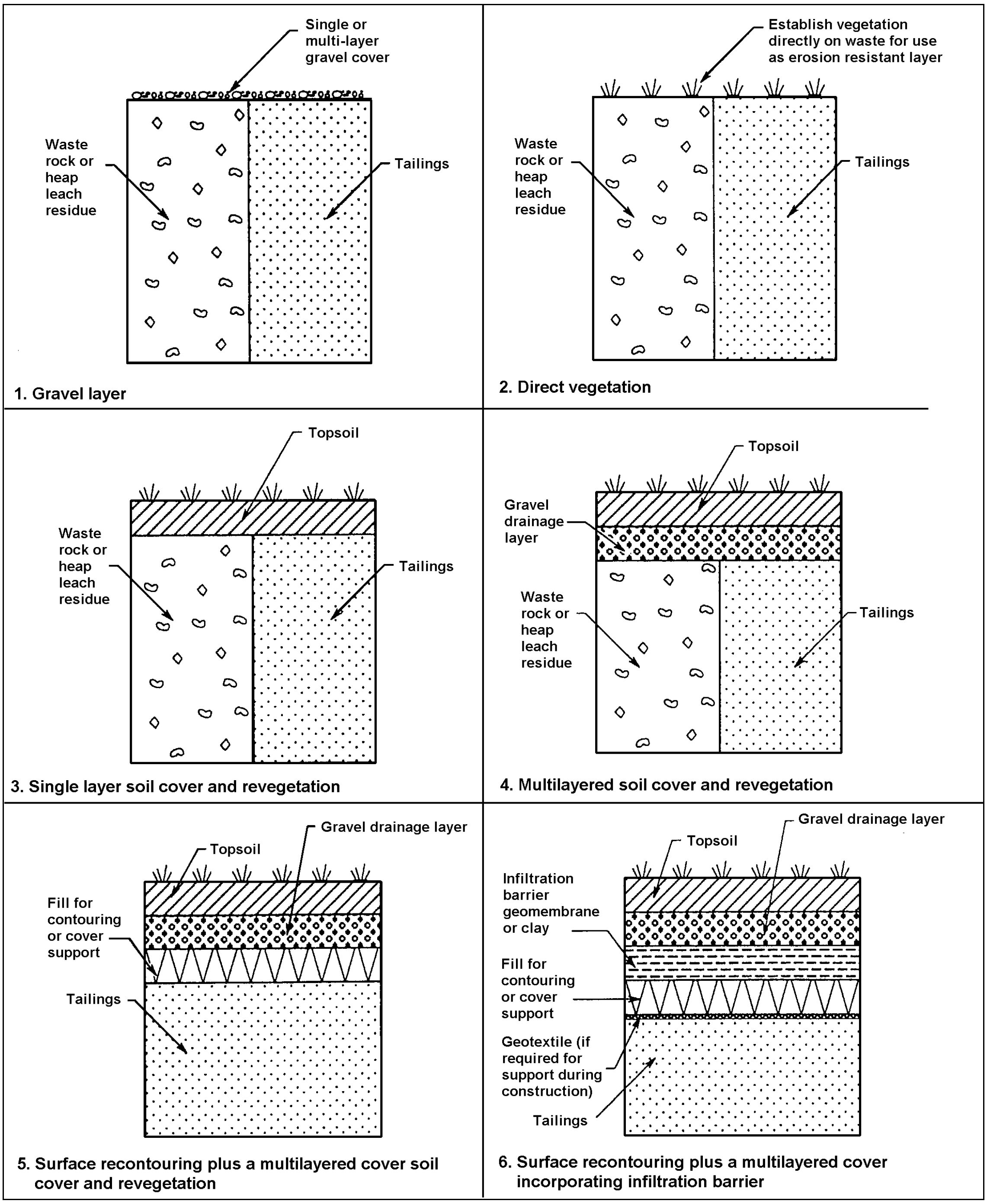

Figure 3.1: Typical covers for tailings management areas
[11, EPA, 1995]

3.1.5.1.2 Physical stability

All anthropogenic structures that remain after mine closure must be physically stable. They should pose no hazard to public health and safety as a result of failure or physical deterioration, and they should continue to perform the function for which they were designed. The structures should not erode or move from their locations, except where such movement does not endanger public health and safety nor cause detrimental effects to the adjacent environment. This means that full account must be taken of extreme events, such as floods, winds or earthquakes, as well as other natural perpetual forces, such as erosion, in the design periods and factors of safety proposed. Monitoring of structures is aimed at demonstrating that there has been no physical deterioration or deformation.
[100, Eriksson, 2002]

Differences from conventional practice arise in several areas, and these are considered in turn:

Extreme events
All dams, tailings and otherwise, are designed to remain stable under the influence of some chosen magnitude of floods and earthquakes, such as the Probable Maximum Flood (PMF) or Maximum Credible Earthquake (MCE). The corresponding design values are established within the framework of the hydrometeorological and seismotectonic understanding of the region, and are thus a function of the state of knowledge at the time they are derived. However, this state of knowledge continually changes as understanding of technical factors improves and occurrences of large floods and earthquakes accumulate. So, the original design estimates change over time as well, but they will always increase in magnitude and never reduce. As time goes on, the largest event to have been experienced can always be exceeded but can never be made smaller. In fact, the bulk of dam safety expenditures for most owners of conventional hydroelectric dams are devoted to improving spillways and foundations to accommodate these new and higher values. For tailings facilities under post-closure circumstances, this kind of upgrading will have to be performed perpetually. Without this it will be impossible to sustain the extreme event estimates that future knowledge provides.
[13, Vick,]

Cumulative damage
A related factor involves cumulative damage from repeated occurrences of extreme events, or progressive processes like internal erosion, that degrade dam stability over time. For earthquakes, conventional dam safety practice is to undertake repairs immediately after a damaging event. For tailings facilities the repairs may be physically impossible to accomplish. For conventional dams, draw-down of the reservoir can be required to repair major damage and is also an important emergency response. But a reservoir containing tailings solids cannot be reduced in level. Moreover, a tailings dam will experience repeated occurrences of extreme events during the indefinite future, their number depending on time and recurrence rate. For major earthquakes in some mining regions this is in the order of only hundreds of years. An example of the cumulative effects of seismic shaking is provided by La Villita Dam in Mexico, which has experienced progressively increasing crest settlements during four separate episodes of major seismic shaking in just 30 years. Cumulative damage also results from simple deterioration with age. No concrete structure - spillway, decant facility, or tunnel lining - lasts forever without continuing maintenance and repair.
[13, Vick,]

Climate change
The effects of long-term climate change are of intense interest and great uncertainty. Yet for a tailings dam to remain stable in perpetuity requires somehow that the influence of these changes on floods and spillway capacity be accurately predicted, something that even climate experts are not able to do. Climate change may also affect both physical and chemical stability in other ways. Frozen conditions are relied upon to reduce ARD reaction rates at some mines in arctic and subarctic regions, where certain tailings dams also depend for stability on the presence of frozen ground. And it goes without saying that permanent submergence requires sufficient water, even during sustained drought, notwithstanding any future changes in climate.
[13, Vick,]

Geologic hazards
While tailings dams are designed to accommodate geologic hazards known to exist at the time they are constructed, in the indefinite future they will eventually be subject to the full suite of geomorphic processes operating at their sites. These include the kind of landslides and rock avalanches notorious in the Alps, destructive debris flows and landslides characteristic of the Andes, volcanic activity in Central America and the Pacific Rim, soil creep common to New Guinea, and karst collapse in any number of areas. Like the occurrence of extreme events, the damaging effects of these processes is only a question of time and recurrence rate, a factor particularly difficult to predict for most large-scale geologic phenomena. Even more benign

processes of alluvial deposition will eventually fill water conveyance facilities unless they are continually cleared of sediment and debris.
[13, Vick,]

3.1.5.1.3 Biological stability

The biological stability of the closed site is closely related to its final land-use, whereas the stability of the surrounding environment will be primarily dependent upon the physical and chemical characteristics of the site. All three are linked because biological stability may significantly influence physical or chemical stability. For example, plant roots will inhibit erosion by binding the soil surface and the development of a healthy plant cover over a wetland treatment area will increase the surface depth of organic matter creating the anoxic conditions necessary for water treatment. The rehabilitation of most sites involves the revegetation of large areas of restored land, which can often be of a poor quality in terms of sustained plant growth. It is important, therefore, that the methods of amelioration and cultivation of the soils or soil forming materials, together with the species chosen will result in the development of a sustainable plant cover. This should be appropriate to the chosen land-use and may play an important part in maintaining the physical and chemical stability of the site, for instance by stabilising the soil cover and preventing erosion. Monitoring is aimed at demonstrating that plant growth has been successful in the first instance, but over a period of several growing seasons has developed into a self-sustaining plant community.
[100, Eriksson, 2002]

Conventional dam safety practice recognises the detrimental effects of burrowing animals and root penetration as matters to be addressed with continuing maintenance. Other problems may be more unexpected. As that country's national symbol, the beaver is ubiquitous to Canada, and its habits are well known to engineer and biologist alike. Its propensity to undertake its activities in response to the sound of running water has been acknowledged as a serious long-term closure issue for tailings dams through blockage of diversion facilities, and has been documented as a cause of tailings dam failure in the past. In Europe we should note that the European beaver, which became extinct in Sweden in the 1870's, was reintroduced in the 1920's and is now thriving successfully.

These factors show at a more detailed level the extent to which long-term dam safety depends on the need for continuing maintenance, modification, and repair, and conversely how difficult it is to assure stability in the long term.
[13, Vick,]

3.1.5.1.4 Successive land use

The successive use of a closed site is determined by the following factors:

- pre-mining or current land use surrounding the site
- any expected future changes in surrounding land use
- the reasonably expected post-operational use of the mine site
- viability of re-using site infrastructure and facilities
- the extent of any environmental impacts
- the need to safeguard against physical, chemical and biological hazards (both anthropogenic and naturally occurring).

From this there are a number of different options that are considered for most sites. These include the following:

- natural recolonisation of the site by local vegetation
- planting of commercial forestry plantations
- development for agriculture
- encouragement of alternative industrial activities
- use of infrastructure facilities as part of the commercial development in the region.

Whatever the final choice, the sites are usually rehabilitated so that the ultimate land use and morphology of the site is compatible with the surrounding area or with the pre-mining environment. This does not preclude maintaining the area as an industrial or commercial site if this is appropriate.

3.1.5.2 Specific closure issues

3.1.5.2.1 Heaps

The geometry and related stability of heaps is dependent on the type of material, the construction method and local topography.

Potential problems and hazards associated with heaps include:

- unstable slopes
- formation of toxic leachate leading to downstream contamination
- generation of ARD
- pollution of surface water and/or groundwater
- fires - spontaneous combustion
- injury to livestock, native fauna and the public
- dust pollution and wind erosion
- visual impact.

It is common practise to obtain full information on the geology and prior to closure. Should there be a risk of seismic activity or other natural of man-induced destabilising events, all measures and structures implemented are designed and constructed adequately.
[100, Eriksson, 2002]

3.1.5.2.2 Ponds

Slurried tailings are generally discharged into a containment site, a pond, where they are isolated from the surrounding environment thus preventing potential impacts on this environment. The impoundments are generally constructed using natural topography and dams within which the management of the tailings is controlled. In some cases, tailings are also discharged into natural water body such as a lake or sea.

Determination of the type of impoundment and the site for a specific site relies on the following factors:

- topography
- natural hazards
- local climate and water balance
- volume of tailings
- extent of tailings consolidation
- toxicity of tailings
- environmental concerns of the tailings and process water
- amount of suitable material for capping
- available topsoil
- economics.

Potential problems and hazards associated with tailings ponds include:

- unstable slopes leading to collapse or dam failure
- seepage or leakage of leachate leading to downstream contamination
- generation of ARD
- pollution of surface water and/or groundwater
- injury to livestock, native fauna and the public
- dust pollution and wind erosion.

It is common practise to obtain full information on the geology and prior to closure. Should there be a risk of seismic activity or other natural of man-induced destabilising events, all measures and structures implemented are designed and constructed adequately. A comprehensive report on the hydrology and geochemistry and the geotechnical aspects of the site is prepared.
[100, Eriksson, 2002]

3.1.5.2.3 Water management facilities

Water management facilities include all facilities at or associated with a mine site utilised to control, store, treat and convey water for the purposes of process and domestic use and the diversion discharge and treatment of excess water. These may include:

- ponds/dams
- reservoirs
- spillways
- intake structures
- diversion ditches
- culverts
- pipelines
- pump houses
- treatment plants
- settling ponds
- dewatering systems.

Potential problems and hazards associated with the closure of water management facilities include:

- contamination of surface water and/or groundwater
- uncontrolled water discharges leading to flooding, alteration of natural hydrological regime
- injury and/or death to livestock, native fauna and the public.

An inventory of all the equipment and facilities present at the site or in use for the purpose of handling and/or treating water arising from the site is usually compiled. The status of the above is documented and the location of all indicated on maps and site plans. Full information on the hydrological conditions and related mine workings is obtained prior to closure. Should there be a risk of seismic activity or other natural of man-induced destabilising events, all measures and structures implemented must be designed and constructed adequately.

Water management facilities are usually decommissioned and, where possible, removed from the site to prevent unacceptable levels of contaminated water from being discharged off site. It is good practice to remove those facilities requiring maintenance during the closure phase especially when safety stability and environmental impacts are at risk if neglected. The site decommissioning plans integrate any reusable components into the post-mining land use, the water management system and/or drainage pattern for the area.

Water management at a mine site is likely to have altered the natural hydrological regime. Water storage of impoundment facilities generally change the naturally occurring surface water and alter the flow rates and volumes moving through natural water channels. Re-watering of the natural hydrological regime involves the cessation of pumping from underground wells to allow flooding of the mine workings and the pumping to surface and treatment of this water until it no longer poses a threat to groundwater quality. A large portion of the exposed surface area in the abandoned underground workings may be pyritic and may be subject to oxidation prior to the initial flooding of the mine. Water may be used to flush the mine of impurities to reduce sulphates and metals to reduce risk of contamination. This continues until normal groundwater quality is restored.
[100, Eriksson, 2002]

3.1.5.2.4 Progressive rehabilitation

Progressive rehabilitation during operation has the following advantages:

- costs are spread over a longer period and may be recovered from mining revenues
- closure measure activities can be integrated into the daily operational activities of the mine
- a shorter closure implementation period will result
- monitoring programmes are integrated into routine environmental management
- successful techniques can be incorporated into the final closure plan
- adverse environmental effects are minimised.

Examples of progressive rehabilitation are:

- underground backfill
- rehabilitation of waste-rock dumps and strip mining
- ongoing vegetation of tailings dams
- backfill of open pit.

3.1.5.3 Applied methods for closure of tailings and waste-rock management facilities

The implemented decommissioning measures will depend completely on site-specific conditions and situation. Nonetheless, the above set criteria need to be fulfilled (physical, chemical, biological stability as well as the land-use criteria). This means that installation containing non-reactive tailings or waste-rock will not require any measure to ensure chemical stability but still have to fulfil the requirements on physical and biological stability as well as the criteria for land-use.
[100, Eriksson, 2002]

3.1.5.3.1 Non reactive tailings and waste-rock

Here needs to be elaborated a section on the basic measures to take to fulfil the requirements on physical and biological stability as well as the criteria for land-use. In this section measures like re-sloping (re-contouring or landscaping), re-vegetation, assure stability (re-sloping, lowering of dam height, dewatering, etc) should be discussed.

Please provide information.

Please provide definition of "reactive tailings".

3.1.5.3.2 Reactive tailings and waste-rock

This section is based on the State-of-the-art-report MiMi (1998) State-of-the-art-report on "Prevention and control of pollution from tailings and waste-rock products" [95, Elander, 1998]. Some additional case studies have been added. The entire report can be found in Annex 3.

There is a number of prevention and control options developed for potentially ARD generating mining waste, applicable for the operational as well as the closure phases of the mine life. As described in Section 2.4.2.2 three types of prevention and control measures can be distinguished:

- prevention of the generation of ARD
- control of contaminant migration; and
- collection and treatment of contaminated drainage.

Section 2.4.2.2 shows examples of ARD prevention methods and the principle on which their function is based.

Water covers
A water cover, or "wet cover" uses free water as a oxygen diffusion barrier. The oxygen diffusion coefficient is 10^4 times less in water than in air. This implies that if a water cover can be established the sulphide oxidation can be almost eliminated. The prerequisites for a water cover are:

- a positive water balance which can guarantee a minimum water depth at all times
- long-term physically stable dams (if not a pit, natural lake or sea has been used for the deposition of the tailings
- long-term stable outlets with sufficient discharge capacity even during extreme events
- a water depth within the pond deep enough to avoid re-suspension of tailings by wave action (break waters can be used to reduce the required water depth).

Furthermore, it is positive if there is a natural stream entering the pond, which can supply organic material, sediments flora and fauna to the decommissioned system. This will further improve the performance of the water cover (additional diffusion barrier added by the sediments) and speed-up the re-colonisation of the system.

Examples of sites where implemented are Stekenjokk and Kristineberg.

Stekenjokk constitutes a pioneer site within the area of decommissioning of tailings ponds containing sulphide tailings. The decommissioning was done in 1991 which allows for more than ten years of evaluation of the results. The Stekenjokk decommissioning project has been described in detail by Broman and Göransson (1994). The implemented measures at Stekenjokk are schematically described in the figure below (from Broman and Göransson, 1994).
[100, Eriksson, 2002]

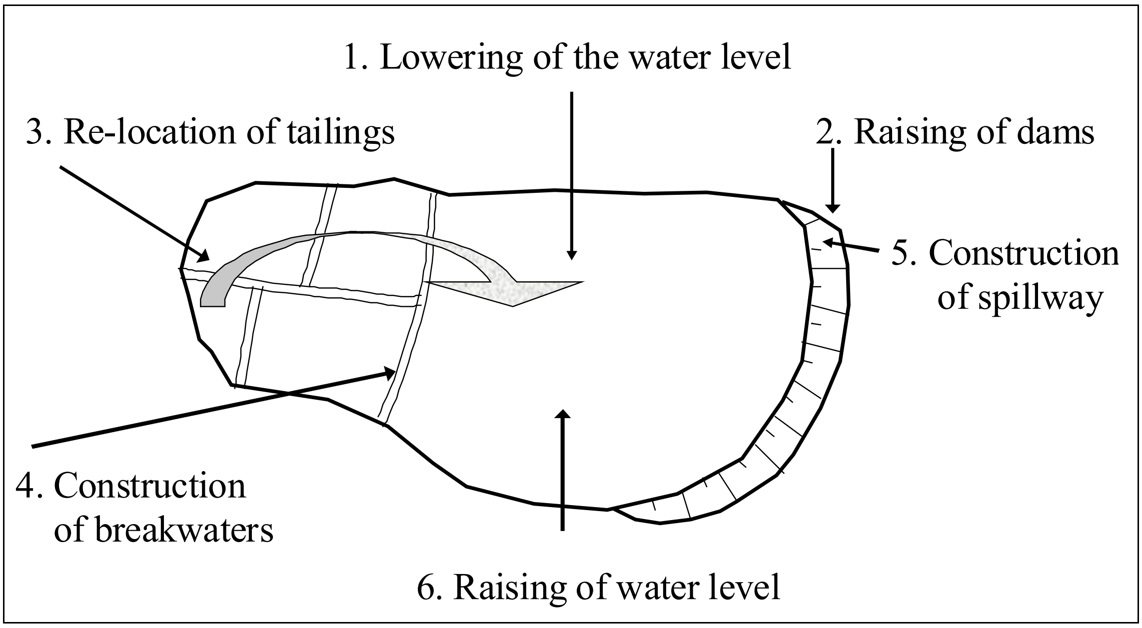

Figure 3.2: Implemented measures at Stekenjokk TMF
[100, Eriksson, 2002]

The results of the measures have been followed up. A mass balance calculation using data from the first 8 years of follow-up and assuming sulphate can be used as a tracer for sulphide oxidation has been presented. The analysis indicates that the water cover efficiently reduces the sulphide oxidation rate of the deposited tailings. Expressed as oxygen flux through the water cover to the tailings, the upper limit of the sulphate outflow of the pond correspond to an upper limit of the effective oxygen flux of 1 x 10^{-10} kg O_2/m^2s. This is comparative to, or better than, what is obtained in engineered composite dry cover solutions. Eight years of follow-up show that the objectives of the decommissioning project have been surpassed. Similar results have previously been reported from studies of tailings subaqueously deposited in natural lakes. The water cover is both efficient and cost effective compared to a solution including a dry cover. The implemented water covers had an investment cost of USD 2 /m^2 compared to studied dry covers USD 12 /m^2. Furthermore, no borrow pits needed to be opened for the extraction of cover material.

The uncertainty about water covers is to assure the long-term stability of the dams (if there are any at the particular site). It can be argued that it is not possible to completely eliminate sulphide oxidation since the water cover will always contain oxygen. However, the results indicate that the sulphide oxidation rate is negligible at Stekenjokk. A steady trend of falling sulphate concentration in the discharge of the pond has been observed. After 10 years the sulphate concentration in the pond effluent is close to background values.

The decommissioning at **Kristineberg** pond 4 is not yet completed but the measures taken are carefully followed up by the MiMi research project and are reported at www.mimi.kiruna.se.
[100, Eriksson, 2002]

<u>Wetland establishment</u>
Wetland establishment uses the same principle as the water cover but can manage with less water depth as the plant cover stabilises the bottom whereby re-suspension of tailings can be avoided. Less water in the pond reduces the potential risk in case of a dam failure. The prerequisites are the same as for water covers but with the additional requirement of adding organic matter to enhance the establishment of the wetland vegetation in the pond.

It should be noted that the principal idea of a wetland establishment is not the treatment of the water but the establishment of a self generating and sustainable cover that reduces the requirements for the water depth and that acts as a oxygen consuming barrier when organic matter is deposited on top of the water saturated tailings.

No designed sites are known. Examples of sites where wetlands are considered/planned to be implemented are Lisheen and Kristineberg.
[100, Eriksson, 2002]

Dry cover
The active part of a dry cover is the low permeable layer with high water content which acts as an oxygen diffusion barrier (sealing layer). The sealing layer is protected from drying out and from mechanical destruction through the application of a protection layer. The protection layer is vegetated.

The short-term efficiency of a dry cover can decrease in the long term as a consequence of different destructive processes that may cause cracks or other discontinuities in barrier layers. Such processes are erosion, frost action, drying, differential settlements, root penetration, digging animals and man-made intrusion [95, Elander, 1998].

Examples of sites where dry covers have been implemented are Apirsa (Aznalcollar), Aitik, Saxberget, Kristineberg, Enåsen.

Decommissioning of tailings ponds at the Saxberget mine in central Sweden, which were decommissioned between 1994 - 1996 using a composite dry cover, has been described. Two separate ponds had been used for different periods, the West pond for the period 1930 - 1958, and the East pond for 1958 - 1988. The West pond occupies an area of 18 ha, while the East pond is twice the size, 35 hectares. In total, the tailings amount to 4 Mt, with a composition of about 2 % S, less than 1 % Zn and 0.5 - 1 % calcite. This mineral composition suggests that the material is potentially acid-generating even though the tailings in the East pond produce a circum-neutral pH drainage at present.

The ponds are located on a permeable glacial formation, which was predicted to cause a rapid fall of the ground water table as soon as the supply of tailings slurry ceased. Large amounts of tailings would then be exposed to the oxygen in the atmosphere. During the production period, the mobilisation of zinc was estimated at 3 tonnes per year. Studies showed that after depletion of readily available buffering minerals, the pollution load would increase considerably if the oxygen supply to the material could not be controlled.

Modelling of the future mobilisation of metal indicated an annual mobilisation of up to 600 tonnes of zinc in the ponds. Due to precipitation and adsorption processes at neutral pH levels, the amount was estimated to stay at 3 tonnes per year as a net transport for several years to come. However, the predicted high future pollution load called for remedial actions. As the hydrogeological situation excluded flooding of the ponds, the only realistic option remaining was a cover designed to reduce the oxygen transport to the tailings.

As the proposed project would be the second one of its kind in Sweden, and certainly the largest, there was no real practical experience at the time of developing the remediation plans. Therefore, a number of options had to be investigated. In general terms, the cover was designed in accordance with principles defined within the Swedish EPA:s investigation programme aiming at long-term, low maintenance remediation solutions for mining waste. This called for a cover with at least two components, one low permeability sealing layer, and one protective layer on top of the sealing layer.

The tailings were covered with 0.3 m compacted clayey till as a sealing layer and 1.5 m unsorted till as a protection layer. The protection layer is vegetated by grass and birch.

The key component was the sealing layer. For this purpose, a number of solutions were considered. One of them was compacted municipal sewage sludge, which was found to possess favourable hydraulic properties. For practical reasons, mainly the time factor, this alternative was rejected.

Another option was the use of fly ash from power stations in the form of “cefyll”, a concrete-like product which had been investigated, and used, in a similar project. The major drawback for this alternative was the cost, as the source for the fly ash - coal-fired power and thermal plants in the Stockholm region - was too distant.

Investigations of glacial till occurrences in the area showed large amounts of clayey till close to the mining area. As this material was found to have excellent hydraulic properties, and the cost was the lowest of all alternatives, this became the sealing material selected.

Modelling of oxygen and water transport coupled to solubility calculations yielded figures for metal transport. Based on these calculations, the specifications for the permeability of the sealing layer were established; 0.3 m with a permeability of 5×10^{-9} m/s.

The extent of the protective layer was subject to discussion. The mining company claimed that 1 m of unclassified till would constitute sufficient protection against frost and root penetration. The EPA argued in favour of a thicker cover, and finally it was agreed that a 1.5 m protective layer should be used.

The layout of the tailings area was designed to adjust to the surrounding landscape as much as possible. Surface run-off water is led in a small stream winding along the West pond. The drainage from the West pond overflows to the East pond and forms large areas of shallow wetlands. In this way water saturation is maintained in the sealing layer, and it gives the area an attractive and varied appearance. Excess water is discharged through a stone paved outlet down the former dam slope.

Follow-up results show a positive trend in the development of the contaminant load from the area. It is, however, too early to draw any conclusive results about the performance of the cover. [100, Eriksson, 2002]

<u>Oxygen consuming cover</u>
An oxygen consuming cover uses a low permeable layer with high water content as oxygen diffusion barrier. The low permeable layer and possibly also the protective layer has a high content of organic matter which, when degrading consumes oxygen and thereby reduces oxygen transport to the underlying sulphides. The access to large quantities of suitable organic matter is a prerequisite for this method to be viable.

Example of sites where implemented are Galgberget (Central Sweden) and Garpenberg (Central Sweden)

[95, Elander, 1998] describes the decommissioning of the Galgberget tailings pond using an oxygen consuming cover in the following manner (for more details see Annex 3):

At Galgbergsmagasinet, a tailings pond in Falun, Sweden, a cover with a high content of organic material was constructed from paper mill sludge, fly ash and wood waste. On the top of the tailings pond a totally 1 m thick layer of fly ash mixed with paper mill sludge was laid out and compacted in two layers and thereafter covered with a 0.5 m layer of wood waste and coarse till. This cover is believed to form an effective barrier against oxygen transport partly due to consumption of oxygen in the cover and partly due to a physical barrier effect in the compacted low permeable mixture of fly ash and paper mill sludge. The hydraulic conductivity of the mixture was measured in the laboratory at $\leq 5 \times 10^{-9}$ m/s and the water retention capacity was measured and considered satisfactory to maintain a high degree of saturation in the barrier. Other possible positive effects are inhibiting of the acidophilic leaching bacteria due to the high content of calcium hydroxide in the fly ash that will raise the pH in the percolating water, and the formation of a sustainable environment for sulphate-reducing bacteria producing hydrogen

sulphide that precipitates metals. However, there is also a risk that the combination of organic compounds and iron hydroxides in the upper (oxidised) part of the deposit could produce bacterial iron reduction that would dissolve co-precipitated heavy metals. The ongoing follow-up indicates that the oxidation of sulphides has decreased and that the pH at the site is higher than at the reference site. No evidence of any significant bacterial sulphate reduction has yet been noticed.
[100, Eriksson, 2002]

Raised groundwater table
An intermediate solution for water saturation without creating open ponds is the saturation of the actual deposit by raising the phreatic level. This method is being practised in two ponds in Kristineberg, both containing strongly weathered material. As the material is entirely water saturated, further oxidation is inhibited. This is accomplished without the complex of problems connected to flooding. The basis for such a measure is careful groundwater modelling, taking into account the influence from surface water management and groundwater raising dams.\

The benefit of the method is, apart from the reduced thickness of the cover, the lack of need for compaction of the cover and the drastically reduced quality requirements on the cover material.

Example of a site where this method has been implemented is Kristineberg.
[100, Eriksson, 2002]

Addition of buffering material
Addition of buffering materials (e.g. limestone) is normally practised before applying a dry cover. This helps to immobilise and weathering products readily available at the time of the decommissioning of the site.

It is also a theoretically possible solution a decommissioning method as an addition of enough buffering material would delay or even eliminate a drop in pH and the production of ARD. However, to accomplish such a long-term buffering effect in a potentially ARD generating deposit normally requires large amounts of buffering materials which need to be brought in to the site at prohibitively high cost.
[100, Eriksson, 2002]

Selective material handling
Selective material handling needs to be applied during operation in order to be effective. By selectively depositing reactive and non-reactive tailings or waste-rock the decommissioning of the non-reactive part can be significantly reduced. It might even be possible to find alternative use for the non-reactive fraction.

Example of a site where this method has been implemented is Aitik (see Section 3.2.4.2).
[100, Eriksson, 2002]

De-pyritisation
Similar to selective material handling, but done as part of the mineral processing in the mineral processing plant. Pyrite can be separated by flotation and handled separately. This method can be suitable if the ARD potential of the bulk amount of tailings can be altered significantly (i.e. converted from ARD generating to non-ARD generating) by lowering the pyrite content.

3.1.6 Liners

There is a trend to apply liners to tailings management facilities. This section aims at describing the liner alternatives and the pros of cons of liners.

Liners systems are designed to restrict seepage of leachate through the base of the tailings storage area. All liner systems have a leakage rate which will depend on:

- the magnitude of the hydraulic head above the liner
- the thickness and effectiveness of the liner material
- the length of time the hydraulic head is applied to the liner.

It is important to be aware of the hydro-geological background and the geochemical features of the tailings to be managed [11, EPA, 1995].

The use of liners is often debated subject. The advantage is the possibly high reduction of seepage. However, critics say that it is not predictable for how long the liner will function properly. The alternative is to handle seepage from the commencement of use.

However, as mentioned in the previous section, measures to restrict infiltration into the deposit may be preferred over low-permeability bottom **liners** with accompanying hydraulic gradients that promote contaminant transport (the so-called “bathtub” effect) [13, Vick,].

A long experience in stacking potash tailings is necessary to be able to apply the appropriate tailings management methods. As an example, the use of clay liners underneath the heap can result in stability problems. For the extention of the heap in the Fulda area, the authorities demanded an impermeabilisation of the ground with an artificial clay liner of 0.6 m. As the heap expanded over this sealed ground, rapid movement of that part of the heap on top of the clay liner was observed to an extent that the safety of the employees on top and in front of the heap was threatened and operation had to stop. An investigation concluded that any material with low shearing strength must not be used for the sealing of potash tailings heaps.
[19, K&S, 2002]

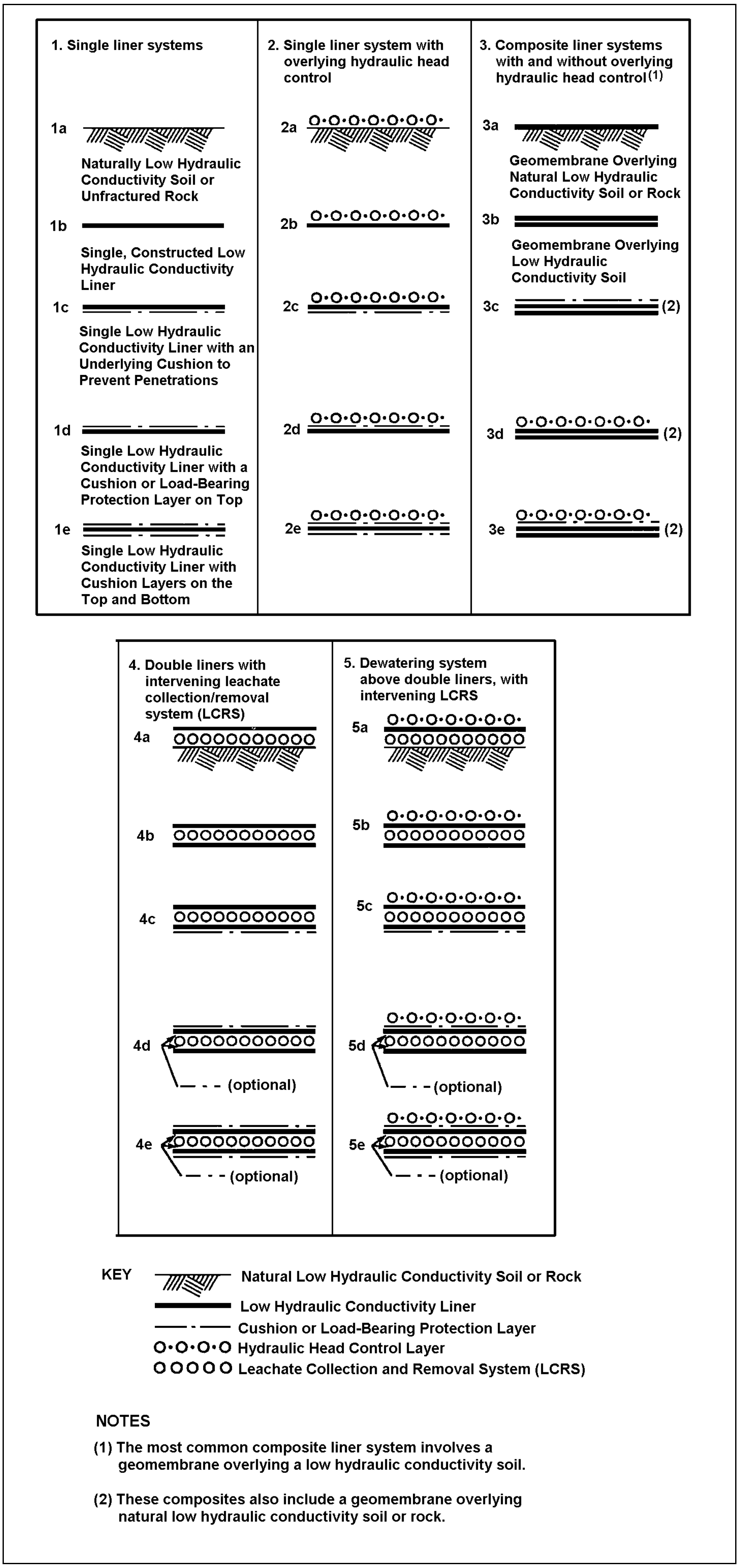

Figure 3.3: Types of liner systems potentially available to tailings
[11, EPA, 1995]

3.2 Metals: Mineral processing, tailings and waste-rock management

3.2.1 Mineralogy and Mining techniques

3.2.1.1 Aluminium

The Bauxite deposits of **central Greece** are red lenticular bodies in the form of three bauxite horizons. Vanadium, manganese, nickel, cobalt, chromium, zinc, copper, phosphorus and sulphides can be found in the ore at low levels or as trace elements. The amounts of ore have been equally mined from underground and open pit mines until recently. The tendency for the future is towards more underground mining because of the increase in the stripping ratio, and the emerging environmental aspects connected to open pits.
[90, Peppas, 2002]

In the underground mine room and pillar mining is applied, sometimes in combination with cut and fill if the ore body is thicker than 8 m. Ore bodies with a stripping ratio of 6 - 8 m^3/t are mined in open pits via conventional drilling, blasting and loading [90, Peppas, 2002].

The **Sardinian alumina refinery** receives its ore from the Australian Weipa mine. Compared tp European ores this ore is very soft and is mined by large ripping front-end loaders [33, **EURALLUMINA**, 2002].

In the Hungarian **Bakony region** six bauxite mines are in operation, which all send their bauxite to the **refinery in Ajka**. The bauxite is of the kaarst type in the form of lenticular or pod-like deposits. Mining is undertaken by open pit (drilling/blasting/loading) at a stripping ratio of 6.3 m^3/t or underground using sublevel caving [91, Foldessy, 2002].

The following table shows the chemical composition of bauxite processed in European refineries.

Area:	**Central Greece**			**Weipa**		**Bakony**
Ore type:	**Upper horizon**	**Middle horizon**	**Lower horizon**	**Grade A**	**Grade D**	
	%	%	%	%	%	%
AL_2O_3	55 - 60	50 - 55	45 - 50	54	57	45 - 52
SiO_2	2 - 5	6 - 12	15 - 25	4	7	5 - 10
Fe_2O_3	16-22	18 - 22	10 - 20	15	67	18 - 22
CaO	0.3 - 1.2	0.2 - 1.0	0.2 - 1.0			0.4 - 1
TiO_2	2 - 4	2 - 4	2 - 4	3	2	0 - 3
LOI[1]				25	27	16
Moisture				12	10	16 - 19
1. Loss on ignition						

Table 3.4: Chemical compositions of the bauxites from different deposits
[90, Peppas, 2002], [33, EURALLUMINA, 2002], [91, Foldessy, 2002]

3.2.1.2 Base Metals (cadmium, copper, lead, nickel, tin, zinc)

Mining of primary sulphide ores dominates the base metal mining for Cu, Zn and Pb in Europe (Almagrera, Asturiana de Zinc, Pyhäsalmi, Aitik, Garpenberg, Boliden, Lisheen, Tara). The sulphur content varies significantly between the sites as does the grade of the value mineral.

Some examples of different mineralogies are descbribed below.

- The ore at **Pyhäsalmi** is massive and coarse grained. The ore contains 75 % sulphides on average; 3 % chalcopyrite, 4 % sphalerite, 2 % pyrrhotite and 66 % pyrite plus minor amounts of galena and sulphosalts. Barytes and carbonates are the main gangue minerals [62, Base metals group, 2002].

- At **Hitura** nickel mine the ultramafic complex consists of three separate, closely-spaced serpentinite massives surrounded by magmatised mica gneiss. The main ore minerals are pentlandite, chalcopyrite and pyrrhotite, but in some places mackinawite, cubanite and vallerite are abundant. Pyrite occurs only in joints with carbonates [62, Base metals group, 2002].

- An other example is **Aitik**, where the open pit mining area is divided into footwall, main ore zone and hanging wall. The contact between the main ore zone and the hanging wall is sharp, being a thrustfault. The contact between the footwall and the ore zone is gradual and grade dependent. The main ore minerals are chalcopyrite, pyrite and pyrrhotite, occurring as dissemination and veinlets. The footwall consists of biotite-amphibole gneiss and intrusions of quartz-monzodiorite (the footwall has lower than 0.26 percent copper.) The main ore zone comprises biotite schist/gneiss and muscovite schist. The hanging wall consists of amphibole-biotite gneiss and pegmatite and is barren in copper. The value mineral in the orebody is chalcopyrite. The mean copper concentration in the ore is 0.4 % and the cut off grade 0.16 %. Further more the ore contains small amounts of gold (0.2 g/t) and silver (3.5 g/t) [63, Base metals group, 2002].

- At **Lisheen** the sulphide mineralisation that forms the orebody occurs at the base of dolomitic limestone. The metalliferous minerals are pyrite, marcasite, sphalerite and galena, but also, in smaller concentrations chalcopyrite, tennatite, native silver, arsenopyrite and gersdorffite. The gangue material are dolomite together with barytes, calcite, shale, illite and quartz [75, Minorco Lisheen/Ivernia West, 1995].

- At the **Las Cruces Project**, which is currently in the planning and permitting phase the value mineral is chalcocite, a secondary sulphide copper mineral, in massive pyrite [67, Base metals group, 2002].

Both underground mines and open pits are represented in the base metal mining sector in Europe. The used mining methods underground are room-and-pillar and various other techniques using backfilling. The ore production capacity in the underground mines is between 65000 and 1100000 tonnes/yr. In open pit mining the production (ore and waste-rock) in 2001 was between 1200000 and 43700000 tonnes. In underground mining almost all waste-rock produced is directly used as backfill in the mine. In some cases, waste-rock was extracted from existing waste-rock dumps and transported under ground. In open pit mining, backfilling was not possible in the majority of the cases, however, at Asturiana de Zinc a mined out part of an open pit was backfilled using waste-rock. The various mines and the mining techniques they apply as well as their ore and waste-rock production are listed in the table below.

Mining area	Mine	Mining method	Ore production	Waste-rock deposition
Almagera	Aguas Teñidas	Underground (cut and fill)	300000	0[1]
	Sotiel	Underground	700000	0
Asturiana de Zinc	Mina Reocín	Open pit/Underground	1100000	2500000[2]
Pyhäsalmi	Pyhäsalmi	Underground (cut and fill)	1097173	0[3]
	Mullikkoräme	Underground	64000	0
Hitura	Hitura Mine	Underground (cut and fill)	518331	0[3]
Zinkgruvan	Zinkgruvan	Underground (cut and fill)	850000	0[4]
Aitik	Aitik Mine	Open pit	17700000	26000000[4]
Garpenberg	Garpenberg Mine	Underground (cut-and-fill)	310000	0
	Garpenberg Norra	Underground (cut and fill)	709000	38400[5]
Boliden Mining Area	Maurliden	Open pit	224400	875700
	Renström	Underground (cut and fill)	160500	-104000[*]
	Petiknäs	Underground (cut and fill)	553000	-15700[*]
	Åkerberg	Underground	32000	-21000[*]
	Kristineberg	Underground (cut and fill)	503600	4600[3]
Lisheen	Lisheen	Underground (cut and fill)	1110000[6]	7000
Tara	Tara	Underground (blasthole open stoping)[7]	2000000[7]	

1. Waste-rock used in back-fill + schists from borrow area
2. Waste-rock used to fill out mined out open pit.
3. Waste-rock used in back-fill
4. 65 % deposited separately for alternative use
5. Used for dam construction
6. Source: [76, Irish EPA, 2001]
7. Source: [74, Outokumpu,]

[*]: A negative number indicates that waste-rock has been removed from existing deposits and brought underground for backfilling purposes.

Table 3.5: Information on mining technique, ore and waste-rock production of base metal mines Year 2000 figures for Almagrera, Asturiana de zinc, Pyhäsalmi and Hitura; year 2001 figures for the Aitik, Garpenberg and Boliden mining areas.

The Aitik site is a typical example of open pit mining:

*Drilling:*The drilling equipment consists of rotary drill rigs. The bench height is 15 m and subdrilling 3 m. The drilled burden and spacing are 8 m x 10.5 m. The diameter of the drillholes are around 300 mm. The rate of drilling is normally about 17 m/h, but in the hard parts of the ore it can be less than 10 m/h. Water pumped from the open pit varies between 3 - 15 m^3/min.

Charging and blasting: Emulsion explosive is pumped from a truck into the blast holes. Non-electric detonators are used for the initiation of the blast. The size of each round is about 600 kt and blasting takes place once a week. The benches are planned with a final pit slope angle of 47° in the footwall (following the foliation) and 51 - 56° in the hanging wall.

Loading and transportation: In Aitik 5 shovels are used. Three rope shovels and two hydraulic shovels. A wheel loader completes the loading fleet. The haulage is carried out by 17 trucks (172 t and 218 t trucks).

In-pit crushing: The ore is transported by trucks to the primary crushers in the pit, 165 m below the surface. From bins below the crusher the ore is loaded on a conveyor belt on 195 m level which takes the ore to the mineral processing plant. The inclination of the conveyor is 15°, the width 1800 mm and the capacity 4000 t/h. The total stockpile capacity on surface is around 50000 t. Two gyratory crushers are used for primary crushing. The intake opening of the crusher is 152 cm and the diameter of the inner surface at the bottom is 277 cm. The fragmentation of the crushed ore is depending on the setting of the crusher but normally the width is set to 160 - 180 mm. The largest pieces are thus between 350 and 400 mm but variations occur caused by different ore characteristics. Each day 40000 to 60000 tonnes are crushed.
[63, Base metals group, 2002]

Both **Garpenberg** and **Garpenberg Norra** are underground mines. Their technique is described here as examples for base metal underground mining.

The used mining method is cut-and-fill. The coarse fraction of the tailings is used as backfill and as a platform when mining the ore above. At present mining is done at a depth of between 400 and 870 m in the Garpenberg mine and in Garpenberg Norra between 700 and 990 m.

Blasting is done using emulsion explosives. Loading and hauling is carried out using diesel vehicles. The ore is crushed with an in-pit crusher before it is skipped through a shaft to the surface. A covered 500 m long conveyor belt transports the ore from the Garpenberg mine to the mineral processing plant. From the Garpenberg Norra mine the ore has to be trucked approximately 2 km to the mineral processing plant.
[64, Base metals group, 2002]

3.2.1.3 Chromium

The chromium ores at Kemi are associated with a mafic-ultramafic layered intrusion within the contact between migmatite granite and schist. The formation starts at the town of Kemi and extends approximately 15 km NE, with a maximum width of 1500 m. The compact chromite-rich horizon appears 50 - 200 m above the bottom of the formation. The thickness of the continuous chromite horizon varies from a few millimetres to a couple of meters, but in the Nuottijärvi-Elijärvi area, the chromite layer contains eight consecutive veins, which are economically viable over a distance of 4.5 km. Both host and waste-rock are serpentinite and talc-carbonate rock. Idiomorfic chromite is the only ore mineral appearing in economic quantities. The average content of the ore is 26 % Cr_2O_3 and Cr/Fe ratio 1.55.

The Kemi chromium mine is an open pit mine with a waste-rock to ore ratio of 5.5:1. The mine production in 1999 was about 250000 tonnes.
[71, Outokumpu, 2002]

3.2.1.4 Iron

The mining of iron ores, mainly hematites and to a lesser extent magnetites, occur predominantly in open pits. Magnetite mines of significant size are situated in the USA, ex USSR and Sweden. LKAB of Sweden also has the two remaining iron ore underground mines Europe.

Mining operations normally consist of preparation including stripping or drifting, drilling, blasting and transportation prior to processing.
[49, Iron group, 2002]

3.2.1.4.1 Underground mines

The magnetite ore body in **Kiruna mine** is about four kilometres long, has an average width of 80 m and extends to an estimated depth of around two kilometres at an incline of roughly 60˚. The main haulage level is at a depth of 1045 metres. Mining of the ore body between the 1045 m and 775 m levels will continue until about the year 2018. To date, about 940 Mt of ore have been extracted from the Kiruna ore body. Approximately 20 - 23 Mt crude ore is mined every year from the ore, sending approximately 5 Mt to the coarse tailings facility and 1.7 Mt to the fine tailings facility.

The ore body is divided into 10 blocks. Each block has its own group of shafts, each consisting of four shafts, except for the two northernmost blocks (the Lake Ore), which have three. In total, the Kiruna mine has 38 such shafts. Each shaft in a group is about 30 metres from the next. The 10 mining blocks are accessed via five separate ramps. An extension of each ramp is cut into the two neighbouring blocks on one side. By linking the blocks in this way, five smaller "mines" are created. Each block has its own air intake and exhaust shafts. The geographic division of the orebody into five mines enables greater mining efficiency. Since the mines are well separated from each other, ore can be extracted from one mine while blasting or maintenance are taking place in another. The mining operation passed the 775 m level in the summer of 1999. Mining will take place above the 1045 m level until the year 2018. The ore body between 775 m and 1045 m is divided horizontally into nine slices, each of which is 27.5 metres high. The distance between ore passes is 25 meters. Each blast brings down about 10000 tonnes of ore.
[49, Iron group, 2002]

The **Malmberget mine** consists of about 20 ore bodies, 10 of which are currently being mined. Most of the ore base is magnetite, but there are also occurrences of non-magnetic hematite ore. Malmberget's newest main haulage level is at a depth of 1000 m. Up until now, about 350 Mt have been extracted from the ore bodies. About 12 Mt crude ore are mined from the ore bodies every year, of which 5.6 Mt of tailings are generated.

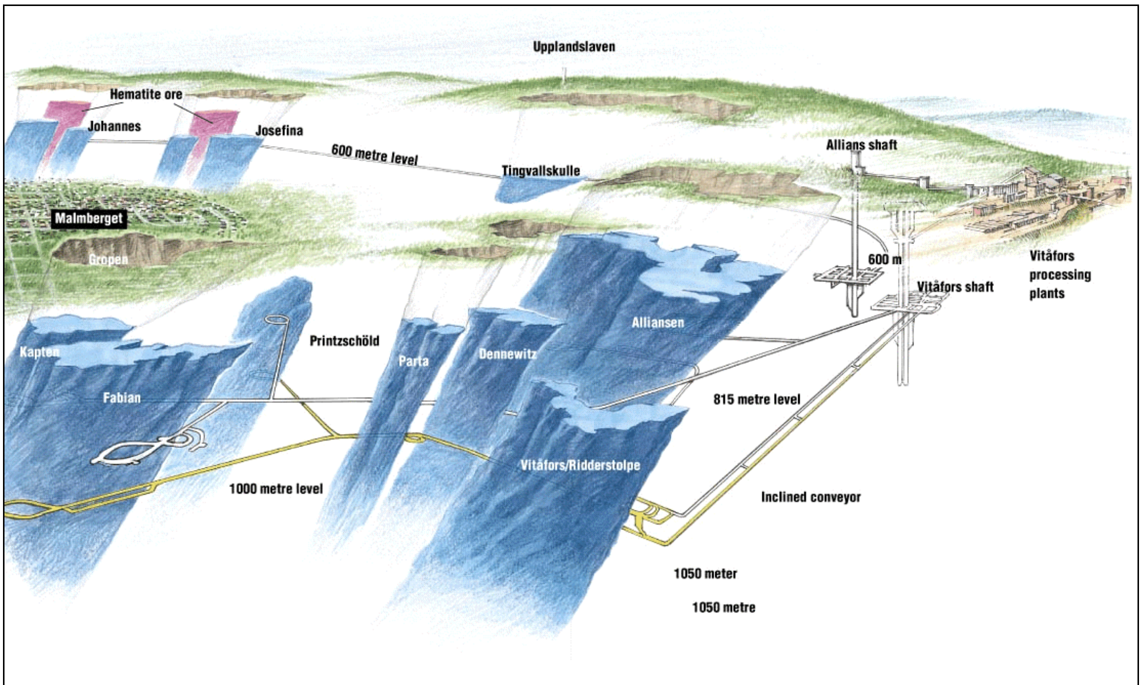

Figure 3.4: Illustration of the Malmberget ore deposit
[49, Iron group, 2002]

The ore field is 4.5 km in an E – W and 2.5 km in a N – S extension. In the western part of the mine, the ores form more or less continuous undulating bands of lens-shaped ore bodies. The ores in the eastern part of the mine exhibit a more complicated, intensively folded, tectonical structure. The ore bodies are steep with great local variations. The thickness of the ore bodies varies between 20 and 100 metres. The host rock is acidic to intermediate highly deformed and metamorphosed volcanic rocks, now appearing as ”leptites” (fine-grained feldspar-quartz rocks) and gneisses. The ore field is generally metamorphosed to lower amphibolite faces. In the western part of the ore field, local higher metamorphosed grade occurs.

Both Swedish mines use large-scale sub-level caving as their mining method.

Preparation/Development

At **Kiruna** as a first step development drifts are driven. A drift is a dead-end tunnel that goes straight into the rock. Preparation, or development, involves building new sections of the mine from which ore can be extracted. A development drift passes straight through the ore body. Drilling is done with electric- powered hydraulic drill rigs. Rounds of up to 60 holes, each five metres deep, are drilled. These holes are then charged with explosive and blasted. Rounds are blasted during the night. Ore from these blasts is loaded out with wheeled loaders. Then, the next round is drilled, etc., until the entire development drift is ready. Drifts can be up to 80 metres long. If necessary, the walls and roofs are reinforced with rock bolts and/or gunned with concrete. Once the development work is done, or when a number of crosscuts in the same area are ready, the next step starts in the production chain, and that is production drilling and blasting.

Production

When a number of development drifts have been cut, production drilling of a 'slice' can begin. The slice is 27.5 metres high. This is done with remote-controlled production drill rigs. The operators control several rigs in the production area by remote control from control rooms. The rig drills upwards into the ore, forming fan-shaped patterns with ten holes each. The holes are normally 40 - 45 metres long, and straight, so that subsequent loading with explosives and blasting can be done efficiently. When a pattern of holes has been drilled, the rig is moved back three metres, then drilling of the next pattern begins. About 20 of these patterns will be drilled in an 80-metre long drift. Once this is completed, loading of the holes can begin.

A robot injects explosives into the drill holes in one pattern. Blasting is done every night. Each round brings down about 10000 tonnes of ore. When the blast has been ventilated, loading with wheeled loaders (LHD) can begin. Then, the next pattern is charged, etc. The procedure is repeated until the entire ore pass has been mined out. Electric wheeled loaders load and carry the ore to vertical shafts (ore passes), located along the ore body. Each loader carries a bucket payload of 17 - 25 tonnes and tips its load to an ore pass. By gravity the ore falls down to bins, located just above the main level.

In the Kiruna Mine there are also electric loaders which are remote controlled. The operator sits in front of a monitor, in a control room, and 'drives' the machines in the production area. The machines navigate with the help of rotating lasers and reflectors on the walls of the drifts. Information, e.g., the position of the machine, is sent via a number of wireless base stations to the control system in the control room computer.

The main haulage level in the Kiruna Mine is at the 1045 m level. Ore is tapped via remote control from the bins into railway cars. A driverless train, consisting of an engine and 24 cars, carries the ore to one of four discharge stations. When the train passes the station, the bottoms of the cars open and the ore falls down into a crusher bin from which it is fed to one of four crushers. The ore is crushed into lumps of about 100mm in diameter. Nine locomotives and about 185 cars are operated on the main level. Each train carries about 500 tonnes of ore.

Mining in **Malmberget** takes place at several different levels, since there are many ore bodies. The main haulage levels are at 600, 815 and 1000 m. There are crushers at each level. Twelve large mine trucks, with payload capacities of 70 to 120 tonnes, are operated at these levels. The trucks are driven to vertical shafts. Drivers control loading from inside the cab of the truck. The

fully-loaded truck is then driven to a discharge station and the ore is emptied, sideways, into a crusher bin. This is also controlled from the cab of the truck. The ore is fed into the crusher and crushed into lumps of about 100 mm in diameter.
[49, Iron group, 2002]

3.2.1.4.2 Open pit mines

The valuable mineral at **Erzberg** is the iron-mineral sideroplesite and the gangue mineral is ankerite "Steirischer Erzberg"-deposit. The iron content of the wholly liberated sideroplesit is 42 %, however this occurs only below 100µm.

The Steirischer Erzberg is an open pit mine with a yearly production of 3.8 Mt/yr of which 1.2 Mt is waste-rock. Conventional drilling and blasting (AN-pump slurry) are used. Transportation is done with wheel loaders and trucks. Within the pit 20 benches with an average height of 24 m are in operation.
[55, Iron group, 2002]

3.2.1.5 Manganese

No data has been supplied for this section. Please provide information.

3.2.1.6 Mercury

No data has been supplied for this section. Please provide information.

3.2.1.7 Precious Metals (Gold, Silver)

Mineralogy
Various geological settings and mineralogical characteristics are represented in the precious metals sites
:

- complex sulphide ores where Cu, Zn and Pb are complementary or even the main value minerals (Boliden)
- sulphide mineralisation comprised of pyrite, arsenopyrite, galena and sphalerite where the contained gold is submicroscopic (<1 µm), finely disseminated in the pyrite and arsenopyrite lattices (refractory gold) (Olympias Gold)
- low sulphidation epithermal quartz and breccia veins in andestic host rock (Ovacik Gold Mine)
- strongly altered volcanics: quartz, sericite and andalusite rich rocks or schists (Orivesi)
- native gold with copper sulphides in skarn and brecha jasperizadas (Río Narcea)
- gossum (Filón Sur).

The differing mineralogy requires different mining and mineral processing techniques to obtain optimum gold recovery.

Mining techniques
Various mining techniques are represented, both underground (with and without back-filling) and open pit mining are used. The open pits are in two cases planned to become underground mines with time. There are several examples where gold is extracted from a tailings stream off a base metal mineral processing plant or from old waste-rock dumps and tailings ponds.

3.2.1.8 Tungsten

The host rock of the Mittersill deposit consists of quartz lenses, laminated quartzites, pyroxenites, orthogneisses, amphibolites, hornblendites and granites. The tungsten bearing mineral at Mittersill is scheelite ($CaWO_4$). The main gangue minerals are quartz, silicates (mica, talk, biotite, hornblende, amphibole, pyroxene, etc.), carbonates, apatite and sulphides. The content of sulphide minerals is < 0.5 %. The most frequent sulphide mineral is pyrrhotite. Less frequent are pyrite, chalcopyrite, galena and molybdenite.

In 1975, the mining operation in Mittersill started with an open pit operation. In 1979, the underground deposit was developed. The open pit operation ceased in 1986. Today 450000 tonnes of ore are mined yearly in the underground mine with an average WO_3-content of 0.50 %.

The whole mining operation in Mittersill is situated in a protected landscape. Therefore all the social facilities, workshops and warehouses of the are installed underground. The ore is crushed underground. The mine and the mineral processing plant are connected by a 3 km long gallery. The ore is transported from the crushing station to the mineral processing plant by a conveyor belt system.

The main mining methods used for the extraction of the massive orebody are:

- sublevel stoping
- sublevel caving
- cut and fill.

The waste-rock which is mined during development of the orebody is dumped into open stopes underground. There are no waste-rock dumps on the surface. Besides that, tailings from the mineral processing plant are pumped to the mine for refilling open stopes.

3.2.2 Mineral processing

3.2.2.1 Aluminium

As mentioned in Section 2.3.2.1 the Bayer process is used to treat bauxite: it is applied in all alumina refineries in Europe.

The Bayer process is based on a continuous recirculation of caustic solution, which plays the role of dissolving agent for the hydrate-alumina within the bauxite as well as transport medium carrying all the solids through the various process stages. In the first stage, the bauxite goes through a wet grinding stage, resulting in a slurry with 50 % solids. This is preheated to 100 ºC and held in holding tanks to make the silica more reactive. Caustic liquor returning from the previous cycle is then re-concentrated and heated up. At the subsequent leaching (or digestion) phase the bauxite slurry is mixed with the caustic liquor at high temperature (250˚C). Gibbsite and Boehmite rapidly dissolve, leaving the inert part of the bauxite (the red mud) undissolved.

Clarification of the pregnant liquor is carried out by thickeners and filtration. The mud is separated in 2 steps. First, so-called sands (particles over 150µm) are removed by cycloning the liquor and separating the solids in screw-classifiers. The mud is then settled in large thickeners.

The clarified pregnant liquor is then pumped to the precipitation phase to produce solid hydrate. The hydrate is then calcined to produce alumina. The liquor is strengthened with fresh soda make-up and returned to another cycle of bauxite attack.

The separated mud is extracted from the decanters‘ cone at around 30 % solids, and pumped to a continuous 3 or 4 stages countercurrent washing unit. This consists of 4 washers in series where most of the caustic liquor accompanying the mud is recovered.

3.2.2.2 Base Metals (Cadmium, Copper, Lead, Nickel, Tin, Zinc)

In the processing of the primary sulphide ores all plants use similar processing techniques, namely:

- crushing
- grinding
- flotation
- drying of concentrates.

Flotation can be done in various ways, e.g., selective flotation or bulk/selective flotation, depending on the characteristics of the ore, the market demands, cost of flotation additives, etc. Two possible options for the same mineral processing plant are illustrated in the figures below for the Zinkgruvan mineral processing plant.

The **Zinkgruvan** mineral processing plant, which was constructed in 1977, is located next to the mine. It operates continuously with an annual throughput of 850000 tonnes. The choice of process and technology is based on a large number of test works with the actual zinc and lead ore. Autogenous grinding in combination with bulk/selective flotation (see Figure 3.5) of the ore has been chosen as the main process technique and has been used at Zinkgruvan since 1977.

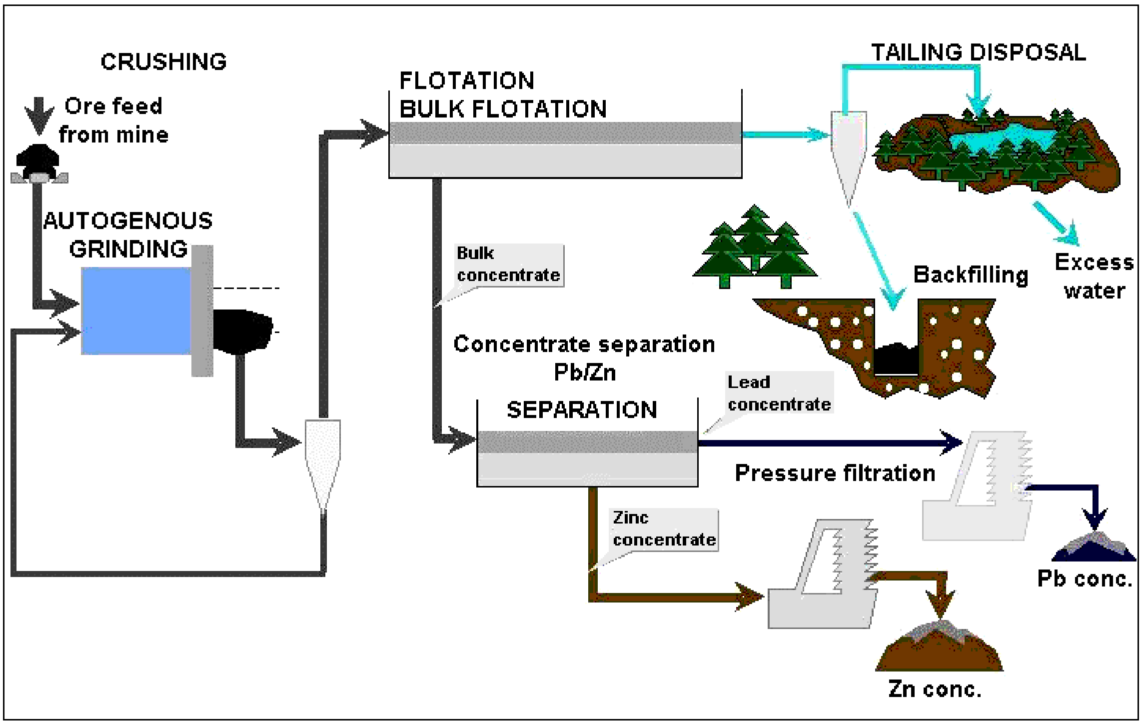

Figure 3.5: Bulk/selective flotation circuit at Zinkgruvan site
[66, Base metals group, 2002]

An alternative flotation method which could be used in the case of changes in the ore composition would be stepwise selective flotation (see Figure 3.6) This would require slightly different process chemicals but is otherwise similarly economical and technically feasible.
[66, Base metals group, 2002]

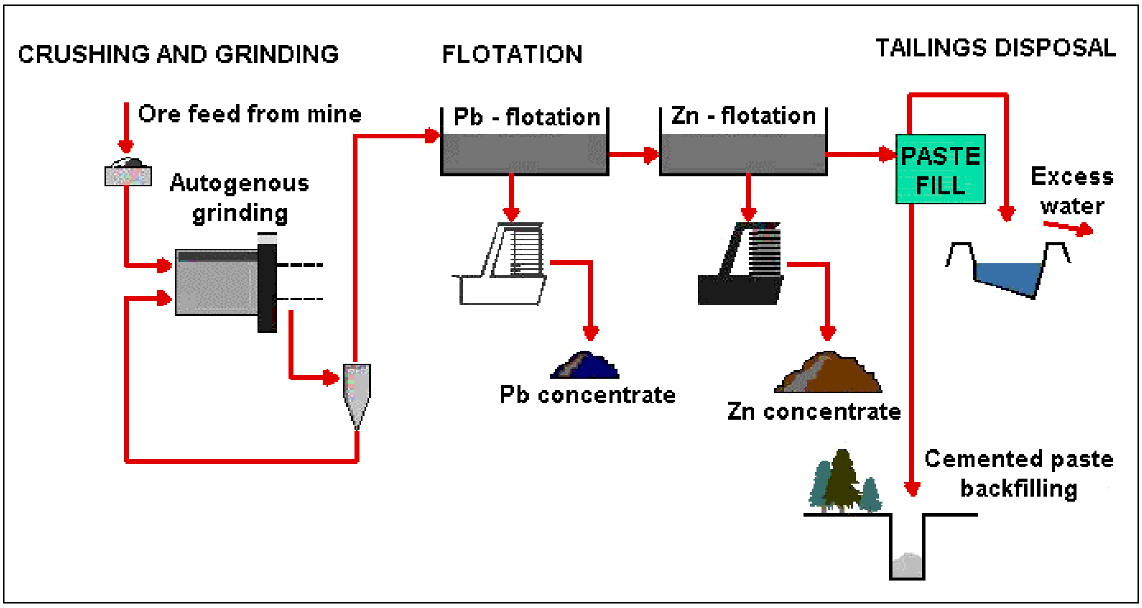

Figure 3.6: Possible selective mineral processing circuit at Zinkgruvan site
[66, Base metals group, 2002]

At **Asturiana de zinc** a pre-concentration is done using gravimetric methods before grinding. Tailings are pumped as a slurry to the pond systems. The coarse fraction of the tailings, which is used in the backfilling, is separated from the fines using cyclones [54, IGME, 2002].

In the **Las Cruces Project** leaching with sulphuric acid is the proposed process method followed by Solvent Extraction and Electro Winning (SX-EW). Tailings are supposed to be dewatered using filtration and will be sent to "dry" lined cells [67, Base metals group, 2002].

The mineral processing set-up of the nickel ore in **Hitura** is similar to the sulphide ores as shown in the figure below.

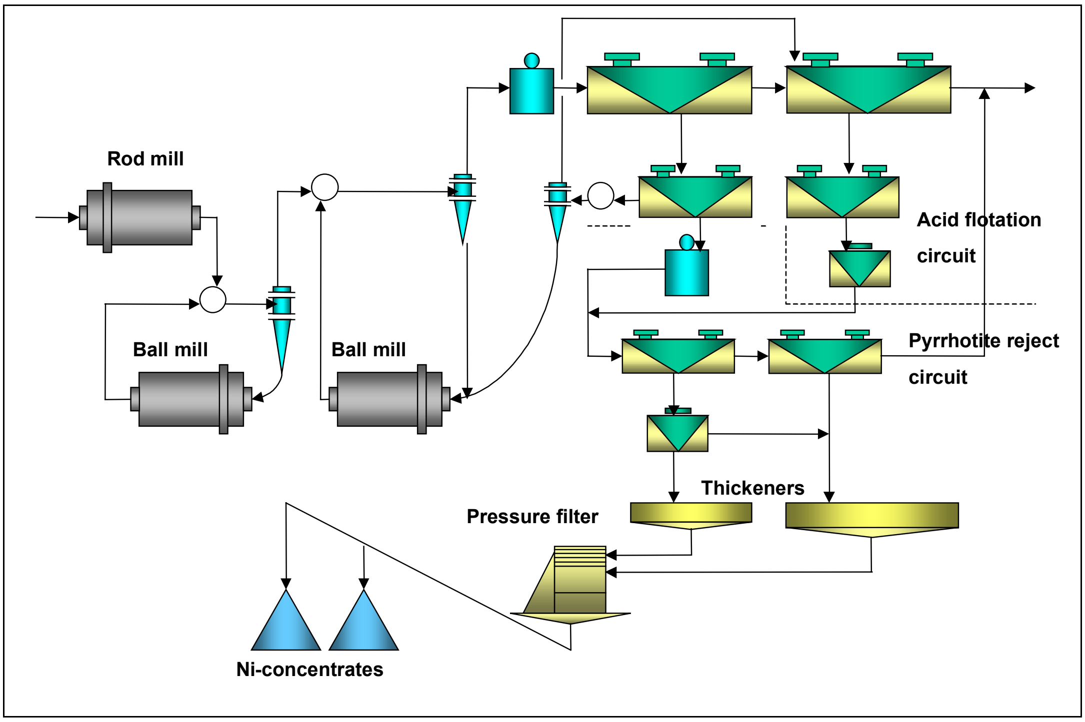

Figure 3.7: Mineral processing flowsheet at Hitura site
[62, Base metals group, 2002]

3.2.2.2.1 Comminution

Size reduction at all sites is done by crushing and grinding using various types of crushers and mills.

At **Pyhäsalmi** mine the comminution is by:

- one stage crushing with a jaw crusher located in the underground mine to three fractions (lumps, pebbles, fines)
- autogenous grinding in three stages (balls are used in the tertiary stage)
- in the grinding circuit 5 ball mills (Ø3.2 x 4.5 m).

At the **Hitura** mine size reduction is achieved via

- crushing in three stages with a jaw crusher, a gyratory crusher and a cone crusher. The crushing circuit is also includes a screen operating in an open circuit
- grinding in three stages with a rod mill (Ø 3.2 x 4.5 m) in the primary stage and two ball mills (Ø 3.2 x 4.5 m) in the next stages.

[62, Base metals group, 2002]

At **Aitik**, where an in-pit crusher is used, there are five milling lines, each made up of an autogenous mill followed by a pebble mill. Each grinding circuit operates in closed circuit with a screw classifier, which feeds back into the autogenous mill.

B-section, comprising two 300 t/h mill lines, is the oldest primary grinding facility. All the mills are run at 75 % of critical speed. C-section is a single 460 t/h line. The autogenous and pebble mills are run at 76 % and 73 % of critical speed, respectively. D-section, another two 460 t/h lines both run at 75 % of critical speed.

Data B-section:
Two AG mills, 6 m diameter, 10.5 m long, installed power 3600 kW.
Two pebble mills, 4.5 m diameter, 4.8 m long, installed power 1250 kW.

Data C section:
One AG mill, 6.7 m diameter, 12.5 m long, installed power 6600 kW.
One pebble mill, 5.2 m diameter, 6.8 m long, installed power 2500 kW.

Data D-section:
Two AG mills, 6.7 m diameter, 12.5 m long, installed power 6000 kW.
Two pebble mills, 5.2 m diameter, 6.8 m long, installed power 3000 kW.

The total grinding capacity is about 50000 tonnes/d, but the actual throughput depends on the grindability, the hardness, of the ore. Energy consumption averages around 11 - 12 kWh/t. The grinding is done at 55 % by weight solid material. The finished ground product from the screw classifier has a k_{80} value of 180 µm and about 25 % are smaller than 45 µm.
[63, Base metals group, 2002]

At **Zinkgruvan** a primary crusher is situated underground. From a temporary storage at ground level, normally containing about 9000 tonnes, the ore is transported to the secondary crusher where two size fractions are produced:

- >100 mm as pebbles for the AG mill
- 25 – 100 mm is recycled
- < 25 mm to AG mill.

An optimum mixture of the two size >100 mm and <25 mm fractions is then fed to the AG mills. Autogenous grinding is used to generate a product with 90 % < 100 µm at 40 % solids.
[66, Base metals group, 2002]

Ore delivered to the **Boliden** mineral processing plant comes either crushed or uncrushed. A jaw crusher with an opening of 220 mm is installed to crush, if necessary, run-of-mine ore (mostly open pit ore). The size distribution of the ore varies from time to time, all from very small rocks to rocks of 200 - 300 mm size. This depends mainly on ore type mined.

All ore is stored in 4 underground bunkers. Storage capacity varies between 1500 to 4500 tonnes of ore. The underground bunkers make it possible to blend and mix ores if wished. Underground storage is beneficial during winter, minimising freezing problems. Ore from the bins is fed to the mill by feeders and conveying belts.

The mineral processing plant uses autogenous grinding, a method where the coarse ore is used as grinding medium (see Section 2.3). The primary autogenous mill is followed by a pebble mill, which receives the grinding pebbles through a continuos draw from the output end of the primary mill. Between mills magnetic separators are installed to clean the pulp of metal scrap from mines. The coarse material is sent back to the mills after screening and hydrocycloning. Both grinding circuits are equipped with Reichert cones, spirals and shaking table for gravity separation of gold.

The throughput is between 92 and 110 tonnes/h and circuit depending on the ore. Energy consumption is about 22 kWh/t. The grinding result varies between 50 - 80 % <45 µm.
[65, Base metals group, 2002]

The **Las Cruces** project proposes using:

- a primary jaw crusher
- secondary and tertiary cone crushers
- ball mills.

The predicted average grain size after comminution is 100 % <100 µm.

At **Lisheen** the ore is continuously fed from the surface stockpile into a grinding circuit consisting of a semi-autogenous grinding mill, secondary ball mill, and closed circuit cyclones [73, Ivernia West,].

The above information on comminution is summarised in the following table

	Pyhäsalmi	Hitura	Aitik	Zinkgruvan	Boliden	Las Cruces	Lisheen
Crushing in pit/ug	ug jaw	jaw	in-pit cone	ug crusher	jaw	jaw	ug crusher
Crushing in mpp		cone cone		sec crusher		cone cone	
Grinding	3 stage AG	RM BM	AG PM	AG	AG PM	BM	SAG BM
	BM	BM					
lines	1	1	5	1	2		
Throughput per line (t/h)	150	90	500	115	100		

ug = underground
jaw = jaw crusher
cone = cone crusher
mpp = mineral processing plant
AG = autogenous grinding mill
RM = rod mill
BM = ball mill
PM = pebble mill
SAG = semi-autogenous grinding mill

Table 3.6: Equipment types used for comminution, number of lines and throughput

3.2.2.2.2 Separation

At the **Pyhäsalmi** mine the separation is done using a flotation circuit composed of:

- Cu-flotation 12 x 16 m^3 cells and 16 x 3 m^3 cells
- Zn-flotation 16 x 16 m^3 cells, 2 x 8 m^3 cells and 8 x 3 m^3 cells. Also a regrinding mill (Ø 2.75 x 3.2 m) is included in the Zn-flotation circuit
- Pyrite flotation 2 x 50 m^3 cells, 2 x 38 m^3 cells, 4 x 16 m^3 cells and 32 x 3 m^3 cells (in reserve 1 x 60 m^3 and 28 x 3 m^3).

All flotation cells are OK type (Outokumpu).

Backfilling material is separated from the tailings with a hydrocyclone (Ø 500 mm) before pumping the fines to the tailings pond.

Reagents added to the process are

- grinding: lime, $ZnSO_4$, SIBX, frother
- Cu-flot.: lime, $ZnSO_4$, SIBX, frother, NaCn
- Zn-flot.: lime, $CuSO_4$, SIBX, frother, NaCN (cleaning)
- Pyrite-flot.: H_2SO_4, SIBX
- dewatering: flocculent (thickeners), HNO_3, CH_3COOH (filters)
- tailings: lime (neutralising).

At **Hitura** mine the separation is done by flotation:

16 x 16 m^3 cells in rougher flotation and scavenging, 8 x 3 m^3 cells and one 12 m^3 in cleaning. And 10 x 3 m^3 cells in the pyrrhotite rejection circuit. All flotation machines are OK-type (Outokumpu). An automatic process control system with two on-line x-ray analysers (6 slurry lines).

Dewatering is done using two thickeners for Ni-conc. (Ø 25 m + Ø 10 m) and a pressure filter (25 m^2).

The reagents added to the process at Hitura are:

- grinding: SEX
- flotation: H_2SO_4, SEX, frother, CMC, lime (cleaning).

[62, Base metals group, 2002]

At **Aitik** the flotation is divided into two steps, one circuit for bulk flotation and one cleaning circuit. The bulk flotation consists of four parallel lines of nine Outokumpu OK38 (cell volume 38 m^3) flotation cells in each line. The cleaning step consists of four columns (all 3 m in diameter, two 11.5 m high and two 9.8 m high), four OK50 (cell volume 50 m^3) and 12 Sala cells.

The final ground pulp from each screw classifier is reporting to a central mixer where frothers and collectors are added and the pH-value is increased to 10.5 by adding lime. In the bulk flotation chalcopyrite and pyrite are floated together. Each flotation line is divided into two steps, where the first four cells are used for rougher flotation and the last five as scavengers. In the rougher flotation a bulk concentrate is achieved with 10 - 15 % Cu. The rougher concentrate from the four lines reports to the cleaning circuit. The scavenger concentrate (1.3 % Cu) is reground in a pebble mill.

In the cleaning circuit the chalcopyrite is separated from the pyrite by regrinding and further addition of lime. The rougher concentrate together with returned products from the separation circuit is reground in a ball mill in closed circuit with cyclones. The cyclone overflow reports to the columns. The concentrate from column 1 and 2 normally holds 20 - 25 % Cu and is mixed together for continuos cleaning in two steps in small conventional cells. The final concentrate contains 28.8 % Cu, 8 g/t Au and 250 g/t Ag. The concentrates is dewatered with a thickener, drum filters and oil burned rotating kilns. Dried concentrate is shipped in containers by truck 20 km to the railroad and then by rail 400 km to the smelter.

The mineral processing plant runs on 100 % re-cycled water from the tailings pond system and recovers 90 % of the copper, 50 % of the gold and 70 % of the silver. It is equipped with a distributed control system and an on-line analysis system.
[63, Base metals group, 2002]

At **Zinkgruvan** the flotation process is done in two steps, as illustrated in the Figure 3.5, bulk flotation followed by zinc and lead separation. Process chemicals are added in both steps to allow for the separation processes.

In the bulk flotation sulphuric acid is added in order to lower pH to approx. 8 from its natural level of approx. 9. As collector for the desired minerals, galena and sphalerite, sodiumisopropylxanthate (Sodium Xanthate) and as frother methylisobutylcarbinol (MIBC) are used. In the bulk flotation circuit separate regrinding is done to improve the purity of the concentrate. The bulk concentrate recovers 98 %, 95 % and 85 % of the total ore content of zinc, lead and silver, respectively.

To the zinc/lead separation step sodium hydroxide is added to increase pH to about 12 where the best results are achieved. The zinc concentrate is directly produced, whilst the lead concentrate requires additional flotation in multiple steps in order to reach the final lead concentrate.
[66, Base metals group, 2002]

At the **Las Cruces Project** pressurised leaching with sulphuric acid followed by solvent extraction and electro-winning (SX-EW) is the proposed method for copper recovery [67, Base metals group, 2002].

At **Lisheen** the ground ore is fed into a lead circuit, and then into a zinc circuit. The lead and the zinc circuit use conventional flotation cells while the zinc circuit also uses flotation columns. The zinc circuit utilises a regrind step to assist in the production of a high-grade concentrate and maximise metal recovery and an acid leach circuit to ensure low levels of magnesium oxide in the concentrate [73, Ivernia West,]. Process water is recycled and supplemented with water reclaimed from the TMF.

3.2.2.3 Chromium

At Kemi the ore from the mine contains 11 % iron and 25.5 % Cr_2O_3. After the mineral processing the concentrate contains between 35 % Cr_2O_3 in the coarse fraction (lumps) and 44 % of Cr_2O_3 in the fines.

The flowsheet of the Kemi site is given below:

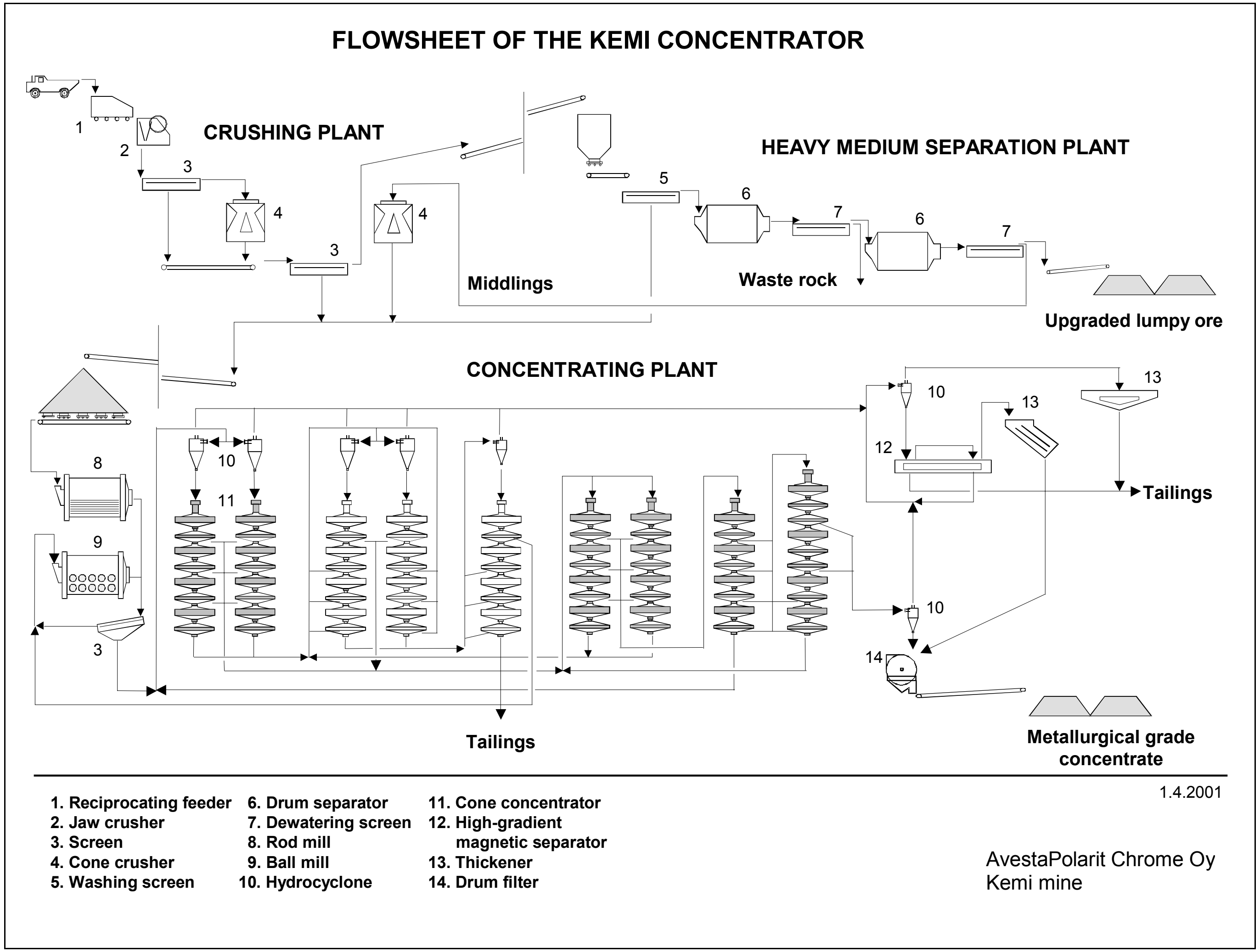

Figure 3.8: Flowsheet of the mineral processing plant at Kemi
[71, Outokumpu, 2002]

As can be seen in the flowsheet the chrome containing mineral is recovered via density separation in cone mineral processing plants. The process steps will be explained in the following sections in more detail.

The mineral processing plant runs at 207 t/h.

3.2.2.3.1 Comminution

Size reduction at Kemi is carried out as follows:

- crushing in three stages with a jaw crusher and two cone crushers
- grinding in two stages with a rod mill (Ø 3.2 X 4.5 m) and a ball mill (Ø 2.7 X 3.6 m)
- a Derrick screen for classification.

[71, Outokumpu, 2002]

3.2.2.3.2 Separation

The following equipment and techniques are used at Kemi to separate the mineral from the gangue:

- two drum separators and three dewatering screens in heavy medium separation plant for lumps
- nine cone separators and a high-gradient magnetic separator in the concentrating plant for
- fine material.

[71, Outokumpu, 2002]

3.2.2.4 Iron

Typically after the extraction the ore is then crushed in various stages to the required size followed by either screening to final products, lumps and fines, or further treatment.

The choice of the mineral processing methods depends on the ore type, chemical composition, fineness, etc. Most common methods used are magnetic separation, high intensity when concentrating hematite ores and low intensity for magnetite, as well as gravity separation and flotation.

The grade of the ore and the treatment method influence the amount, type and composition of the tailings.
[49, Iron group, 2002]

At the **Steirischer Erzberg** the ore-processing plant processes 1.7 Mt ore per year of which 0.98 Mt becomes concentrate, 0.7 Mt coarse tailings (co-deposited together with waste-rock) and 0.1 Mt fine tailings. 0.9 Mt ore per year is sold directly as low-grade ore without processing. The waste-rock dumps, located in two valleys, cover an area of about 400 ha and contain at present about 550 Mt waste-rock have been dumped in two valleys. The tailings facilities, where the fine tailings are deposited, cover about 40 ha and are divided into 6 tailings ponds of which 4 are in operation. Until 2002 about 5.2 Million m^3 (9.4 Mt) of tailing materials had been deposited in total. An overview of the operation is given in the figure below.

Figure 3.9: Steirischer Erzberg
[55, Iron group, 2002]

3.2.2.4.1 Comminution

In the case of the **Kiruna and Malmberget** operations there are in pit crushers (product 100 % passing 100 mm) and secondary crushing for sinter feed production. In pit crushing, secondary crushing, autogeneous mill/ball mill and pebble mill are applied for pellet production [49, Iron group, 2002]. At the **Erzberg** operation two gyratory crushers (product 100 % passing 120 mm) and secondary crushing are applied [55, Iron group, 2002].

3.2.2.4.2 Separation

The **Kiruna and Malmberget** operation use dry magnetic separation followed by wet magnetic separation for the sinter feed production. Dry magnetic separation, wet magnetic separation, hydrocycloning and flotation are applied for the pellets production (in Malmberget no flotation is required).

Material discarded from the dry magnetic separation is coarse (from gyratory crushing) and transported via a conveyer belt to silos from where it is trucked to the tailings dumps. Material discarded from the wet magnetic separators is transported via pipeline to a thickener. The overflow water from the thickener is re-cycled back to the process as process water. After the thickener, water is added to the tailings in an aqueduct as a transporting agent, and discharged into tailings via the fixed aqueduct with a liquid/solid ratio of 19:1. In Malmberget and in Svappavaara were pump system are used the liquid/solid ratio is between 9:1 and 17:3
[49, Iron group, 2002]

At Erzberg the coarse fraction, 8 - 30 mm and 30 - 120 mm, are separated by dense media separation. Finer fractions, 1 - 4 mm and 1 - 8 mm, are separated by dry high force magnetic separation. The concentrate is further crushed to <8 mm. The fines, 0.1 - 1 mm, are dewatered via screw classifiers are hauled together with coarse tailings from the dense media separation and the high force magnetic separation to heaps within the mining area. Blending of the concentrate with “direct ore” (ore that is not processed) is done in the final crushing and screening.

The process water, mainly the overflow from the screw classifiers, is treated in three 32 m thickeners. The overflow is recycled back to the process, whilst the thickened slurry is pumped to tailings pond.
[55, Iron group, 2002]

Process chemicals are used in the case of the use of flotation process: as a depressant, fatty acids; froth agent, pine oil or equivalent; and as an activator, sodium silicate.

3.2.2.5 Manganese

No data has been supplied for the section. Please provide information.

3.2.2.5.1 Comminution

3.2.2.5.2 Separation

3.2.2.5.3 Others

3.2.2.6 Mercury

No data has been supplied for the section. Please provide information.

3.2.2.6.1 Comminution

3.2.2.6.2 Separation

3.2.2.6.3 Others

3.2.2.7 Precious Metals (Gold, Silver)

Various mineral processing techniques are used, mainly due to their different suitability for different mineralogy. Depending how the gold occurs in the ore it is necessary to use different methods to liberate the gold so that it can be extracted. The gold can in many cases be recovered in the copper concentrate and separated from the copper in the subsequent smelting process. Native gold can be gravimetically concentrated and recovered. Gold in oxide form can be directly leached with cyanide. Refractory gold may require oxidation, e.g. bio-oxidation, in order to liberate the gold and make it accessible for CN leaching.

3.2.2.7.1 Comminution

Common for all operations is that the ore needs to be crushed and ground before the gold can be liberated. In some cases this is done in the previous recovery of base metals. Tank leaching requires a finer grain size in order to allow for relatively short residence times in the leaching tanks. Heap leaching allows for coarser grain size as the leaching time is much longer. In heap leaching relatively coarse grain size (even conglomeration may be necessary) is desired to allow for oxygen inflow and to secure a sufficiently high permeability of the heaped material.

Used type of equipment in the Comminution is, e.g., various types of crushers, and various types of mills such as dry semi-autogenous mills, ball mills, autogenous mills etc.

Orivesi Mine uses the following equipment in the comminution process:

- crushing in three stages with a jaw crusher and two cone crushers
- grinding in two stages with a rod mill (Ø 3.2 X 4.5 m) and a ball mill (Ø 2.7 X 3.6 m)
- Derrick screen for classification.

[59, Au group, 2002]

The Boliden comminution circuit is described in Section 3.2.2.2.1. Both grinding circuits are equipped with Reichert cones, spirals and shaking table for gravity separation of gold.

For the tank leach operations it is commonly required to reach a grain size of 50 - 80 % < 45 µm or in some cases, if the gold is extremely finely disseminated, even below 40 µm to achieve optimum liberation.
[50, Au group, 2002]

3.2.2.7.2 Separation

Used methods used in Europe are standard mineral processing and hydrometallurgical.

The mineral processing methods represented are:

- flotation, where the gold reports mainly to the copper concentrate (gold recovered from the concentrate in the smelting process)
- heavy-media separation for lumps using drum separators and dewatering screens
- cone separators and high-gradient magnetic separator for fine material
- Reichert cones, spirals and shaking table for gravity separation of gold.

In the schematic figure below en example of a mineral processing plant is given. This plant, with a relatively low throughput of 35 t/h, produces a concentrate containing 125 g Au/tonne. The leaching of some of the gold concentrate is to reduce the content of impurities (Tellurium (Te) and Bismuth (Bi)). This step is aimed at dissolving Bi and Te away from the concentrate. The tailings from this process are led to a separated ditch in the old TMF (used during nickel mining phase). Because the water from the leaching process is acidic, lime is added to neutralise it. Bi is precipitated in these circumstances, but most of the Te remains in solution. The leaching process has been in use only when necessary, depending on the ore characteristics. There is no outlet from the ditch, thus the water evaporates and filtrates into the old tailing material. According to analysis on seepage water outside the TMF area no significant concentrations of Te have been found. Currently the leaching process is not in operation, because the quality of the ore has changed and Bi and Te are no longer problematic.

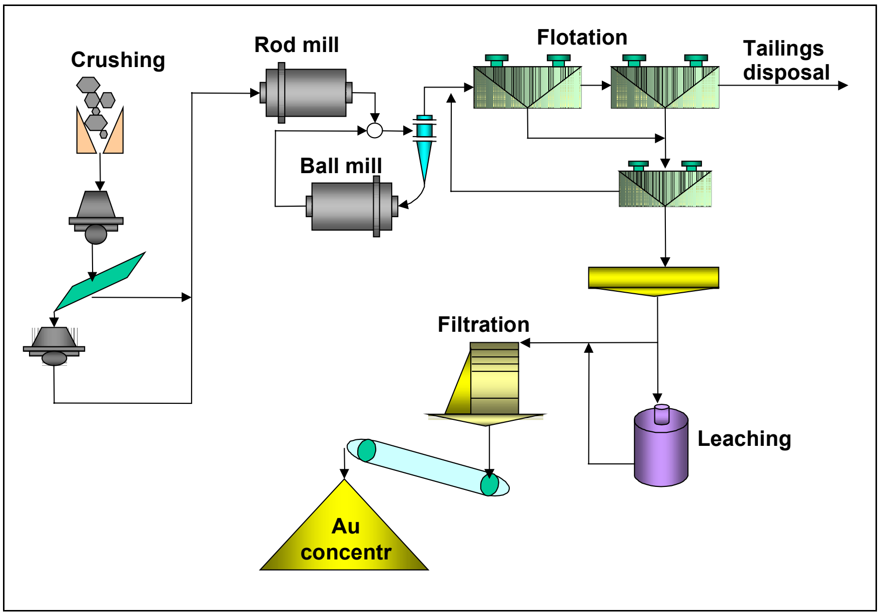

Figure 3.10: Schematic flowsheet of an example gold mineral processing circuit
[59, Au group, 2002]

The hydrometalurgical methods used are:

- CN leaching in tanks using the Carbon In Pulp method (CIP) (e.g. Ovacik Gold Mine)
- CN leaching in tanks using the Carbon In Leach method (CIL) (e.g. Boliden and Río Narcea)
- bio-oxidation and pressure oxidation followed by CN leaching using the CIL method (all processes in closed tanks) (e.g. Olympias Gold Project)
- heap-leaching using CN solution followed by Merrill-Crowe process where the gold is precipitated on zinc powder (e.g. Filón Sur).

CN leaching and the management of CN in general is surrounded by a large amount of security measures to prevent accidents and environmental impacts. The design of the plant also includes several technical solutions aimed at prevention of accidents and environmental impact, such as:

- the incorporation of a cyanide destruction circuit integrated into the leach plant. This circuit has a design capacity twice the actual requirement
- the tailings pond system constitutes a second cyanide treatment facility, serving as a backup to the cyanide destruction circuit
- flotation plant tailings and the leach circuit effluent are combined prior to cyanide destruction to prevent an increase of pH, which may cause dissolution of already precipitated cyanide complexes
- a back-up system for lime addition is installed
- the storage tank for hydrochloric acid is well separated form the cyanide storage and preparation units
- sulphur dioxide is delivered in liquid form. The storage capacity for sulphur dioxide is limited, and the tank is placed away from the leach plant
- a berm is placed between the sulphur dioxide and the LPG tanks, to avoid mixing of the two gases in the unlikely case of simultaneous leaks in both tanks
- the leach circuit is connected to a collection pond with volume exceeding 1000 m^3, i.e. more than the containing capacity of one leach tank (800 m^3)
- the leach tanks are placed in a concrete trough with a surrounding berm, which also functions as a collision barrier. The capacity of the trough exceeds the volume of one leach tank. The floor is heated to avoid build up of snow and ice during winter
- leach tanks placed outdoors are open. Indoor equipment is connected to a gas extraction system with a scrubber operating with NaOH- solution
- backup power generators are installed
- all spills are pumped back to the circuit.

[50, Au group, 2002]

The hydrometallurgical processes mentioned above all require further processing in order to achieve a sellable product, i.e., transfer of the gold and silver from the activated carbon into doré containing gold and silver. A complete gold tank leaching plant constitutes of the following principle stages:

- Cyanide leaching (CIL-process or CIP-process)
- Gold refining (eluation, electro-winning, smelting and doré production)
- Cyanide destruction (e.g., Inco-SO_2/Air)
- Reagents preparation (lime and sodium cyanide).

A complete plant is schematically illustrated in the figure below. This particular plant (Boliden), that was commissioned in year 2001, recovers gold and silver from a tailings steam resulting from a base metal mineral processing plant. The system is designed for a throughput of 800000 t/yr with a gold production of 850 kg/yr. The recovery is approximately 80 % of the gold. The recovery of gold increased by 50 % after the installation of gold leaching.

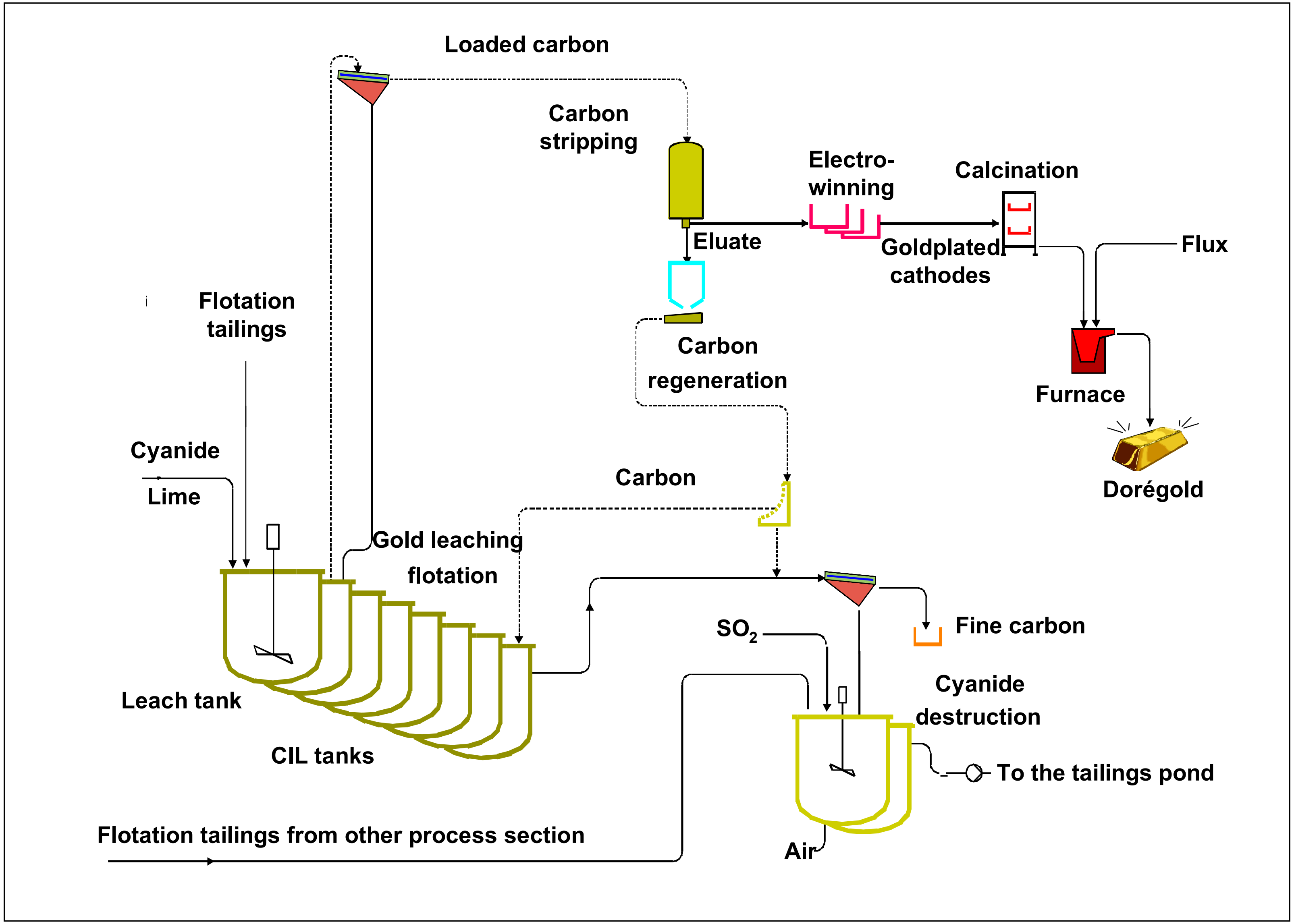

Figure 3.11 Schematic drawing of CIL process
[50, Au group, 2002]

At all sites where tank leaching is practised, the tailings and tailings water undergo detoxification prior to discharge into the tailings pond.

3.2.2.8 Tungsten

Due to the fine intergrowth of scheelite with the gangue minerals, the ore has to be treated by flotation. Using gravity separation would result in high losses of scheelite and the operation would not be economical.

3.2.2.8.1 Comminution

The ore is crushed to <14 mm by means of a three stage crushing system which is situated underground. It is then stored in two underground ore bins before being transported to the mineral processing plant by a conveyor belt system situated in a 3 km long gallery. Just beside the mineral processing plant a stockpile is positioned to secure the supply of the process with ore in case of discontinued production at the crushing plant.

The top size of the feed is further reduced to <10 mm in a one stage crushing system consisting of a cone crusher which operates in closed cycle with a vibrating screen. The crushed ore is stored in two ore silos from where the ore is fed to a single stage ball mill at a feed rate of 80 – 82 t/h. To achieve sufficient liberation of the scheelite from the gangue, the ore has to be ground to 80 % passing 180 µm.The mill discharge is pumped to a classification system, which consists of screens and a cyclone. The fines with a top particle size of 500 µm are pumped to the flotation process, the coarse fraction is recycled to the ball mill.
[52, Tungsten group, 2002]

3.2.2.8.2 Separation

Flotation consists of one rougher bank and four cleaning stages. A concentrate with an average grade of 40 % WO_3 is produced. The rougher tailings are pumped to a cyclone. The cyclone underflow, which contains coarse and intergrown scheelite is recycled to the ball mill for regrinding, the cyclone overflow represents the final tailings stream. The collectors used for flotation are fatty acids (carboxylates), alkyl sulphonates and alkyl sulphate.

A schematic flowsheet of the processing plant is given in the figure below.

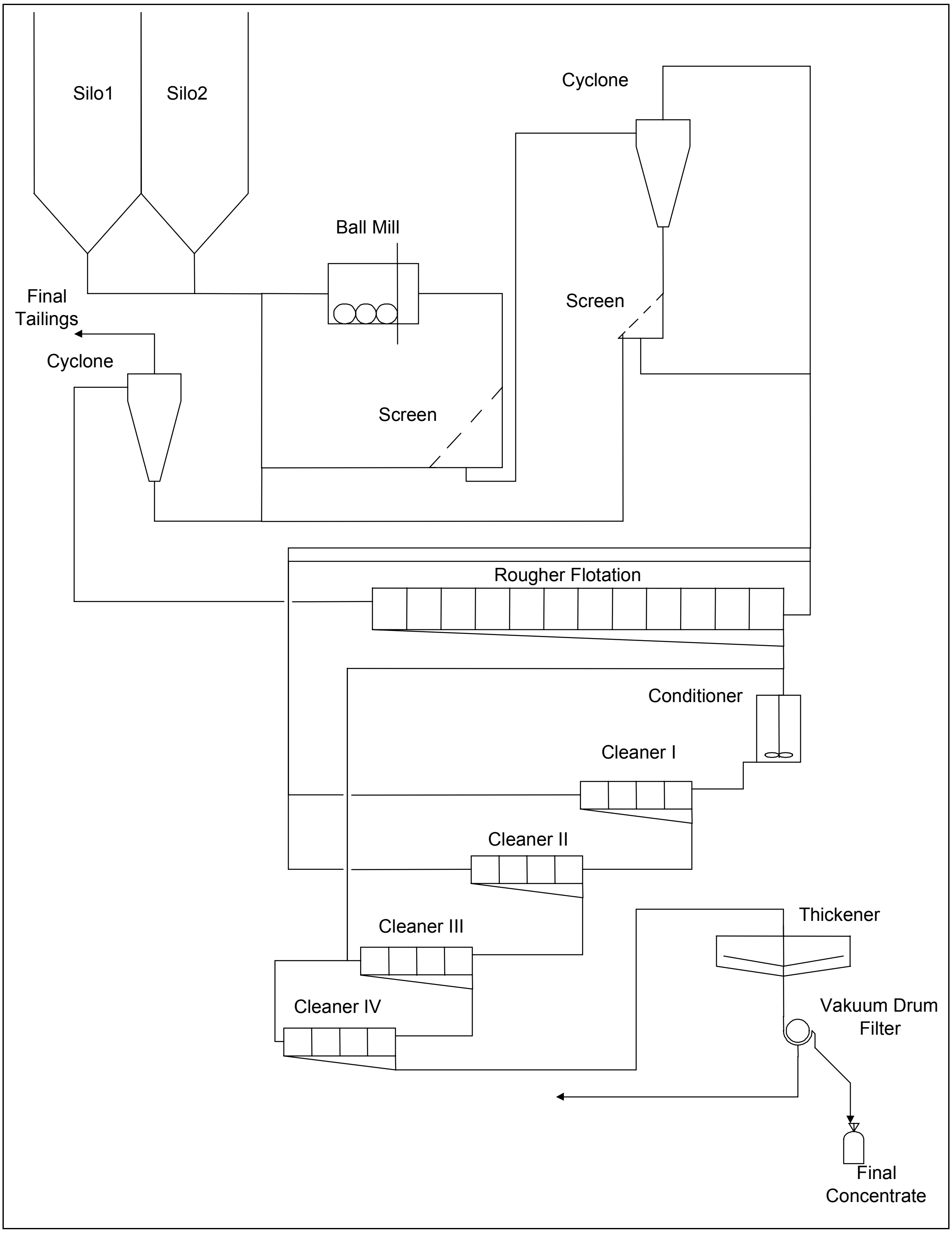

Figure 3.12: Flowsheet of Mittersill mineral processing plant
[52, Tungsten group, 2002]

3.2.3 Tailings management

3.2.3.1 Aluminium

On a worldwide basis it can be said that 4 - 6 tonnes of bauxite yield 2 tonnes of alumina and 1 tonne of aluminium. The European refineries that import bauxite use high grade bauxite, in order to reduce shipping costs. The following figure shows the mass flows for European refineries.

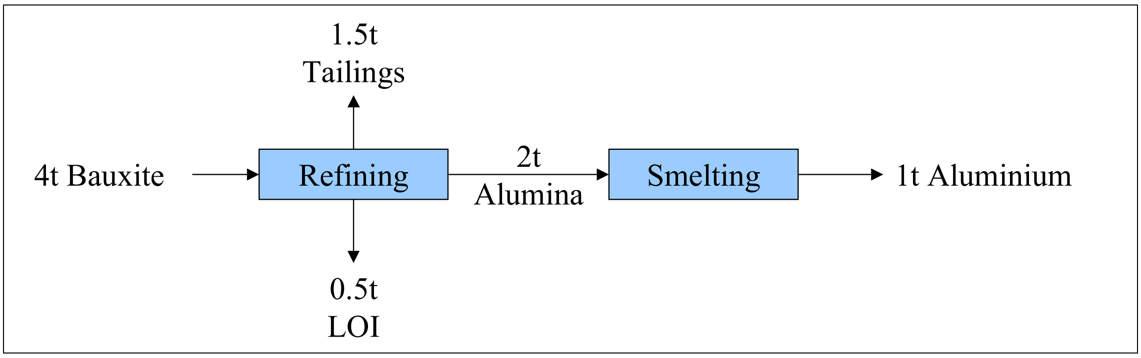

Figure 3.13: Typical mass flow from Bauxite to Aluminium (dry basis)

It should be noted that LOI stands for "loss on ignition" or "water of crystallisation."

3.2.3.1.1 Characteristics of tailings

The alumina tailings consist of two major parts. The fine fraction, 80 – 95 % of total, called 'red mud'. The coarser fraction is commonly referred to as 'process sand'. These two portion represent 97 %-100 % of the total tailings. In some cases the remaining 3 % consist of salt cake, from a salting-out liquor purification process and sludge (principally aluminium hydroxide) from the underflow of the clarifier.

Red mud

The following figure shows some red mud size distributions of alumina refineries.

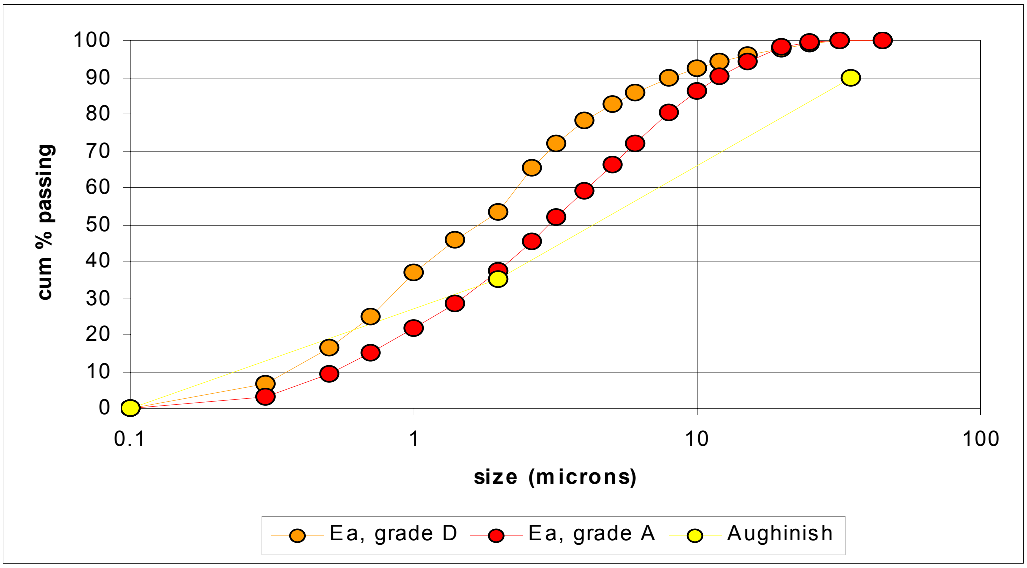

Figure 3.14: Size distribution of red mud at the Sardinian (Ea) and Aughinish sites [89, Teodosi, 2002], [22, Aughinish,]

If the red mud is pumped as thickened tailings it usually has a solids content of 55 – 60 %, "matures" at the TMF, which in the case of thickened tailings is often referred to as a "stack", over a period of 3 – 6 months to a solids content of 68 – 70 %. In the case of the **Aughinish** refinery the initial permeability of the red mud is between 1 x 10^{-8} to 1 x 10^{-9} m/s. It decreases as the mud matures. The average density of dry mud solids is 3.3 kg/m^3 [22, Aughinish,]. The benefit of this technique is that the tailings are physically stable upon discharge onto the stack.

However, precipitation run-off and seepage water will have an elevated pH, due to the residual caustic liquor, and must therefore be neutralised.

At the **Sardinian site** the red mud, resuspended in seawater, is pumped to the pond at 20 – 25 % solids, with the magnesium chloride of the seawater neutralising the red mud. After settling and evaporation the solids content increases to 65 – 72 %. The ratio of tailings, in the case of the Sardinian refinery is 0.78 tonnes dry tailings per tonne of alumina. Considering the slurry consolidates at 60 – 65 % solids in the pond, this corresponds to about 1.3 tonnes of wet material per each 1 tonne of alumina produced or 0.8 m^3/tonne of alumina produced.
[89, Teodosi, 2002].

The neutralisation of the red mud results in chemical stability of the tailings. The trade-off here is that, as in the case of all slurried tailings impoundments, physical stability of the dams must be taken care of.

The solids content of the tailings for both options are shown in the following figure.

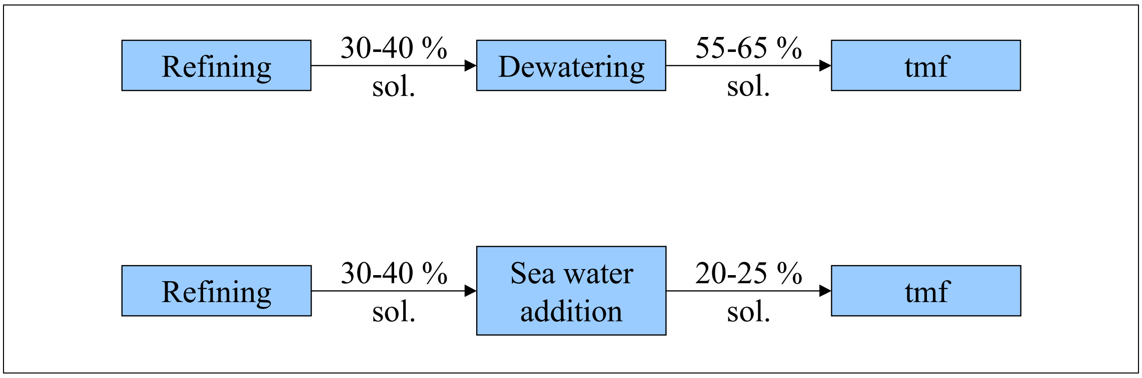

Figure 3.15: Solids content (in % solids by weight) of tailings in case of thickened and conventional management schemes.

In both cases the tailings mature to about 70 % solids. In general dewatering can be carried out in vacuum filters (yields 65 % sol., e.g. Aughinish) or in deep thickeners (yields 55 % sol.).

Some chemical analyses of red mud from different sites are shown in the following table.

Site:	Sardinia	Bakony	Aughinish
	dry wt. %	dry wt. %	dry wt. %
Fe_2O_3	18	40	47
Al_2O_3	26	18	17
TiO_2	6	4	12
SiO_2	20	15	7
Na_2O	12	8	5
CaO	8	7	8
LOI	9	7	3
Misc. Trace elements	1	1	1

Table 3.7: Constituents of red mud
[89, Teodosi, 2002], [91, Foldessy, 2002], [27, Derham, 2002]

Despite the repeated washings, the solution entrained within the red mud contains small amounts of caustic (sodium hydroxide), which causes the elevated pH characteristics, and alumina. Most of the caustic converts to sodium carbonate and sodium bicarbonate on the tailings stack.

The following table shows an example of a more detailed analysis of red mud including the trace elements.

Analysis of Aughinish Alumina Red Mud

Name	formula	% dry		METHOD OF ANALYSIS
Principal Compounds				
Titanium dioxide	TiO_2	9.93%		X-RAY FLUORESENCE
Iron Oxide	Fe_2O_3	46.18%		
Silica Quartz	SiO_2	8.11%		
Sodium Oxide	Na_2O	4.39%		
Calcium Oxide	CaO	4.41%		
Alumina (aluminium oxide)	Al_2O_3	16.50%		
Loss on Ignition LOI (includes crystalline water)		9.26%		
subtotal			**98.78%**	
Secondary Compounds				
Zirconium dioxide	ZrO_2	0.15%		
Zinc Oxide	ZnO	0.01%		
Vanadium pentoxide	V_2O_5	0.17%		
Phosphorus pentoxide	P_2O_5	0.43%		
Manganese Oxide	MnO	0.05%		
Magnesium Oxide	MgO	0.07%		
Potassium Oxide	K_2O	0.04%		
Chromium trioxide	Cr_2O_3	0.26%		
subtotal			**1.18%**	
BASIC TOTAL =			**99.96%**	
Misc. Trace elements ex analysis by EOLAS				
Sulphur		0.12%		**Inductively Coupled Plasma Spectrometry**
Arsenic (just at detection limit, therefore approximate)		0.005%		
Tin	<	0.005%		
Mercury	<	0.005%		
Antimony		0.019%		
Lead		0.020%		
Gallium		0.006%		
Bismuth	<	0.005%		
subtotal =			**0.19%**	
GRAND TOTAL (discrepency of different dates & methods of analysis) =			**100.15%**	

Figure 3.16: Detailed analysis of red mud including trace metals
[32, Derham, 2002]

Process sand

Size distribution curves for process sand are shown in the following figure.

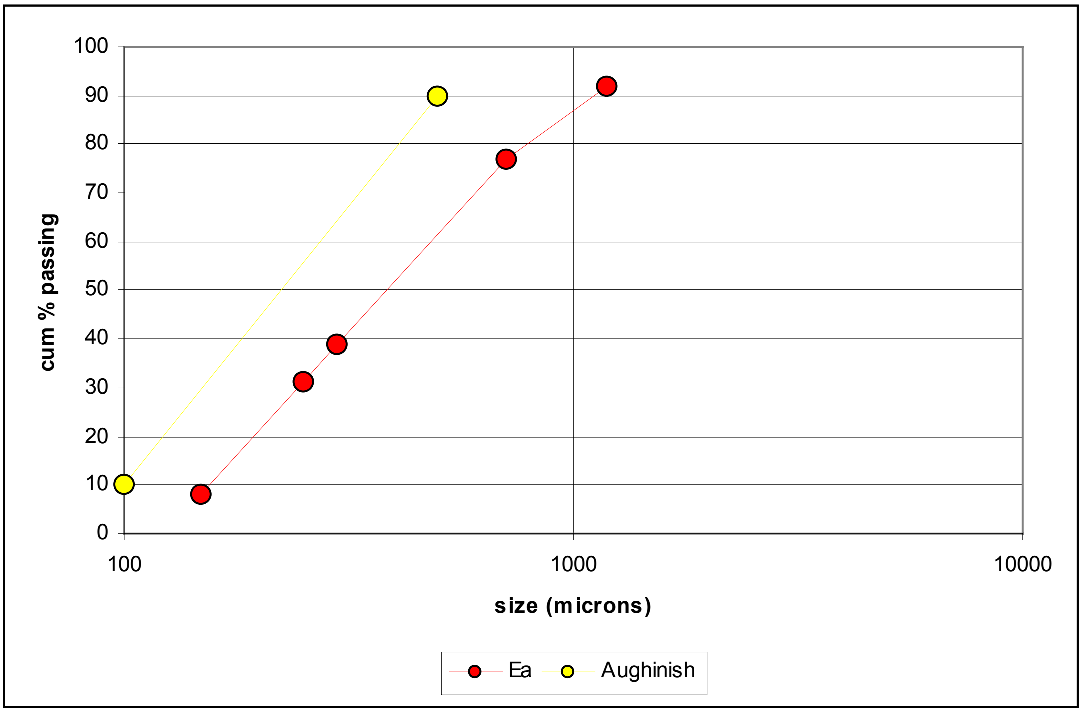

Figure 3.17: Size distribution of process sand at the Sardinian (Ea) and Aughinish sites [89, Teodosi, 2002], [22, Aughinish,]

The following table shows the components of the sand fraction:

Site:	Sardinia
	dry wt. %
Fe_2O_3	14
Al_2O_3	40
TiO_2	3
SiO_2	16
Na_2O	12
CaO	1
LOI	12
Misc. Trace elements	2

Table 3.8: Constituents of tailings sand
[33, EURALLUMINA, 2002]

The permeability is estimated to be 100 times higher than that of the red mud [22, Aughinish,].

At the Sardinian site leaching tests of the red mud and the sand were undertaken [33, **EURALLUMINA**, 2002].

<u>Others</u>
Salt cake is dumped as a 70 % solids cake. Clarifier sludge is pumped to the stack as a 2 to 3 % solids slurry. Salt cake consists of organic degradation products from humates in the bauxite, including sodium carbonate, sodium sulphate and sodium oxalate.
[22, Aughinish,].

3.2.3.1.2 Applied management methods

For the management of tailings from alumina refining thickened tailings as well as conventional slurried tailings are applied. Some refineries discharge the tailings into the sea. The others store them on land on “stack” in the case of thickened tailings and within dammed ponds in the case of slurried tailings.

Generally the design of red mud stacks using the **thickened tailing method** includes pervious perimeter rock fill dams and sealing of the underlying surface. A perimeter dam for the collection of surface run-off typically surrounds the stack. The upstream construction method for the stacks is used, since the dewatered red mud is sufficiently stable.

Due to the very low permeability of the red mud, the principal risk of seepage arises from ponding of caustic surface water run-off in exposed areas prior to covering with mud and seepage from standing water in the perimeter ditch. This can be handled by sealing surface and ditches with liners, such as glacial till or synthetic liners combined with a drainage system. Seepage analysis for typical and worst case conditions are undertaken in order to properly design these facilities.
[22, Aughinish,]

The main differences between the use of thickened and slurried tailing can be summarised as follows:

The **slurried tailings management** involves much more water being treated with the mud. This method has the advantage that the slurry is easily pumpable by standard centrifugal pumps at relatively low pressure in the pipeline. The water available to suspend the mud can be seawater, if available in the refinery vicinities, with the associated neutralisation of residual caustic. Pumping can be done over relatively long distance (several kilometres) between the refinery and the pond without the danger of a pressure drop along the pipeline.

Thickened tailings management must be associated with good recovery of the caustic mother liquor, as the management at the pond will not involve further neutralisation. The density and viscosity of the paste is so high that the dewatering is carried out preferably at the TMF unless the stack is located adjacent to the refinery. In case the two sites are distant from each other, pumping is done at low density prior to dewatering at the pond site, to produce the thick slurry right at the pond feed, in which case the surplus water has to be pumped all the way back to the plant. Therefore, this technique involves an investment related either to a high pressure pumping station, such as membrane pumps, or the investment of a deep thickener installed and operated at the pond, i.e. far from the refinery.

The compaction of the decanted and aged slurry does not show any significant differences to “matured” paste. In both cases the figures are around 70 % solids.

In the case of **Sardinian** refinery, the utilisation of mud suspended in more water, to 20 % solids, is is required of flue-gases desulphurisation by wet scrubbing, the so called Sumitomo process. The mud slurry to be used in the absorbers needs to have its solid content well diluted in, in order to protect the perforated dishes of the absorber against early plugging by solids deposition.
[89, Teodosi, 2002]

The following aspects are of importance in the design of the facility:

- short distance between refinery and pond to reduce pumping costs
- availability of surface area
- need to store tailings on land, as opposed to discharging into the sea, in order to protect fishery
- vicinity to the sea, because of the need for seawater to neutralise the tailings
- low risk of aquifer contamination
- strong winds in the area, therefore it is beneficial to have wet tailings.

The location of the TMF can be seen in the following picture

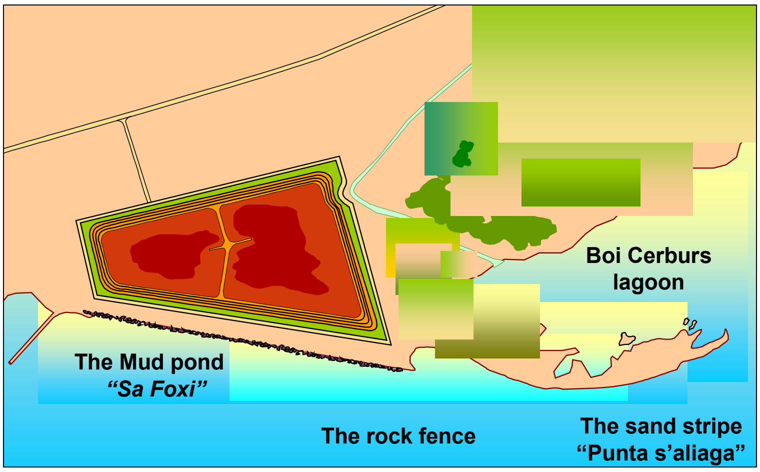

Figure 3.18: Location of TMF at Sardinian refinery
[33, EURALLUMINA, 2002]

The "rock fence" is supposed to protect the TMF from waving action.

A cross-section of the dam can be seen in the following figure.

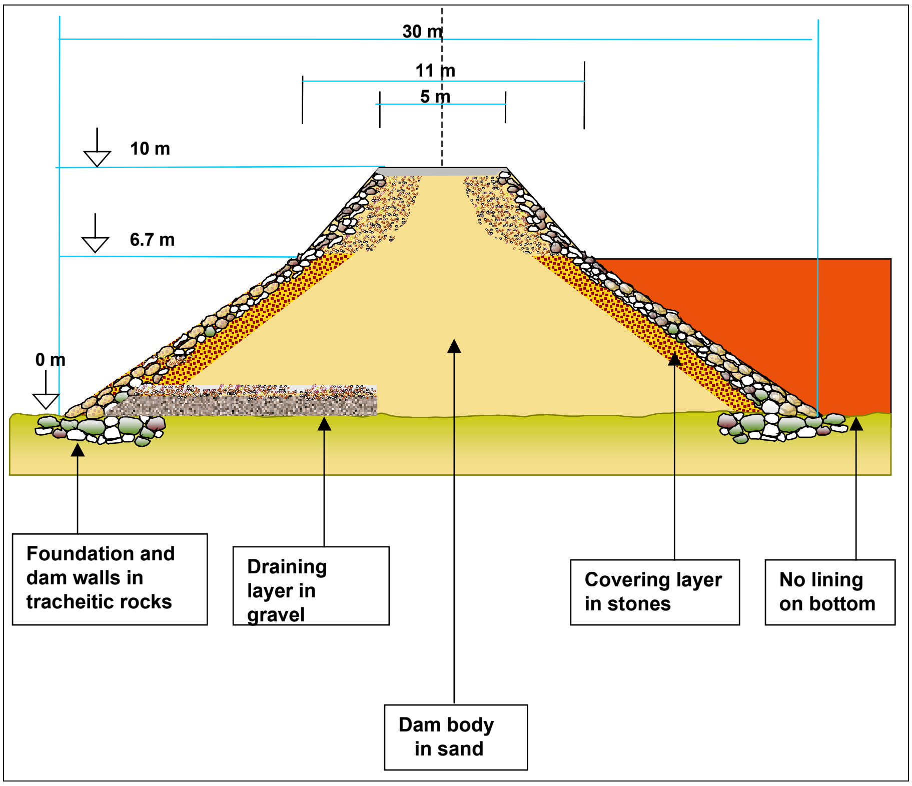

Figure 3.19: Cross-section of tailings dam at Sardinian site
[33, EURALLUMINA, 2002]

The concept of this original dam design is filter the seeping liquid whilst the tailings remain within the impound. Hence good drainage (to 70 %) is achieved.

Further raises of the dam, using the upstream method, were carried out as shown in the following figure.

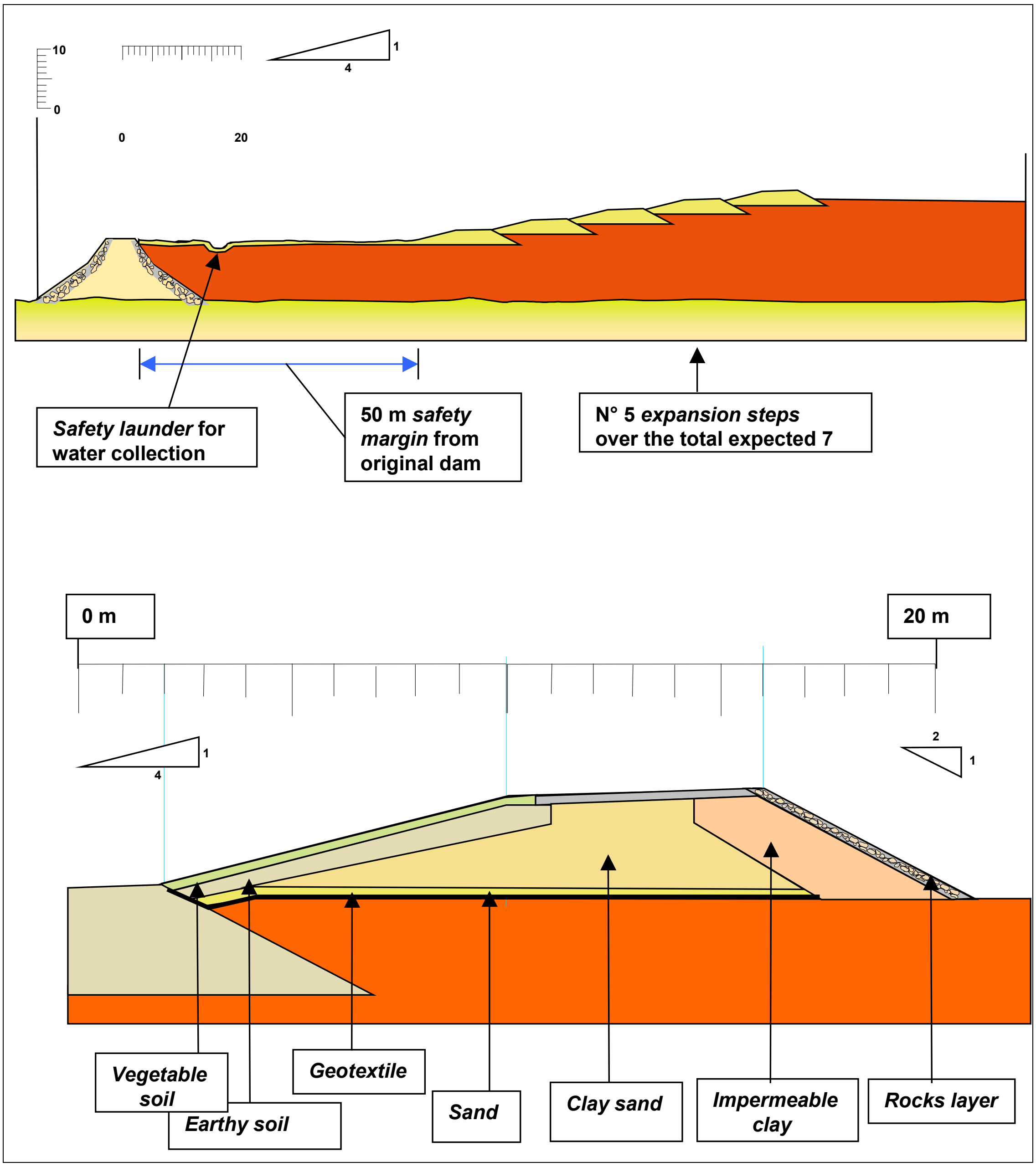

Figure 3.20: Cross-section of dam raises using the upstream method
[33, EURALLUMINA, 2002]

The mud is distributed along the perimeter of the facility with discharge every 50 m. To achieve an even distribution different discharge points are used every 24 h. The sands and other process residues are transported to the TMF by trucks and discarded in a dedicated area of the TMF. [33, **EURALLUMINA**, 2002]

In the **Ajka** refinery "cassettes", paddock-style tailings ponds for the collection of red mud, are built from gray slag derived from the nearby thermal power plant. The dams have 1:1 – 1:1.5 slope ratios (see figure below). Their final height is maximum 10 m. The red mud is transported to the TMF via pipeline at 20 % solids. The distance is 3 - 4 km. The supernatant water from the pond is re-used in the process. The circular movement of the discharge pipe achieves an even distribution of the red mud in the cassette. The supernatant water in the cassettes prevents the development of larger dry surfaces and the drying of the red mud.
[91, Foldessy, 2002]

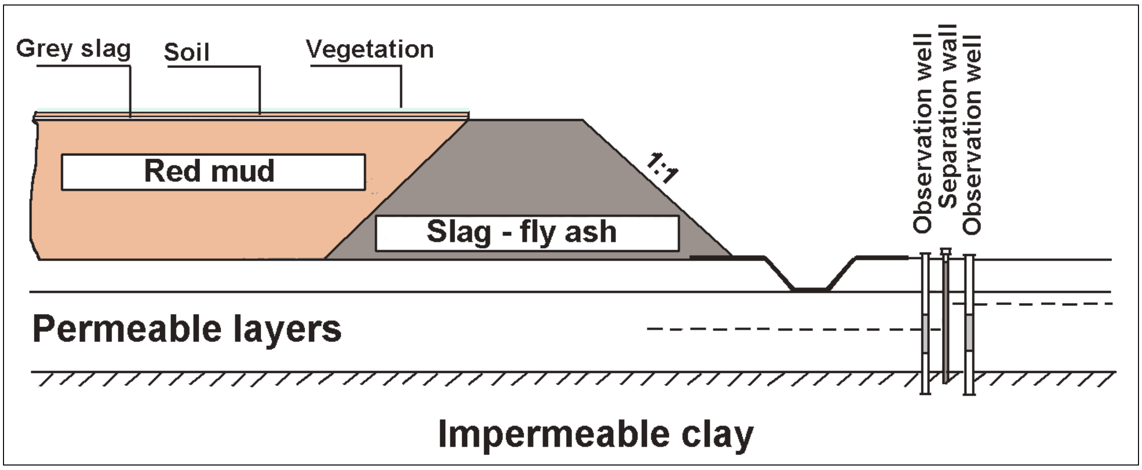

Figure 3.21: Cross-sectional view of TMF as Ajka showing dam, pond, observation wells, separation wall and ground conditions as well as the soil cover upon closure
[91, Foldessy, 2002]

An impermeable clay layer is found 10 m beneath the tailings management facilities. For this reason no sealing was used during the construction of the cassettes. In the 1980s it was revealed that groundwater pollution had developed in the layer between the bottom of the cassettes and the clay layer. To contain this pollution an impervious wall into the impermeable clay layer has been built around the cassettes. In the inner side of this sealing wall a drainage system collects seepage water and groundwater, which is pumped back into the cassette.

In the surrounding area 240 groundwater observation wells were drilled. These serve to measure the level and sample the groundwater for chemical analysis. Groundwater level measurements are repeated monthly, chemical analysis of groundwater samples for 8 - 10 components is done in every quarter. This system ensures the early detection of any damage of the separation wall, and also monitors the migration of the pollution plume.
[91, Foldessy, 2002]

3.2.3.1.3 Safety of tailings facility and accident prevention

The control program at the Sardinian site includes the following:

- inspection tour of TMF every 2 h
- daily overall inspection inside and outside of TMF by supervisor
- performance control of external water collecting pumps on daily basis and recording of flow measurements
- monthly sampling at external piezometer network, with analysis of pH and metals
- checks of dam stability twice a year
- annual tracing of coastal profile to check erosive trend
- daily change of discharge points
- checking of water balance
- continuous recording of meteorological conditions
- continuous measure of pH upon exit from mud filtration unit, before pumping to the TMF.

The staff working in the TMF area have been trained in specific annual training courses. An emergency procedure exists.

Overflowing and pipe bursts are the major risk considered at this site. Hence, 7 pumps are spread around the perimeter of the embankment to pump back overflowing tailings water [33, **EURALLUMINA**, 2002]. However, if the dam overflows the whole dam will burst which will not be prevented by those pumps. Several pipe bursts have occurred at this site.

3.2.3.1.4 Closure and aftercare

If mud stacking is carried out on one stack progressive restoration is not practical, since most of the surface will be used for dumping red mud. At the time of restoration a flat surface with an incline of 2.5 % is created which is ideal to allow effective run-off of precipitation without erosion. Furthermore, the stack is accessible for construction equipment [22, Aughinish,]. The mud stack is restored with a vegetative cover. This has been successfully demonstrated at several sites. Revegetation of the perimeter slopes, built with borrowed rock-fill (e.g. limestone) is common practice and usually designed in a way that this vegetation matches the looks of the surroundings [22, Aughinish,]. Vegetative covers will also be applied in the case of conventional tailings ponds [33, **EURALLUMINA**, 2002]. At the Ajka site the dewatered tailings are covered by a 0.5 m thick layer of slag from a power plant and another layer of soil [91, Foldessy, 2002].

In the after-care phase the run-off has to be treated prior to discharge until the chemical conditions have reached acceptable concentrations for discharge into surface waters. Also access roads, drainage systems and vegetative cover (including re-vegetation if necessary) are maintained. Furthermore groundwater quality measurements must be continued.
[22, Aughinish,].

3.2.3.2 Base metals (cadmium, copper, lead, nickel, tin, zinc)

Tailings are used in the backfill of underground mines at 5 mineral processing plants (Boliden, Zinkgruvan, Garpenberg, Pyhäsalmi, Lisheen). At these sites 16 – 52 % of the tailings are backfilled into the underground mines. One site, Asturiana de zinc, backfills an old open pit using 94 % of the tailings. The tailings that are not used for backfilling need to be managed in ponds. For the Las Cruces project, it is proposed to deposit dewatered tailings in lined cells. At Almagrera the coarse fraction of the tailings (33 %) is roasted and sulphuric acid produced. The cinders are then leached and copper extracted in an SX-EW process. The cinders are deposited in a cinders dam. The remaining 2/3 of the tailings are deposited into a tailings pond.

The tailings production and management method at the various mineral processing plants is summarised in the table below.

Site	Mining method	Tailings production (Mt)	Tailings used in back-fill (%)
Almagera	Underground	900000	0
Asturiana de Zinc	Open pit/Underground	950000	94
Pyhäsalmi	Underground	213816	16
Hitura	Underground	518331	0
Zinkgruvan	Underground	850000	50
Aitik	Open pit	17700000	0
Garpenberg	Underground	910000	50
Boliden Mining Area Lisheen	Open pit/Underground	1457000 910000	29 50
Tara		1680000	52

Table 3.9: Percent of tailings backfilled at base metal operations

Almagrera uses waste-rock and rock from quarrying (schist) in their back-fill and no tailings. Asturiana de Zinc is filling out a mined out open pit, which explains the high backfill percentage. Zinkgruvan and Garpenberg run backfilling operations which results in 45 – 50 %

of the tailings used in the backfill. The Boliden Mining Area received ore from one open and a series of underground pits. If the ore from the open pit is subtracted from the tailings production the percentage of back-filled tailings is 34 %. This value is misleading since during year 2001 large quantities of waste-rock were brought back underground at Renström, Petiknäs and Åkerberg mines (a total of 140000 tonnes of waste-rock was brought back underground during 2001).

Probably the newest TMF in Europe was constructed recently at the Lisheen mine. This pond was constructed on flat land (paddock style) on a peat bog and is fully lined. Even though it is designed for a maximum amount of 10 Mt of tailings, it is expected to store a total of 6.6 Mt of tailings over the project's life [75, Minorco Lisheen/Ivernia West, 1995].

3.2.3.2.1 Characteristics of tailings

Base metal ores usually contain several metalliferous minerals. Often copper, lead and zinc are mined together. Typically base metals are mined as sulphides. Hence acid rock drainage is a major issue in the management of tailings and waste-rock. Long-term chemical stability is therefore a challenge. The tailings are in the form of a slurry and the ponds and dams can have large dimensions. Many underground mines backfill a proportion of the tailings slurry.

A suite of metalliferous complexes and process chemicals are included in the tailings slurry. Hence physical stability is also of major importance for this sector.

At **Asturiana de Zinc** the tailings are in the form of a slurry, a mixture of water and dolomite, with 65 % solids content and with a solids density of 2.75 t/m^3. The tailings are alkaline at the time of discharge (pH 6.5 to 8) and are reported to be easily compactable and not reactive (due to their alkaline nature).

At **Almagrera** there are two types of tailings. The fine fraction of the tailings and the cinders resulting from the roasting and leaching of the coarse fraction of the tailings. The tailings are mainly pyrite and ARD generating. The cinders are easily leached with water. The tailings have a 66 % solids content and the compact density of the tailings material is 4.0 t/m^3 (mainly pyrite). Upon discharge into the tailings pond the tailings have a pH of approximately 9 but pH in the pond is around 3.2.

At **Pyhäsalmi** the chemical composition of the tailings has been determined and leaching behaviour (max. solubility /DIN 38614-S4 by Kuryk`s method and long-term behaviour) have been investigated in laboratory scale simulation tests. Neutralisation capacity vs. acid formation potential of material has been determined. Also wind erosion tests have been done on laboratory scale. The most significant problems are the contents of heavy metals (As, Cd, Cu, Pb, Zn) and sulphur, resulting in ARD generation properties. Alternative processing methods to change the characteristics of tailings have been considered. One example is the selective flotation of pyrite in the tailings to achieve a final S-content of less than 1 %. It was decided that it is technically possible, but economically not viable. The process would generate a product (pyrite) that is impossible to sell and that requires special techniques and arrangements to deposit or destroy.

Mixing of peat with the tailings when it is pumped to tailings area to create reducing conditions has also been investigated. The test was stopped because of technical difficulties, but the idea is to continue the investigation on laboratory scale. The down side of this technique is the fact that a natural resource is "consumed".

The particle size distribution of tailings material is 65 % <74 µm.

At the **Hitura** the same tailings examinations as at Pyhäsalmi have been performed. The most significant problems with the tailings are the contents of Cu and Ni. The tailings will not produce ARD because the buffering capacity of the tailings is higher than acid formation potential.

The particle size distribution of the tailings is 60 % <74 μm.
[62, Base metals group, 2002]

At **Aitik** the main issue for the closure and decommissioning plans for the tailings pond is the possible acid generating potential. Due to an early assumption that the material would produce ARD, a number of options to change the composition of the material have been investigated. In its crude form, the tailings have an ABA value of -13 kg $CaCO_3$/t, ruled by the pyrite content (0.9 % S). Flotation tests as well as sampling of various products in the mineral processing plant have yielded a range of samples with sulphur content ranging from 0.12 % for de-pyritised tailings to 31 % for the pyrite flotation product. These samples have been subjected to humidity cell tests in different campaigns.

The results from kinetic tests and modelling indicate that the silicates in the tailings constitute a substantial acid consuming capacity. More important, however, is the sulphide oxidation rate in the field. The dissolution of silicates is capable of consuming the acid produced by pyrite oxidation up to a certain rate. Below that rate, the carbonates are slowly consumed, but above that same rate, the carbonates are depleted after some time, after which the silicates alone are unable to neutralise the acid generated.

Field oxygen flux measurements have been carried out to illustrate the material's behaviour in field scale. The results indicate acid production will take place, corresponding to the silicate acid consumption capacity of only the top 20 cm layer of tailings. In lower strata, no acid will be produced, indicating a vast excess of buffering capacity.

In Aitik, where frost conditions prevail for seven months of the year, the kinetics differ significantly from the conditions in the laboratory and during the actual field test. To verify that the tailings do not possess ARD capabilities, also column tests have been carried out, under conditions which are representative for the unfrozen period at Aitik. In this test, the measured oxygen consumption rate was 50 % below the lowest oxygen consumption rate calculated from sulphate export in the humidity cell experiments.

Parallel to those tests, hydrogeological modelling of the groundwater flow within the pond have shown that over 90 % of the volume will be permanently water saturated, which is technically equal to sub-aqueous tailings management. Only minor areas at the upstream and downstream dams may become unsaturated at times. To address the situation, a solution has been worked out suggesting a wetland be established in the lower parts of the tailings pond. Unsaturated areas in the lower parts of the pond would then be avoided, leaving the problem for only a few percent of the tonnage, at the upstream dam, unresolved at present.

A possible solution for the remaining, upper part of the pond, is pyrite separation and selective management of pyrite (de-pyritisation). Such a solution, however, does not eliminate a possible problem, it only concentrates the pyrite into a high-potential acid generating material, which calls for a technical solution of high quality and low risk. Such a solution could, e.g. be deposition of this material in the bottom of the mined out open pit upon closure whereby it would remain permanently covered by water.
[63, Base metals group, 2002]

At **Garpenberg** the tailings were investigated with regard to composition and weathering characteristics. Used methods were mineralogical investigations, full rock analysis, acid base accounting (ABA) and kinetic weathering tests (extended humidity cell tests conducted between 1995 and 1999) in combination with predictive modelling. All results show that the tailings will not produce ARD. The metal concentrations in the pore water of the tailings will be solubility limited at the naturally high pH within the pond even if the tailings are allowed to weather with full access to atmospheric oxygen. The metals mobilised by sulphide oxidation at the surface of the tailings will be immobilised by absorption and precipitation as they are transported through the tailings. Based on these results it was concluded that no measures were necessary in order to limit the mobilisation of metals by weathering from the deposit at closure.

The tailings presently produced show larger variations in mineralogy as other parts of the orebody are mined with higher sulphide content, primarily higher content of pyrrhotite (FeS). According to sampling and analysis done during year 2001, it is predicted that these “new” tailings will produce ARD (see the detailed analysis in the table below).

Following the development of the weathering characteristics of the tailings is considered important, even though the planned decommissioning method (flooding) is well suited for potentially ARD producing tailings. Therefore sampling and testing of the tailings will continue in the future.
[64, Base metals group, 2002]

Element	Concentration (mg/kg)
As	56.3
Ba	338.8
Be	0.45
Ca	30933
Cd	18.6
Co	6.1
Cr	3.2
Cu	317.7
Fe	65533
Li	4.6
Mn	4163
Mo	2.9
Ni	7.8
P	149
Pb	4011
S	44600
Sn	<5
Sr	19.6
V	9.5
Zn	7051

Table 3.10: Average results of tailings analysis at the Garpenberg site (2001)
[64, Base metals group, 2002]

Some of the key information regarding the tailings deposited in the tailings pond can be listed as follows:

- 500000 tonnes of tailings/yr
- 20 % solids
- typical particle size distribution (% passing) (K_{50} = 20 µm, K_{80} = 64 µm).

size (µm)	Cumulative % passing
500	100
350	99.8
250	99.7
180	99.4
125	97.5
90	93.3
63	79.1
45	68.1
20	50.8
10	31.6

Table 3.11: Size distribution of tailings at Garpenberg site
[64, Base metals group, 2002]

Some of the key information regarding the tailings used as backfill at Garpenberg are:

- 450000 tonnes of backfill/ year
- 80 - 85 % solids.

size (µm)	cumulative % passing
250	96.6
180	86.8
90	46.4
45	18.8

Table 3.12: Typical size distribution of backfilled tailings at Garpenberg site
[64, Base metals group, 2002]

The **Boliden** mining area consists of complex sulphide mineralisations. Mining in the area started at year 1925 and to date approximately 30 mines have been exploited in the area. The tailings in the pond consequently have variable chemical characterisations and physical-chemical properties. The characteristics of the tailings produced today are summarised in the tables below. The fine fraction after cycloning is deposited to the tailings pond and the coarse fraction is used as backfill in the underground mines.

Size	Total tailings	Cyclone overflow to pond
µm	cumulative % passing	cumulative % passing
350	100	100
250	99.9	100
180	99.7	100
125	97.8	100
88	93.5	95.6
63	85.9	87.8
45	76.6	78.3
20	53.2	54.4
-20	0	0

Table 3.13: Particle size distribution of tailings at Boliden site
[65, Base metals group, 2002]

The tailings have the following composition before cycloning and CN leaching:

- Au: 0.85 g/t
- Ag: 24.9 g/t
- Cu: 0.10 %
- Zn: 0.40 %
- Pb: 0.13 %
- S: 17.8 %

More than 50 % of the tailing consist of particles less then 2 µm. The tailings slurry pumped to the tailings pond contains 20 - 25 % solids. The density of the tailings, as placed in the pond, is 1.45 t/m^3.
[65, Base metals group, 2002]

At **Zinkgruvan** the tailings contain mainly quartz, feldspar and calcite. Small quantities of sulphides are also present (sulphur content of <0.25 %). The calcium content is approximately 8 %. The ratio between sulphur and calcite is <0.1 implying well buffered tailings that will not produce ARD. Weathering tests have also shown that the tailings have a low weathering rate. The composition of the tailings is given in the table below.

Mineral	**Weight-%**
SiO_2	62.4
TiO_2	0.3
Al_2O_3	11.8
Fe_2O_3	0.6
FeO	2.9
MnO	0.7
MgO	2.2
CaO	7.0
BaO	0.01
Na_2O	0.6
K_2O	4.9
$H_2O^{110-350}$	0.1
CO_2	2.1
B_2O_3	0.1
FeS	0.5
ZnS	0.2
PbS	0.1
Other minerals	3.3
TOTAL	**100**

Table 3.14: Chemical analysis of tailings at Zinkgruvan site
[66, Base metals group, 2002]

The tailings have an in-situ permeability of 10^{-5} - 10^{-6} m/s and an in-situ density of 1.35 - 1.45 t/m^3.

For the **Las Cruces Project** it is predicted that the tailings generated during the estimated lifetime of the project will be approximately 4 Mm^3 (or 15 Mt). The tailings are pyritic and are predicted to be ARD generating. The average grain size is estimated to be 100 µm. The tailings will be deposited "dry" after dewatering, with an approximated moisture content of 7 - 8 % [67, Base metals group, 2002].

The tailings are delivered to the TMF at **Lisheen** at about 35 % solids and contain zinc, lead, some process reagents and metal salts and have a grain size of 80 % passing 95 µm. The density of the tailings on a dry basis is 3.5 g/cm^2, the in-situ density is about 1.7 g/cm^2. ABA was performed at the permitting state and the tailings are predicted to be acid generating [75, Minorco Lisheen/Ivernia West, 1995].

3.2.3.2.2 Applied management methods

At **Asturiana de Zinc** 94 % (900000 out of 950000 t/yr) of the coarse tailings, which are filtered to 15 % moisture content, are used to backfill an old open pit. The remaining 50000 t/yr, because of limited filter capacity, are deposited in a tailings pond. The capacity of the pond is 2.6 Mm^3 and it currently contains approximately 2.5 Mm^3 of tailings. The dams are constructed of borrow material. The pond is built on top of natural soil. The decant water is discharged to the recipient after having passed a series of clarification ponds. No water is recycled back to the mineral processing plant. 100 % of the required 2.2 Mm^3 of process water are pumped from the mine [54, IGME, 2002].

At **Almagrera** the coarse fraction of the tailings (33 % or 300000 t/yr) are roasted and sulphuric acid is produced. The cinders are then leached with sulphuric acid and copper extracted in a SX-EW process. The cinders are deposited in a cinders pond. The remaining 66 % of the tailings (600000 tonnes fines) are deposited into a tailings pond. The dam is constructed without lining directly on the natural ground. It is an earth dam with a core of compacted clay. The volume of the dam is at present 3.2 Mm^3. Leakage through the dam is back pumped into the pond. Clarified water is pumped to a water treatment plant (liming) and treated before discharge. The emergency outlet is constructed in natural bed-rock.

The cinders are deposited in a cinders dam.
[61, IGME, 2002]

At **Pyhäsalmi** 16 % of the tailings are used in the backfilling of the mine, the remaining 84 % (180000 t/yr) are deposited in a tailings pond. This relatively low backfill percentage can be explained by the fact that only the coarse tailings are suitable for backfilling. The total area of tailings management facilities is about 100 hectares including three tailing ponds. Two of those ponds (pond B and D in the figure below) are used parallel for settling of solids and to decant clarified water to the third pond (pond C in the figure below). The residence time of the tailing water in the area is about two months.

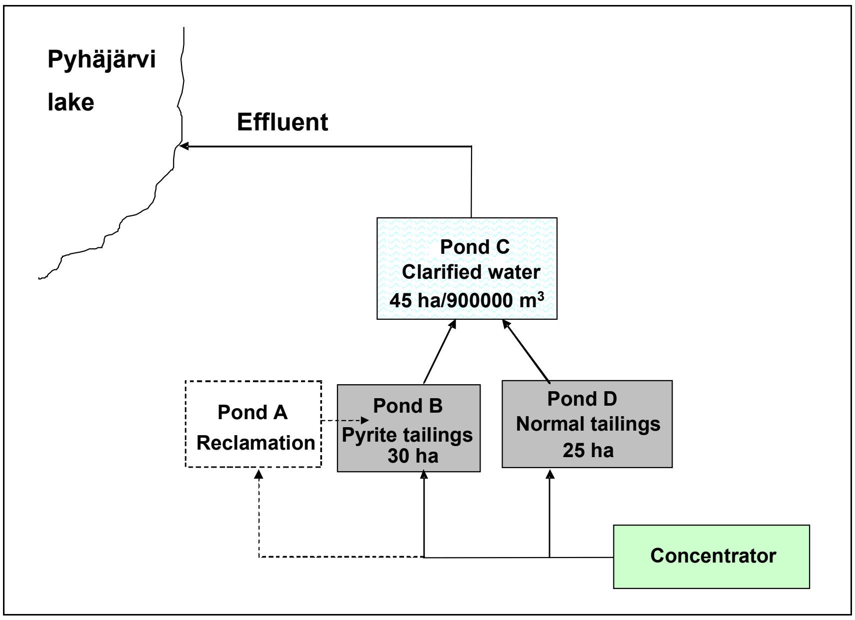

Figure 3.22: TMF set-up at Pyhäsalmi site
[62, Base metals group, 2002]

Pond A in the figure above is completely filled and is not in use any more. Reclamation work for this pond has been started in 2001. It will be covered with 80 cm thick layer of soil material (30 cm clay and silt and 50 cm moraine). The central part of the pond will be remaining under water.

Before construction of the tailings area the soil had been studied. The soil is considered sufficiently impermeable (silt) to prevent leakage to ground water and stable enough to carry the load of tailings material. Base line studies were also performed on the down-stream lake systems.

The tailings area has been built paddock-style on a flat terrain. The base dam has been made of moraine. The tailings are distributed from spigots around the first tailings pond and the clarified water is led forward from the centre part of the pond via a decant tower. The necessary raises of the tailings dams are done with tailings material. The dam of the clarification pond is constructed of moraine and lined with broken rock to prevent erosion. The area is surrounded by a ditch to collect seepage water, which is pumped back to the tailings pond.

The distance from the mineral processing plant to the TMF is about 500 m and 200 m to the nearest lake.

The annual rainfall in Pyhäsalmi is about 650 mm. The mean temperature during a year is 1 - 3 ºC. The maximum temperature in summer time is 30 °C and the min. temp in winter is –35°C. During 5 months in a year the temperature is under zero and during 6 months above zero.

The tailings management area was designed in the beginning of the 60's and no closure or aftercare plans were taken into account in the design stage.

The operational routines include daily control of the facility, regular monitoring of the phreatic surface level in the dams, monitoring of discharged water and audits of the facilities.

At **Hitura** the tailings area, 110 ha in total, is divided into three ponds. The tailings (480000 tonnes in 2000) are discharged into the first pond. The two others are clarification ponds. The solids settle in the first pond and the clarified water is decanted via a tower and led to the next pond from the central part of the tailings pond. Clarified water is re-used in the mineral processing. Only the excess water is led to the river system. The tailings pond is off-valley-site type. The starter dams have been made of moraine. The tailings are distributed with spigots. The dams are raised every 12 to 15 months using tailings.

The dams of the clarification ponds have been made of moraine and are lined with coarse gravel to prevent erosion. The distance from the mineral processing plant to the TMF is about 500 m. The distance from the tailings area to the nearest river is about 3 km.

Problems with infiltration of water from the tailings pond into the groundwater exist. Groundwater and drainage water are pumped into the pond in order to control the groundwater flow and minimise the impact.

The annual rainfall in Hitura is about 550 mm. The mean temperature during a year is 1 – 3 °C. The max. temp. in summer time is 30 °C and the min. temp in winter is –35 °C. During 5 months in a year the temperature is under zero and during 6 months above zero.

Before construction of the tailings management area the soil was investigated, but apparently not carefully enough, because in one location infiltration to ground water is occurring. The affected groundwater is monitored in groundwater monitoring wells located downstream of the tailings pond and the back-pumped water is sampled.

No special requirements exist for the closure and aftercare of the tailings pond.

The operational routines include control of the facility three times a day, regular monitoring of the phreatic surface level in the dams, monitoring of discharged water and audits of the facilities. No risk assessment has been made at the site.
[62, Base metals group, 2002]

At **Aitik** the tailings are pumped to a 14 km^2 (7 km x 2 km) tailings pond. Four pipelines (rubber lined steelpipes) are used for this purpose of which only two are normally in use. All four lines are equipped with five pumps in series. Total installed power for each line is 2000 kW. The water from the tailings pond feeds into a clean water clarification pond.

The tailings pond is limited by the topography (valley-site type) and four dams (A-B, C-D, E-F incl. the E-F extension, and G-H), see figure below. The tailings are pumped as a slurry to the discharge area along dam A-B. This process, spigotting, leads to an accumulation of the coarser particles close to the dam A-B, while the finer fractions are successively settling along the pond towards the downstream dam, where separated water is collected. The active water volume in the tailings pond is normally about 2 Mm^3, which occupies about 1/5 of the pond's area. The water was, until a dam failure occurred in 2000, decanted through a spillway (tower type) at the dam E-F extension for final clarification in the downstream located clarification pond.

The clarification pond is located west of the tailings pond, downstream dam E-F and dam E-F extension. The pond's area is 1.6 km^2 and the holding capacity is about 15 Mm^3. This pond serves as

- the final treatment step for the process water
- serves as a reservoir for process water
- and as a buffer water from spring snowmelt and precipitation events.

The freeze-in of water in the tailings during winter is a climatic effect that is of particular importance for the water balance. At excessive precipitation and snowmelt, water is discharged from the pond to the receiving streams. Discharge of water is possible, when necessary, also from the recycling water channel.

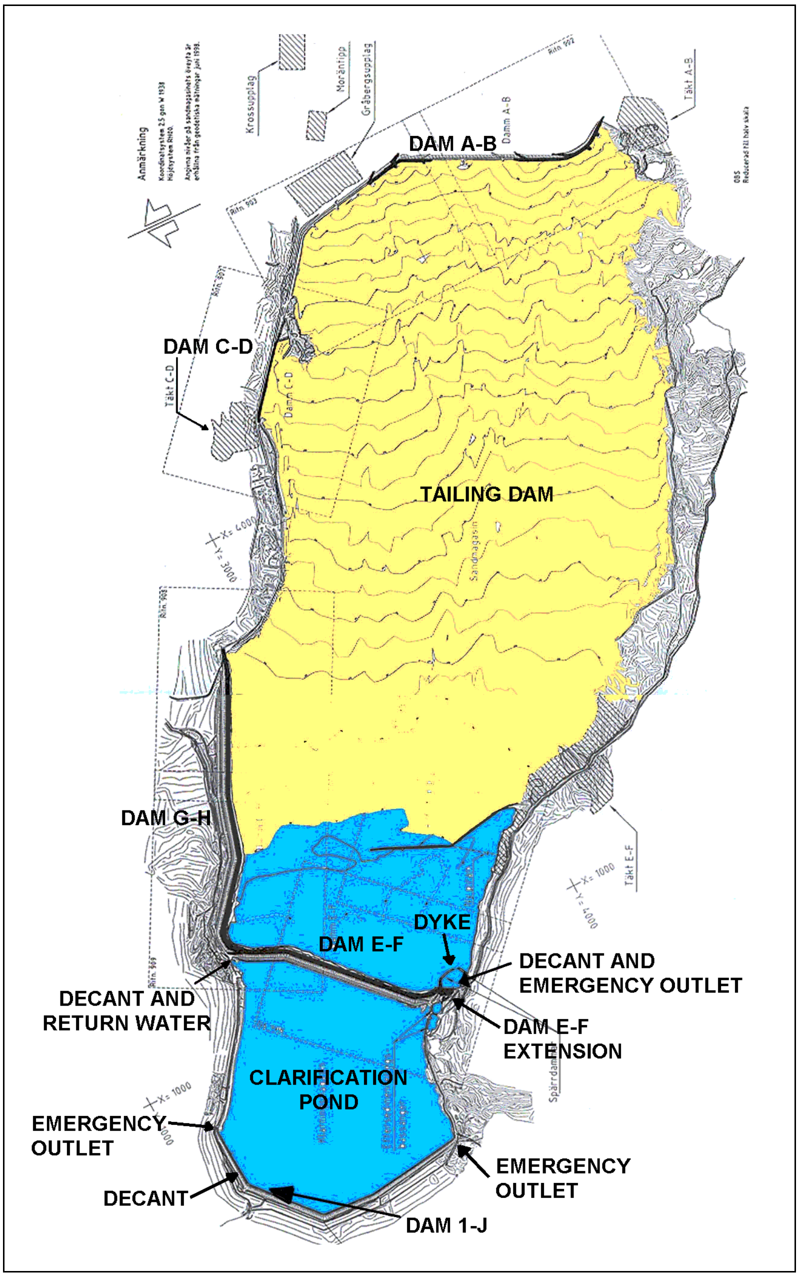

Figure 3.23: Year 2000 situation of Aitik tailings and clarification ponds
[63, Base metals group, 2002]

The dams surrounding the pond were constructed starting 1966 and have been raised since then mainly applying the upstream method. Each raise comprises about 3 m. Material used for the raises have been till for sealing cores and waste-rock for the support fill. For the construction of the dam E-F extension, which started 1991, the downstream method has been used, with the crest of the dam moving outwards from the pond.

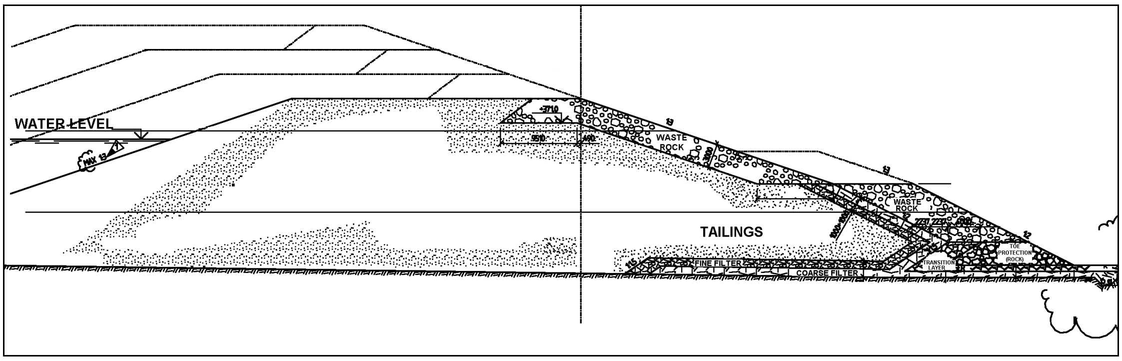

Figure 3.24: Cross-section of dam E-F2 at Aitik
[63, Base metals group, 2002]

All mining voids created at **Garpenberg** are backfilled with waste-rock from development works and tailings. The concentrates constitute about 10 % of the ore processed which means that 90 % become tailings. 50 % of the tailings are used for backfilling. When the ore is blasted, crushed and ground the volume increases by about 60 % which means that the volume of tailings in Garpenberg is about 145 % of volume of mined ore. There are no possibilities to backfill more tailings underground for geometric reasons.

The tailings are cycloned in order to separate fine and coarse particles. The coarse particles are filtered to remove water and allow transport by trucks and mixing with cement to stabilise the tailings in one of the mines. After mixing with water, the cemented backfill is transported hydraulically to mined-out areas of the mine and excess water is removed by a draining system.

The tailings pond presently used in the Garpenberg area is located approximately 2 km south-west of the mineral processing plant. Before applying for the latest permit to increase the height the tailings pond, various alternative tailings management methods were investigated, such as:

- thickened tailings and
- sub-aqueous discharge into a lake.

These alternatives were disqualified because of costs (thickened tailings) and the public opinion against sub-aqueous deposition.

The presently active part of the tailings pond covers approximately 35 ha. The lifetime of the pond depends on the tailings production rate but is approximately 8 years assuming the present production rate. The tailings have an effective density of 1.5 t/m^3. The dam is raised using the downstream method (see Figure 3.25).
[64, Base metals group, 2002]

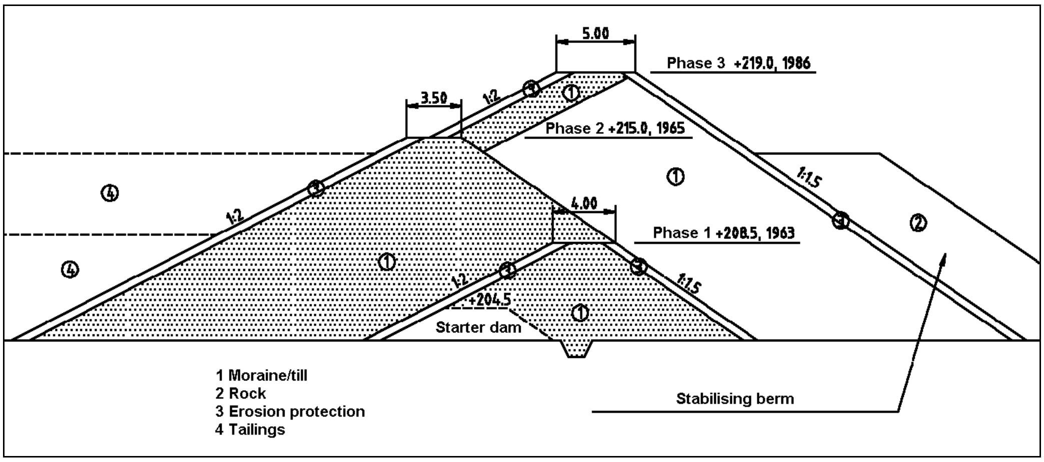

Figure 3.25: Cross-section of dam at Garpenberg before latest raise
[64, Base metals group, 2002]

Investigations are ongoing in order to determine the possibilities to raise the dam using the upstream method.

The water discharge from the pond in 2001 was 4.55 Mm^3. Of this 50 % were re-used in the mineral processing plant. The remaining 50 % were discharged to surface waters. The catchment area for the tailings pond is 1.56 km^2.
[64, Base metals group, 2002]

The tailings management at the **Boliden** area has been described in the Section 3.2.2.7.

At **Zinkgruvan** the mining method used requires backfilling. Up until 2001 hydraulic backfill had been used. This type of backfilling requires a drainage capacity of the tailings of at least 5 cm/h. This is why the coarse fraction had been extracted from the tailings using cyclones whereby the fraction > 50 µm was returned to the mine. In this way approximately 50 % of the tailings were backfilled using hydraulic backfill. The fine fraction of the tailings had been pumped to the Enemossen tailings pond.

A change in mining method to using “panel stoping” requires paste backfill. This removes the requirement of the drainage capacity of the fill and thereby allows the use of the fine fraction of the tailings in the backfill. In this way it is anticipated that up to 65 % of the tailings will be possible to backfill. Furthermore, the tailings pumped to the tailings pond will also contain the coarse fraction which will enable the use of the tailings in the construction of the dams. This method is now implemented in Zinkgruvan so the hydraulic backfilling is not in use any longer.

The tailings that are not backfilled are pumped together with the process water from the mineral processing plant to the tailings pond, located 4 km south, in pipelines. The solids sediment in the tailings pond and the supernatant water is led by gravity to a clarification pond 1 km from the tailings pond for additional clarification. In order to evenly fill up the tailings pond and to avoid dusting and oxidation of the tailings, the spigotting points are continuously moved along piers constructed of waste-rock. Water is re-circulated back to the mineral processing plant from the clarification pond (see water balance). Water is also discharged through a pipeline and a tunnel to the recipient. The tailings pond and the clarification pond are formed by natural basins (valley site type).

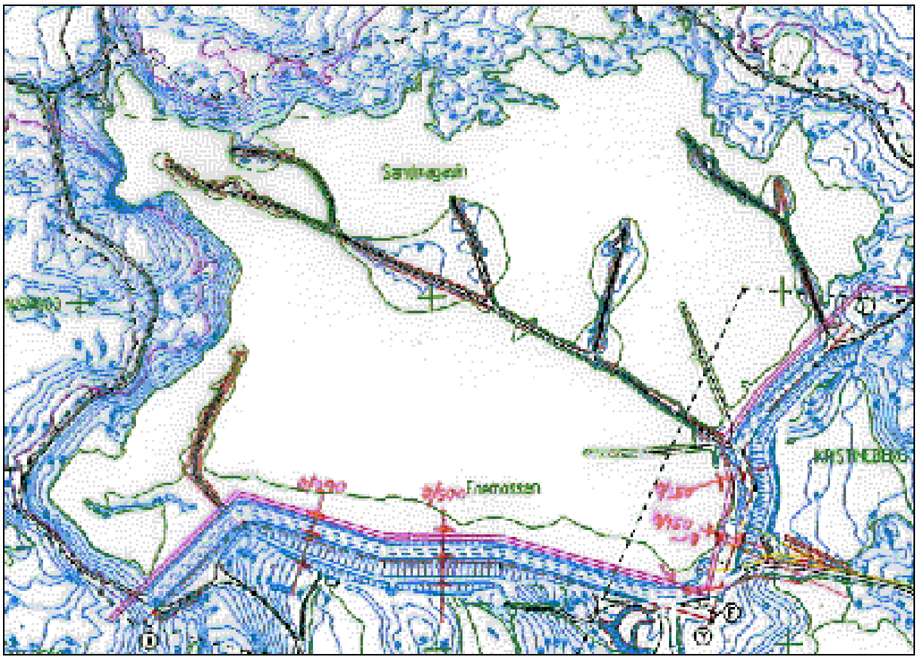

Figure 3.26: Top view of the Zinkgruvan TMF
[66, Base metals group, 2002]

The tailings pond is constructed in a valley and surrounded by natural slopes and two dams. The pond is founded on a peat bog and currently covers approximately 50 ha. At its final height it will cover approximately 60 ha. The embankments are of zoned construction, comprising erosion protection rock-fill on the upstream face, an inclined low permeability till core, a filter layer of sized screen rock and a downstream shoulder of rock-fill. The characteristics of the dams and the tailings pond is given in the table below.

Characteristic data	Dam X-Y	Dam E-F
Used capacity Dec. 2000	5.7 Mm^3	
Permitted capacity (from 1981)	7.0 Mm^3	
Total tailings pond area	50 ha	
Total clarification pond area	16 ha	
Volume of material in dams	380000 m^3	170000 m^3
Material from external borrow area	70000 m^3	30000 m^3
Dam height	27 m	17 m
Crest length	800 m	400 m
Crest width	16 m	16 m
Slope angle of upstream slope	1:1.5	1:1.5
Slope angle of downstream slope	1:1.5	1:1.5
Width of stabilising berm	7 m	7 m
Slope angel of downstream side of berm	1:1.5	1:1.5

Table 3.15: Characteristic data for the existing dams X-Y and E-F at Zinkgruvan site
[66, Base metals group, 2002]

To avoid dusting and oxidation sub-aqueous discharge is practised. However, to lower the phreatic surface a 30 - 50 m beach with a height of 0.1 - 0.5 m above the water level close to the dam is required. When discharging tailings under water, the angle of repose is significantly steeper than in the case of discharge above the water level. In order to evenly fill up the pond the spigotting points are continuously moved along piers constructed into the pond. The beach is irrigated during the dry period of the year (spring-summer-autumn). During periods with no snow during the winter dusting cannot be entirely avoided, even though several methods of temporary covering etc have been tried.

The decant system is tower-type. Decant water flows by gravity to the clarification pond. 50 % of the decant water is re-used in the mineral processing plant. An emergency outlet is constructed, which automatically discharges the water if the level increases above a certain level. The installed discharge capacity is 0.7 m^3/s (not counting the emergency outlet discharge capacity) which corresponds to the 100 year rain event and a maximum increase of the water level in the pond of 0.5 m.

The dams E-F and X-Y are constructed as conventional dams for tailings ponds. The foundation of the dams is on natural bedrock partly covered with moraine or peat soil. Excavations were done below the dams down to natural bedrock or at least 4 m into the moraine for the connection between the low permeable core of the dam and the underlying foundation. The low permeable core is constructed of compacted moraine from a borrow pit area. The permeability of the moraine is between 1 x 10^{-8} and 1 x 10^{-9} m/s. During the construction of the dams QA/QC was conducted continuously on the moraine and the filter material, mainly compaction tests/control and material characterisation (grain size distribution).

Hydrogeological studies of the area show that the bed-rock in the area contains several fracture zones. The fractures are permeable and drained which results in some drainage of the water from the pond. The tailings in combination with the permeability of the dams also results in some seepage from the pond. The water balance for the pond is given in the figure below.

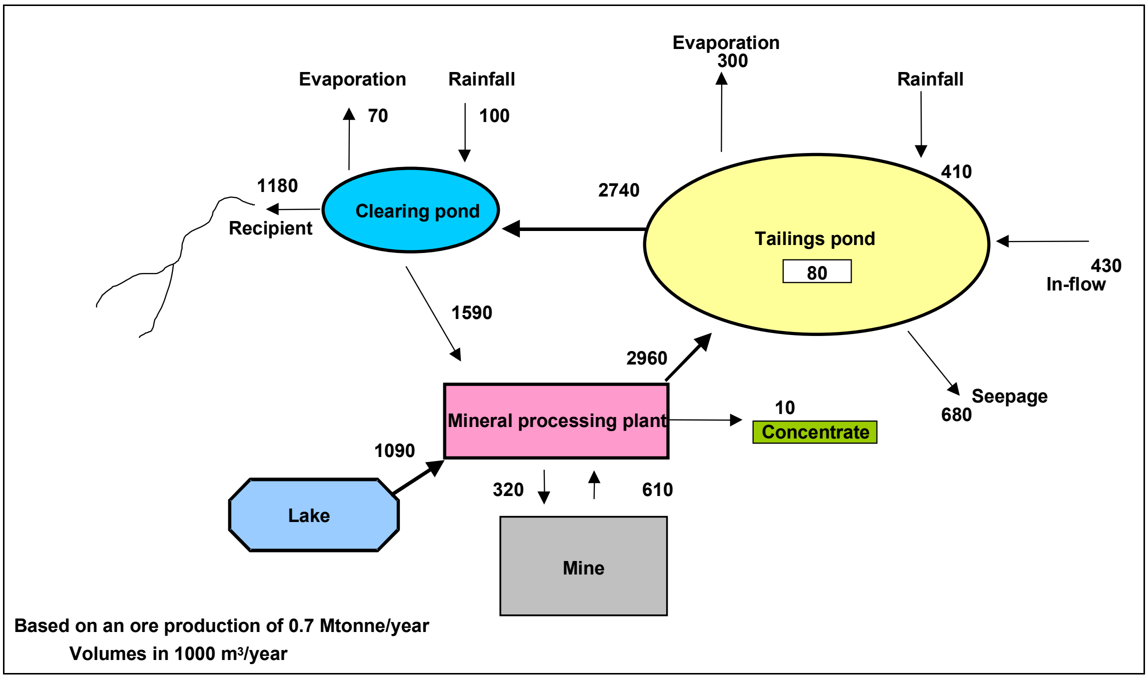

Figure 3.27: Water balance for the Zinkgruvan operation
[66, Base metals group, 2002]

In the design phase of the **Lisheen** TMF all primary methods of tailings management were discussed and evaluated. The decision-making process will be explained in the following paragraphs.

<u>The design of a new TMF at Lisheen</u>
In the decision-making process that led to the preferred method of tailings management the various methods available were investigated for the basic construction and more detailed design criteria for the TMF. This process is described in the following.

Primary tailings management methods
Three primary methods of tailings management were investigated in the design phase:

- into a surface water body such as a lake, river or sea
- into the mine workings as backfill
- into a surface tailings pond.

The first of these options was considered environmentally unacceptable. However lake deposition, under managed conditions has been accepted as best practice in several northern Canadian operations. The operator had adopted the philosophy that the most desirable tailings management strategy is to maximise the use of tailings as backfill in the underground workings. This was thought to have the advantages of.

- minimising the volumes of tailings to be managed on surface
- supporting the hanging wall so that surface subsidence is minimised
- managing of the tailings in an underground environment that will be permanently under water after closure, hence oxidation will be avoided
- maximising the recovery of ore.

The layout of the mine and the mine sequencing make it possible to backfill 6.9 Mt tailings underground. The balance of 6.6 Mt of the tailings from the first stage of mining and an additional mass of 3.4 Mt to allow for possible changes to the backfill volume, reducing of the in-situ dry density and a possible second stage of mining, must therefore be managed in a surface impoundment.

The topography at Lisheen, within a reasonable distance from the ore processing plant, is such that no significant valleys or hillsides were available as potential tailings pond sites, and thus a ring-dyke impoundment (paddock-style) was proposed.

Other considerations
It had been identified that the tailings have a potential to generate acid if exposed to oxygen, and that the tailings pore water contains some metal ions. These two facts led to the decision that:

- an tailings pond/dam system to retain water so that the tailings are discarded and stored under water was needed
- the tailings need to be dealt with in a pond that is as impermeable as possible to minimise seepage into the groundwater system.

To satisfy these requirements a low or very low permeability liner with attenuating capability was considered necessary. The extensive bogs in the area contain peat which has a low permeability, making the use of it as a component of a composite liner very attractive. Peat has the added advantage in that it can attenuate the release of many of the likely contaminants in any seepage that may occur.

In order to identify the strength of the peat, its permeability in both the uncompressed and compressed states and its attenuation properties, a programme of tests was commenced.

Selection
It had been established that the maximum mass of tailings to be managed on surface will be 10.0 Mt and the TMF should incorporate a low permeability barrier between the tailings and the local groundwater system. Using average topographical features and a reasonable thickness of tailings an area of 80 to 120 ha will be required. This area is based on the conservative in-situ dry density of 1.6 t/m^3, though subsequent design is based on 1.8 t/m^3, and a relatively low average height of approximately 10 m of tailings.

Since the tailings were found to be net acid generating it was decided that the containment facility must prevent oxidation of the pyrite and be lined to restrict the seepage of water into the groundwater system. Two methods of achieving this were discussed, namely: to provide a composite artificial liner if the site is on farm land or make use of the low permeability and high attenuation potential of compressed peat, as part of a composite liner, if the site is on a bog.

Methodology
The selection of a site for the TMF involved the assessment of the economic, environmental and engineering considerations. The objectives of the selection process were thus to minimise the impacts on the local community and the environment while satisfying the engineering requirements in the most economical way.

The site selection process was carried out in four stages, namely:

1. a regional search for a topographical bowl or valley that would favour a tailings management scheme within a radius of 15 km of the ore processing plant site
2. a localised search to eliminate unsuitable areas within an 8 km radius. This radius was based on pumping considerations and the lack of good topographical sites in the area immediately beyond this radius
3. identification of possible locations
4. a detailed assessment of the possible locations.

[75, Minorco Lisheen/Ivernia West, 1995]

Description of TMF
The TMF was constructed on a bog which consists of up to 4 m of peat overlying a glacial till on limestone bedrock. The limestone is a geotechnically competent lower

carboniferous dolomitised Waulsortian formation with no major faulting, and a low palaeokarst potential. The site investigation found no open or infilled cavities and, for this reason and due to the minimal drawdown that takes place below the TMF, dewatering of the nearby mine does not cause reactivation of palaeokarst features even if these are present.

The TMF consists of an earth embankment forming a dam around the impoundment area. Complete removal of peat from the embankment footprint was performed and the entire embankment is constructed on firm till or bedrock.

The perimeter of the TMF is a wide embankment consisting of zoned, engineered fill with a cross-section designed and built to act as a water retaining structure. The dams are constructed of compacted fill material from borrow pits with upstream and downstream slopes of 1:3 and 1:2 respectively. The dam crest is 6 m wide to provide access during construction and operation. A cross-sectional view of the dam is shown in the following figure.

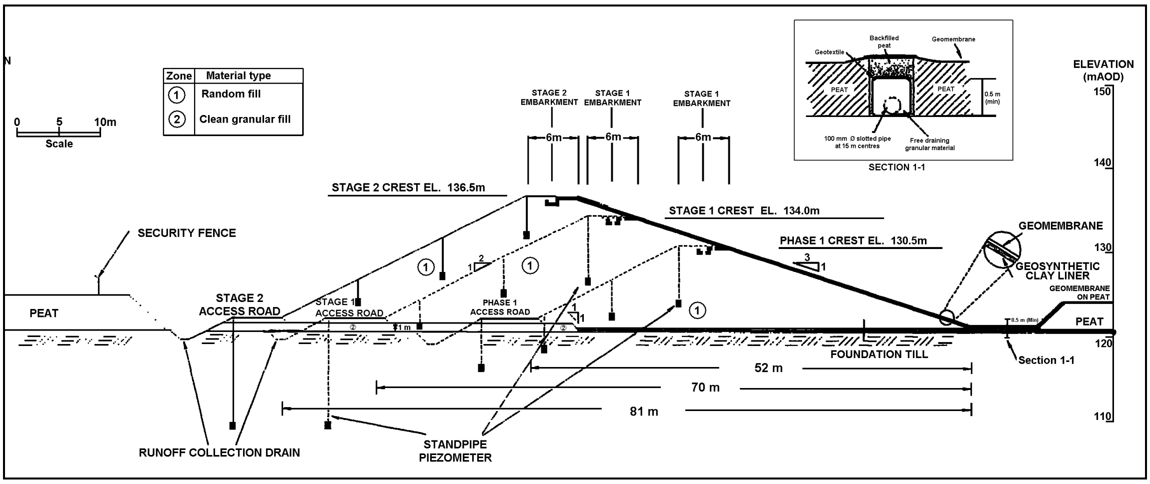

Figure 3.28: Cross-sectional view of dam at Lisheen TMF. Pond is to the right of the dam [75, Minorco Lisheen/Ivernia West, 1995]

The dams have been designed to a maximum height of 15.5 m above the till which lies beneath the bog. This allows for the eventuality that additional capacity may be required, due to the discovery of additional ore reserves or reduction in the in-situ dry density of the tailings or change to the backfilling quantities. The dams are constructed initially to a maximum height of 9.5 m to provide for the 2.8 Mt of tailings which will be stored on surface in the first six years of operations.

Most of the impoundment area will be underlain by the bog. Peat in the bog is generally of sufficient thickness and has the required physical and chemical characteristics to limit seepage and remove various metalliferous constituents from the seepage water. When loaded by the tailings, the peat will compress to become a natural liner with, a permeability of less than 1 x 10^9 m/s. The permeability and strength of the peat are adequate to enable it to act with a geomembrane to form a composite liner capable of containing the tailings and its porewater. A small volume of seepage, estimated to be 34 m^3/day, could pass through the composite liner due to punctures in the geomembrane. It is likely that the majority of this water will be collected in the perimeter drains and pumped back into the impoundment.

Around the inner perimeter of the dams in areas where the peat is less than 1.5 m thick, and on the embankments, a geosynthetic clay liner was placed below the geomembrane, to complete the containment system. A series of 100 mm diameter slotted drainage pipes were installed around the inner perimeter at the level of the base of the peat. These drains will extend from the start of the blanket drain beneath the embankment to 50 m inside the toe of the embankment and will collect some of the water that will be released during compression of the peat and also collect some of the seepage water.

At start-up, prior to deposition of any tailings, the impoundment was covered with water to a minimum depth of 1 m to provide cover over the tailings. Tailings were placed below the water surface by a floating distribution system which was moved slowly back and forth across the impoundment to produce a relatively even layer of tailings so as to minimise differential loading on the peat liner.

Tailings transport water is be returned to the ore processing plant for re-use, and any surplus water in the TMF is treated in the mine water treatment plant prior to discharge into the river system. Due to the net annual precipitation of approximately 450 mm and the low volumes of seepage water there is generally a surplus of water in the tailings impoundment.

The seepage and run-off water from the dams are collected in the surface drain around the TMF and pumped back into the impoundment.
[75, Minorco Lisheen/Ivernia West, 1995]

In short, for the design of the liner and the dams, the following factors were considered:

- stability
    - dam stability
    - foundation stability (in this case peat)
- seepage
  Seepage rates are calculated based on different defect scenarios
- seepage quality
  It is concluded that the seepage water will in general meet drinking water standards partly due to the fact that the peat has the ability to bind metal ions
- decant water and water balance
- tailings conveyance and discharge.

It was decided to discharge the tailings sub-aqueously to avoid oxidation of the sulphides. This will be achieved via floating pipelines (see figure below).

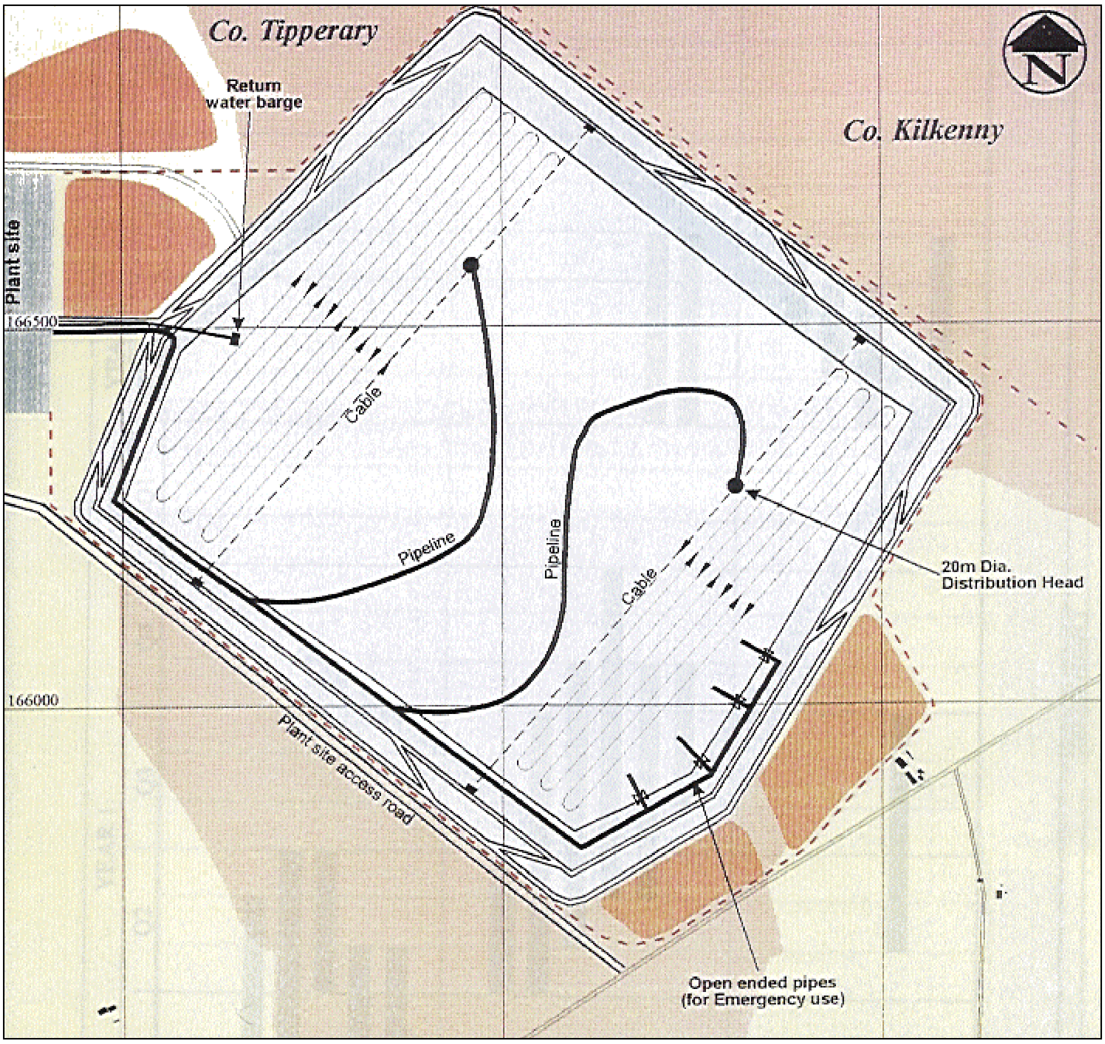

Figure 3.29: Tailings distribution system at Lisheen
[75, Minorco Lisheen/Ivernia West, 1995]

The distribution heads at the end of each pipeline are connected to a reversible, electrically driven winch (see figure below) which passes over a main pulley.

Figure 3.30: Electrically driven winch controlling the tailings distribution pipeline at the Lisheen TMF

Lisheen uses an LLDPE (Linear Low Density Polyethylene) membrane as part of the liner system. The following program was carried out during the installation of the liner

- soil testing of embankment fill material
- destructive & non destructive testing of LLDPE liner
- destructive & non destructive testing of welds on liner
- geosynthetic clay liner testing
- micro gravity survey for potential kaarst features
- liner leak location survey.

The field quality control forms for the TMF liner used:

- geosynthetics inventory control form
- geomembrane panel deployment log
- geomembrane trial seam log
- geomembrance seam log
- geomembrane seam pressure test log
- geomembrane seam vacuum(spark) test log
- geomembrane defect log
- geomembrance log
- geomembrane destructive test record
- geomembrane seam destructive sample log
- gcl panel log
- gcl accessory bentonite test record
- failed destructive sample tracking log.

[41, Stokes, 2002]

However, recent inspections have shown that several leaks and tears have developed in the synthetic liner membrane [76, Irish EPA, 2001]. These have been subsequently repaired.

The operation practices an 'Open Door Policy', which includes:

- environmental information office in the community
- all monitoring data available monthly and annual reports to authorities
- complaints register
- annual schools project.

[41, Stokes, 2002]

3.2.3.2.3 Safety of tailings facility and accident prevention

At **Pyhäsalmi** and **Hitura** the underlying soil was investigated before the dam construction commenced. The system has been designed and constructed so that the surface of water in the tailings area can be kept in balance and the excess of water in case of rainfalls etc. can be removed in a controlled manor, i.e. the ponds have been designed on a calculated water balance. Engineering and stability are controlled by external experts before raising of all dams at the Hitura site. No formal risk assessments have been made at either of the sites regarding the tailings facilities.

The TMF area is controlled daily by the operators of the mineral processing plant and inspected annually by an independent expert and at 5-year intervals by the dam safety authority. The comments are recorded in a 'Dam Safety Document', which is a compulsory document for all similar types of tailings management areas in Finland.

A documented emergency plan does not exist, but it is expected, that an emergency plan will be developed in the near future according to new legislation.
[62, Base metals group, 2002]

The tailings ponds at **Aitik**, **Boliden** and **Garpenberg** follow the routines for dam safety worked out within the OMS manual for tailings ponds (see Section 3.1.4.1). Furthermore, each site follows specific monitoring and surveillance routines. For example at Garpenberg the pore pressure in the dams is monitored on a weekly or monthly basis in 13 piezometers installed in the dam (manual monitoring). Each measured value is compared to an ”alarm level” at which a thorough follow-up investigation is conducted to detect why an abnormal value was obtained. At the discharge point an automatic water level indicator is installed which is coupled to the information system of the mineral processing plant. Every day, the dams are inspected by personnel from the mineral processing plant. The inspections include the slopes, the discharge from the polishing pond and the pipes for sand transportation [63, Base metals group, 2002], [64, Base metals group, 2002; 65, Base metals group, 2002].

At **Zinkgruvan** a risk classification of the tailings pond and the clarification pond has been done according to the RIDAS system (Guidelines for dam safety developed by the Hydro power industry). According to this classification system, where the dams are classified according to the possible consequences of a dam failure, the dams of the tailings pond (E-F and X-Y) are classified as type 1B and the dams of the clarification pond are classified as type 2.

This classification dictates what (minimum) safety measures and control programs need to be followed. In the case of the dams at Zinkgruvan some of the applicable measures are:

- audits of the class 1 dams at least every 3 years and the class 2 dams every 6 years
- class 1 dams need to be able to discharge the 100 year flow event as well as store a class 1 flow event. Class 2 dams need only to be able to discharge the 100 year flow event
- monitoring of class 1 and 2 dams according to the table below.

Parameter	**Consequence class 1B**	**Consequence class 2**
Seepage	X, Continuously	Every 6 months
Movements of the dam crest	X, Every 6 months	(X, Annual)
Movements of the slopes	(X, Every 6 months)	(X)
Pore pressure in the core	(X, Annual)	(X)
Water level in support filling	(X, Every 6 months)	(X)
Water level in the foundation	X, Every 6 months	(X, Every 6 months)
X = measuring should be compulsory where it is feasible. () = measuring is important but can be excluded under some circumstances.		

Table 3.16: Basic measuring to be performed at new dams
[66, Base metals group, 2002]

The stability of the two dams have been controlled by external experts. Results showing a safety factor of 1.5 and 1.6. Nonetheless, a dam safety improvement program is running and comprises among other things installation of piezometer readings, flattening the dam slope from 1:1.5 towards a slope 1:2.5 – 1:3.0 and monitoring of the seepage water flow.

A number of incidents have occurred over the years mainly due to inner erosion of the dams. This has led to changed routines with regard to the deposition technique of the tailings in the dam. In order to lower the pore pressure and thereby avoiding any further inner erosion of the dams a >30 m wide beach is maintained in the upstream side of the dams. The pore pressure level is monitored frequently (monthly, but more often if any unnormal levels are monitored) in installed piezometers in the dams.

A control program for dam safety has been agreed on with the competent authority which contains the following main components:

- yearly external audits of the tailings pond, dams and clarification pond. This inspection also includes pipelines for water and tailings as well as discharge facilities
- weekly inspection of the dams by the environmental department at the site. At these inspections the dams are checked for possible damages, water levels, ice pressures, high precipitation events. Dam leakage flow is measured at the toe of the dams (stable around 5 - 10 l/s). All observations are registered in a logbook
- yearly environmental audits of the entire site that also include the tailings pond facilities
- yearly visits by expert from the competent authority
- established ways of regular communication agreement with the consultant that has designed the dam.

From 2001 on piezometer readings in order to register the hydraulic gradient over the dam have been included in the monitoring program. In total 21 manually monitored piezometers have been installed. In addition, 3 control wells have been constructed to better monitor and control seepage water flow and quality. The dam seepage flow collection and measurement facilities are shown in the figures below. Instrumentation for reading the electrical potential gradient in order to register water streaming through the embankment dams offers an additional method to monitor dam conditions.

Figure 3.31: Ditch for collection and flow measuring of seepage water alongside dam [66, Base metals group, 2002]

Figure 3.32: Another ditch for collection and flow measuring of seepage water alongside dam [66, Base metals group, 2002]

A dam safety manual is currently being prepared in order to cover all issues connected to the tailings management. The manual will cover the following areas:

- dam safety organisation
- emergency and contingency plans
- risk assessment, environment impact and consequence classification
- design and construction
- hydrology and decant system
- systematic monitoring
- plans for closing the facility
- official permits and other documents of importance.

[66, Base metals group, 2002]

At Lisheen the following monitoring scheme is applied for this TMF:

Location	Parameter	Monitoring Frequency	Analysis Method/Technique
Piezometers in TMF Embankment	Water level pH Conductivity Pb, Zn, As, Fe, Cu, Hg, Co, Cr, Mg, Mn, Cd, Ni, CN, Sulphide & Sulphate	Weekly Weekly Weekly Monthly	Dip Meter Electrometric Electrometric Standard Method Note 1
Hydrostatic pressure cells on base of TMF	Hydrostatic pressure	Monthly	Agreed method (c.f. condition 7.4.12)
TMF Retaining Wall	Standard walk-over condition & stability checks Embankment Settlement/ movement Annual safety inspection report	Weekly Quarterly Annually	Visual Survey of seven fixed movement monitoring stations Agreed standard.
TMF embankment crest	Tailings distribution system	Twice daily	Visual
TMF	Tailings settlement/peat consolidation	Bi-annual	Agreed geophysical methodology. (c.f. Condition 7.4.11)
TMF	Volume of tailings disposed Tonnage of tailings disposed Used Capacity Remaining Capacity	Continuous Monthly Annual Annual	Flow meter Dry Density Agreed method Agreed method
Use of spigot distribution system	Period and volume/tonnage Efficiency of distribution	Continuous during use	Record Log Visual
Tailings distribution heads	Depth to tailings	Continuous	Agreed method (c.f. condition 7.4.13)
TMF Perimeter Drain (min. six N o selected locations).	Water level pH Conductivity Pb, Zn, As, Fe, Cu, Hg, Co, Cr, Mg, Mn, Ni, Cd, CN, SS, Sulphide and Sulphate,	Weekly Weekly Weekly Monthly	Dip meter/gauge Electrometric Electrometric Standard Method Note 1
TMF perimeter groundwater monitoring wells (Inner and outer rings)	Water level pH Conductivity Pb, Zn, As, Fe, Cu, Hg, Co, Cr, Cd, Mg, Mn, Ni, CN, Cl, PO4, Cr, NO_2, NO_3, Na, DS Sulphide & Sulphate,	Monthly Monthly Monthly Monthly	Dip meter/gauge Electrometric Electrometric Standard Method Note 1

Figure 3.33: Example of monitoring scheme of TMF
[41, Stokes, 2002]

In the following two dam failures are described. The descriptions provide useful suggestions for safe management of tailings management facilities.

The Aitik dam failure

On the night of September 8, 2000, a dam failure occurred at the Aitik site. The failure took place in a section of the dam which separated the tailings pond from the downstream clarification pond. The event led to the discharge of 2.5 Mm^3 of water from the tailings pond section into the clarification pond. The subsequent rise of the water level in the clarification pond, 1.3 m, caused a controlled discharge of 1.5 Mm^3 clarified water into the receiving streams. This resulted in a temporary rise of the suspended solids content in the river system downstream.

The event is especially worrying because it happened at a tailings dam that had been closely monitored by manual and automatic monitoring systems before the accident. In addition, the dam was operated according to a recently developed OMS-manual.

Two theories to explain this event have been developed:

According to the first theory, the filter layers in the dam were not performing properly, so that the pore pressure within the pond increased causing erosion or sliding failures in the support fill, eventually resulting in a complete dam failure. Detrimental leaks with elevated pore pressure as a result may also have occurred

- along the discharge culvert through the dam
- through the narrow upper section of the impermeable core
- underneath the sheet pile at the culvert
- through cracks in the bedrock
- from the right side of the breach.

According to the second theory, inner erosion occurred along the discharge culvert, possibly in combination with openings in the joints between culvert elements and/or collapse of the culvert. Break-in of water and soil to the culvert, probably caused a sinkhole in the dam with flow directly from the pond into the culvert. The failure escalated and ended with overtopping of the dam and, eventually, a complete failure.

It will probably not be possible to fully eliminate one theory in favour of the other, mainly because the dam was completely eroded away. The operator, however, interprets the results as leakage in connection with the culvert being the main cause of the failure. The reasons for this conclusion are:

- the culvert was founded on gravel, 16 – 50 mm, and at the last reconstruction covered by filter cloth. Leaks through joints and/or in the gravel have occurred which is proven by investigations after the accident, when tailings were found, that did originate from the accident
- the culvert was not equipped with a longitudinal reinforcement, and could therefore not withstand tension.

In addition, some conditions indicate that high pore pressure was not the main cause:

- as late as the evening before the failure, no visible leaks could be observed along the toe of dam E-F extension. This indicates that the failure occurred rapidly
- calculations show that before the accident the dam had a safety factor exceeding 1 even at increased pore pressure.

The operator has therefore concluded that leaks and/or collapse of the culvert are the most likely causes of the accident. However, it cannot be ruled out, that also increased pore pressure caused by deficient filter function may have contributed to the accident.
[63, Base metals group, 2002]

As one consequence of this accident more stable culvert has been constructed, which will in the future be replaces by an overflow system built around the dam in natural rock. Future dams will be built with two filter layers, coarse and fine.

Please provide more detailed information on culvert, overflow and new dam design.

The Aznalcollar dam failure

The Aznalcollar event has been described in many places. Here only the main causes for the failure and the conclusions will be described.

On the night between April 24 and 25 1998 a 600 m section of the downstream dam of the tailings pond suddenly slid up to 60 m. The slide created a breach in the dam through which water and tailings were flushed out. In a few hours 5.5 Mm^3 of acid and metal rich water flowed out of the pond. The amount of tailings that was spilled has been estimated to be between 1.3 and 1.9 Mt. Due to the fine particle size of the tailings (k_{80} < 45 µm) they were easily transported in suspension with the flood wave.

The direct cause of the accident was a fault in the marls 14 m below the ground surface (see figure below). This fault was the result of surplus pressure in the interstitial water of the clay due to the weight of the dam and the tailings deposit.
[68, Eriksson, 2000]

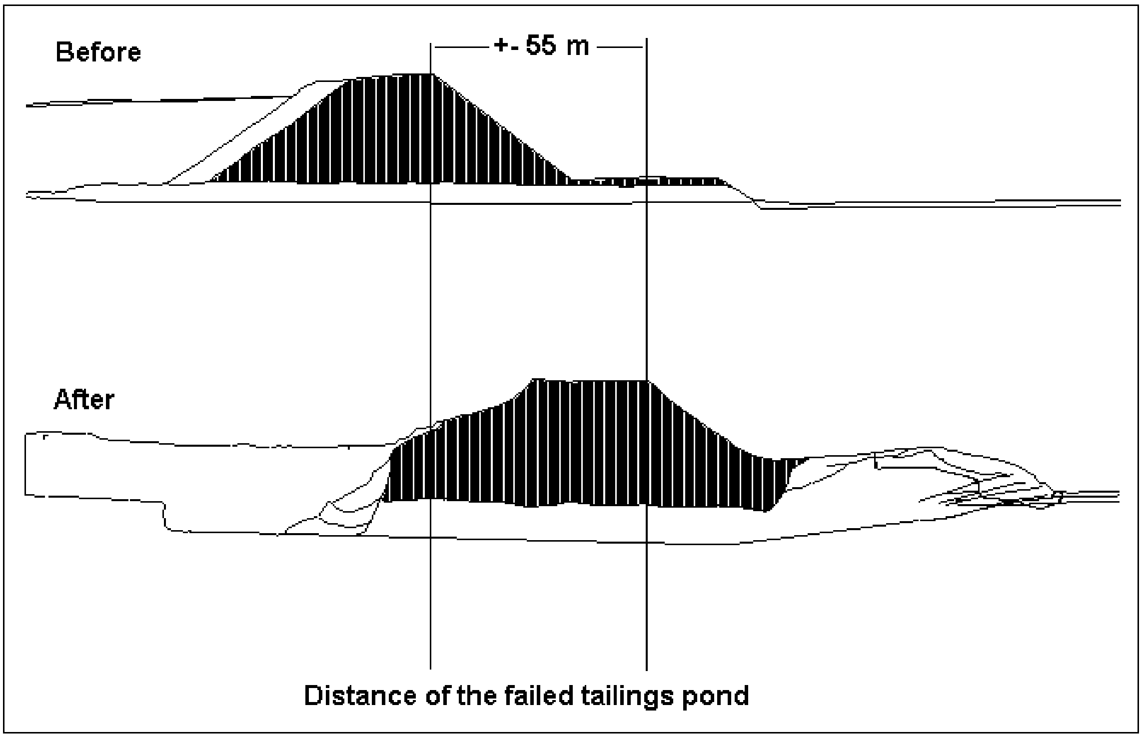

Figure 3.34: Cross-section of the tailings dam
[68, Eriksson, 2000]

One of the conclusions was that a good baseline study, conducted before the accident, would have significantly facilitated the evaluation of the effects of the accident [68, Eriksson, 2000]. Another conclusion that can be drawn is that a close and thorough investigation of the TMF's foundation has to be prepared and evaluated prior to the dam construction.

3.2.3.2.4 Closure and aftercare

At **Pyhäsalmi** the closure plan for the first filled tailings pond (pond A) has been worked out and presented to environmental authorities, but it is not yet officially approved. The work, however, has already started and the costs are estimated to be about EUR 1 million for this pond. No detailed plans for the other ponds exist, but the total closure and aftercare costs for Pyhäsalmi tailings area is estimated to EUR 5.4 million. The costs are reviewed every year. The EUR 5.4 million for closure have been reserved in the income statement of the company to cover the closure and aftercare costs. This money, however, has not been deposited. So, in case of economical difficulties of the company, no assurance mechanism exists.

Production is planned to continue for at least an additional 15 years. Hence, it will be possible to gather experiences for long-term behaviour of the material and the dams at pond A. This experience will be utilised for planning the closure of the other dams in the future.

How the tailings management area will be monitored in the future, after the closure, is not yet determined. The main target of the aftercare work will have to be to prevent ARD generation from the tailings (5 – 10 % sulphur) and to avoid the need for collection and treatment of drainage water for an indefinite period of time.

At pond A the tailings will be covered with 80 cm of soil. The lower layer will be clay and silt material (about 30 cm thick) and the upper layer will be made of moraine. The thickness of the cover was determined taking into account site-specific design criteria and the locally available materials. Other cover materials were also considered like peat, sand etc., but the choice was done based on economical and technical facts again taking into account the materials locally available. The central part of pond A will remain water covered. A system to control the level of the water surface has to be constructed and will include a decant tower and a culvert. Finally, the surface of the treated area will be covered with suitable vegetation.
[62, Base metals group, 2002]

A draft plan for closure and aftercare has been developed at **Hitura**, which has not been approved by the authorities [62, Base metals group, 2002].

The decommission plan for **Aitik** focuses on the three main parts of the operations, the waste-rock areas, the tailings pond and the industrial area including the open pit. For the tailings, evaluation of the weathering properties are still going on. Results so far indicate that no wet cover is required. The measures planned are therefore limited to fertilising and sowing with herbs, grass and trees to prevent wind erosion of the top layer. Dams around the tailings deposit and the clarification pond will be re-sloped at an angle of 1:3.0 and the slopes will be sown with grass.
[63, Base metals group, 2002]

At **Garpenberg** according to hydro-geological modelling results, the higher section of the Ryllshyttan tailings pond will be almost completely saturated with groundwater. Limited areas along the west and south dams will have a partly unsaturated top-soil.

According to the decommissioning plan, the tailings pond will be covered with vegetation. With numerous references from other sites, it is anticipated that seeding directly on the tailings surface with the addition of nutrients will be a cost efficient and realistic alternative. If problems occur measures to reinforce vegetation, such as application of an organic cover or similar, will be taken. The areas along the dams that will remain unsaturated will be covered, should acid conditions develop. The dams, which potentially contain acid producing material, will be covered using a 1.1 m thick engineered soil cover, containing a 0.4 m compacted clay layer as the sealing element. The dams will be re-sloped to 1:2.5 – 1:3.0 before covering and revegetated after covering. The lower section of the tailings pond (the part that is now active) is situated so that a positive water balance can be guaranteed and will thus remain water covered.

For several years, contacts have been maintained with a nearby paper mill, regarding possible use of organic waste products for reclamation purposes. These contacts resulted in a test programme, which was launched after the upper section of the pond was completed in 2000. The paper mill produces organic sludge and a fly-ash product, a combination with properties making the material suitable as cover material. The supply of material is sufficient for covering the entire pond area within 5 – 10 years, with a potential for a robust and environmentally friendly technical solution.
[64, Base metals group, 2002]

The decommissioning plan for the **Boliden** tailings pond is described in the gold section.

The existing and the indicated ore reserves are estimated to give **Zinkgruvan** a mine life for at least another 15 years of operation. Plans for rehabilitation of the areas affected by the mining operation are designed according to the present status of the rehabilitation technique. Since the technology and the requirements from authorities are changing continuously this closure plan can be considered a model, developed from today's demands and standards.

as a part of a large corporate group this closure plan has been developed in conjunction with the company's closure planning guideline, health, safety and environment policy and communities policy.

The rehabilitation of the previous tailings area in Åmmeberg started in 1982 with the construction of a 18 hole golf course and was finalised in 1991 when a marina, a beach area and residences were arranged in the centre of the area. A monitoring program for the recipient of water from the golf course area, is running and is a part of the control program.

Until the currently operating facilities are decommissioned the closure plan will be reviewed at least every five years.

The current tailings impoundment is planned to be dewatered and covered. Once the area has been restored and rehabilitated the land will be handed back to the original owners. At that stage when the land will be handed back it can be used for the same purposes as pre-mining i.e. forestry.

The time schedule for the rehabilitation work is depending on the life of mine and will consequently not be started until the mining operation has ceased, now estimated to be around 2025. Depending on the choice of how to extend the tailings impoundment area, which is estimated to reach permitted volumes around 2007, the need for rehabilitation of the existing tailings impoundment may occur earlier. If the authorities demand a new tailings impoundment to be constructed rehabilitation of the existing facilities will be performed.

In the application for a new permit an extension of the existing tailings impoundment in Enemossen is the primary alternative. This facility can technically, by means of raising the dam, store tailings quantities corresponding to another 25 years of ore production. A dam raise to a height corresponding to life of mine will imply that rehabilitation measures are not applied before mine closure. An exception to this is the downstream walls of the dams that may be rehabilitated before final restoration.

A "wet" cover is not possible at the existing pond as the catchment area is to small to guarantee a permanent water surface covering the area. Hence, a "dry" till cover must be arranged in order to reduce infiltration and diffusion and to prevent water and oxygen to reach the tailings (see Section 2.4.2.2.9).

When the pond has been dewatered the dams will no longer be subject to water pressure. Instead the dam walls can be classified as stable earth-formations with groundwater pressure. From this point on the dams cannot be flooded or be subject to inner erosion, which are normally the two most common reasons for dam failure. During times of high water flows it is important though, that water is prevented from entering the pond.

Measures will be taken to secure the physical and chemical stability of the dams and the tailings stored within the pond. Long-term stability and access for big equipment can be obtained by flattening the dams slope from the current 1:1.5 to 1:2.5 - 1:3.0. The major part of the material needed to flatten the slopes out will be put in place in conjunction with the continuos raise of the dams.

The slopes and the surface of the pond will be vegetated to withstand erosion and to aesthetically blend into the surroundings.

The final rehabilitation of the Enemossen tailings impoundment can be summarised as follows:

- excavation of by-pass ditches along surrounding natural slopes, approximately 2000 m
- dewatering and consolidation of the pond
- contouring of the pond surface
- flattening of the down-stream dam walls
- placing of dust control cover
- placing of the final cover
- revegetation of the cover.

Below is a suggested cover design for Enemossen. This proposal is based on recommendations from the authorities, international practice and experiences from other rehabilitation projects in similar settings. The design of the cover may change over time since closure is far off in the future.

The suggestion below has been chosen in order to fulfil its purpose with good margin. It has been assumed that the following material shall be used to form a cover from top to bottom:

	0.2 m	Top soil
	0.5 m	Protective cover of moraine
	0.2 m	Drainage layer of moraine
	0.2 m	Sealing cover of material with low permeability
	0.2 m	Dust control layer of crushed rock or sand and gravel
	-	Tailings

Table 3.17: Structure for cover of Zinkgruvan TMF
[66, Base metals group, 2002]

The water surface of the clearing pond will be lowered to a level that can be maintained by natural precipitation within the catchment area. At this level minor areas with tailings will be exposed, mainly in the upper (south) part of the pond. In these areas it is thought to be enough to use a simplified type of the cover compared with the cover used at the tailings impoundment. It is assumed that a simplified cover can consist of 0.2 m of topsoil and another 0.2 m of moraine.

Where the thickness of the tailings in the clearing pond is limited one possibility is to clear that area from tailings. Cleared tailings-material will be put in the tailings impoundment.

4 km of tailings- and water pipelines from the process plant to the tailings impoundment will be dismounted and removed including foundations and frames.
[66, Base metals group, 2002]

At **Lisheen** the closure plans have been developed as part of the initial permitting procedures and will be reviewed annually. It is expected that 5 years active care and 10 years passive care will be necessary. For the TMF a permanent water cover is believed to be the best solution. A closure funding(incl. perpetual aftercare) of about EUR 14 million has been in place with the authorities since construction commenced – IR£11million
[41, Stokes, 2002]

At Aznalcollar the emergency program after the accident evolved into a complete decommissioning of the failed dam and the entire pond. This included:

- diversion of the nearby river
- building an impermeable seepage cut-off wall around the north and east sides of the dam
- installation of a hydraulic barrier including a back-pumping system on the inside of the cut-off wall
- cutting and re-sloping the dam to 3:1 and covering it
- remodelling of the tailings surface to minimise the infiltration and to control the surface run-off
- construction of a vegetated composite cover over the remodelled tailings surface. Starting from the tailings, the cover consists of a geo-textile, 0.5 m waste-rock, 0.1 m blinding layer, 0.5 m compacted clay, 0.5 m protective soil layer and vegetation.

[68, Eriksson, 2000]

3.2.3.3 Chromium

To date only one contribution, the Finnish Kemi site, has contributed to this section.

3.2.3.3.1 Characteristics of tailings

The chemical composition has been determined and leaching behavior (max. solubility /DIN 38614-S4 by Kuryk`s method and long-term behaviour) have been investigated in laboratory scale simulation tests. Also wind erosion tests in laboratory scale have been done. In the tailings material the most significant contents are Cr and Ni, which occur as insoluble compounds and are considered by the operator not to cause any negative effects.
[71, Outokumpu, 2002]

3.2.3.3.2 Applied management methods

The TMF consists of three active and three decommissioned ponds and a total area of 120 ha. The tailings are pumped from the process to a first pond where the solids settle before the supernatant water is led to one of the two clarification ponds. Water is re-used in the process. Excess water is led to the river system. Out of the decommissioned ponds one has been covered and landscaped. The two others still need to be landscaped.

The distance between the mill and the TMF area is about 1 km. A stream runs just beside the ponds. The quality of water in the stream is poor as it is coming from a moss area. Very close to the mine and the TMF there is a moss protection area belonging to Natura program. So, in respect of flora and fauna the area is sensitive. Drainage water leaks directly to the stream without any special collecting ditch and control system.

No baseline studies have been done.

The TMF has been built on flat land with paddock-style dams. The starter dams have been made of moraine and are founded on stable and low permeable soil. The supporting body has been made of broken rock. For improving the stability of the dams there are counter banks, where necessary.

The tailings from the process are distributed directly from the tailings pipe around the first tailings pond. The outlet is moved periodically so that the pond will be equally filled. The dams are raised annually with moraine and broken rock as a supporting body. External experts are used, when plans to raise the dam are made.

The dam of the clarification pond has been made of moraine and lined with broken rock to prevent erosion.

The tailings management area was designed in 1960`s and no closure or aftercare plans were taken into account at that time. However risk assessment has been performed more recently.
[71, Outokumpu, 2002]

3.2.3.3.3 Safety of tailings facility and accident prevention

The system has been constructed so that the surface of water in the tailings area can be kept in balance and the excess of water in case of rainfalls etc. can be removed in a controlled manner.

The tailings management area is inspected daily by the operators of the mill. The dams are inspected annually by an external expert and at 5-year intervals by the dam safety authority. The comments have to be recorded in a Dam Safety Document.

A documented emergency plan does not exist, but it is expected, that an emergency plan must be created in the near future according to new legislation.
[71, Outokumpu, 2002]

3.2.3.4 Iron

3.2.3.4.1 Characteristics of tailings

Iron ores are usually mined as oxides. Two tailings fractions, coarse and fines, are generated in the mineral processing step. The coarse tailings are managed on heaps and the fines are pumped into ponds. The tailings and waste-rock, if the iron is mined as oxides, are not acid generating.

The tailings from iron ore production are well characterised in the **Kiruna** area with regards to
- mineralogy
- geochemistry (kinetic leaching tests, trace element analysis)
- mechanical/geotechnical properties.

The tailings material at **Malmberget** has not been characterised.
[49, Iron group, 2002]

Example results from Kiruna are given in the tables below.

Element	Average Concentration (wt. %)
SiO_2	33.82
TiO_2	1.21
Al_2O_3	6.82
MnO	0.15
MgO	6.9
CaO	15.7
Na_2O	2.02
K_2O	1.89
V_2O_5	0.06
P_2O_5	8.1
Fe_xO_y	16.5
Total	
Fe	11.6
P	3.55
S	0.35

Table 3.18: Average concentrations in wet-sorting tailings from Kiruna and Svappavaarra
[82, Iron group, 2002]

Element	Wet-sorted tailings (ppm)	Other tailings (ppm)
As	3.67	18.1
Ba	168	205
Be	8.25	6.10
Cd	0.14	0.10
Co	94.2	67
Cr	13.4	23.5
Cu	356	211
Hg	<0.0400	0.060
La	107	331
Mo	15.4	11.8
Nb	11.9	<12.0
Ni	82.4	56.5
Pb	9.35	7.56
S	4990	4130
Sc	48.2	26.7
Sn	36.8	31.1
Sr	30.3	80.4
V	523	290
W	11.9	<12.0
Y	40.6	170
Yb	7.78	15.4
Zn	53.5	42.5
Zr	114	161
Notes: Samples marked with < are below detection limit, the number indicate the detection limit		

Table 3.19: Average trace element concentrations for wet-sorting tailings and other tailings material at Kiruna and Svappavaarra
[49, Iron group, 2002]

Mechanical/geotechnical properties have been mainly with the intention to use the tailings as construction material of the dams. It was concluded that the tailings need to be cycloned in order to fulfil the requirements for dam construction due to the grain size distribution.

Tailings from Kiruna and Svappavaarra have also been sampled at various depths in existing tailings ponds with the following typical results:

- wet-density analysis for the undisturbed tailings samples are in the range of 1.71 - 2.30 t/m^3
- calculated dry density is in the range of 1.66 - 1.97 t/m^3 for the undisturbed tailings samples
- compact density: 3.2 t/m^3
- friction angles range from 19° to 26.5°.

Samples of tailings material collected from the gravity separation circuits (excluding particles from the pellet production) show the following grain size distribution:

size (µm)	cumulative % passing
700	100
60	75
2	5

Table 3.20: Size distribution of tailings from gravity separation
[49, Iron group, 2002]

Sampling of tailings material collected following the separation by screw classifiers show the following, slightly finer, grain distribution:

size (µm)	cumulative % passing
60	91
40	80
2	8.8

Table 3.21: Size distribution of tailings after separation by screw classifiers
[49, Iron group, 2002]

Samples are collected on a frequent basis from the tailings deposition stream in order to evaluate the efficiency of the separation method.

3.2.3.4.2 Applied management methods

Note: The coarser part of the tailings which is co-deposited with waste-rock, is regarded as waste-rock and will be described under Section 3.2.4.4.

The **Kiruna** (which has tailings ponds in Kiruna and Svappavaarra) and **Malmberget** operations use hydraulic transport methods of their tailings to tailings ponds (pumping through pipelines or by gravity flow in aqueducts). Conventional earth dams are used with a low permeable core of compacted morain, filters and supporting fill of mainly waste-rock. The three major tailings ponds are described in detail below where key information for each tailings pond is also summarised in tables. Both sites follow a very similar tailings management since the material being deposited and the meteorological, geological and hydrological settings are relatively similar.

At Kiruna and Malmberget operations the tailings deposition method is through a constructed ditch using a high water to solid ratio. The discharge point has been kept in the same location throughout the life span of the tailings dams. In order to reduce the liquid/solid ratio and to change the distribution of the tailings, the use of a mobile spigotting point or cyclones have been discussed with the next tailings dam increase.

The freeboard at these tailings dams are 2 meters at two of the facilities and 1.2 meters at the third one. The freeboard is based on withstanding a 100 years, 24 hours rainstorm event without invoking any extra pumping. Discharge to the tailings dams is controlled by a relatively constant operation system producing a constant flow of tailings.

The tailings dam facility at the Kiruna operation was originally completed in 1977 and the dam height was increased in 1984 and 1992 using the centre line method. A new raise is being discussed since the current configuration will be full (within the safety limits) by end of 2003. The current dam height at Kiruna tailings dam is 15 metres.

A discharge point was added to the subsequent clarification pond in 2000. The dams are built using the upstream method with an impervious core of moraine and waste-rock as support material.

There are two outlets from the tailings dam in order to direct the overflow water from the tailings dam to the clarification pond. The outlets consist of two funnels connected to a 1400 mm diameter pipeline ending in a control station downstream of the dam. There are two discharge points and one overflow channel constructed on the clarification pond to handle excess water. The outlets consist of two monks connected to a 1400 mm diameter pipeline ending in an overflow well. The overflow channel is 13.5 m wide with a slope of 1:2 along the dam line.

From an overflow well and the overflow channel, water can flow to a ditch that leads to the river. Water is directed in 1200 mm pipeline starting upstream from the dam wall inside the clarification pond to a pump station. Water is pumped from this station back to a storage pond near the processing plant via the return water pipe line system. Water can also be pumped from the pump station to the channel leading towards the river.
[49, Iron group, 2002]

	Tailings dam			Clarification pond	
Dam type	Valley Dam			Valley Dam	
Dam area	2.2 km^2 4.2 km^2			0.66 km^2 0.96 km^2	
Tailings volume	9 Mm^3			---	
Water volume	4.7 Mm^3 7.4 Mm^3			1.6 Mm^3 2.3 Mm^3	
Dam body	C-D	O-R	R-B	R-S	S-F
Dam type	Upstream	Upstream	Upstrea m	Upstream	Upstream
Highest height, m	8	15	15	11	13
Dam length, m	1450	2560	1040	1440	850
Dam width, m	15	15	15	15	15
Lowest freeboard, m	2.0[1)]	2.0[1)]	2.0[1)]	2.0[1)]	2.0[1)]
Upstream slope	1:1.8	1:1.8	1:1.8	1:2	1:2
Downstream slope	1:1.4	1:1.4	1:1.4	1:1.5	1:1.5
Dam construction volume, Mm^3	0.66	1.58	0.86	3.00	0.39
Discharge arrangement			2 monks	overflow discharge point	2 funnels
[1)] DG is 3.0 m under lowest dam top and is allowed reduced to 2.0 m.					

Table 3.22: Characteristics of the Kiruna tailings dam system
[49, Iron group, 2002]

All dams (tailings pond and clarification pond) are built with a central vertical impervious core with a width of 4 m. Downstream of the impervious core is a 1.5 m thick fine filter (0 - 6 mm or 0 - 8 mm grain-sizes range), a 1.7 m thick coarse filter (grain-size range of 0 - 100 mm) and a downstream support consisting of waste-rock. Upstream, the impervious core is waste-rock filling for support (grain-size range of 0 - 35 mm) and an erosion layer of waste-rock (grain-size range of 0 - 100 mm). The upstream dam slope is 1:1.8, and the down-stream dam slope is 1:1.4. Elevation increases are performed using the centreline method maintaining the 4 meters impervious core.

At the other TMF at **Kiruna (Svappavaara)** there are three dams in the deposition system: a tailings dam, a clarification pond, and a recipient pond. In addition to these constructed systems, a natural lake, acts as a water resource. The pond valley site type using two dams. The clarification pond is also valley site type with one dam. The recipient pond is off-valley site type.

In order to maintain operation and reduce the need for increased area for tailings deposition, the tailings dam has been raised several times since initial construction. It was originally constructed in 1973. The dam has been raised 11 times since then using the upstream method and reaches 15.5 m. It contains approximately 15 Mt (dry weight) of tailings material.

The recipient pond was the first dam built, and put in use in 1964. The idea with the construction was that the tailings material would naturally settle upstream of the recipient pond, and the recipient pond would collect the drained water. Water was funnelled from the pond to a lake. Because of the steepness of the terrain between the processing plant and the recipient pond, the tailings settled closer to the recipient pond than was expected. Therefore, a blocking dam was constructed using borrow material. The recipient dam since then has worked as a clarification pond. Water is led from the tailings dam via a blocking dam to the clarification pond.. This did not work properly in the spring of 2001 due to the formation of ice-lenses, hence an overflow trench was constructed in the fall of 2001. The water runs from the clarification pond via a channel to the recipient pond, and from there via a channel and outflow pipes to a pump station. Water is then pumped into the river and from the river via a high reservoir back to the processing plant. When there is a lack of water within this system further surface water can be added. There is usually no excess water within the deposition system. However, water can overflow from the recipient reservoir to several surface waters.
[49, Iron group, 2002]

	Tailings pond		Clarification pond	Recipient pond
Dam type	off-valley		off-valley	off-valley
Dam area	1.2 mm^2		0.7 mm^2	0.42 mm^2
Tailings volume	4.5 mm^3		1.5 mm^3	0.2 mm^3
Water volume	0.4 mm^3		4.5 mm^3	0.45 mm^3
Dam section	soil dam	blocking dam	soil dam	recipient dam
Dam type	upstream	upstream	downstream	downstream
Max. height	15 m	15.5 m	21 m	10 m
Dam length	2030 m	1100 m	2350 m	800 m
Dam width	8.3 m	12 m	7.2 m	6.0 m
Smallest freeboard	2.0 m (+364.0 m)		1.8 m (+358.2 m)	2.5 (+340.0 m)
Upstream slope	1:2	1:1	1:2	1:2
Downstream slope	1:1.5	1:3 / 1:7	1:1.5	1:1.8
Tailings dam volume	ca 0.36 mm^3	ca 0.5 mm^3	ca 0.46 mm^3	ca 0.17 mm^3
Discharge arrangement		overflow	2 standipes	1 standpipe

Table 3.23: Characteristics of the Svappavaara tailings dam system
[49, Iron group, 2002]

<u>Tailings pond</u>

- Soil dam

The starter dams comprises a homogeneous moraine material with an erosion cover of 0 - 100 mm grain size. The erosion cover is 1 m thick on the downstream slope and 1.5 meters thick on the upstream slope. The slope angle is 1:1.5 and 1:2 for the downstream and upstream slopes, respectively. Height increases of the dam have been constructed based on the upstream method with a 4 meter thick impervious core consisting of moraine material. There is a one meter thick transition layer on both sides of the core with a grain size of 0 - 100 mm. The erosion cover on the downstream side is approximately 0.5 meter thick with a grain size range of 0 - 100 mm. The upstream support fill and erosion cover consist of material with a grain-size range of 0 - 200 mm and 0 - 500 mm, respectively. A 2 m increase in dam height, using the downstream method, is planned for the summer of 2002.

- Blocking Dam

The blocking dam is made with a dam of waste-rock without an impervious core. The dam is built with the upstream method with a grain-size range of 0 - 500 mm.

Clarification pond

The clarification pond is built with a soil dam constructed as a conventional dam. The starter dam is made with a homogeneous moraine material with an erosion cover consisting of material with a grain size of 0 - 100 mm. The erosion cover is 1.0 meter thick on the downstream slope and 2 meters thick on the upstream slope. The slope angels are 1:1.5 and 1:2 for the downstream and upstream slopes, respectively. Further height increases have been constructed based on the centreline method.

Recipient pond

The dam at the Recipient Reservoir is built as a conventional dam and raised using the centreline method. The vertical impervious core consist of, at the top, a 3 meters thick moraine material. On both sides of the impervious core, is a 2 m thick fine-sand filter consisting of a material with grain size range of 0 - 32 mm. Outside the fine filter is a coarse filter material with grain size of 8 - 64 mm. On top of the core and the fine filter is a 0.5 meter thick horizontal layer consisting of bark. The support material consists of blasted rocks on both sides. The downstream slope angle is 1:1.8 and the upstream slope angle is 1:2.
[49, Iron group, 2002]

There are five dams within the **Malmberget** mining operation; tailings dam, clarification pond, pond for biological degradation, a reserve pond and a buffer pond. Only the two first dams are described in this document.

The tailings pond was constructed in a lake. The tailings pond consists of primarily two dams of different design, the B-A dam and the C-D-E-F dam. Water is funnelled through a decant tower from the tailings pond into the clarification pond. Water is then pumped from the clarification pond back to the processing plant.

The tailings dam at Malmberget was originally constructed in 1977 and has been increased in height five times since then. The height of the dam reaches 35 meters. It will be full by the end of 2002 and a height increase has been designed using the upstream method. This height increase will secure the tailings deposition for another 25 years assuming today's production rate of 1.5 Mt/yr. The whole pond currently contains approximately 16 million tonnes (dry weight) of tailings.

The following table lists the characteristic data for the Malmberget TMF. The tailings dam and clarification pond were constructed using the natural terrain with a main dam at the end of the valley.
[49, Iron group, 2002]

	Tailings dam		Clarification pond
Dam type	valley dam		valley dam
Dam area (foot print)	1.8 mm^2		0.12 mm^2
Tailings volume	16.8 mm^3		0 mm^3
Water volume	0.4 + 1.2 mm^3		0.25 mm^3
Dam section	b-a	c-d-e_1-f	j_1-j_2
Dam type	up/downstream	downstream	centre line
Maximum height	13 m	35 m	14 m
Dam length	700 m	2500 m	1100 m
Dam width	40 m	40 m	8.0 m
Smallest freeboard	1.2 m	1.2 m	0.5 m
Upstream slope	1:2	1:2	1:1.5
Downstream slope	1:1.5	1:1.5	1:1.5
Tailings dam volume	app. 0.2 mm^3	app. 2.5 mm^3	app. 0.2 mm^3

Table 3.24: Characteristic data for the Malmberget tailings and clarification ponds and dams [49, Iron group, 2002]

Tailings Dam

The dam was designed to span the width of a lake, thus blocking the lake water. On the inside this blocking dam is designed as an upstream dam to level 271 (Figure below). The upstream dam is built with a 7 m thick impervious core of moraine material with permeability of 10^{-8} m/s. The impervious core is slanted 1:1.5. Below and above the impervious core is 1 meter thick filter with grain size of 0 - 100 mm and permeability of 1 x 10^{-3} – 1 x 10^{-4} m/s.

From level 271, the dam is built using the downstream method with an inside slope of 1:2 and an outside slope of 1:1.5. Between the support material and the impervious core is a 1 meter filter as described above. On top of the core is a 1 m thick erosion layer consisting of material with grain size 0 - 70 mm and permeability of 1 x 10^{-5} m/s.
[49, Iron group, 2002]

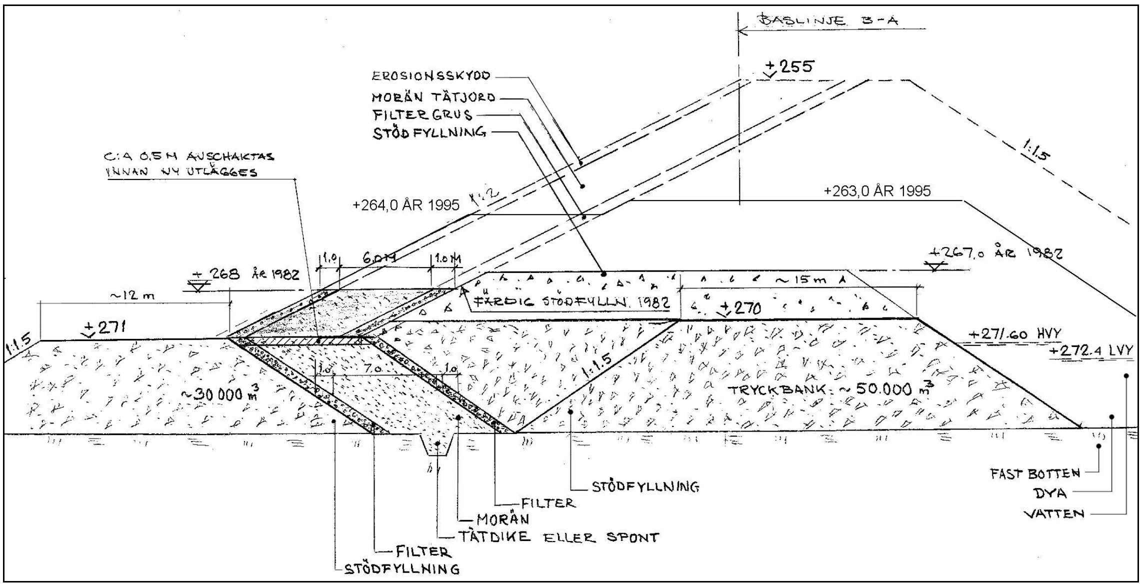

Figure 3.35: Cross-section of Malmberget tailings dam [49, Iron group, 2002]

Clarification pond
The dam of the clarification pond is designed as a conventional dam with a four metre thick impervious core of moraine material. On each side of the core is a 1 metre thick filter layer. Outside this is a support material and on top an erosion layer. Both the support material and the erosion layer are made of coarse dry tailings material. Outside and inside slope are 1:1.5 [49, Iron group, 2002].

At the **Steirischer Erzberg** the tailings ponds are built on top of the 50 to 100 m high waste-rock dumps and are constructed to be of low permeability but use infiltration areas to drain off the clarified water. A series of ponds is used in order to allow for the draining of one pond while another pond receives the tailings slurry. The draining water infiltrates through the waste-rock dump and mixes with the water in a stream that flows underneath the dump. This is described in more detail below.

The total volume stored in all ponds is about 9.4 Mt (5.2 Mm^3). In total the TMF occupies about 40 ha. The distance between the processing plant and the active TMF varies between 500 and 2000 m. The tailings have to be pumped from an altitude of 745 m to an altitude of 873 m and 980 m respectively.

In the first half of the 20th century the area served as a waste-rock dump area for the mining operation. This buried the stream in this valley for practically its the total length. The method applied at that time - trail bound transportation with comparatively high dump heights - resulted in a high proportion of big sized blocks at the bases of the dump, due to size segregation. The base of the dump was constructed by removing the topsoil and installation of a bottom layer of big rock blocks. Accordingly, a sufficient permeability for the dewatering of the valley was achieved and has remained intact until present. The majority of the drained water from the dump emerges at the toe of the dump. The dump material is mainly ankerite and limestone.

The fundamental design criteria were stability and tightness to water. All dams are constructed from carbonate tailings (0.15 - 120mm) and a schist ("Werfener Schiefer") rock layer on the inner side of the dam. Sealing is done by establishing a compressed layer of schist ("Werfener Schiefer") and tailings, which provide, according to the experiences of the company, sufficient impermeability. In order to prove the suitability of the materials and techniques used for dam construction, comprehensive studies were conducted, comprising both in-situ and laboratory tests (geotechnical parameters, permeability, internal friction angle, etc.)

Investigations have shown, that the stability of the dam construction is almost independent of the tailings situation inside the pond, if a sufficiently impermeable seal layer made of compressed schist and tailings is put in place before starting the discharge of tailings. Accordingly the impermeability of the sealing layer is of great importance.

During design and construction attention was paid to the execution of the sealing layer and the drainage of the supernatant water on top of the pond. Depending on the dam material for each pond a particular position is selected for discharge of the water from the pond. These areas are 20 to 30 m in length and consist of weathering resistant materials of appropriate fragmentation in order to warrant the necessary permeability.
[55, Iron group, 2002]

3.2.3.4.3 Development of new deposition methods

The construction of a drained cell pond is currently investigated. If the results from this test project are positive, the method will be modified to suit large-scale applications. The technique is based on the grading of waste-rocks taking place down slope from the truck dumping. This grading results in a pervious/well draining filter dam. Constrained cells can be constructed with hydraulic tailings deposition. The filter dam then contains the tailings material, while process water is drained.

A collection ditch or walls will be constructed around the filter ponds to collect the draining water. Collected water will be directed towards the existing tailings dam. The suggested location of these draining cell ponds will result in the tailings dam acting as a clarification pond for suspended material transported through the filter dam.

Some of the tailings material will pass through the filter dam to the existing tailings dam. This may result in a need for a height increase of the existing tailings dam during the planned 16 year deposition period, depending on the efficiency of the filter dam. It is necessary to have high filtering efficiency (sand deposition within the cell) to make the drained cell deposition a viable method. The height increase that may be necessary (max. 1 - 2 meters over the 16 years period depending upon the dam efficiency) can be constructed on the existing dam.

An advantage of this draining technique is that an increase of the footprint of the existing tailings dams is not necessary or the height increase of the current dams can be delayed. Also since the drained cell is a “dry” system the tailings can be stacked higher. Because the water from the tailings deposition is drained, a failure of the filter dam is less likely. If it fails, the effect of the failure is reduced because water content is lower compared to the current system and tailings material escaping the cell ponds will be trapped in the current tailings dam. With the current conventional dam system the coarse tailings are treated like waste-rock and trucked to the waste-rock dump, which is very cost and labour intensive. An economic benefit for the operator is that with this new method both coarse and fine tailings can be pumped to the new TMF as a slurry.
[49, Iron group, 2002]

3.2.3.4.4 Safety of tailings facility and accident prevention

At **Kiruna** and **Malmberget** discharge to the tailings dams is controlled by a relatively constant operation system producing a constant flow of tailings. The dams are inspected several times a week following an operation, inspection and maintenance (OIM) manual that has been developed for all three facilities. The inspections include evaluation of water level in the dams and the overflow ditches/funnels. All observations are logged in the field log book so that changes can be evaluated. Monthly and yearly inspections are also implemented according to the OIM manuals.

A risk assessment for tailings accidents has been performed in order to evaluate the potential for harm to human life and property damage. It focuses on failure of the dam and the effect of physical destruction of the tailings material. Since the material, as described earlier in this document, is chemically stable, the risk of chemical contamination with a is not relevant.

The OIM manuals developed at Kiruna and Malmberget are described below.

<u>General</u>
During year 2001 Operation, Inspection and Maintenance (OIM) manuals, similar to the OSM manuals described in Section 3.1.4.1, were implemented for three large tailings dams. These manuals were developed in order to avoid dam failures, or if failure would take place, the emergency response to reduce the effect from a dam failure. The three manuals are very similar and will, therefore, be described together. Another objective of these manuals is to facilitate and document future design changes. The manuals are to be updated yearly. The content of these manuals are as follows:

- dam design
- dam classification relative to safety
- possible actions for safety improvements
- operation, inspection and maintenance routines
- emergency plan for dam incidents
- risk assessment for possible dam incidents.

The condition of the dams during operation can be classified in four different levels:

- normal operation, where there is no indication of changes in conditions
- tightened operation, when there may be some indications of dam fractures, high rainfall or process water output etc.
- disturbed operation, when there is an unusual high water level in the dams, distinct dam fractures, and water leakage; and lastly
- incident, where operation likely will be halted.

The following paragraphs describe monitoring/dam inspection routines and dam failure emergency plans.

Monitoring and inspections of tailings facility
A water table monitoring system has been installed in the dams to monitor fluctuation of the pore pressure, which is critical in evaluating the stability of the dam. There are nine measuring points for the pore pressure at the Kiruna tailings dam, 53 at Svappavaara and four for the tailings dams and Malmberget. A weather station is located at the nearest airport which measures rainfall, temperature, wind speed, and wind direction.

The OIM manuals describe the most critical parameters for inspection and maintenance. These include the monks and channels at the dams, and the pump systems, needing frequent inspection and maintenance. The manuals suggest inspections of these critical areas by trained personnel three times a week. All results from the inspections are noted in a field book. All changes in the processing routines, discharge volumes and change of personnel or material etc. are also noted. The manuals also require group meetings once a week for the OIM personnel, where all information obtained during the week is presented and discussed for necessary improvements.

A monthly inspection is performed in order to evaluate the safety of the dams and possible improvements to the dams to maintain a high level of safety. The inspections are to be performed by the person responsible for the tailings dam together with the person performing the routine inspections. In addition to going through the routine inspection areas, water levels are inspected. These water levels include:

- water level within the pond
- water level (water table) within the dams
- water level in the drainage system.

The dams are inspected for erosion features on the inside and outside slopes. This is especially important during the spring with thawing and frost. Changes in the dam leakage and solid transport in the leakage water are also checked, where an increase in leakage can be an early indication of fractures in the dam.

A yearly inspection (audit) is performed by an outside dam-design and maintenance specialist. At this yearly inspection all the field notes and the monthly inspection reports are reviewed in addition to a field inspection. The yearly field inspection goes through the same checklist as the monthly inspection. In addition to the inspection, possible dam settling is measured. The yearly inspection report summarises all the measurements collected throughout the year, evaluates the results; and suggests possible improvements or adjustments to the dams and to the daily and monthly inspections. The yearly inspections also review and evaluate the dam calculations behind the dam designs including the operation and maintenance data.

Emergency plans for three levels of increased risk of operation listed above have been developed. These three levels, as described above, require different responses, which are summarised below.

Tightened Operation: When the conditions indicate increased risk of a possible dam failure; increased leakage, unusual high water level in the dams etc., the dams will undergo more frequent inspections, every day, to evaluate if the conditions are improving or getting worse. All observations are noted in the field book by the person responsible for dam safety.

Disturbed Operation: If the disturbances on the dams are larger than described above, e.g., caused by long term heavy rainfall with a highly increased water level within the ponds, severe erosion through the funnels and overflow channels, fractures on the dams, etc., the setting is called disturbed operation. During this setting preventive measures (preventing a dam failure) are required. The OIM Manuals describe possible scenarios and the suggested measures for those scenarios as an aid for a quick response. All observations and measures are to be described in detail in the field log book by the person responsible for the dam safety. The most common measures are to repair fracture and erosion areas and lowering of the water level within the ponds. Filling of fractures and sinkholes in the dams is suggested, performed in co-operation with a dam specialist.

Incident: If an incident takes place, a setting where a temporary stop in the mining operation is likely, a suggested action plan to aid in decision making has been established. An incident has to be followed up with a report that includes the reason for the incident and what actions were taken to mitigate the incident. The routines described are primarily for disturbance/failure of the culvert, which is considered very serious for dam safety. A catastrophe plan has been developed to be implemented if the incident is of a magnitude that loss of life or property is likely. The catastrophe plan outlines the emergency management organisation within mining company and the community.

For the safe operation of the tailings ponds located on top of the waste-rock dumps at the **Erzberg** a series of monitoring and supervision measures are provided, focusing on crucial parameters. Parameters observed on a regular basis comprise:

- surface water level inside the dams (piezometer measurements)
- water level in the ponds
- subsidence measurements (surveying).

Operational instructions are also provided and cover:

- visual observations
- drainage control and documentation of drainage failures and maintenance works
- water monitoring
- monitoring of dam stability by surveying fix points
- monitoring of water-level within the dams.

The water quality is regularly analysed at sampling points defined by authorities and internal analysis of water quality is done according to needs. However, due to the fact that the discarded tailings are recognised as safe in respect to geochemical environmental aspects the environmental monitoring has merely documentational and preventional character.
[49, Iron group, 2002]

3.2.3.4.5 Closure and aftercare

For the three large tailings ponds at **Kiruna** and **Malmberget** formal closure plans have not been submitted for approval by the regulatory authority. A closure plan will be developed in co-oporation with local and regional regulatory agencies. Parts of the tailings dam system that may be decommissioned prior to mine closure, will be covered and re-vegetated, and if ponding takes place, water pumping and regrading may be performed.

At **Erzberg** some small tailings ponds have been decommissioned. No approved closure plan exists for the ponds in operation, however, studies have been conducted and closure concepts have been developed. The methodology used for the closed ponds has been dewatering and soil covering, followed by re-vegetation. Re-vegetation directly in the dewatered tailings has also been done successfully. These measures effectively eliminate dust emissions from the ponds, water contamination is not an issue (30 years of monitoring results) as the tailings are chemically stable and no reagents are used in the mineral processing. The closed ponds are continuously supervised and surveyed. Alternative use for the tailings material is being investigated and there are expectations to find a market for the tailings.

3.2.3.5 Manganese

No data has been supplied for this section. Please provide information.

3.2.3.5.1 Characteristics of tailings

3.2.3.5.2 Applied management methods

3.2.3.5.3 Safety of tailings facility and accident prevention

3.2.3.6 Mercury

No data has been supplied for this section. Please provide information.

3.2.3.6.1 Characteristics of tailings

3.2.3.6.2 Applied management methods

3.2.3.6.3 Safety of tailings facility and accident prevention

3.2.3.7 Precious Metals (Gold, Silver)

3.2.3.7.1 Characteristics of tailings

The untreated tailings from gold mineral processing using cyanide contain different compounds, depending on the process used, ore type, cyanide dosage, degree of aeration, etc. The composition of tailings will also change as the ore changes [24, BC CN guide, 1992].

During a CIP/CIL leaching process a small portion is lost to the mineral processing plant atmosphere by volatilisation. Some will react with whatever other cyanide consumers may be present in the ore to produce complexes such as the ferrocyanide, thiocyanate, cyanate and cuprocyanide complexes. During leaching, gold is removed from the solution by adsorption onto carbon, and some cyanide may be removed with it. The remaining unreacted cyanide, together with the products with other cyanides consumers, is discharged with the tailings. The cyanide in the tailings may be treated for cyanide removal (most European sites) or left as is for removal by natural degradation in the tailings pond (international standard). Any cyanide entering the carbon stripping circuit would either be periodically bled back into the leach circuit or destroyed during reactivation of the carbon in the carbon kiln [24, BC CN guide, 1992].

The untreated tailings stream from a CIP/CIL process consists of a tailings slurry with elevated levels of cyanide, complexed metals, cyanate and thiocyanate. It may also contain arsenic and antimony, depending on the ore and mineral processing.

It is common practice to have regular control of other material characteristics (the parameters determined varies somewhat from site to site) such as, e.g.:

- grain size distribution
- solid to liquid ratio
- ARD-characteristics
- mineralogy
- trace element content.

The above-mentioned parameters are used to determine leaching characteristics of the material which has an important influence in the operational management and suitable decommissioning methods for the tailings. For this purpose all sites using tank leaching have carefully evaluated ARD-generation characteristics for their tailings. The Boliden mineral processing plant, with 18 % sulphur and low carbonate content has to deal with potentially ARD-generating tailings [50, Au group, 2002].

The **Boliden** mining area consists of complex sulphide mineralisations. Mining in the area started at year 1925 and to date approximately 30 mines have been milled in the area. The tailings in the pond consequently have varied chemical characterisations and physical- chemical properties. The characteristics of the tailings produced today are summarised in the tables below. The fine fraction "after cycloning" is discarded into the tailings pond and the coarse fraction that is extracted from the cyclones is used as back-fill in the underground mines.

Size	Total tailings	Cyclone overflow to pond
µm	**cumulative % passing**	**cumulative % passing**
350	100	100
250	99.9	100
180	99.7	100
125	97.8	100
88	93.5	95.6
63	85.9	87.8
45	76.6	78.3
20	53.2	54.4
-20	0	0

Table 3.25: Particle size of tailings at Boliden mine
[50, Au group, 2002]

The tailings have the following composition before cycloning and CN leaching:

Au	0.85 g/t
Ag	24.9 g/t
Cu	0.10 %
Zn	0.40 %
Pb	0.13 %
S	17.8 %

More than 50 % of the tailing consist of particles less then 0.002 mm. The tailings slurry pumped to the tailings pond contains 20 - 25 % solids. The density, as placed in the pond, of the tailings is 1.45 tonnes/m^3.
[50, Au group, 2002]

At Ovacik detailed characterisation of samples has shown that the tailings and waste-rock will not produce ARD as illustrated in the figure below.

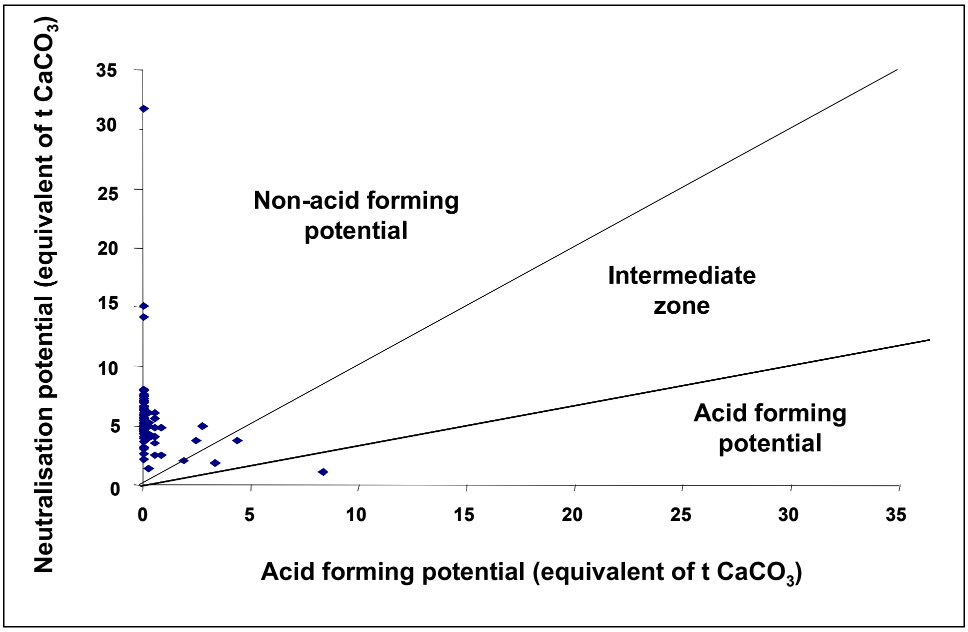

Figure 3.36: Acid forming potential vs. neutralisation potential graph of samples from Ovacik site [56, Au group, 2002]

3.2.3.7.2 Applied management methods

At the Filon Sur heap leach operation, the tailings (the heap of leached material) are left in-situ and decommissioned. The heaps are built on a pad with a synthetic liner. Leachate or "pregnant solution" is collected in a small pond before it is pumped to the plant for gold and silver precipitation. The leachate is then pumped to a conditioning pond before it is re-used in the leaching process. Very little information is available at the moment to evaluate how tailings and waste-rock management and decommissioning is done and planned, thus it will not be further described at this stage. No material characterisation is reported [57, Au group, 2002].

All other sites, using CIL or CIP to leach the gold in tanks, produce tailings in a slurry form that, after CN-destruction is applied, are pumped via pipelines to tailings ponds. The commonly used process to destroy CN is the SO_2/air process. In general this treatment results in a total CN concentration in the treated slurry stream of < 1 mg/l. One site (Ovacik) that measures WAD CN reports concentrations < 1 mg/l.

At present, one site (Boliden) uses the coarse fraction of the tailings as backfill in underground operations. These tailings are extracted from the tailings stream in cyclones situated after the CN-destruction plant. The tailings used for backfill are also analysed for total CN (typically less than 1 mg/l).

50 % of the sites use lined tailings ponds and 50 % uses unlined tailings ponds. Various dam types are used to confine the ponds.

At the **Ovacik** gold mine, with an ore production of 0.3 Mt/yr, the tailings are stored in a 1.6 Mm^3 capacity pond with a 30 meter high downstream rockfill embankment and clay-geo-membrane composite lining system. As described earlier the tailings are treated for cyanide destruction and heavy metal precipitation utilising oxidation with SO_2 followed by ferric sulphate treatment [56, Au group, 2002].

A conceptual drawing of the TMF is given below:

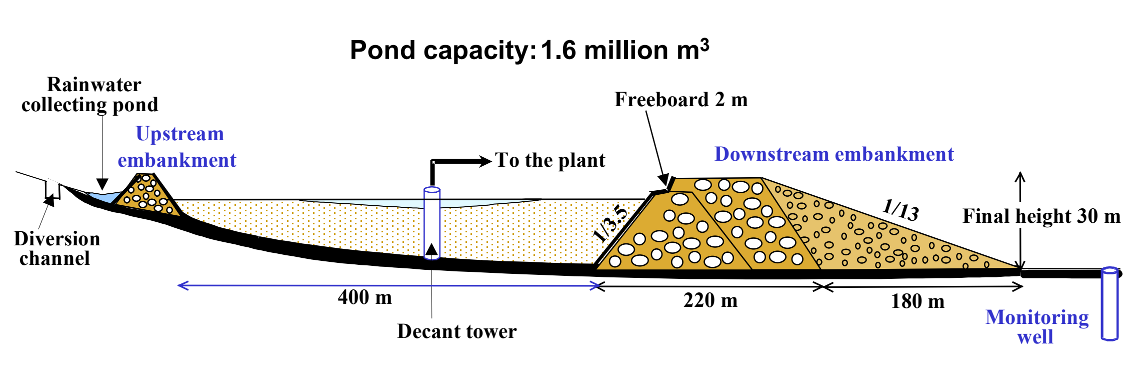

Figure 3.37: Cross-sectional drawing of Ovacik tailings pond
[56, Au group, 2002]

The tailings pond is located in a valley within two hundred metres from the process units. Rockfill dam construction materials (mainly andesites) were obtained from the overburden excavation in the open pit. The region is an arid zone where evaporation plays an active role in water deficit for the pond during the summer season. The TMF was designed as a "zero" discharge unit where water in the pond is re-circulated for re-use during the operation of the mine. Because of the low cyanide concentration in the pond (less than 1 mg/l WAD), HCN volatilisation is negligible. The geo-technical and seismological investigations in the TMF area before and after the construction revealed presence of a suitable setting for the rock-fill embankments and the reservoir stability. The embankments were constructed as conventional dam structures.

Topsoil was scraped and stored on site for future use in site rehabilitation. During closure of the pond, tailings will be dewatered and the top will be covered with rock and soil and subsequently re-vegetated.

In selecting the TMF location, the main factors taken into consideration were:

- minimised land and soil disturbance
- close proximity to the process plant
- use of overburden and waste-rock in the embankments in an efficient way to minimise the foot-print
- storage of topsoil for vegetative cover upon closure
- cyanide destruction and heavy metal precipitation for tailings
- re-use of process water in the process
- zero-discharge of water from the TMF.

It was the company policy to select tailings dams of rock-fill type for its increased stability and easy maintenance (as opposed to using the coarse tailings). Selection of the clay-geo-membrane composite liner system was made to achieve an effective containment and to expedite the regulatory approval and permitting process.

From the geotechnical point of view, the dams were designed to withstand an earthquake induced horizontal acceleration of 0.6 g. With the placement of the overburden and the host rock on the downstream slope of the main dam, the became less than 10˚, increasing the factor of safety of the dam structure to 2.23 compared to the usual 1.2 used internationally for water retention dams.

The base of the tailings pond is covered with a composite liner system of 50 cm compacted clay, overlain by a 1.5 mm thick High Density Polyethylene (HDPE) geo-membrane, 20 cm of another compacted clay and 20 cm gravel filter layer. Drainage pipes are placed in the filter layer to drain the water towards the decant tower from where it is pumped to the process plant for re-use. The following figure shows the set-up of the composite liner system.
[56, Au group, 2002]

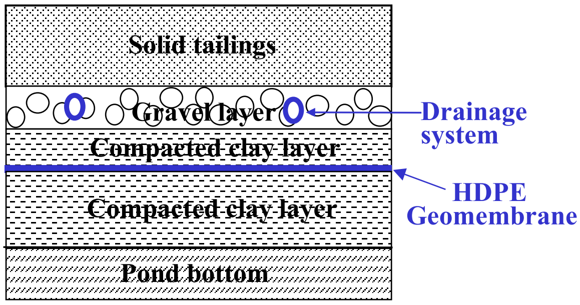

Figure 3.38: Composite liner set-up at Ovacik site
[56, Au group, 2002]

Deposition of tailings is carried out via pipelines discharging into the pond area near the downstream embankment. During the mine operation, a minimum of 2 m of freeboard is provided in the TMF design. The tailings water is pumped back to the process through a decant tower constructed at the near centre of the pond. TMF design includes surface run-off retention behind the upstream dam and a diversion channel for excessive flood waters (for 100-year flood conditions).

The tailings area of the **Orivesi** mine consists of two tailing ponds. The tailings from the process are pumped into the first pond (37 ha), where the solids settle and the clarified water is led forward from the other end of the pond. The second pond (14 ha) is for storing clarified water. Water is re-used in the process and only the excess is led to the river system. The starter dams have been made of moraine. The tailings are spigotted to one side of the first pond and the clarified water is led forward from the other side.

The dams of the clarification pond have been made of moraine and lined with broken rock and coarse gravel to prevent erosion. The tailings management area has been designed in the beginning of 70`s and no closure or aftercare plans have been taken into account at that time. The tailings pond is, however, used only occasionally when the tailings are not deposited into the old mined-out underground nickel mine.
[59, Au group, 2002]

A schematic figure of the system is given below.

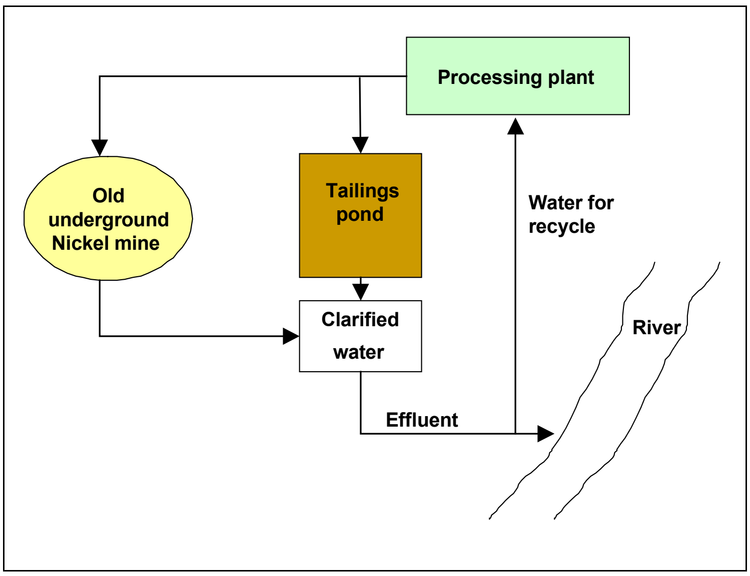

Figure 3.39: Schematic illustration of tailings and effluent treatment at Orivesi mine
[59, Au group, 2002]

The base dam of the tailings pond has been constructed of moraine and there is drainage collection outside the dam to collect seepage water. The necessary raises of the dams are done using moraine for the core and the tailings material as supporting fill.

The TMF was originally constructed for a nickel mining operation. After 20 years of operation the nickel mine was closed, but the mill has been used since to treat gold ore from Orivesi mine located 85 km from the plant. The distance from the mill to the tailings management area is about 500 m. The distance from the tailings area to the river is about 600 m. The surrounding area is not used for agriculture, but the nearest house is only 200 m from the tailings area. The operator does not consider dusting from the tailings management area a problem, because the material on the surface of the area has formed a hard layer. The drainage water is collected by a ditch system and is led directly to a river, because, according to the operator, it does not contain "significant" contamination. Detailed information on dust emission measurements or analysis of water discharge have not been provided.
[59, Au group, 2002]

The **Boliden** base metal mineral processing plant received a total of 1.58 Mt of ore from five different mines during 2001 in order to produce copper, lead and zinc concentrates. Coarse gold is also extracted using shaking tables. Depending on the ore type part of the tailings produced (approx. 50 %) are further processed in the gold leaching plant. The gold leaching plant generated 0.8 Mt of tailings in 2001.

Out of the five mines four are underground mines and one is an open pit. The underground mines use the coarse fraction (> 125 µm) of the tailings for backfilling. The amount of tailings used for backfilling is dependant on the production level in the mines and the production status. During preparation work in the mines a significant amount of waste-rock is produced and used for backfilling. It should be noted that approx. 33 % of the ore comes from an open pit, where no back-filling is done during operation. Subtracting this amount of ore the percentage of back-filling is close to 50 %.

The tailings that are not used for backfilling are sent to the tailing pond that has been used since the 1950's. The area has previously a lake. The amount of tailings in the pond is currently approx. 16 Mm^3 and covers a surface area of 260 ha. According to the existing operation levels, the existing tailing pond can be used for 4- 5 more years. The tailings are pumped to the pond and discharged at various outlet points in order to allow for uniform filling of the pond.

The tailings are confined within the pond by five dams. Another dam is also constructed downstream of the tailings pond to cut off the lakes natural outflow to create an additional clarification volume. The pond area is currently 260 ha and after a dam raise in the summer of 2002 the area will be 280 ha.

The tailings pond catchment area is 8 km^2. The inflow of surface run-off has been estimated to be 1 Mm^3 during a dry year and 3 Mm^3 during a normal year. The pond receives approximately 4.5 Mm^3/yr of process-water from the mineral processing plant.

The tailings pond is approximately 3 km from the concentrating plant. Tailings are pumped via 2 separate pipelines, one to the north and one to the south of the pond. Downstream of the pond slaked lime is added to the discharged water to increase the pH to 10 - 11. All water from the pond is discharged to waterways downstream. No re-circulation of process water is done at the moment.

Water sampling for monitoring water quality is done on a regular basis according to a control program. Sampling is done both upstream and downstream of the tailings pond, as well as around the industrial area. Sampling consists of stream analysis and ground water samples.

The dams (see cross section in the Figure 3.40) were constructed initially in 1979 to +216.2 m as a centre line type dam with a vertical impervious core and support fills on both upstream and

downstream sides of the dam. In 1995 the dam was raised to +220 m as a downstream dam (see the figure). A final raise is ongoing to +225 m to be finalised year 2002. A discharge channel constructed in natural ground will replace the discharge tube.
[50, Au group, 2002]

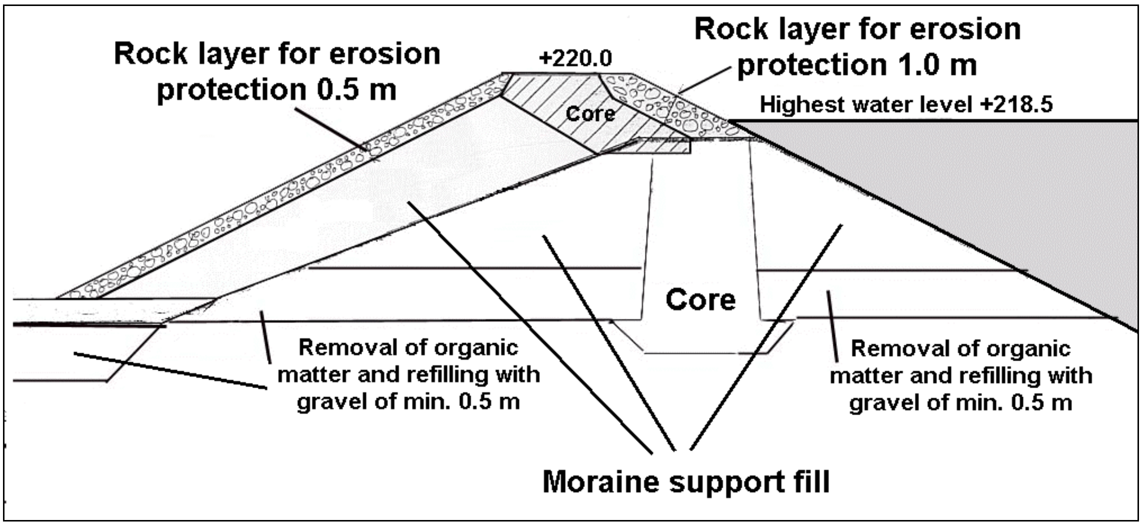

Figure 3.40: Cross-sectional view of dam at Boliden site
[50, Au group, 2002]

Drainage through and under the dams is collected in a collection ditch and lead to the clarification pond. Drainage through and under the other dams is back-pumped into the pond.
[50, Au group, 2002]

At **Río Narcea** the tailings are deposited into a lined tailings pond after CN-destruction. The present volume of the deposit is 2.4 Mm^3 and the pond is continuously raised according to requirements. The dams are built out of compacted clay with a supporting fill of waste-rock. The pond has an impermeable composite liner system composed of compacted clay and a 1 mm HDPE liner. The pond is surrounded by channels for collection and deviation of surface run-off. Collected surface run-off is diverted into three sedimentation ponds for clarification before discharge [58, Au group, 2002].

3.2.3.7.3 Cyanide treatment techniques

On as worldwide scale, natural degradation is still the most common treatment method of treating cyanide in gold leaching effluents, although it is often supplemented by other treatment processes. Especially in dry and sunny climates as e.g. in South Africa, natural degradation is usually the only treatment method.

The following table lists the currently applied cyanide treatment alternatives.

Treatment Process	Stage	Applications	Comments
Natural degradation ▪ Neutralisation by CO_2 absorption ▪ HCN volatilisation ▪ Metal cyanide complex dissociation ▪ Metal cyanide precipitation	**C**	**TP,SW**	Application is limited to site-specific factors (e.g. arid, sunny) and regulations
Oxidation Processes ▪ Alkaline Chlorination ▪ SO_2/air process ▪ Hydrogen Peroxide (Degussa, DuPont)	 **C** **C** **C**	 **TP,SW** **TP,SW** **SW**	 Displaced by SO_2-air and H_2O_2 due to cost, inability to remove iron Universal application, slurry treatment can result in elevated reagent consumption Not applicable to slurries due to reagent consumption
Adsorption ▪ Activated carbon adsorption	 **D**	 **SW**	 Limited to low CN concentrations, site-specific
Biological treatment ▪ Biodegradation	 **C**	 **SW**	 Limited to low CN concentrations, site-specific, may require supplemental heat.
Cyanide Recycle			
▪ AVR	**C**	*Is this applied?*	
▪ Electrolytic recovery	**D**	*Is this applied?*	
▪ Ion exchange	**D**	*Is this applied?*	
▪ Resin-in-pulp	**D**	**TP**	
TP = discharge into tailings pond; SW = Discharge into surface water C = commercial ; D = Development			

Table 3.26: CN treatment processes

The SO_2/air process, which is used in all European sites to treat the slurry prior to discharge into the TMF is usually described using the following reactions:

Oxidation:

$CN_{free} + SO_2 + O_2 + H_2O => OCN^- + H_2SO_4$
$M(CN)_4^{2-} + 4\ SO_2 + 4O_2 + 4H_2O => 4OCN^- + 4H_2SO_4 + M^{2+}$
$M^{2+} = Zn^{2+}, Cu^{2+}, Ni^{2+}, Cd^{2+}$ etc.

Neutralisation using lime:

$H_2SO_4 + Ca(OH)_2 => CaSO_4 \times 2H_2O$

Precipitation:

$M^{2+} + Ca(OH)_2 => M(OH)_2 + Ca^{2+}$
$2\ M^{2+} + Fe(CN)_6^{4-} => (M)_2\ Fe(CN)_6$
M= Zn, Cu, Ni, Cd, Fe, etc.

The presence of copper ions catalyses the reactions. The influence of sulphur dioxide is not fully explained, but it is assumed, that some intermediary compounds are generated, that accelerate the reactions. On site uses ferric sulphate to even further stabilise any heavy metals.

The CN destruction is capable of reducing the WAD CN concentration in the slurry from 140 mg/l to below 2 mg/l, if the copper content in the ore is not too high. At high copper concentrations several stages of CN destruction may be necessary.

The characteristics of tailings from the various sites varies according to the mineralogy and mineral processing technique described above. However all sites using tank leaching destroy CN, using the SO_s-method, before discharging the tailings to the pond. The following table describes the CN concentrations of the sites using CN at several stages [50, Au group, 2002].

Site:	**Boliden**	**Ovacik**	**Filon Sur**	**Rio Narcea**	**Olympias**
Leach: Free CN WAD CN Total CN pH					
Measurement frequency					
Min					
Max					
Discharge from Detox: Free CN WAD CN Total CN pH	 0.87				
Measurement frequency	bi-weekly				
Min	0.31				
Max	1.94				
In TMF Free CN WAD CN Total CN pH					
Measurement frequency					
Min					
Max					
TMF discharge: Free CN WAD CN Total CN pH	 0.06				
Measurement Frequency					
Min	0				
Max	0.33				

Please provide information missing in table

Table 3.27: CN levels at European sites using cyanidation

The Orivesi Mine, where only mechanical mineral processing is practised, does not use any CN. At the Boliden mineral processing plant monitoring of the CN-destruction and the water quality of the discharge from the tailings and clarification pond was done during year 2001. Results showing that 99.5 % of the CN was destroyed. Further degradation of CN occurs naturally in the tailings pond. Similar results are reported from Ovacik and Rio Narcea.

3.2.3.7.4 Safety of tailings facility and accident prevention

At the **Ovacik** site a full risk assessment has been done, stability calculations have been performed and the design has been done using external experts. As described above, the design aims at assuring stability for seismic load, static stability, flood events and any other relevant parameter detected in the risk assessment.

The tailings facility is under daily surveillance for environmental monitoring and structural integrity. The site is routinely audited as per the mother company's environmental policies and an Ovacik Gold Mine Environmental Management System report prepared. The mine will be subject to the annual internal environmental audit program using the company's assessment process to assess the effectiveness of the environmental management systems and the level of environmental performance at the operation. An external audit by an independent expertise group was conducted during the trial operations.

Similarly, management plans on other issues such as health & safety, tailings storage, mine closure and rehabilitation, emergency action and community relations are in place.
[56, Au group, 2002]

At the **Orivesi** mine the tailings facility is inspected daily as part of the operational routines at the site. No formal risk assessment has been done, however, the dam undergoes annual audits by independent experts and every 5th year it is audited by the dam safety authority (Mine Inspector). The comments are recorded in dam safety document, which is a compulsory document for all similar types of tailings management areas since the mid-80`s.

In the construction phase of the tailings facility the soil characteristics were investigated. The system has been constructed so, that the surface of water in the tailings area can be kept in balance and the excess of water in case of rainfalls etc. can be removed in a controlled manner. There are no instruments installed to monitor the phreatic level in the dam body. A documented emergency plan does not exist. It is not clear if the environmental impact of the backfilling of tailings has been assessed.
[59, Au group, 2002]

The tailings pond at the **Boliden** mineral processing plant is managed according to an OSM manual (see Section 3.1.4.1) designed according to guidelines for dam safety, developed by the Swedish Association for Hydropower Operators (RIDAS). In 1997, when Boliden initiated a dam safety project for tailings dams it was decided to use RIDAS as a guideline where applicable to tailings dams. Changes would then be made when necessary, rather than developing new guidelines for tailings dams. Other mining companies have followed the same route [50, Au group, 2002].

At **Río Narcea** the dams are controlled using piezometers and inclinometers. The tailings pond undergoes regular audits by external experts. Risk assessment has been performed [58, Au group, 2002].

3.2.3.7.5 Closure and aftercare

At **Orivesi** a plan for closure and aftercare has been developed recently concerning the mine site and the industrial area. Only a draft plan has been made concerning the tailings management area. The main idea is to cover old tailings material from the nickel process with the tailing material from the gold process. A total of EUR 0.6 million has been reserved for closure [59, Au group, 2002].

At **Ovacik** mine rehabilitation will be done concurrent with the operation to the extent practicable. Topsoil removed during construction is retained on site for subsequent rehabilitation. A conceptual mine closure and rehabilitation plan has been prepared and will be reviewed annually during operation. Upon closure of the mine, the tailings pond area will first be covered by rock, gravel, clay and topsoil and then replanted with trees. Prior to the operation of the mine, a financial assurance bond was submitted to the competent authority to secure rehabilitation and closure in accordance with the operation permit protocol [56, Au group, 2002].

At **Boliden** a water cover solution has been chosen for the closure of the tailings pond. The dams around the tailings pond have been raised to their final height. The pond will be filled up in 5 years time after which it will be water covered according to existing permits. Apart from the water cover of the open tailings surface, the dams will be re-sloped to 1:3, covered and re-vegetated, long term stable outlets arranged and breakwaters constructed in shallow water depths to avoid re-suspension of tailings by wave action. All dams will receive additional long-term stable erosion protection. Back-pumping of seepage water will be carried out until the water quality has improved sufficiently to allow for its direct discharge. Water treatment will be conducted by straight liming at the outlet during the same time period, which is expected to last < 8 years.

Water cover as a decommissioning method has been used at various sites within Boliden. The water cover established at Stekenjokk in 1991 as been monitored and follow-up in detail, showing very good results, see Appendix XX (Eriksson et al., 2001).

An alternative decommissioning technique currently being evaluated is wetland establishment. This would allow for a higher sand level in the pond (better use of existing pond), less water stored in the pond (less risk) and a self generating organic oxygen consuming cover the top of the tailings.

Boliden is also trying out an alternative method called “water saturation” or “raised groundwater level” which basically is applicable where the natural groundwater level in the tailings is very shallow. By applying a simple soil cover the groundwater level can then be raised to permanently cover the tailings and eliminating sulphide oxidation (see Section 3.1.5).
[50, Au group, 2002]

At **Río Narcea** the tailings pond will be dewatered and using soil that has been temporarily stockpiled at the edge of the pond. Re-vegetation will be done and the area will be returned to original land-use (pasture). Pore water, containing CN concentrations < 1 mg/l WAD CN, will be collected through the installed underdrains in the pond and analysed before discharge.

For the **Lisheen** operation, due to the acid generating potential of the tailings, a permant water cover will be maintained by fixed spillways to provide long-term chemical stability. Erosion protection of the dams will by achieved by vegetation and, if necessary, by a rock cover [75, Minorco Lisheen/Ivernia West, 1995].

3.2.3.8 Tungsten

The tailings stream at **Mittersill site** represents 99 % of the initial process feed. At the present throughput of 450000 t/yr a storage volume of 250000 m³ is needed every year.

The Mittersill site operates two tailings management systems:

- a tailings pond approximately 10 km away from the mineral processing plant in a valley
- a backfilling system with a maximum capacity of 35 % of the mineral processing plant feed.

The tailings ponds cover an area of 34 ha of which 20 ha have already been reclaimed.

3.2.3.8.1 Characteristics of tailings

The chemical behaviour of the tailings has been characterised. The test procedures were:

- performing leachate tests
- determination of the total content of heavy metals by leaching the solids with aqua regia

The following tables show the results of these tests.

Parameter Leachate	Test results
pH	7.8
Conductivity, mS/cm	0.8
Ca, mg/l	10
Mg, mg/l	9
Al, mg/l	0.17
Sb, mg/l	<0.01
As, mg/l	<0.05
Ba, mg/l	<0.5
Be, mg/l	<0.005
B, mg/l	<0.01
Pb, mg/l	<0.05
Cd, mg/l	<0.005
Cr total, mg/l	<0.05
Fe, mg/l	<0.1
Co, mg/l	<0.01
Cu, mg/l	<0.01
Mn, mg/l	<0.01
Ni, mg/l	<0.05
Hg, mg/l	<0.001
Se, mg/l	<0.01
Ag, mg/l	<0.05
Th, mg/	<0.01
V, mg/l	<0.01
Zn, mg/l	<0.5
Sn, mg/l	<0.05
F, mg/l	<0.01
PO_4, mg/l	0.6
SO_4, mg/l	156
CN, mg/kg dry solids	n/d
F, mg/kg dry solids	n/d
NO_3-N, mg/kg dry solids	0.8
NO_2-N, mg/kg dry solids	n/d
Anionic surfactants, mg/kg dry solids	<0.05
Total hydrocarbons-C, mg/kg dry solids	n/d
Hydro-Carbons, mg/kg dry solids	n/d
Extractable organic halogens, mg/kg dry solids	n/d

Table 3.28: Leachate test results of tailings at Mittersill site
[52, Tungsten group, 2002]

Parameter total content	Test results (mg/kg dry solids)
As	7
Cd	<0.5
Co	<0.5
Cr	31
Cu	<0.5
Ni	22
Hg	n/d
Pb	12
Zn	82
THC	n/d
HC	n/d
PAH	n/d

Table 3.29: Heavy metal contents of tailings at Mittersill site
[52, Tungsten group, 2002]

The following figure shows the grain size distribution of the feed to the mineral processing plant and the tailings.

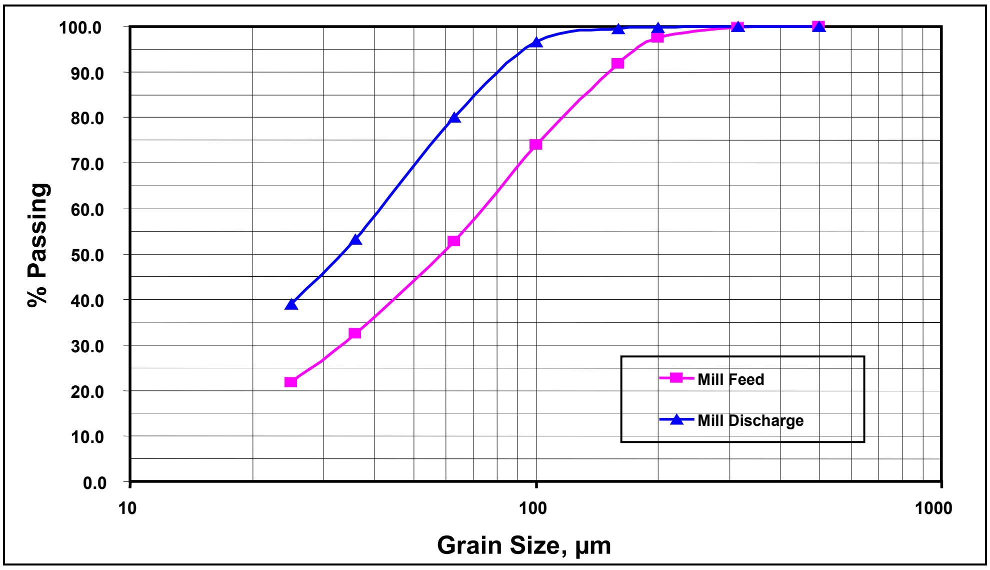

Figure 3.41: Size distribution of feed to mineral processing plant and tailings at Mittersill site
[52, Tungsten group, 2002]

3.2.3.8.2 Applied management methods

The backfilling system was installed in 1987 and consists of a lamella thickener, a piston diaphragm pump and a steel pipeline which connects the mineral processing plant with the different levels of the underground mine. The backfill has to be pumped over a distance of 3000 m and up to a maximum height of 280 m.

The currently operated tailings pond is situated south of the little village of Stuhlfelden. The start-up of the tailings ponds was in 1982. Until this time the first tailings pond, the “Felbertal” –pond, situated just on the opposite side of the mineral processing plant was in operation. The final height of this first tailings dam was 24 m. The dam was built using the upstream method. Every 8 m a drainage system was installed. The starter dam consists of borrow material, the second and third stage were built using tailings.

The tailings ponds in Stuhlfelden are built using upstream method. The final height of the tailings dam Stuhlfelden I & II was 16 m. The dams IVA and IVB will reach a final height of 24 m. The starter dams of ponds I and II with a height of 4 m were constructed using borrowed material. The starter dam of tailings pond IVA was built with tailings. To prevent erosion, the surface of the dam is covered with humus and revegetated. On one side the area is limited by a slope. Two roads which cross the slope 30 and 60 m above the pond prevent uncontrolled entering of surface water into the tailings pond area. Prior to construction of the starter dam, the area was investigated by geotechnical engineers. Where necessary, the foundation of the starter dam was reinforced. The construction was surveyed by geotechnical engineers and reviewed by the water and mining authority.

In spring and summer, the water surface in the pond is kept high enough to prevent dust emissions from the tailings pond area. In autumn water is discharged to the nearby stream. To prevent dusting from the tailings pond area, an automatic sprinkling system was installed. The sprinkling system is started and monitored from the central control room of the plant. During shutdowns of the mineral processing plant standby teams are on duty to control the tailings pond area. The nearest river, the river Salzach is approximately 600 m away from the tailings ponds.

3.2.3.8.3 Safety of tailings facility and accident prevention

The dams are raised in 2.5 m sections every year. The height of the layers applied to the dam surface is 0.5 m. The dam is divided in sections of 50 m. From every profile 4 samples are taken from the applied layer. The compaction is checked by using the proctor method. From one sample of every profile a particle size analysis is performed. The construction, monitoring, sampling and the data are controlled by a civil engineer and the federal authority.

For monitoring settlements of the tailings pond piezometers were installed. The ground movements are checked yearly. The data are controlled by the federal authority.

Monitoring of the TMF is performed 3 times a day by the process supervisors. In case of heavy rainfalls and failure of the barriers, excess water can be discharged through an emergency outlet.

To prevent erosion of the dam by the slurry, the inner surface of the dam is covered by a geo-textile.

3.2.3.8.4 Closure and aftercare

It is planned to cover the pond surface with humus and grass. After reclamation the land is given back to the land owners. The tailings of the Mittersill operation readily dewaterFrom experience, after a period of 2 – 4 years the tailings pond will be dewatered and consolidated.

Partial reclamation of the tailings pond is already performed during operation. The dam is constructed with he final inclination. The outer dam surface is already covered with humus and reclaimed.

3.2.4 Waste-rock management

3.2.4.1 Aluminium

No data has been supplied for this section. Please provide information.

3.2.4.2 Base metals (cadmium, copper, lead, nickel, tin, zinc)

At all sites, where the ore is mined underground, the small amounts of waste-rock from development works remain underground.

3.2.4.2.1 Characteristics of waste-rock

The waste-rock at **Asturiana de Zinc** is mainly dolomite (limestone). At the initial stage of the open pit mining clay (marl) and topsoil were also generated and stored separately for future use at the decommissioning phase.
[54, IGME, 2002]

The **Aitik** waste-rock has been subjected to extensive testing such as material characterisation, field-scale transport modelling, hydro-geological tracer tests, mineralogy and geology. The suite of tests performed include:

- whole rock analysis
- mineralogical investigations
- ABA
- kinetic testing like batch test, column tests, humidity cell test, large scale column weathering tests
- tracer tests to determine the water flow paths within the waste-rock
- effective surface area determinations.

Field characterisation includes

- in-situ measurements of oxygen concentration as a function of depth within the heaps
- temperature profiles within the heaps
- field-scale tracer tests
- determination of effective diffusion coefficient
- water flow and quality measurements
- water balances.

All this characterisation work has been used in various scientific exercises and in the waste-rock management planning of the Aitik site. Activities performed are, e.g., predictive modelling of water quality evolution with time, equilibrium and kinetic modelling of pore water and drainage composition, mass-balance calculations, coupled hydro-geological and transport modelling. Due to the extensive testwork done it has even been possible to use the information from Aitik in order to try to solve one of the biggest scientific challenges within this area - -namely the scale dependency between laboratory tests and the actual field scale conditions.

From these results it can be concluded that at Aitik two types of waste-rock are generated – about 65 % that will not generate ARD and 35 % which have the potential of producing ARD. It is a very small percentage that will actually produce ARD, however it is not feasible to separate it from rock that may produce ARD.

These results led to the decision to try to separately deposit the waste-rock that does not produce ARD and thereby minimise the surface area on which ARD-producing waste-rock is deposited. Since 1999 Aitik mine has used a new waste-rock dump for selective deposition of sulphide free waste-rock. This dump is named “The environmental waste-rock dump”. The results have also been used in order to develop an adequate decommissioning plan for the waste-rock dumps.

The environmental waste-rock is frequently tested and has to have less than 0.1 % S and 0.03 % Cu with a NP/AP ratio exceeding 3 to be accepted to use outside the mining area and for deposition in the deposit for “environmental waste-rock”. Tests conducted by different

laboratories have showed that the waste-rock quality is usable as ballast material for roads and railways as well as for use in asphalt.
[63, Base metals group, 2002]

Within the **Boliden** area (5 operating mines) waste-rock is managed based on detailed characterisation, mainly focusing on weathering characteristics. ARD producing waste-rock is preferably used directly in back-fill. In the case of open pit mining, ARD generating waste-rock is separately deposited and in the case of the Maurliden mine, the ARD generating material is temporarily stored in deposits and will be back-filled into the mined out open pit at closure where it will permanently be covered by water.
[65, Base metals group, 2002]

At **Zinkgruvan** the mineralogical composition of the waste-rock is given in the table below (based on microscopic analysis). The waste-rock consists of mainly quartz and feldspar (>70 %) and may contain traces of sulphide minerals. The ratio of carbonates to sulphur is >10:so the waste-rock has a high buffering capacity and will thereby not produce ARD. The waste-rock is regularly sampled and analysed for Zn and Pb content which over a large number of samples have been found to be 0.3 % and 0.2 % respectively. The density of crushed waste-rock is 1.75 tonnes/m^3 whilst the compact density of the rock varies between 2.6 and 2.7 tonnes/ m^3.
[66, Base metals group, 2002]

Mineral	**Fraction %**	**Mineral**	**Fraction %**
Quartz	32.8	Epidote	0.4
Plagioclase	1.0	Zoizit	3.1
Mikrocline	27.3	Calcite	2.5
Biotite	4.3	Titanit	0.3
Muscovite	1.6	Zircon	0.3
Hornblende	11.7	Apatite	0.1
Diopside	9.9	Other	0.5
Garnet	4.2	**Total:**	**100 %**

Table 3.30: Waste-rock mineralogy at Zinkgruvan
[66, Base metals group, 2002]

3.2.4.2.2 Applied management methods

At **Asturiana de Zinc** 2.5 Mt of waste-rock are produced. The waste-rock is deposited into an mined out part of the open pit. The old waste-rock dumps generated in the initial phase of the open pit mining are being covered with soil and re-vegetated. Restauration is done using clay (marl) and top soil separately stored for this purpose [63, Base metals group, 2002].

The waste-rock deposits at **Aitik** are situated east and west of the mine and cover an area of approximately 400 ha. In 2001 26 Mt of waste-rock were extracted from the mine, of which 67 % were separately deposited due to its low sulphur and metal content.

Today’s strategy is to avoid expanding the stockpile area containing sulphidic waste-rock. In 1999, a new waste-rock dump area, was opened. This dump is designated for non-sulphidic waste-rock exclusively, to allow for less extensive decommissioning procedures according to the permit. Furthermore, the quality of the rock opens opportunities for it’s utilisation as construction material.

Selective managing of waste-rock has been identified as a potential for cost savings and possible revenue if low sulphur material can be isolated. The bedrock from the hanging wall has a lower sulphide content and is therefore more suitable for selective management than rock from other parts of the mining area. The material consists of amphibole-biotite gneiss, which is intruded by pegmatite veins. The amphibole-biotite gneiss is characterised by a varying degree of amphibole banding, with a matrix of amphibole, biotite, quartz and to lesser extent plagioclase. The pegmatites contain mostly feldspar and quartz. The thrust fault makes a sharp contact between the hanging wall and the ore zone, making it easy to follow the contact. It is known that the hanging wall is barren of copper, and earlier mapping from diamond drill holes shows no change in the bedrock. The analyses carried out show low copper and sulphur content.

A new test procedure to secure the quality of the waste-rock was developed, including chemical analyses, acid base accounting (ABA-test) and humidity cell tests on drill core material on the forthcoming waste-rock mining areas in the hanging wall. This work led to further investigations. Drill chip samples from the production drilling were collected and tested for several different blasts, with positive results. Today, routines are implemented for testing this type of bedrock for every blast aiming at rapidly classifying the material for deposition on the new waste-rock dump. The material should be amphibole-biotite gneiss or/and pegmatite. Copper grades, sulphur content and the ABA-test are not to exceed the recommended values. All results are stored in databases.

In the latest waste-rock deposition plan of 1999, the conditions for selective management of various waste-rock fractions are regulated. The criteria for selective deposition of sulphide free waste-rock are less than 0.1 % S, less than 0.03 % Cu and a NP/AP ratio exceeding 3. Analyses shall be conducted on accumulated samples from minimum 8 drillholes representing 150000 t of waste-rock. To secure the quality, waste-rock within 30 m from the ore zone needs to be excluded.

The decommissioning method involves covering of the sulphide free waste-rock dump with 0.3 m of till and/or other material as vegetative layer. The decommissioning is undertaken progressively, and establishment of vegetation will be commenced within two years after completed deposition of each terrace.

Surface runoff and drainage water in collection ditches is collected and re-used in the mineral processing plant as process water. Collection ditches receiving effluents from old sections of the waste-rock dumps are receiving drainage water with high metal content and low pH. The quality of the water in the diversion ditches is strongly influenced by the local quaternary geology, with elevated sulphide content in the till.

At Aitik the hydrogeological investigations showed that the dumps are not hydraulically connected with the pit. The whole area, on which the dumps are located, is covered with a 10 m layer of low permeable glacial till on top of the bedrock. Virtually all the infiltrated water leaves the dumps at the toe, and is easily collected in ditches. Acid drainage with elevated content of copper was identified during the 1970's. Detailed field investigations in 1992 – 1993 estimated the annual amount of copper leaving the dumps to be 80 tonnes, of which 55 tonnes originated from the old marginal ore stockpile. The corresponding overall amount of sulphate was 4000 tonnes annually. During recent years, the bulk part of the marginal ore has been reprocessed and the influence on the pollution load of this undertaking is presently being evaluated.

A critical component of the decommissioning plan is to develop measures addressing the ARD situation. An engineered cover was soon identified as the only realistic way to deal with the waste-rock dumps, and between 1993 and 1996, a project using modelling tools to design a cover for reducing the flux of water and oxygen into the waste-rock was undertaken. The goal was a 99 % reduction in oxygen flux into the dump. Hydraulic properties of potential cover materials were measured and a number of cover designs involving layers of moraine and tailings sand were investigated. Following the modelling program, a cover design was selected for the waste-rock dumps. Physical tests of the glacial till in the area, i.e. the stockpiles and the overburden that has been or will be removed in the future, indicated that this material would be suitable for engineering of a cover suitable as a gas diffusion barrier of relevant quality.

A number of possible cover alternatives were evaluated. The results indicated that a 1 m layer of compacted moraine with a hydraulic conductivity of $1.5x10^{-7}$ m/s would reduce the oxygen transport into the dump to $1.2x10^{-9}$ kg O_2/m^2s - less than 1 of the reference case without cover. From this result, the estimation, based on weathering tests, was made that the reduction in copper pollution load would be of the same order of magnitude, resulting in a copper release of less than 1000 kg/yr.

An estimation of the influence of freezing, which could possibly affect the long-term performance of the cover, was that frost would penetrate the cover to a depth of 0.7 m. The penetration of frost is strongly depending on the depth of the snow cover, which at Aitik is considerable during winter. The snow reduces the frost penetration compared to normal engineering standards, which generally are conservative, assuming the ground is cleaned from snow.

To enhance the establishment of vegetation and to further secure the structure's resistance against frost penetration, it was concluded, that an additional top layer of 0.3 m of uncompacted till should be applied. An illustration of the decommissioned waste-rock dump and the proposed cover is shown in the figure below.

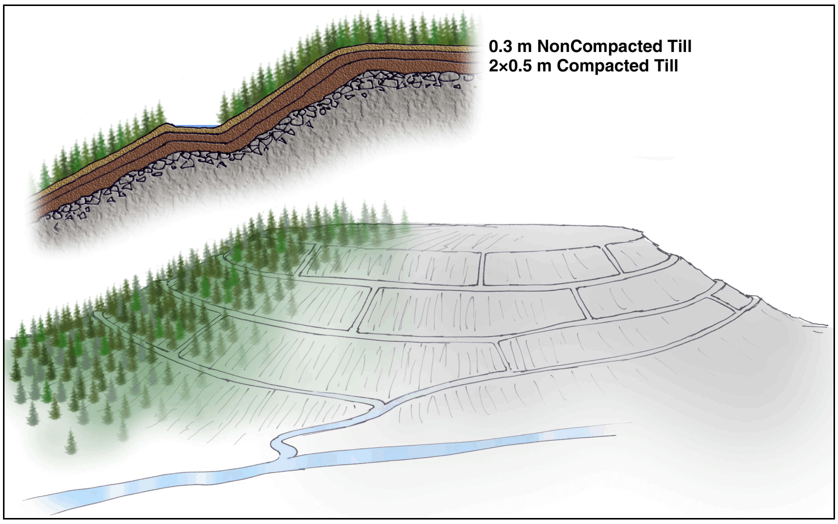

Figure 3.42: Structure of waste-rock dump cover and illustration of decommissioned waste-rock dump at Aitik site
[63, Base metals group, 2002]

The permit of 1997 allowed Aitik to commence the cover placement in 1997, with a 14 hectares area of the East waste-rock dump. This cover consisted of 1 m of moraine, distributed in two 0.5 m layers, which were compacted individually, and 0.2 - 0.3 m of topsoil. According to the permit, the maximum hydraulic conductivity was $2x10^{-7}$ m/s, and the compaction rate 93 % proctor. The surface was finally sown with grass during the fall the same year.

To divert surface runoff water, channels were constructed along the benches and down the slopes, using geo-textile and till. It soon became obvious, that a different solution regarding the surface water needed to be developed, as erosion from snowmelt water severely damaged the cover. Replacement of new till and erosion resistant waste-rock was the immediate solution, but for future cover steps, surface water management solutions must be designed in a way that does not endanger the integrity of the cover.

Placement of cover on the slopes, on the other hand, did not constitute any problem. The angle of the slope, 1:3, was shallow enough to allow normal operation of the conventional construction machinery.

In the coming years, additional sections of the waste-rock dumps will be covered in order to reduce the exposition of the waste-rock for oxidising conditions and to minimise the material handling and costs. Therefore cover placement will be synchronised with the overburden removal in connection to future mine development.

Since 1999, the Aitik mine has used a new waste-rock dump for selective deposition of sulphide free waste-rock. This dump so far has received 40.0 Mt of waste-rock and is frequently tested to verify that the permitted values, less than 0.1 % S and 0.03 % Cu and a NP/AP ratio exceeding 3, are met. Tests conducted by different laboratories on chip value, brittleness, ball mill hardness and particle density have furthermore showed that the waste-rock quality is sufficient for ballast material for roads and railways as well as for use in asphalt.
[63, Base metals group, 2002]

In the **Boliden** underground mines large quantities of waste-rock are moved directly to mined out areas within the mine. Only the waste-rock that is not used for backfilling is brought to the surface. In the case of open pit mining all waste-rock has to be brought up to he surface and deposited. At closure some of the waste-rock, e.g. highly acid generating rock, may be backfilled into the mined out open pit.

During year 2001 the following amounts of waste-rock have been used for backfill and deposited within the Boliden mining area.

Mine	**Waste-rock used in backfill (kt)**	**Waste-rock deposited (kt)**
Renström	82.1	-104.0
Petiknäs	103.4	-15.7
Kristineberg	127.6	4.6
Maurliden		875.7
Åkerberg	24.3	-21.0

Table 3.31: Amounts of waste-rock backfilled and deposited in the Boliden area

Waste-rock from deposits at the Petiknäs and Åkerberg mines has been brought back underground to be used in the back-fill (hence negative values). The waste-rock dumps at the Renström mine have shrunk significantly as material from the dumps has been used in the construction of a regional public road.

Generally it can be concluded that the managed waste-rock quantities are relatively limited, with the exception of the Maurliden open pit mine.

Waste-rock is managed based on detailed characterisation, mainly focusing on weathering characteristics. ARD producing waste-rock is preferably used directly in back-fill. In the case of open pit mining, ARD generating waste-rock is separately deposited and in the case of the Maurliden mine, the ARD generating material is temporarily stored in deposits and will be backfilled into the open pit upon closure where it will be permanently covered by water. All waste-rock deposits are surrounded by diversion ditches and drainage collection ditches. If required the drainage is treated before discharge.

Topsoil and moraine are deposited separately for future use in the decommissioning of the site.
[65, Base metals group, 2002]

At **Zinkgruvan** about 0.2 M of waste-rock are produced annually in preparation works. At the end of the mine life ore production will be possible for a couple of years without any waste-rock generation. The waste-rock is used for the construction of the dam of the tailings pond, as backfill in the mine and is also sold commercially outside the mine. About 0.5 Mt of waste-rock

is stored on the surface close to the old open pit as noise barrier around the east part of the industrial area. Any surplus of waste-rock is stored in deposits that are managed by an external entrepreneur who crushes and sells the material to third parties. During the years 1996 - 2000 58 % of the waste-rock has been sold to external users.
[66, Base metals group, 2002]

3.2.4.3 Chromium

Currently at **Kemi** waste-rock is deposited in three separated areas close to the mine. From 2003 on the mine production will develop gradually towards underground mining. The annual amount of waste-rock will therefore decrease and by the end of the decade all waste-rock will be directly backfilled in the underground mine. Waste-rock material from the old waste-rock dumps will also be used as backfilling material in the future.

3.2.4.3.1 Characteristics of waste-rock

The mineralogical knowledge exists, but no tests have been done [71, Outokumpu, 2002].

3.2.4.3.2 Applied management methods

The most important design parameters have been

- high stability of strata
- low permeability of the underlying strata
- short transport distance from mine
- good possibilities for material use in the future.

Drainage from the waste-rock dump area is not specifically monitored, but the emissions are included in the emission figures (see Section 3.2.5.3.3), because the emissions are calculated according to regular samples taken from the stream both above and below the mining site.

Part of the drainage water is collected in a ditch and it is led with other drainage waters from the industrial area to the tailings management area. There is also one part of the drainage that drains directly to the nearby stream.
[71, Outokumpu, 2002]

3.2.4.3.3 Site closure and aftercare

No plan for closure and aftercare has been made. No money has been reserved for the closure and aftercare.

The expected life time of Kemi Chromium Mine is tens of years. Therefore no closure plans have been developed, as it can be assumed that both the technical and the economic plans will be developed. There are no requirements to reserve money for the closure and aftercare.

As described above, waste-rock material will be used as backfilling material in the underground mine in the future. However, all the stock-piled waste-rock will not be required as backfilling material. No alternative use for the waste-rock can be foreseen. A plan for landscaping has been made, but no further closure plans exist.
[71, Outokumpu, 2002]

3.2.4.4 Iron

Two of the mining operations are underground mines. As a result, only smaller amounts of waste-rock are excavated for access tunnels. However, the dry magnetic separation tailings are included in the discussion of waste-rock, since the management of these coarse tailings is more typical of waste-rock than tailings.

At the **Kiruna** and **Malmberget** operations the coarse tailings are transported on a conveyer from the processing plant belt to silos and from there hauled to the so called "waste-rock" facility using dump trucks. The coarse tailings are dumped on heaps approximately 15 m high and at the natural angel of repose. In total these two sites manage about12 Mt/yr of "waste-rock" this way.

At **Erzberg** approximately 1.9 Mt/yr of "waste-rock" are managed 0.7 Mt of which are the coarse tailings from the dense media separation and 1.2 Mt of actual waste-rock coming directly from the open pit mine.

3.2.4.4.1 Characteristics of waste-rock

The **Malmberget** waste-rock (the coarse tailings) has not been characterised, however, the waste-rock at **Kiruna** has been tested for leachability and acid-base accounting (ABA), in addition to characterisation of the ore and country rock during exploration. Detailed mineralogical and trace element analysis is given under the tailings section (Section 3.2.3.4). Tests have also been performed to evaluate the amount of unexploded explosives left in the waste-rock material.

The leachability and ABA investigation indicated that the finer fraction of the waste-rocks (from the sorting plant) had the highest sulphide content (1.4 - 3 wt. % S). The neutralising capacity from calcite is, however, higher than the acid producing potential from the sulphides. The leach tests performed (humidity test), indicate that acid being produced due to sulphide mineral oxidation is neutralised by the calcite. The investigation also indicated that silicate minerals present in the test material also act as a neutraliser. The leach tests indicate that sulphate, calcium and magnesium are the main constituents leaching from the waste-rock.

The nitrate/ammonia leaching tests indicate that the ammonium nitrate left over from un-detonated explosive, is easily leachable and is primarily leached by the first infiltrating rainwater on the waste-rock.

Geotechnically, the waste-rock is stable. The coarseness of the material and truck dumping stabilise the material during deposition. The chemical weathering is very slow in the northern Sweden sub-alpine climate. Generation of clay minerals due to weathering is extremely slow. Therefore, no alternative deposition method have been considered.
[49, Iron group, 2002]

At the **Erzberg** site the waste-rock has not shown any sign of leaching and has been mineralogically characterised as follows:

- ankerite
- limestone
- schist ("Werfener Schiefer", "Zwischenschiefer"): quartz 46 %, dolomite 14 %, hematite 6 %, mica 4 %, feldspar 0.18 %, phyrophyllite 30 %
- porphyroid (small amounts): mica 8 %, quarz, 63 %, feldspar 5 %, chlorite 25 %
- fragmentation: 0 - 1500 mm.

Ankerite, limestone and porphyroid are quite resistant against weathering. On the contrary schist shows a rather high degree of weathering, in particular due to the meteorological conditions at the site.
[55, Iron group, 2002]

3.2.4.4.2 Applied management methods

There were no baseline studies performed prior to developing the waste-rock management facilities at two of the sites. However, at one site, an advanced design was carried out based on site investigations. The locations of all dumps have been chosen to be as close as practically and technically possible to either mine or processing plant.

For two of the sites the waste-rock management facility is located near the processing plant and extends to mined out open pits. In fact, at one site, the coarse tailings from dry magnetic separation were discarded into the mined out open pit over a short time period using a conveyor belt system. This is not done any more because of dusting problems.

At **Kiruna** and **Malmberget** the waste-rock is deposited on a thin soil cover or directly on bedrock. The bedrock consists of primarily volcanic rocks, trachytes, trachy-andesite, rhyolites, and rhyodacites. These rocks are very competent resulting in low risk for collapse into the underground mining operation [49, Iron group, 2002].

At **Erzberg**, due to the alpine location of the mine, space is scarce. The previous waste-rock dump was in operation until middle of the 20th century. After closure the tailings ponds was built in this dump area. As the capacity of the dump was exhausted, it was necessary to provide new dumping facilities. Based on investigations done by he company and in close co-operation with the local community, landowners and involved authorities a new area for the waste-rock dump was identified. This new waste-rock dump is located in a small valley close to the mining operation. The rivulets in this valley were over dumped while care was taken to warrant sufficient permeability for the water. Soil and loose material have been removed down to the competent rock. This formation is permeable and sits on top of an impervious bed, which consists of schist and porphyries. In the valley the base rock consists of porphyries, clay schists and carbonates. The total area of the dump is about 400 ha. Up to 2002 about 550 Mt of waste-rock have been dumped at this facility. The dump extends from the level of 1230 m to the toe of the end dam at a level of 821 m. The dump comprises several dump areas and has a total vertical extension of more than 400 m. The maximum height of a single dump slope is 70 m. The end dam, which is situated at the lowest part of the valley has a height of 147 m. The distance from the mining faces to the dump varies between 500 m and 1500 m in linear distance. Hauling distances for truck haulage are up to 3 km.
[55, Iron group, 2002].

<u>Design and construction</u>
As mentioned above, Erzberg needed to locate the waste-rock dump area in a valley due to the topography in the area. For planning and operation of the WRMF particular care was taken due to the specific situation of this dump with respect to:

- dumping at a mountain-slope area
- dumping on top of rivulets
- distance to residents
- alpine climatic conditions.

Therefore the planning of the project considered three key factors:

- ground conditions (geological, hydro-geological)
- waste-rock characterstics
- dumping method.

Many options for dealing with mining, soil mechanics, geology and hydraulic systems were discussed. The following issues were evaluated:

- avoidance of erosion and stability of the dump slopes
- avoidance of accumulation of water behind and inside the dumps
- studies about the flow rate through the dumps at high water flow
- evaluation of the quality of water after percolating through the dumps.

The basis for the design and construction of the WRMF was elaborated by an external consultant. According to the concept worked out, the bottom layer of the dump (valley base) consists of large-sized carbonatic rocks. The cross-section of this layer was designed so that in case of a flood (100-year event) the water can percolate through the dump without problems and without producing an increase of the streaming pressure. In addition an extensive testing program was executed by the responsible authority. Over a 2 years period penetration tests have been conducted which show that the maximum water flow can be managed if the base of the dump is constructed as proposed.

Based on these expert opinions and investigations the WRMF was approved by the mining authority in 1969. The approval comprises a series of strict instructions in respect of design and operation, including:

- before dumping the ground had to be cleared from vegetation, trees, roots and soil
- the dump must not exceed a general slope angle of 31° upon completion
- the cross-section of the lateral ditch for drainage must be designed large enough to handle run-off waters from the slopes
- the total bottom layer of the dump must be made of carbonatic rock blocks of a size between 400 - 1000 mm and must be at least 1.5 m high
- in the area of the previous bed of the rivulet block sizes of at least 700 mm should be used
- in the designated discharge zones only carbonatic rocks must be used
- at the toe area of the dump towards the valley a discharging body perpendicular to the valley must be made
- an appropriate monitoring system has to be implemented to check the phreatic surface within the dam
- the total workings for the dam and the seperate construction phases have to be well documented.

Both design and construction were evaluated by an external expert on the basis of the existing documentation for the closure in 1996. This evaluation showed that all instructions of the authorities had been followed and that there are no indications of any instabilities of the dump slope.

As described above the dumps have been designed to allow for a stream to flow underneath the dump. Apart from this the main factor for the waste-rock dump design is hauling distance from the mining area. As described above, the waste-rock and the dry magnetic separation tailings are transported on trucks and dumped within the waste-rock facility. The dumping is based on the natural angle of repose with no further change of the slopes. This has been the historic way of depositing the waste-rock. Since the material is considered to have only a minor impact on surface and groundwater or the surrounding soils, changes to these practices have not been made. The use of conveyer belts or slurry pumping is frequently being evaluated to replace the truck hauling. However, truck hauling has so far been found to be the most efficient and economic way of transporting the waste-rock.
[55, Iron group, 2002]

Operation
The deposition of waste-rock is similar at all sites. The waste-rock is hauled by trucks from the mining faces at distinct benches via the ramp system and from the dump area to the dump positions. The material is directly dumped from the truck over the dump slope or on top of the dump.

At Erzberg the dump heights vary between 40 and 70 m. With this method dump slopes will be between 33° and 38°. The overall general slope angle is kept lower than 28° [55, Iron group, 2002].

At the **Kiruna** and **Malmberget** sites the dumps are constructed in 15 m high lifts. The truck dumping method results in a gradation where the larger grain sizes roll down to the bottom of

the slope, while smaller grains settle higher up on the slope. This was used in the design of one of the dumps as described above in order to allow for a stream to flow underneath one of the dumps. In additon, there is likely to be some compaction on the top of each lift level due to the driving of the dump trucks. Later on natural compaction of the deeper parts of the waste-rock piles may also take place. None of these different compactions considerably influence the water flow. Most of the rainfall onto the waste-rock is likely to flow vertically through the dumps. When the infiltrating water has percolated through the dumps, a portion of the water will infiltrate the groundwater and a portion will flow on top of the bedrock and be visible as seepage at the toe of the dump. It is common practice to construct ditches at the toe of the waste-rock facility to control the seepage water. At one site, however, the seepage goes directly into the stream that flows under the dump.
[49, Iron group, 2002]

3.2.4.4.3 Safety of waste-rock facility and accident prevention

At two sites the waste-rock is considered to be chemically and geotechnically stable. For that reason, monitoring systems of the waste-rock facilities are not applied.

At the site where the stream flows underneath the waste-rock a monitoring plan is followed including geotechnical monitoring (surveying, piezometer measurements) and environmental monitoring.

3.2.4.4.4 Site closure and aftercare

As a part of the permit process for the waste-rock facility, one company has developed closure plan. As described before the waste-rock dumps are designed with 15 m lifts. The waste-rock on top of each lift is moved inwards leaving a ledge of 30 m. The reclamation concept is to focus on revegetation of the ledges adding soil and seeds of the local vegetation. A small rock berm will be constructed at the edge of each ledge. Water will be added to the revegetated areas in the early stages of the reclamation project.

The top of the waste-rock will slope from the centre to the edge of the waste-rock dumps. The dry coarse magnetically-separated tailings will be spread on top of each lift with a thickness of 0.5 - 0.7 m. On top of this coarse tailings material it is suggested to add a 0.2 m thick soil cover. Growth enhancing organic material is also suggested to be added to the soil.

At another site the reclamation measures to be taken after closure are part of the permit by the authorities. These measures are different for distinct areas and comprise landscaping and tree-planting. However, due to the local situation characterised by

- absence of mineralogical soil
- deficit of nutrients (mainly carbonates)
- coarse fragmentation (due to mining technique and weathering resistance)
- temperature gradient
- steep slope angles.

these measures will be difficult to realise.

Due to these difficulties the company has initiated a research project with specialists (biologists, reclamation experts, forest experts, mining engineers) to develop improved and site-specific reclamation techniques. Another important goal is to achieve site-specific vegetation in order to gain a sustainable reclamation. Within this project the total area is categorised in respect to reclamation aspects. By testing reclamation techniques over a 3-year period the most appropriate methods were selected. After 6 years of observing the vegetation progress sustainability of the measures is supported. Hence the company has the know-how to apply reclamation in the future with high potential of success and in a economic manner. Observed and documented effects of progressive recultivation of the waste-rock dumps are:

- improvement of water balance (percolation and surface drainage rate)
- improvements of visual impact
- increased habitats for flora and fauna
- improvement of bio-diversity in the area.

The methods developed are planned to be used for the areas currently in operation as well.

Long-term supervision for the WRMF is comprised of frequent monitoring of the seepage line within the end dam.

3.2.4.5 Manganese

No data has been supplied for this section. Please provide information.

3.2.4.6 Mercury

No data has been supplied for this section. Please provide information.

3.2.4.7 Precious Metals (Gold, Silver)

At Filón Sur 0.1 Mt/yr of waste-rock are generated. There is no information on how this is handled nor any information on the characteristics of this material [57, Au group, 2002].

At Río Narcea 6 Mt waste-rock were produced in 2001. Approximately 20 Mt waste-rock is stored in waste-rock dumps at the site. Topsoil is separately stored to be used in the reclamation of the site. Waste-rock from mine production will be backfilled in mined out open pits as production goes along. The initial waste-rock dump, from the initial open pit, will be decommissioned in-situ. The waste-rock consists mainly of silicates (granite and sandstone) and various carbonates (limestone) [58, Au group, 2002].

Orivesi uses all waste-rock as back-fill in the underground operations. No waste-rock is hoisted to the surface [59, Au group, 2002].

At Boliden the waste-rock is generated at the 5 mines supplying the mineral processing plant with ore. As these mines are mainly base metal mines, this waste-rock management is described under the section for base metals (see Section 3.2.4.2) [50, Au group, 2002].

At the Ovacik gold mine the overburden and the waste-rock are andesites which are currently used as rock-fill material on the downstream side of the TMF embankment. The waste-rock source at later stages of the mine will be from underground workings (galleries, drifts etc.) and these materials will be used as backfill in the underground voids.

Appropriate ARD potential and geotechnical property tests were conducted on the waste-rock. These tests revealed that the waste-rock does not have ARD potential and is has adequate properties for use in construction of the rockfill dam and retaining structures. Non-ARD potential of the waste-rock allowed the operator to use this material in the retaining structure of the TMF while providing an optimum use of the storage area requirement at the facility. The waste-rock is transported from the open pit area by trucks and placed on the downgradient slope of the TMF embankment and spread evenly and compacted with clay material.

Because of the inert nature of the waste-rock, there is no environmental risk associated with the waste-rock dumping unit at the Ovacik Gold Mine. (as per the probabilistic risk assessment by an independent consultant).
[56, Au group, 2002]

3.2.4.8 Tungsten

At Mittersill the waste-rock which is mined during development of the orebody is dumped into open stopes underground. There are no waste-rock dumps on the surface.

3.2.5 Current emission and consumption levels

3.2.5.1 Aluminium

3.2.5.1.1 Management of water and reagents

At Aughinish and at the Sardinian site water from the TMF is recycled to the process.

At Ajka a total of 1.75 Mm^3 of fresh water are consumed every year, of which 50 % are released into surface water.

The following table lists reagent consumption of alumina smelters

	Site			
	Ajka	Aughinish	Sardinia	
Reagent	consumption	consumption	consumption	consumption
Type:	g/t	g/t	g/t	g/t
NaOH	79167			
H_2SO_4	4167			
HCl	50			
Hg	3			
CaO	39167			
industrial flour	2333			
water glass	19333			

Please provide the missing information

Table 3.32: Consumption of reagents in g per tonne of feed of alumina refineries

3.2.5.1.2 Emissions to air

Air pollution may derive from the stack gases of the high capacity alumina calcinating kilns. Here electrostatic filters are used to separate the suspended solid particles.

Dust blowing of the TMF can be an issue. Spraying and hay spreading are applied in this cases in dry periods.

3.2.5.1.3 Emissions to water

Groundwater monitoring is carried our in wells around the stacks and ponds. If the effluent is discharged into surface water it is also analysed [22, Aughinish,].

3.2.5.1.4 Soil contamination

Due to the very low permeability of the mud seepage into the ground is very limited.

3.2.5.1.5 Energy consumption

The energy consumption related to the tailings management at the Sardinian site consists of the energy used in three pumping stations, to pump the following:

- the tailings slurried in water (fresh seawater and recycled water from the pond) from the refinery site to the pond, and distribution within the dam; power utilisation approx. 230 kW 100 % of the time
- the clarified water from the pond back to the refinery to suspend other mud and reduce the usage of fresh seawater, to keep the total water into balance; power utilisation approx. 60 kW, 70 % of the time
- the fresh seawater necessary to the tailings management, both for neutralisation and solids suspending purposes; power utilisation approx. 100 kW, 30 % of the time

[33, **EURALLUMINA**, 2002]

At Ajka in 2001 the energy consumptions were as follows:

▪ Energy:	127705 MWh or	21 kWh/tonne of feed
▪ Steam:	788300 t or	1.3 tonnes of steam/tonne of feed
▪ Natural gas:	35360000 m^3 or	58.9 m^3/tonne of feed

3.2.5.2 Base Metals (Cadmium, Copper, Lead, Nickel, Tin, Zinc)

3.2.5.2.1 Management of water and reagents

From the **Garpenberg** mine the mine water is pumped to the mineral processing plant and used as process water before it is pumped together with the tailings to the tailing pond system. From Garpenberg Norra the mine water is released to the recipient after clarification.
[64, Base metals group, 2002]

<u>Reagent consumption</u>
The following tables shows the reagents used at base metal mineral processing plants. Note that cyanide can be used for two purposes, as a depressant for sphalerite, pyrite and some copper sulphides or as a leachate for gold.

		Site								
		Aitik	**Almagrera**	**Asturiana de zinc**	**Boliden**	**Garpenberg**	**Hitura**	**Lisheen**	**Pyhäsalmi**	**Zinkgruvan**
Reagent		Consumption	Consumption	Consumption	Consumption	Consumption	Consumption	Consumption	Consumption	Consumption
Group:	**Type:**	g/t	g/t	g/t	g/t	g/t	g/t	g/t	g/t	g/t
COLLECTORS					179[1]					
	Xanthates					209	300	135	250	100 - 120
	Thionocarbamate							10.9		
FROTHERS					28					
	Sylvapine						150		50	
	MIBC							8.8		30 - 40
	Dowfroth							0.9		
ACTIVATOR										
	Copper sulphate				441	433		876	500	
	Sodium sulphide									
	Sodium hydrosulphide									
DEPRESSANTS					90					
	Sodium cyanide				310[2]				4	
	Zinc sulphate				92	306		234	400	30 - 50
	Iron sulphate					47				
	Acetic acid								15	
	Sodiumchromate				30	10				
	Dithiophosphate							55.1		
pH										
	Lime	408			3448	773	350	4368[5]	9000	
	Sulphuric acid						7500	5609[6]	12000[3]	300 - 500
	Sodium hydroxide				30					400 - 600
	Nitric acid								150	
	Hydrochloric acid				1					
FLOCCULANTS										
	CMC						100			
	Other							13.5	1	
OTHERS										
	Soda ash							472		
	"Flotation agents"	19								
	Sulphur dioxide				869[4]					

1. No information about collector type, probably xanthates; 2. Used in cyanide gold leaching; 3. Based on 100 % H_2SO_4
4. For CN destruction after cyanidation; 5. pH and water treatment; 6. pH and to leach

Please provide missing information in this table

Table 3.33: Consumption of reagents of base metal sites

The success of the flotation process is based on the proper use of the prescribed set of chemicals. Any reduction of the prescribed chemicals may affect the financial results of the production significantly. However, it is also necessary that the use of chemicals is kept to a minimum for economical and ecological reasons. To achieve this often the ore grade is measured frequently or even constantly so that the reagent addition can be adjusted accordingly. Newer technologies in this area are cameras that monitor the froth on the flotation cells on-line. Together with expert systems this results in optimisation of the process conditions and therefore higher recovery and a most advantageous addition of reagents [69, Nguyen, 2002].

As an alternative to xanthates as collector there are a number of different brands on the market. These collectors are of the type diaryldithiophosphates. A change into those collectors means for Zinkgruvan a change of the flotation process into a straight selective lead/zinc flotation process. The overall costs for chemicals in that process is twice the costs compared to the actual process used today. This is due to the fact that a set of other chemicals will be used i.e. copper sulphate, sulphur dioxide and slaked lime [66, Base metals group, 2002].

Water consumption

The following table shows the water consumption and percentages of re-used process water of base metal sites.

Site	Ore processed (tonne)	Water consumption (m^3/tonne)	used/re-used in mineral processing plant (%)	of which from tmf (%)	of which from mine (%)
Asturiana de Zinc	1100000	2.0	100	0	100
Almagera	1000000	3.2	0	0	0
Pyhäsalmi	1250000	5.3	0	0	0
Hitura	518331	6.2	100	90	10
Aitik	17700000	1.8	100	100	0
Garpenberg	984000	2.9	68	90	10
Boliden area	1450000	3.2	0	0	0
Zinkgruvan	850000	2.7	63	73	27

Table 3.34: Water consumption and water use/re-use of base metal sites

Note that at the Pyhäsalmi and Boliden sites water is partially re-used within the mineral processing plant.

At **Pyhäsalmi** there is no re-use for process water from the TMF area to the process. The reason being gypsum ($CaSO_4$) in the water causing blocking problems in the pipes. There is only an internal re-use of water in the process, where water from the thickener in the pyrite flotation is returned to the grinding circuit to save sulphuric acid in the pyrite flotation and lime in the Cu-flotation. This amount of water is corresponding to 10 % of the total amount needed in the mineral processing plant.

Fresh water is pumped from a lake. The water balance for 2001 is presented in the figure below.

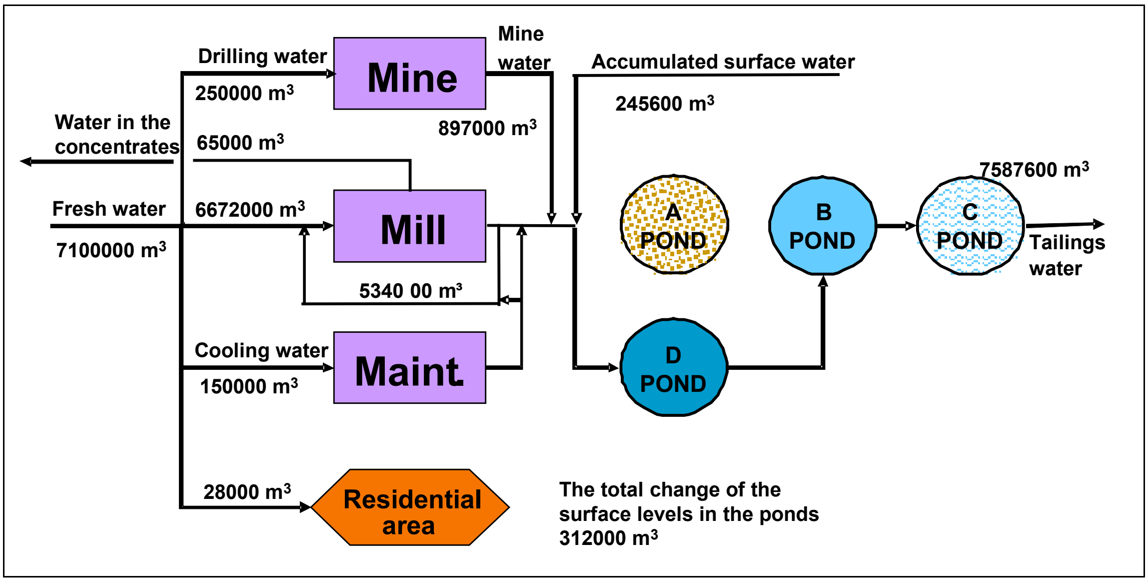

Figure 3.43: Water balance at Pyhäsalmi for the year 2001
[62, Base metals group, 2002]

At **Hitura** the clarified water from the TMF is re-circulated to the process. The amount of this water is corresponding to almost 100 % of the total amount of water used in the process. This system does not save reagents significantly, because flotation chemicals like xanthate and frothers are decomposed in the tailings area and the tailings material consumes the sulphuric acid. The water balance is presented in the figure below.

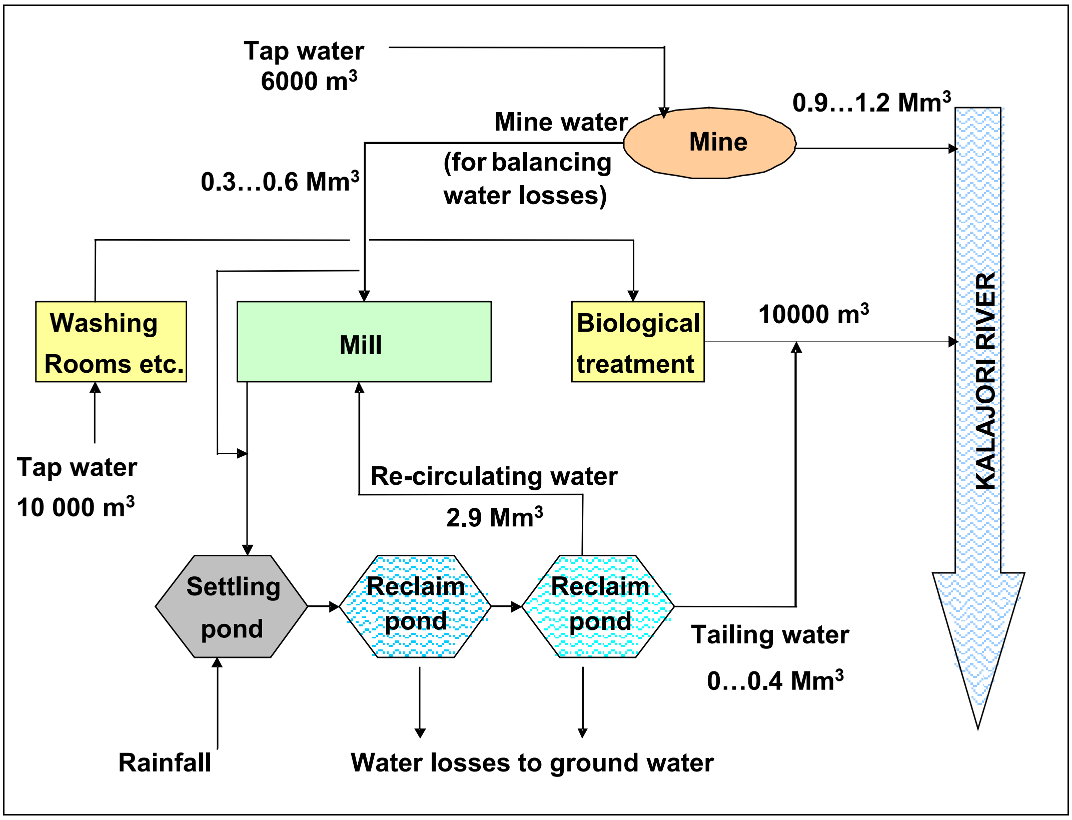

Figure 3.44: Water balance at Hitura
[62, Base metals group, 2002]

It can be seen that, depending on rainfall the amount of water from the tailings pond used/re-used in the mill (mineral processing plant) varies between 88 to 100 % (0 to 0.4 Mm^3 to the river)

The **Aitik** mineral processing plant uses 100 % re-used water from the tailings pond. Due to the tailings pond accident at Aitik in year 2000 it was necessary to use 1.08 Mm^3 of fresh water during the period between February and April 2001. Under normal conditions the entire water consumption, 31.5 Mm^3/yr, is supplied by re-used water from the tailings pond. Approximately 1.8 m^3 of water per tonne of ore processed is used in the process plant. In the snow smelt period excess water is normally released from the clarification pond to the recipient. The released water is of good quality and no water treatment is required (see Section 3.2.5.2.3).
[63, Base metals group, 2002]

At the **Garpenberg** mineral processing plant the consumption of used/re-used water was during year 2001 1.95 Mm^3 and the consumption of freshwater during the same period was 0.93 Mm^3. The discharge from the tailings pond amounted to 4.55 Mm^3. Out of this volume approximately 50 % were re-circulated to the mineral processing plant and re-used as process water. The remaining 50 % were discharged to a lake.

From the Garpenberg mine the mine water is pumped to the mineral processing plant and used as process water before it is pumped together with the tailings to the tailing pond system where water treatment occurs through interaction with the fresh mineral surfaces which effectively absorb any dissolved metals.
[64, Base metals group, 2002]

At **Zinkgruvan** the water consumption in the mineral processing plant is approximately 2.7 m^3/tonne or 2.4 Mm^3/yr in total. The water requirement is covered by freshwater supply from nearby lakes and by recycling of water from the tailings pond (partly process water and partly mine water).

The main consumption of water is in the actual process, in the paste fill and for cooling purposes. The entire water balance is illustrated in the following figure.

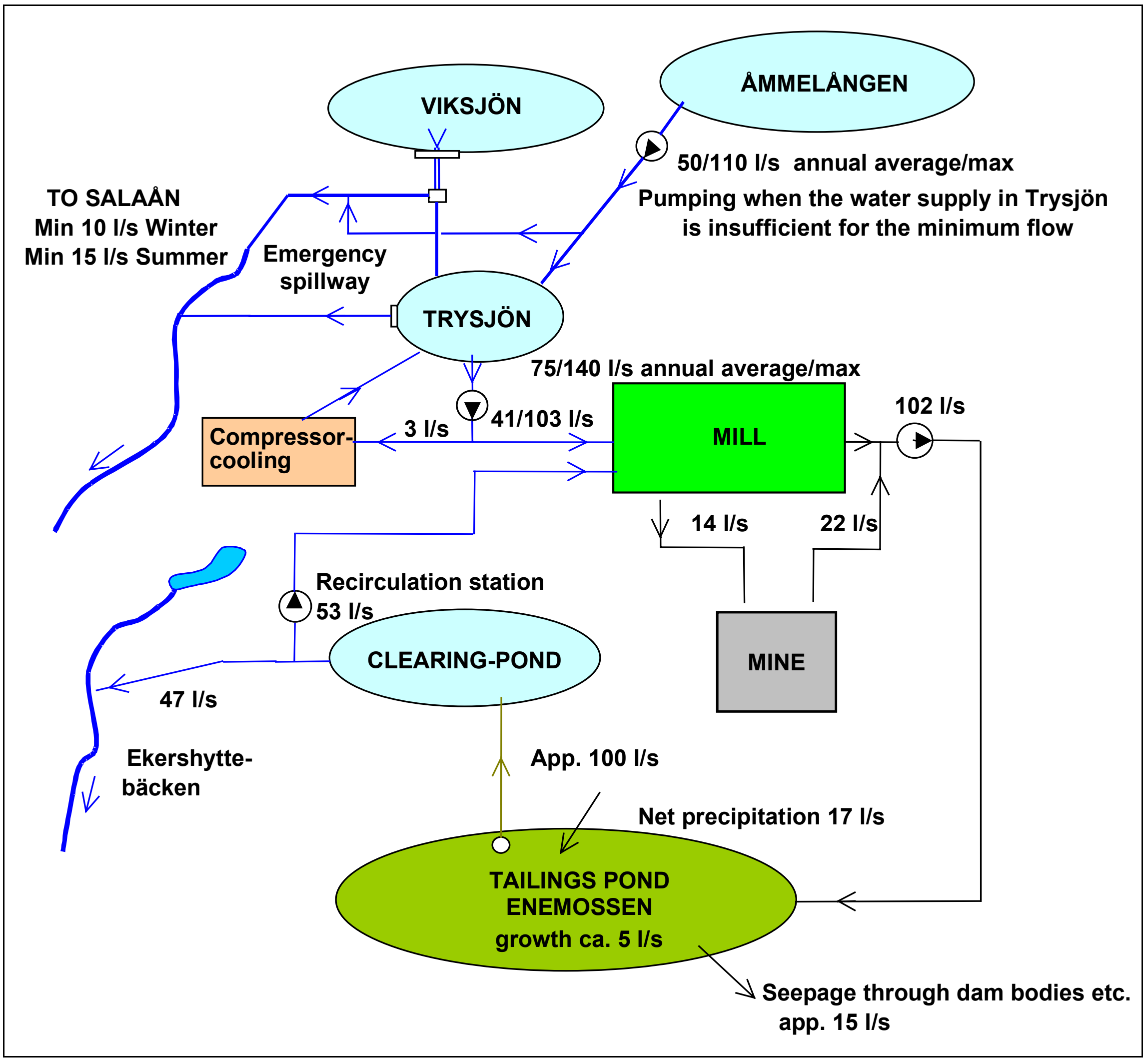

Figure 3.45: Water balance for the Zinkgruvan operations shown as average annual flows and maximum flow during operation
[66, Base metals group, 2002]

At Lisheen process water is re-used and supplemented with water reclaimed from the TMF [73, Ivernia West,].

3.2.5.2.2 Emissions to air

At Lisheen the emissions to the atmosphere are monitored using the following measurements:

- point source
- ambient air
- dust deposition.

[41, Stokes, 2002]

The emissions in 2001 are listed in the following table.

Parameter	Unit	Quantity
Particulates	kg/yr	3375
Nitrogen Oxides	kg/yr	243266
Carbon Monoxide	kg/yr	129546
Carbon Dioxide	kg/yr	186713872

Table 3.35: Emissions to air at Lisheen site
[76, Irish EPA, 2001]

At Pyhäsalmi the main sources of emissions to air have been identified as:
- dust and SO_2 from concentrate drying in the mineral processing plant
- dust from the TMF
- dust from concentrates loading area
- dust from roads and industrial area.

Dust emissions are measured at several collecting points. The main purpose is to survey the area of influence. Since June 2001 emissions have been controlled also with an automatic device, which continuously takes measurements but no exact figures are available yet.

Dusting from the tailings management area is a problem in dry and windy weather. Attempts have been made to prevent this by spaying lime slurry on the banks.
[62, Base metals group, 2002]

At Hitura the main sources of emissions to air have been identified as:
- dust from the industrial area including TMF, crushing plant etc.
- dust from roads.

The area of influence is monitored at several collecting points.

Dusting from the TMF is a problem in dry and windy weather. Attempts have been made to prevent dusting by covering the banks immediately after raising with soil material and using lime slurry on the banks. Also the water surface level in the tailings pond is kept as high as possible in summer time and tailings distribution is arranged so that the beach area is kept as wet as possible.
[62, Base metals group, 2002]

The Aitik site follows a comprehensive monitoring program for emissions to air. At the site there are mainly three sources of emissions to air:
- emissions from the drying of the concentrates
- emissions from blasting and diesel vehicles, and
- diffuse dusting from the whole site including the tailings pond.

However, emissions from blasting, diesel vehicles and drying of concentrates are not part of the scope of this document. It should be noted, though, that drying oven are being gradually replaced by filters.
the site as sedimented particles. The collected samples are analysed for copper and the total weight of sedimented particles (normalised towards the surface area of the collector). The results are summarised for the years 1999 to 2001 in the table below.
[63, Base metals group, 2002]

	1999		2000		2001	
Monitoring point	Sedimented particles	Cu	Sedimented particles	Cu	Sedimented particles	Cu
	mg/m^2month	mg/m^2month	mg/m^2month	mg/m^2month	mg/m^2month	mg/m^2month
S 1	1210	1.5	1910	2.5	3030	2.6
S 7	450	0.4	330	0.3	480	0.4
S 8	394420	21.4	55550	19.8	23440	12.7
S 9	1100	0.7	720	0.3	2610	1.0
S 10	920	0.9	750	0.7	540	0.5
S 11	690	0.7	1200	0.8	480	0.5
S 12	1820	0.8	1360	0.8	1000	0.9
S 13	520	0.3	860	0.5	780	0.4

Table 3.36: Measurements of total sedimented particles and Cu at Aitik
[63, Base metals group, 2002]

At Garpenberg there at mainly two sources of emissions to air – the drying of concentrates and ventilation from the mines (SO_2, NO_2 and CO_2) [64, Base metals group, 2002].

The emissions to air for the Boliden site are discussed in the precious metals section.

3.2.5.2.3 Emissions to water

The following table summarises the total emissions to water from base metals sites.

		Site					
		Aitik	Boliden	Garpenberg	Hitura	Lisheen	Pyhäsalmi
Parameter	**Unit**	Year					
		2001	2001	2001	2000	2001	2000
Discharge	Mm^3	6.44	11.10	2.60	0.08	22.9	6.89
Ca	t	-	-	-	-	-	4727
SO_4	t	-	-	-	254	-	12057
COD	t	-	-	-	-	51.4	334
Solids	t	-	-	6.2	0.9	89.4	47.1
Al	kg	446.0	-	-	-	2465	-
As	kg	1.7[1]	156	18	-	-	-
Cd	kg	-	1	0.8	-	8.1	7
Co	kg	5.3	-	-	-	17	-
Cr	kg	0.2[1]	-	25	-		-
Cu	kg	36.0	72	40	-	28.5	309
Fe	kg	-	-	-	24	1412	9141
Mn	kg	-	-	-	-	565	-
Hg	kg	0.1	-	0.3	-	0.6	-
Ni	kg	5.1[1]	-	-	107	311.9	-
Pb	kg	0.1	191	52	-	263	-
Zn	kg	34.6	1070	586	-	2321	1464
N	t/yr	17.0	-	6.5[2]	-	40892	-
1. Dissolved 2. Year 2000							

Table 3.37: Total emissions to water from base metals sites

Table 3.38 shows the concentrations in the emissions from tailings management facilities.

		Site		
		Aitik	Garpenberg	Zinkgruvan
Parameter	**Unit**	Year		
		2001	2001	2001
pH		7.1	10	7.5
Susp. particles	mg/l	-	2.4	3.1
Mineraloil	mg/l	-	0.1	-
Copper (dissolved)	µg/l	2.1	-	-
Copper (total)	µg/l	7.3	15	2.7
Zinc	µg/l	1.7	218	220
Lead	µg/l	0.02	20	27.3
Cadmium	µg/l	0.004	0.37	0.3
Arsenic	µg/l	0.3		1.9
Chromium	µg/l	0.004	9	<1.0
Mercury	µg/l	0.009	-	<0.1
Iron	µg/l	8	-	-
Aluminium	µg/l	38.5	-	-
N-total	mg/l	2.6	-	5.4

Table 3.38: Concentrations in emissions from base metals sites

The annual total discharge from Zinkgruvan was 1.5 Mm^3.

At Hitura emissions from the TMF to groundwater have been reported. Exact figures are not available. The flow of ground water has been cut and the contaminated water is back-pumped and led to the river [62, Base metals group, 2002].

At Aitik water sampling is carried out at the discharge point (clarification pond) and at 12 sampling stations in the river systems according to the regular monitoring program. The samples are analysed for several metals, pH, N-total, oil, SO^4-S, conductivity and turbidity. Water was, during year 2001, only discharged from the clarification pond to the Leipojoki river. No discharge was done from the recycle pond nor from the recycle channel [63, Base metals group, 2002].

Garpenberg follows a broad monitoring program for surface waters as well as recipient sampling and control, which is carried out within an integrated program for the catchment area (the main river in the area). This program contains water sampling analysis, fish investigations, sediment and bottom fauna investigations. The discharge from the tailings pond is sampled by an automatic sampler every two hours and a composite sample is produced monthly.

Sufficient water quality for the process and for discharge is obtained in the tailings pond/clarification pond system. The main contaminants are Zn and N predominantly coming from the mine water. The mine water containing approximately 4.5 mg/l Zn and up to 50 mg/l of total N. Major reductions in discharge of Zn to the environment have been obtained by pumping the mine water together with the tailings slurry to the tailings pond whereby the Zn adsorbs to the mineral surfaces. Laboratory test work has shown that the method effectively reduces the Zn concentration in the mine water from 4.5 mg/l to less than 0.2 mg/l in 40 min.
N compounds are partially degraded in the tailings and clarification ponds. In 1998 it was estimated that about 10 tonnes of N was added to the system from the mine water.
[64, Base metals group, 2002].

The emissions to water from the Boliden tailings pond are described in detail in the gold section.

At Zinkgruvan the tailings and tailings pond system constitutes a very good treatment facility for the process and mine water due to its high adsorbtion capacity. By fully utilising the characteristics of the system and passing all mine and process waters through the system significant reductions in Zn discharge have been achieved over the last 15 year period as illustrated in the figure below.

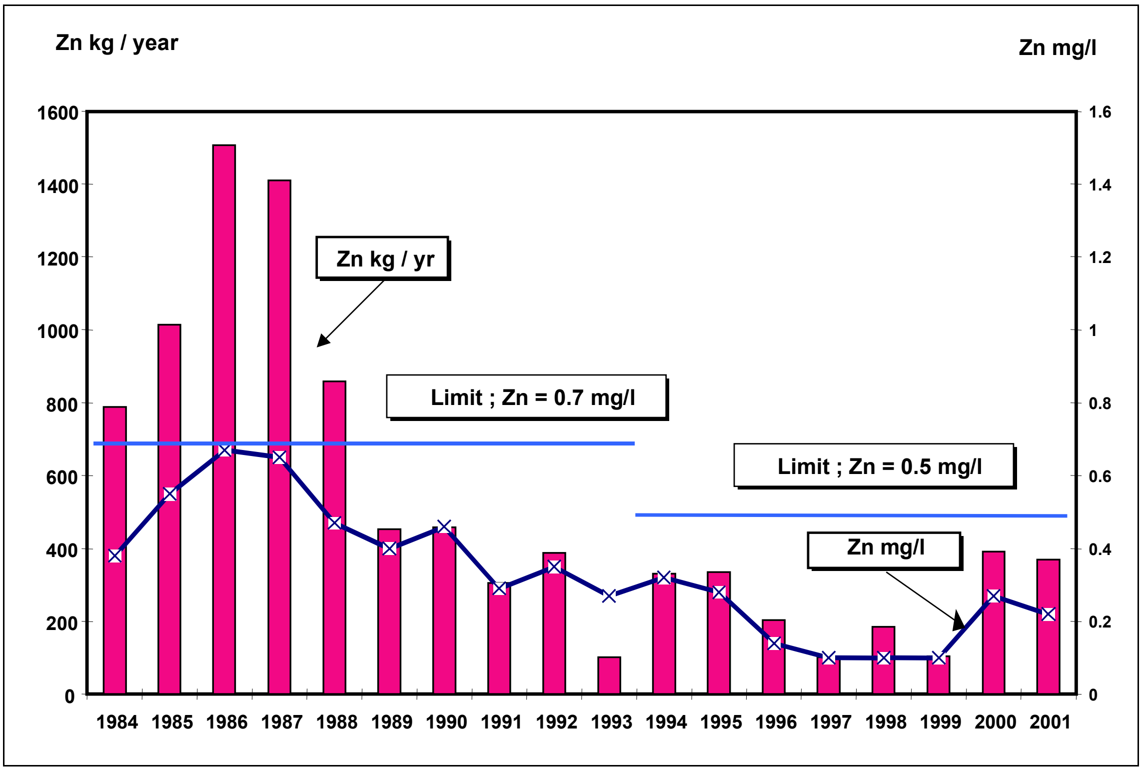

Figure 3.46: Annual average zinc concentration (in mg/l) in excess water from the clearing pond to the recipient and calculated transport (kg/yr) 1984 - 2000
[66, Base metals group, 2002]

At Lisheen arsenic is treated with ferric sulphate if the concentration in the discharge is above 0.0048 mg/l. Thereby the arsenic is precipitated as metastable ferric arsenate compound. Similarly if cyanide is added in the process as a suppressant and the concentrations in the discharge approach 0.048 mg/l the CN will be destroyed [75, Minorco Lisheen/Ivernia West, 1995].

3.2.5.2.4 Soil contamination

At Pyhäsalmi soil contamination in the close environment of the plant has been observed, which was caused by sulphur (pyrite) dusting. No significant contents of heavy metals or chemicals have been reported in the soil.

In an area of about 400 m around the TMF soil contamination was discovered at Hitura.
[62, Base metals group, 2002]

3.2.5.2.5 Energy consumption

The following table summarises the energy consumptions of base metal sites.

Energy consumption	Unit	Site					
		Aitik	**Boliden**	**Garpenberg**	**Hitura**	**Pyhäsalmi**	**Lisheen**
Mine	kWh/t[1]						
Mineral processing plant	kWh/t[1]				32.8	34.9	47.3
	GWh[1]						53.4
Grinding	kWh/t[1]	11 - 12	22				20.6
Dewatering	kWh/t[1]				0.22	3.9	
TMF	kWh/t[1]	2	2	3	1	1.6	
Waste-rock management	kWh/t[1]						
Total electrical	kWh/t[1]	22.1					
Total all energies	GWh	545.5	214.6	123.5			
	kWh/t	30.7	148	126			
Ore processed	Mt	17.77	1.45	0.98	0.52	1.25	1.15
1. Electrical energy 2. Total = mine+mineral processing plant+TMF+waste-rock management							

Table 3.39: Energy consumptions at base metal sites

3.2.5.3 Chromium

3.2.5.3.1 Management of water and reagents

The following table shows the reagents and steel in the mills consumed per tonne of ore processed.

Reagent	Consumption (g/t of ore processed)
Flocculent	13
Steel balls	50
Steel rods	200
Ferro silicon (for heavy media separation)	80

Table 3.40: Consumption of reagents and steel at Kemi site
[71, Outokumpu, 2002]

In the process there are arrangements for internal re-circulation to minimise the consumption of water. Re-use of the clarified water from the tailings management area corresponds to almost 100 % of the total demand of water in the process. Sometimes (usually, when a dam raise is ongoing) it is necessary to add fresh water. The excess of water from the system is removed to the stream without any further treatment.

A water balance is not available.

[71, Outokumpu, 2002]

3.2.5.3.2 Emissions to air

Dust emissions are not regarded as a significant problem. The mineral processing plant has installed de-dusting equipment. The dust emissions from the mineral processing plant have been estimated to be around 1.8 t/yr. The area of influence is assumed to be very limited based on results from moss investigations.

Dust from the open pit and loading area has been estimated to be around 30 t/yr. Also in this case the area of influence is very limited.

At intervals of five years sampling of moss is carried out for determination of heavy metals and suspended particles.

No emissions from the waste-rock dumps to air are specifically monitored. However, any dusting from the dumps is monitored in an integrated way for all emissions to air in the moss-investigations described above.
[71, Outokumpu, 2002]

3.2.5.3.3 Emissions to water

The discharge to the stream is sampled on a monthly basis and is carried out by an external expert, when taking samples from the surrounding streams. The samples are taken, if effluent is occurring.

During year 2000 the total emissions to surface water are summarised in the table below. The year 2000 was exceptionally rainy and wet, which resulted in extraordinarily high amounts of discharge from the pond system. However this did not influence the other parameters listed in the table.

Parameter	Unit	Amount
Discharge from pond system	Mm^3	1.67
Ca	t	191
Solids	t	33
Fe	kg	11000
Cr	kg	79

Table 3.41: Emissions to surface water at Kemi site
[71, Outokumpu, 2002]

Note that Cr is in solid form and therefore included in the solids.

3.2.5.3.4 Soil contamination

No significant contamination reported at Kemi. Limited areas, such as locations for old stockpile of chromium concentrate, are probably contaminated [71, Outokumpu, 2002].

3.2.5.3.5 Energy consumption

The energy consumption for the tailings management is given in the table below for the year 2000.

Process step	Electrical energy consumption (kWh/tonne of ore)
Mineral processing	16.6
Dewatering	1.5
Tailings Management	0.9

Table 3.42: Energy consumption data at Kemi site
[71, Outokumpu, 2002]

3.2.5.4 Iron

All operators follow established monitoring programs agreed on with the competent authorities. Results are reported in required intervals.

The operator of the Malmberget and Kiruna sites has implemented a monitoring system for the environmental effects of emissions. The program contains descriptions of sampling procedures, analysis, and reporting for environmental control. There are instructions and procedures within the company operation system that describe sampling in detail.

Monitoring is according to the following minimum protocol:

- discharge control in one sampling point at least 10 times a year. The analysis includes pH, carbonate nitrate, phosphorous, hydrocarbons and metals
- recipient control is based on two sampling points and in one reference location (for background level) at least 6 times a year. The analysis parameters include pH, carbonate, and phosphorous
- recipient- and surroundings investigations. Investigations of the recipient environment are carried out every 3 - 5 years. The investigations consist primarily of sedimentological and biological evaluations
- evaluation of flooding overflow water from the clarification pond takes place continuously

[49, Iron group, 2002]

3.2.5.4.1 Management of water and reagents

At **Kiruna** the total water intake into the mineral processing plant was 61 Mm^3 in 2001. Of this 3 Mm^3 were captured surface run-off, 9 Mm^3 mine water and the rest, 49 Mm^3, was water re-used from the clarification pond. For the 23 Mt of ore processed in that year, the process uses 2.6 m^3/tonne of ore, of which 80 % are re-ciculated from the pond [51, Iron group, 2002].

In the flotation at Kiruna the following amounts of reagents are consumed in a year:

- collector: fatty acid, 290 tonnes
- depressant: sodium silicate, 1500 tonnes containing 94 tonnes Na and 194 tonnes Si
- conditioner: sodium hydroxide, 60 tonnes containing 35 tonnes Na.

The fatty acid, coming from the flotation process, going to the tailings corresponds to 250 t/yr (86 % of total consumption), of which approx. 63 % are methylic carbon and 27 % carboxylic carbon. The fatty acids are attached to the mineral phases and are transported to the tailing pond were they sediment and decay. The complete aerobic decay can be described by the formulas below:

$CH_2\text{-} + 2O_2(g) + 2H+ = CO_2(g) + 2\ H_2O$

$COOH + ½\ O_2(g) + H+ = CO_2(g) + H_2O$

5 tonnes of flocculent (sodium acrylate) is used in the ponds and contains about 5 tonnes Na and 1 tonne N.

There is no collection of runoff water/seepage from the waste-rock facilities except for a drainage ditch around parts of the dumps. In these two cases the drainage flows naturally into the tailings ponds.

At the **Erzberg** operation the mineral processing plant uses 90 % re-circulated water from the screw-classifiers. Drainage water from the tailings ponds percolates through the waste-rock dump and drains into a stream that flows under the dumps. No chemicals are used in the process. The tailings are inert and do not leach nor weather to any notable degree.

None of these operations have completed water balances. At Kiruna, however, as part of a groundwater investigation to estimate sources for contaminants to a lake, the drainage from the waste-rocks into this lake was calculated to be approximately 1.13 Mm^3/yr.

3.2.5.4.2 Emissions to air

The most severe dust problems at the waste-rock dumps occur on dry days from the crushing, transport and dumping of the waste-rock. The haul roads are then watered to reduce this problem and dumping facing populated areas ceases during windy or dry days. At one site, progressive reclamation minimises the open waste-rock dump area and thereby also the possible dust emissions.

Ponds in operation at **Erzberg** are kept water covered or water saturated. This is possible due to the alpine weather conditions with:

- high precipitation rate of about 1200 mm/yr
- short summer period
- protection by nearby mountains against wind.

At **Kiruna** and **Malmberget** sampling of airborne particles is performed continuously at several locations around the three mining operations and within the residential areas. During the winter, the snow is collected at the sampling points and analysed for particles.

Testing of air imissions the last few years at the three sites indicates that solid particles have been less than 220 g / (100 m^2 x 30 days) for Kiruna, 18 - 220 for Malmberget, and < 200 for Svappavarra residential area. The solid particles trapped in these tests are primarily from other parts of the mining facility and not from the tailings dams. Snow samples are collected during the winter at several collection points. These samples are analysed for airborne particle distribution and reported yearly.

3.2.5.4.3 Emissions to water

At **Erzberg** water discharges are monitored. No negative effects on the downstream water quality have been detected nor have any threshold values been exceeded.

For the other sites the emission to water is variable for each of the large sites. The following sections give a description for each of the sites. Groundwater samples have been collected in order to evaluate transport of nitrate from the coarse tailings facilities.

At **Kiruna** approximately 9 Mm^3 are discharged yearly from the clarification pond to the surface water system. The yearly average discharge rate is approximately 16.8 m^3/min. The discharge rate over the year is highly variable, and follows the natural drainage cycle, however, with some time delays. The total amount of nitrate and phosphate discharged in 2001 was 116 tonnes and 251 kg, respectively, which is in the range of the discharge over the last 10 years. Discharge concentrations for nitrate are approximately 13 mg/l, and for phosphate, the discharge concentrations are approximately 0.03 mg/l (average concentrations for the year). Nitrate comes from the un-detonated explosives and the phosphate comes from the ore.

The following table shows a complete analysis of the discharge of this site.

Parameter	Conc.	Units
Al	10.7	ug/l
Aliphatics	<0.1	mg/l
Aromatics	<0.2	mg/l
As	0.59	ug/l
Ba	31.35	ug/l
Ca	160.7	mg/l
Cd	0.009	ug/l
Cl	123.8	mg/l
Co	0.18	ug/l
Cr	0.049	ug/l
Cu	1.79	ug/l
F	1.71	mg/l
Fe	0.049	mg/l
HCO_3	1.10	mmol
Hg	<0.002	ug/l
K	35.1	mg/l
Conductivity	139.7	mS/m
Mg	20.05	mg/l
Mn	32.36	ug/l
Mo	53.94	mg/l
Na	80.37	mg/l
Ni	0.92	ug/l
NO_3-N	11.33	mg/l
P	25.54	ug/l
Pb	0.0429	ug/l
pH	8.03	
S	141.1	mg/l
Si	3.684	mg/l
SO_4	431.2	mg/l
Sr	551.1	ug/l
Susp. Solid	3.14	mg/l
Tot-N	12.77	mg/l
Tot-P	0.0274	mg/l
Turbidity	1.871	FNU
Zn	0.924	ug/l

Table 3.43: Average concentrations of an iron ore tailings facility discharge to surface waters for 2001

From the **Svappavarra** facilities there is normally no or only marginal direct water discharge of process water to the recipient water system except for leakage through the dams. For the year 2000, approximately 130000 m^3 water were reported discharged during the period from May 23 to June 14, due to an unusually high water level in the clarification pond. Four sampling points are frequently sampled for water quality in connection with the tailings facility.

Water quality in the tailings ponds complies with Swedish and European water quality standards. Water from the tailings ponds discharges into the clarification ponds. Excess water from the clarification pond is used either as process water or for transport of the tailings to the tailings dams. Excess water from this cycle is discharged to the river system according to the discharge permits. In 2000, approximately 80 % of the excess water entering the clarification pond were re-used in to the processing plant, while 20 % were discharged. The amount discharged is 16.7 m^3/min (yearly average). The water quality discharged to the river systems is classified according to the Swedish Environmental Protection Agency as low concentration waters for all three facilities at Malmberget and Kiruna.

Approximately 6168 m^3 water was discharged from the Malmberget facility into the river. The discharge water and the recipient water were monitored and total mass of constituents discharged are estimated on a yearly basis. The processing water constitutes approximately 2 % of the total flow in the river.

At one of the sites a comprehensive groundwater investigation was performed to evaluate contaminant transport from the waste-rock facility to a nearby lake. Four monitoring wells were installed to depths of 2.5 - 3 meters and sampled several times during the summer. The study indicated that there are only minor amounts of constituents transported from the waste-rock facility via the groundwater due to the high acid-buffering capacity of the waste-rock and the sorption capability of the aquifer.

Erzberg has direct discharge of drainage from the waste-rock dumps. After 30 years of monitoring of the surface water, no adverse effects on the surface water quality have been detected.

3.2.5.4.4 Soil contamination

At the **Kiruna** and **Malmberget** sites soil sampling is performed on a regular basis (approximately every 5 years) designed to monitor any contamination originating from atmospheric deposition. The investigation includes analysis/evaluation of ground growing moss near (at various distances and in various directions) the mine facilities. The investigations focus on metal concentrations. The result of this investigation is compared with regional investigations that are performed by the National Environmental Protection Agency.

A water balance calculation has been performed for the tailings dams system, including

- direct precipitation
- surface run-off
- process water discharge
- pump back process water
- evaporation
- discharge to the river system
- groundwater recharge and seepage through the dykes.

Based on this balance the estimated flow into the groundwater from the tailings pond/dam system is 2 m^3/min. However, there is a large uncertainty behind this number since several parameters cannot be measured but must be estimated.

Groundwater studies to evaluate the effect of the groundwater recharge from the TMF have not been performed. However, tailings/clarification pond water quality is monitored regularly, and is considered as having low concentrations. Groundwater contamination from the tailings dam system is unlikely to occur.

There has been no investigation to directly evaluate the possibility of contamination of soil from the waste-rocks facilities. The leaching from these dumps is minor except primarily for nitrate and smaller amounts of sulphate. It is considered unnecessary to investigate soil contamination from the waste-rock facility other than the airborne particle monitoring and the vegetation investigations updated every 5 years.

3.2.5.4.5 Energy consumption

One site reported a unit diesel consumption for haulage of waste-rock: 0.18 litre/tonne (average 2001).

3.2.5.5 Manganese

No data has been supplied for this section. Please provide information.

3.2.5.5.1 Management of water and reagents

3.2.5.5.2 Emissions to air

3.2.5.5.3 Emissions to water

3.2.5.5.4 Soil contamination

3.2.5.5.5 Energy consumption

3.2.5.6 Mercury

No data has been supplied for this section. Please provide information.

3.2.5.6.1 Management of water and reagents

3.2.5.6.2 Emissions to air

3.2.5.6.3 Emissions to water

3.2.5.6.4 Soil contamination

3.2.5.6.5 Energy consumption

3.2.5.7 Precious Metals (Gold, Silver)

In addition to the routine occupational safety and health surveillance, an environmental monitoring program has been established at the **Ovacik** mine. An official surveillance committee assigned by the Turkish Government carries out verification sampling. Environmental monitoring data are compiled in monthly reports and submitted to the regulatory authorities and shared with the community through various means including the national press and other public reports. Environmental sampling locations are presented in the figure below. Data collected for the periodical environmental monitoring are the following:

- dust, noise and vibration
- WAD CN in tailings water leaving the detoxification unit and at the water intake from the tailings pond
- heavy metals (As, Sb, Cd, Hg, Cu, Pb, Zn, Cr) in the tailings water
- indicator water quality, including WAD CN at the 6 groundwater monitoring wells located down-gradient of the tailings dam
- HCN measurements at various locations at the mine, including the tailings pond area.

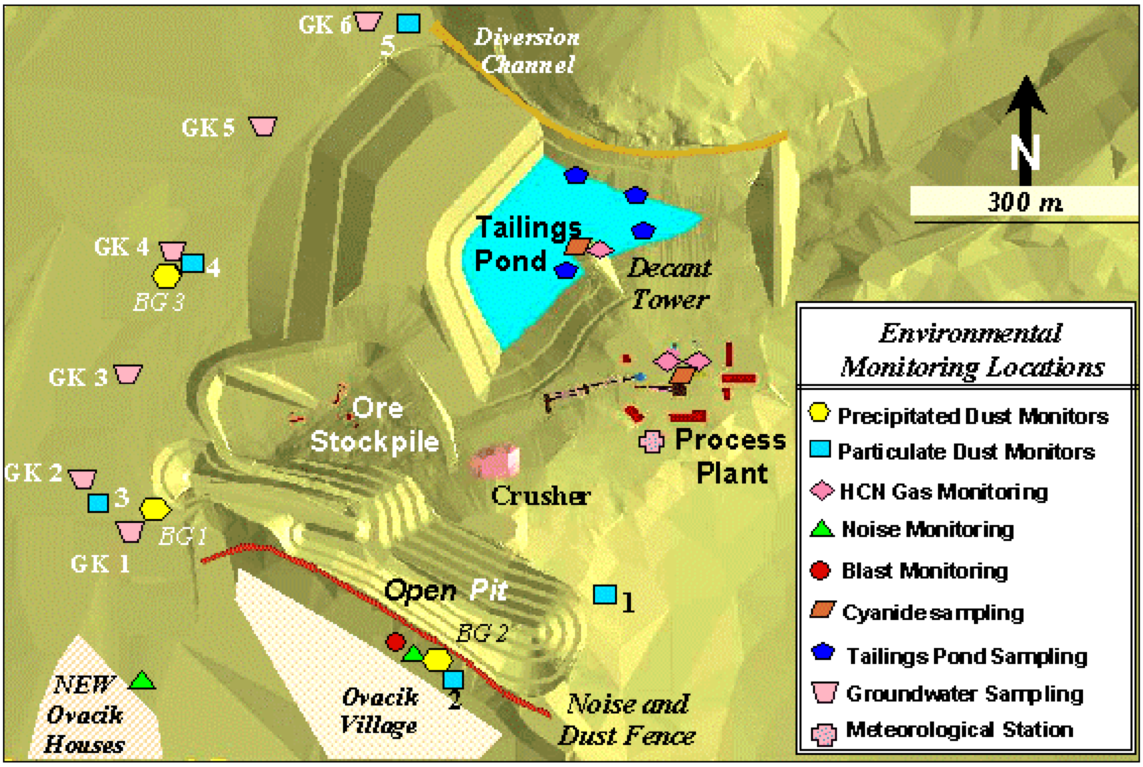

Figure 3.47: Environmental monitoring locations at Ovacik site
[50, Au group, 2002]

The control program followed at the **Boliden** mineral processing plant consists of

- surface (numerous monitoring points with varying frequency, see the figure below) and ground water monitoring (17 monitoring wells with monthly sampling)
- emissions to air (dust and gases)
- CN destruction monitoring (at various points. The discharge from the CN-destruction plant to the tailings pond is sampled 6 times per day and the discharge from the tailings pond daily)
- noise and vibration monitoring
- recipient investigations.

Environmental monitoring data are compiled in monthly reports and submitted to the regulatory authorities and shared with the community through various means including a local reference group that meets regularly at the site to discuss any issues of concern and for general information.

3.2.5.7.1 Management of water and reagents

The design criteria and management system for **Ovacik** tailings pond is set for “zero” release of water to the receiving environmental media. This is possible as the operation is a net consumer of water (due to the arid climate conditions) and re-uses all water from the tailings pond in the process. Mean annual rainfall and evaporation of the area are 728 and 2313 mm, respectively (negative water balance).

The catchment area at the point of the up-gradient dam is approximately 0.6 km^2. Maximum possible flood discharge is calculated as 24.6 m^3/s for the first hour of an extreme rainfall event. In the event of such extreme rainfall, the potential floodwaters coming from the catchment area will be stored in the runoff water pond behind the upstream embankment. The accumulated water will be pumped to the tailings pond or the excess water taken directly into the diversion channel, which is constructed along the north side of the pond.

At the **Orivesi** mine the clarified water from the tailings management area, including the rainfall water, or from the old underground mine is re-used/used in the process. The mineral processing plant is operating only with this water, without any additional water from natural surface waters. Depending on the rainfall it is sometimes (but not every year) necessary to remove excess water from the system by leading it to the river. Recycling also saves small amounts of regents but the savings are not very significant, because the flotation reagents decompose in the tailings management area. A schematic water balance is presented in the figure below.

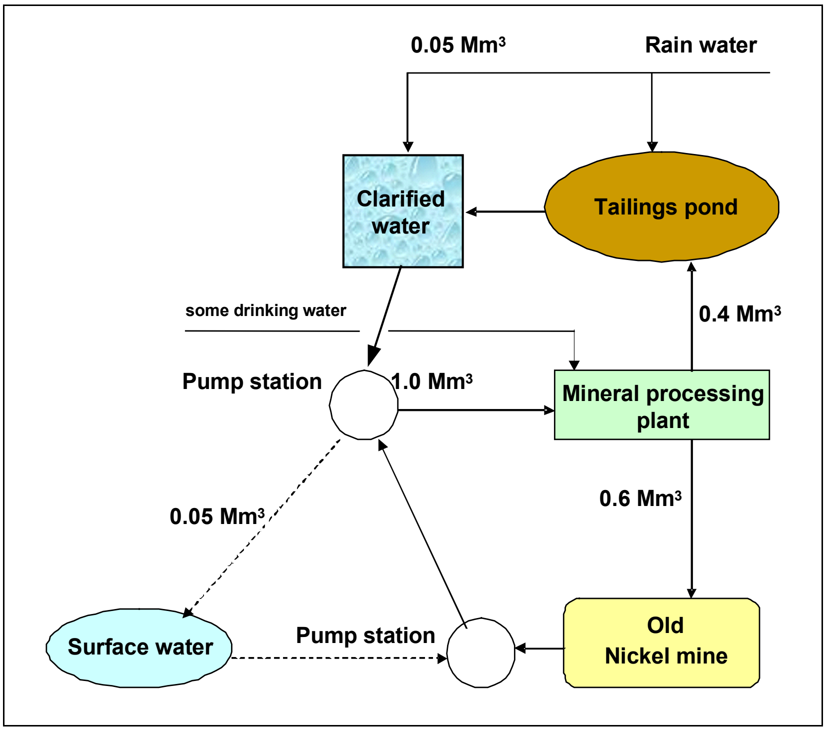

Figure 3.48: Water cycle at Orivesi site
[50, Au group, 2002]

During 2001 the (unit) consumption of reagents at the Orivesi gold mine was given in the table below.

Reagent	Consumption (g/t)
SIBX	50
DTP	50
Dowfroth	8
Flocculant	2
Steel balls	1500
Steel rods	700

Table 3.44: 2001 unit reagent consumption at Orivesi mine

The water consumption at the **Boliden** mineral processing plant is approximately 4.5 Mm^3/yr or 2.9 m^3/tonne of ore. The water is obtained from a lake 2 km north of the mineral processing plant. Some re-circulated water is used in the mill for cleaning and cycloning. Of the total amount of water used in the mineral processing plant about 10.5 % are re-used.

Due to oxidation of thiosalts and depending on the time of the year, the water contained within the pond is of low pH and contains elevated metal concentrations. The discharge from the tailings pond is therefore treated in a straight liming installation installed at the outlet of the tailings pond. A small sedimentation pond has been constructed to collect the precipitates. The pond is dredged bi-annually and the precipitates deposited within the tailings pond. The flow of the discharged water is measured daily. Discharged water volume from the tailings pond is presented in table below.

Year	1997	1998	1999	2000	2001
Flow (l/s)	254	238	186	218	352
Volume (Mm^3)	8.0	7.5	5.9	6.9	11.11

Table 3.45: Discharged water from Boliden TMF from 1997 - 2001
[50, Au group, 2002]

The following figure illustrates the seasonal variations of the water quality in the tailings pond system and the recipient (year 2001 data).

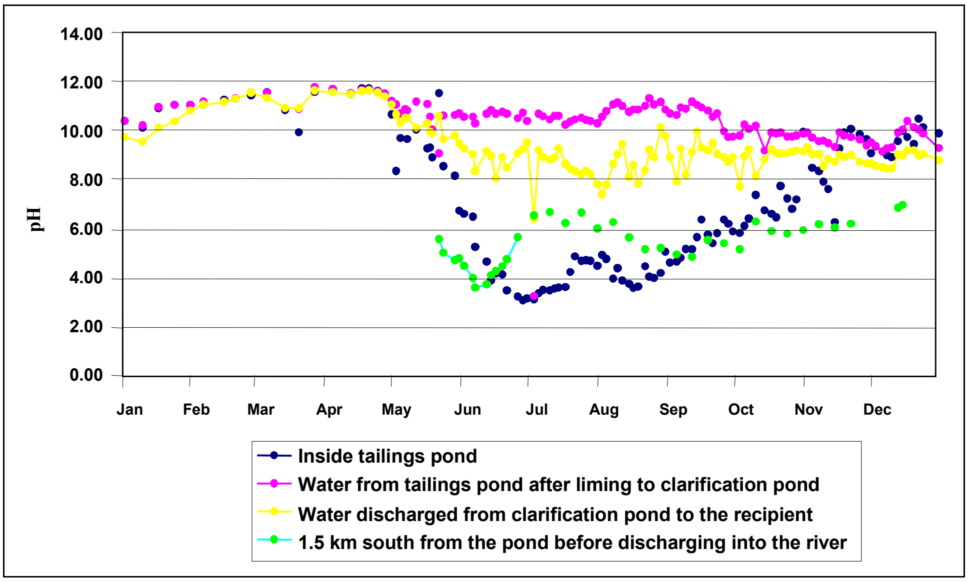

Figure 3.49: Seasonal variations of water quality in the tailings pond and the recipient at Boliden in 2001
[50, Au group, 2002]

Sampling points in the figure above are inside the tailing pond, discharge water from the pond after liming to the clarification pond, discharged water from the clarification pond to the recipient and 1.5 km south of the pond before discharging to the river. The pH in the tailings pond during winter seasons is 10- 11. During spring and summer the pH drops to about 3.5 due to the oxidation of thiosalts and the discharged water is therefore limed as described above.

During 2002 the downstream dam will be raised, the discharge system will be re-built and a new system for flow monitoring will be installed. The discharge from the tailings pond will be re-arranged from a decant tower to an overflow channel in natural ground. A back-up system for discharging water in the tailings pond is in place and will be raised.

A water balance for the Boliden mineral processing plant, the tailings pond and the surroundings is illustrated in the figure below for a year with average precipitation.

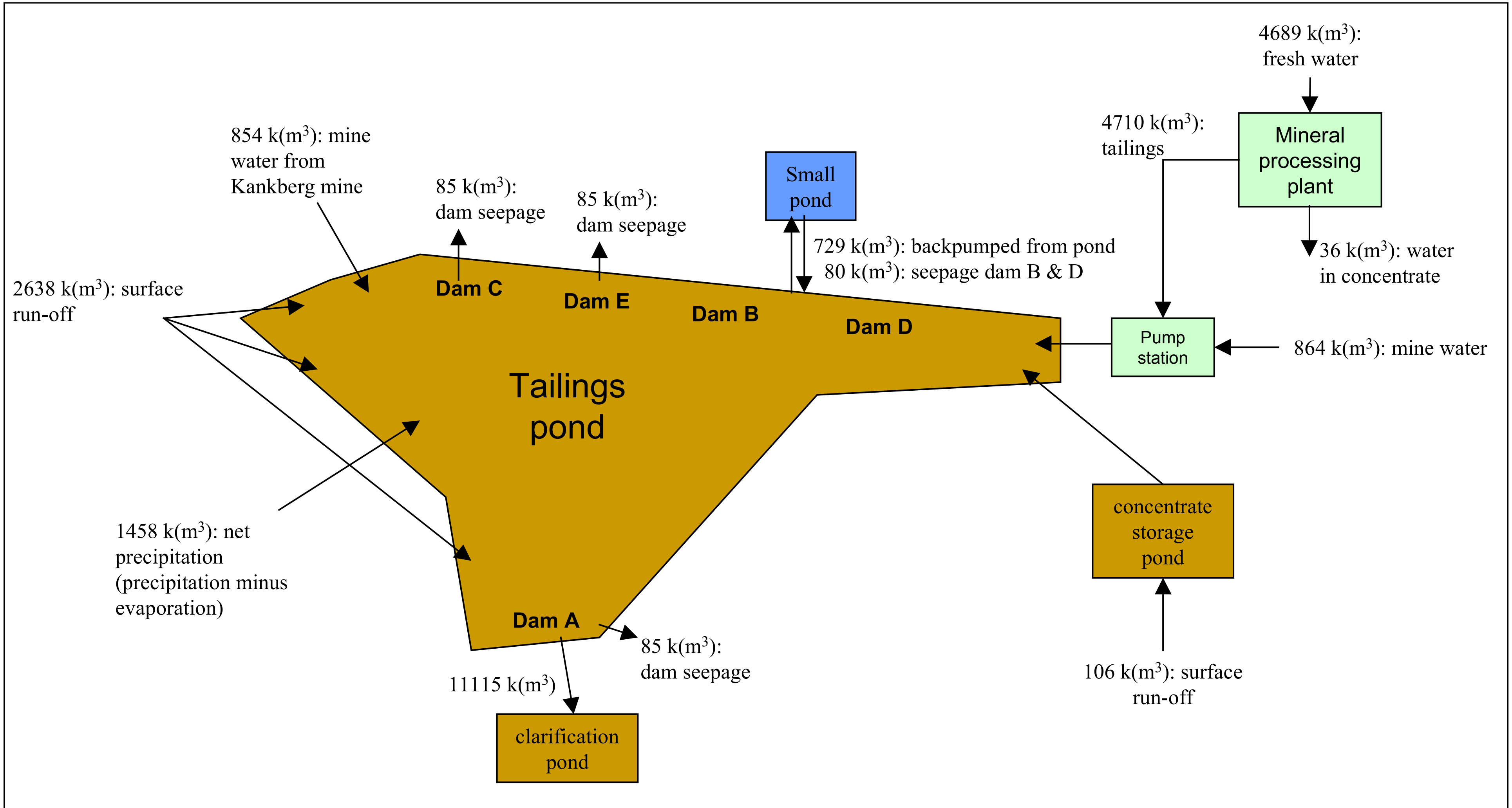

Figure 3.50: Water balance at Boliden site
[50, Au group, 2002]

Within the industry area there is an old open pit and a shaft under the mineral processing plant. Drained water is pumped to the tailings pond to be treated before discharging to the recipient. Drained water from the tailings pond is pumped back to the pond continuously. A small lake north of the tailings pond is continuously pumped in order to maintain a lower water level than the surroundings and thereby capture any possible seepage and pump it back to the tailings pond. Data such as snow depth, rain and groundwater level are collected for the water balance. The data of water in the concentrates is also used for the water balance. The system is used for monitoring the amount of water in the system at present and for coming seasons.

Fresh water consumption is monitored continuously in the process system in the mineral processing plant.

At the Boliden gold leaching plant sodium cyanide is used for collecting precious metals. Sulphur dioxide is used in the destruction of cyanide and lime for pH-regulation, before discharging to the tailings pond. During 2001 the consumption of chemicals related to the recovery of gold used at a treatment level of 0.8 Mt of was as follows:

Lime (gold and base metals)	5000 tonnes
Sulphur dioxide	1260 tonnes
Sodium cyanide	450 tonnes

The CN that is discharged into the tailings pond undergoes further natural decomposition in the pond system. This is the reason for further decreased CN concentrations in the tailings pond and, if discharge occurs, in the discharge from the tailings pond. Values from Ovacik site, where there is no discharge to the recipient, shows that the average WAD CN concentration in the discharge to the pond is 0.33 mg/l while the concentration in the pond itself is 0.19 mg/l. At the Boliden site the total CN concentration in the discharge to the tailings pond is on average 0.89 mg/l, while the discharge from the pond contains only an average 0.06 mg/l total CN.
Natural decomposition of possible trace contents of cyanide is assumed to take place in the tailings pond, following a complex scheme of processes. Examples of such reactions and processes are given in Section 2.3.2.2.4.

3.2.5.7.2 Emissions to air

At the **Orivesi** mine dust emissions are not measured, but it is recognised that some dust emission occurs from the crushing plant.

At the **Boliden** mineral processing plant the emissions to air are closely monitored. During the last years the biggest emission source to air, drying of concentrates, has been completely eliminated by the introduction of filters instead of using ovens. The gold leaching plant has a complete purification plant for all ventilation air. This air passes through a wet-scrubber where any possible HCN is absorbed in a sodium-hydroxide solution at high pH. The CN laden solution is returned to the CIL-process. The regeneration circuit for the activated carbon is equipped with a wet scrubber where lime is added for pH adjustment.

The emission from the gold leaching plant during year 2001 is summarised in the table below. Apart from the emissions reported in the table below the Boliden mineral processing plant reported emissions of 0.1 tonne particles in suspension.

		Emissions				
Date	Operating hours	Particles	CN	Hg	H_2S	SO_2
Regeneration of activated coal	h	kg	kg	kg	kg	kg
2001 – 10 - 16	30	128.550	0.270	0.000	8.700	1.275
2001 – 11 - 22	30	1.350	0.009	0.006	10.050	1.275
Wet-scrubber						
2001 – 11 - 22	1400		4.200			
2001 – 10 - 16	1400		3.080			
2001 – 07 - 03	1400		0.042			
Ovens						
2001 – 12 - 03	437.5	0.013	0.051			
2001 - 09 - 25	437.5	0.001	0.001			
Total		**129.91**	**7.65**	**0.007**	**18.75**	**2.55**

Table 3.46: Emissions from Boliden gold leaching plant

3.2.5.7.3 Emissions to water

No discharge of water has occurred from the **Ovacik** site during year 2001 therefore no direct emissions occur. Groundwater monitoring does not indicate any discharge to the groundwater.

At the **Orivesi** mine the total emissions to surface water for year 2000 are given in the table below.

Parameter	Unit	Year 2000
Tailings water discharge	m^3	780000
Ca	t	-
SO_4	t	680
COD	t	-
Solids	t	15
Cu	kg	10
Zn	kg	-
Fe	kg	-
Cd	g	-
Ni	kg	278
Cr	kg	-

Table 3.47: Emissions to water from Orivesi site

A slight increase of metal contents in groundwater (compared with the contents in the baseline study) have been observed after the nickel mine was closed and the groundwater had reached the original level. The tailings water from the current gold process has not increased the metal contents in ground water.

The emissions to surface water from the **Boliden** site are summarised in the table below for the last 4 years (1998 - 2001). The annual average concentrations are given together with total annual load of each element.

Year	Volume	Cu		Pb		Zn		As		Cd	
	Mm^3	µg/l	kg	µg/l	kg	mg/l	tonne	µg/l	kg	µg/l	kg
2001	11.1	7	72	19	191	0.1	1.07	14	156	0.1	1
2000	6.9	10	70	34	235	0.11	0.77	8	55	0.1	3.0
1999	5.9	8	51	10	59	0.2	1.04	10	58.7	0.1	0.6
1998	7.5	22	134	20	100	0.22	1.33	1	7.5	0.2	1.5

Table 3.48: Emissions to surface water from Boliden site

3.2.5.7.4 Soil contamination

No data has been supplied for this section. Please provide information.

3.2.5.7.5 Energy consumption

The energy consumption for tailings management at **Orivesi** is reported to be 1 kWh/tonne. The total energy consumption at the site per tonne ore processed is 53.5 kWh/tonne.

At **Ovacik** mine the monthly total energy consumption (based on the first 10 months of operation) 1500 MWh. Corresponding to the designed throughput of 0.3 Mt/yr this results in a total energy consumption of 60 kWh/tonne of ore processed. No figures related to the energy consumption for the tailings management are given.

At the **Boliden** mineral processing plant it is estimated that tailings management consumes about 2 kWh/tonne.

3.2.5.8 Tungsten

3.2.5.8.1 Management of water and reagents

No water is recycled from the tailings pond to the mineral processing plant.

3.2.5.8.2 Emissions to air

The average emissions of dust particulates from the tailings pond area are in the range of 50 mg/(m^2 28days).

3.2.5.8.3 Emissions to water

The following table shows the parameters measured in the effluent discharged from the tailings pond.

Parameter	Average Values 1997
Temperature, °C	13.8
pH-value	7.9
Filterable particles, mg/l	
Volume of sediment, ml/l	<0.1
Aluminium, mg/l	0.072
Iron, mg/l	0.285
Tungsten, mg/l	<0.1
Nitrite, mg/l	<0.1
Phosphorus, mg/l	<0.1
Chemical oxygen demand, mg/l	32.3
Total hydrocarbons, mg/l	<1

Table 3.49: 1997 averages of parameters measured in discharge from TMF of Mittersill site [52, Tungsten group, 2002]

Monitoring of the effluent of the tailings pond is performed twice a week by the laboratory technicians. When discharging the water into the nearby river, sampling of the water of the river upstream and downstream is performed daily. These samples are analysed in the laboratory of the mill and by a chemical laboratory. A report is sent to the federal authorities every year.

3.2.5.8.4 Soil contamination

No data has been supplied for this section. Please provide information.

3.2.5.8.5 Energy consumption

No data has been supplied for this section. Please provide information.

3.2.6 Costs

3.2.6.1 Operation

The following table lists the costs for tailings and waste-rock management.

Operation	Sub-operation	Cost interval	Unit	Site
Waste-rock management				
	Hoisting to surface	0.5 - 1	EUR/t	[1]
	Surface transport to dump	0.2 - 0.5	EUR/t x km	[1]
	Dump construction	0.1 - 0.5	EUR/t	[1]
	In open pit mining it is said that "it is difficult to extract waste-rock for less than 1 USD/tonne"			[1]
Tailings management				
	Pumping to pond	0.1	EUR/t	[1]
	Tailings distribution	0.05 - 0.3	EUR/t	[1]
	Dust suppression	>0.1	EUR/t	[1]
	Tailings dewatering	1.0 - 4.0	EUR/t	[1]
	Truck transport to mine/dump	0.5 - 1	EUR/t	[1]
	Tailings pumping and maintenance	0.1	EUR/t	[1]
	Dam raises	0.4	EUR/t	[1]
	Water treatment with lime	0.1	EUR/t	[1]
	Monitoring	0.1	EUR/t	[1]
	Total operating cost	0.8	EUR/t	Boliden[2]
	Capital cost for 7 Mm^3 pond	5.34	EUR million	Zinkgruvan[3]
	Capital cost pumps, 100 l/s	0.45	EUR million	Zinkgruvan[3]
	Tailings pumping	0.11	EUR/t	Zinkgruvan[3]
	Pumping water back to processing plant	0.04	EUR/t	Zinkgruvan[3]
	Pipe wear	0.16	EUR/t	Zinkgruvan[3]
	Piers	0.07	EUR/t	Zinkgruvan[3]
	Total operating cost	0.37	EUR/t	Zinkgruvan[3]
	Dam safety monitoring	0.05	EUR/t	Zinkgruvan[3]
	Dam raises	0.5	EUR/t	Río Narcea[4]
	CN destruction	1.0	EUR/t	Río Narcea[4]
	Others (energy, pipes, maint.)	0.5	EUR/t	Río Narcea[4]
	Total operating cost	2.0	EUR/t	Río Narcea[4]
	Total operating cost	0.6	EUR/t	Kemi[5]
	Total operating cost	0.4	EUR/t	Orivesi[6]
	Total operating cost	0.48	EUR/t	Pyhäsalmi[7]
	Total operating cost	0.3	EUR/t	Hitura[7]
	Total operating cost	0.4	EUR/t	Garpenberg[8]

Sources:
1 = [98, Eriksson, 2002]
2 = [65, Base metals group, 2002]
3 = [66, Base metals group, 2002]
4 = [58, Au group, 2002]
5 = [71, Outokumpu, 2002]
6 = [59, Au group, 2002]
7 = [62, Base metals group, 2002]
8 = [64, Base metals group, 2002]

Table 3.50: Costs for tailings and waste-rock management at metal sites

At the Boliden mineral processing plant the operational cost for deposition of tailings is EUR 0.8 /t. This figure includes the energy cost for pumping the tailings and maintenance (EUR 0.1 /t) and the actual cost to raise the dam (EUR 0.4 /t), water treatment of discharged water from the pond (EUR 0.1 /t) and monitoring costs (EUR 0.1 /t).

At Garpenberg the operational cost for the tailing deposition is EUR 0.4 /t ore processed. This cost includes pumping costs, raising of dams, maintenance of pipelines and pumps, monitoring etc. However, it does not include decommissioning costs.

At Zinkgruvan up to the beginning of 1990's the tailings were managed above the water surface which was less expensive as the pipes could be stationary at one fixed point for a long time. Since the start of discharging mainly under the water surface the costs per unit have been more than double. On the other hand the management under water has given a significant reduction of the metal transport from the pond and less dusting from the tailings area.

The operating costs can be divided into the following items (EUR/m^3):

Pumping of tailings	0.15
Water recycle	0.05
Pipe arrangements, wear	0.22
Piers	0.10

The dam safety monitoring system now underway will add another EUR 0.07 /m^3 and may be complemented with other systems as well.
[66, Base metals group, 2002]

The following table shows some further cost information relevant to the management of tailings and waste-rock.

Operation	Sub-operation	Cost	Unit	Comment/Site
Dam costs	Dam construction	0.05 - 0.5	EUR/t	Scale. site & method dependent[1]
	Dam monitoring	*Please provide*	EUR/yr	[1]
	Dam decommissioning	*Please provide*	EUR/km of dam	[1]
Lining	HDPE liner, 16 ha	7.5	EUR/m^2	Ovacik[2]
Environmental monitoring	One water sample (surface or GW)	220	EUR/sample	Sampling, sample preparation, shipping, analysis and reporting[1]
	Air - sedimentad particles	*Please provide*		
	Air - particles in suspension	*Please provide*		
Installation of monitoring well	Ground water monitoring well	200	EUR/m	Establishment, drilling, lining and rinsing[1]
Backfill	Transport cost, 15 km	0.3	EUR/t	[1]
	Transport cost, 100 km	0.8	EUR/t	[1]
Thickened tailings	Operating costs excluding capital costs	0.15	EUR/t	[3]
	Capital cost thickener, (14 m high)	170000	EUR	[3]
	Total capital cost	2.2	EUR million	[3]
	Of which for dam construction	1.4	EUR million	[3]
Sources: 1 = [98, Eriksson, 2002] 2 = [56, Au group, 2002] 3 = [31, Ritcey,]				

Table 3.51: Cost of other operations relevant to the management of tailings and waste-rock

The following table gives more detailed information on costs for destroying cyanide using the SO_2/air method.

Site	Tonnes/day	Weight % solids	CN_{WAD} (mg/l)		Operating cost	
			Feed	Treated	USD/tonne	USD/Kg CN_{WAD}
A	2800	35	80	0.30	0.35	2.56
B	920	47	175	0.90	0.77	4.28
C	800	45	120	0.50	0.91	6.06
D	2700	40	290	0.15	0.95	2.40

Table 3.52: Operating cost in USD for CN destruction using the SO_2/air method in 2001
[99, Devuyst, 2002]

The operating costs are actual and include the costs of SO_2, lime, copper sulphate and power. Capital costs for these operations are in the range of USD 360000 to 1.1 million installed. Capital costs include reactor, agitator, air compressor, SO_2 delivery system, and copper sulphate delivery system. It does not include the tailings pump box and pump and the lime system (usually already part of the plant). It assumes the system is outdoors, including reagent systems and air compressor. Therefore no additional building facilities need to be constructed, only site preparation and proper foundations. None of the examples in the table make use of a sulphur burner for the source of SO_2. If this was the case, the capital cost would be much higher(about 80 %), but the operating cost would be reduced by about 60 %. The variation in operating costs is due to unit reagent cost for SO_2, lime, copper sulphate and power.
[99, Devuyst, 2002]

3.2.6.2 Closure

The following table lists cost information related to closure cost.

Sub-operation	Cost interval	Unit	Comment/Site
Dump or tailings pond revegetation	0.1 - 0.5	EUR/m^2	Scale dependent[1]
Engineered cover on dump or pond	3.0 - 10	EUR/m^3	Scale and method dependent[1]
Flooding of tailings pond	0.5 - 1	EUR/m^2	Scale and site dependent[1]
Wetland establishment	0.1 - 1	EUR/m^2	Scale and site dependent[1]
Groundwater saturation	0.2 - 2	EUR/m^2	Scale and site dependent[1]
Dewatering of pond	0.7 - 1.2	EUR/m^2	Tara[2]
Revegetation	0.7 - 0.8	EUR/m^2	Tara[2]
Monitoring	1.3 - 1.7	EUR/m^2	Tara[2]
Maintenance	0.1	EUR/m^2	Tara[2]
total reclamation and closure	3.1 - 3.7	EUR/m^2	Tara[2]
Closure (dewatering and cover)	1.8	USD million	Ovacik[3]
Closure (not specified), 37 ha	0.6	EUR million	Orivesi[4]
Closure (water cover, vegetation), 280 ha	1.5	EUR million	Boliden[5]
Closure and aftercare, 100 ha	5.4	EUR million	Pyhäsalmi[6]
Rehabilitation	14.4	EUR/m^2	Zinkgruvan[7]
Apirsa actual costs			
Apirsa tailings pond reclamation	18.5	EUR/m^2	Total cost/total area[1]
clay cover placed	2.9	EUR/m^3	Material it self not included[1]
Protection cover placed	3.1	EUR/m^3	Material it self not included[1]
Resloping of dam	0.9	EUR/m^3	<100 m movement of material (bulldozer)[1]
Resloping of dam	4	EUR/m^3	>100m movement of material (loading transport and placement)[1]
Revegetation with grass	0.05	EUR/m^2	Conventional seeding[1]
Saxberget actual reclamation cost			
Composite unit cost (1995)	7	EUR/m^2	Total cost/total area[1]
Water cover unit cost (1992)	1.5	EUR/m^2	Total cost/total area[1]
Kristineberg actual reclamation costs			
Unit cost water cover	1.5	EUR/m^2	Total cost/total area[1]
Unit cost composite dry cover	6	EUR/m^2	Total cost/total area[1]
Unit cost increased ground water level	4	EUR/m^2	Total cost/total area[1]
Sources: 1 = [98, Eriksson, 2002] 2 = [23, Tara, 1999] 3 = [56, Au group, 2002] 4 = [59, Au group, 2002] 5 = [50, Au group, 2002] 6 = [62, Base metals group, 2002] 7 = [66, Base metals group, 2002]			

Table 3.53: Cost information for closure and aftercare of metalliferous mining tailings and waste-rock management

Reclamation and closure costs estimated for the Tara tailings facility are calculated for a 5-yr. active monitoring phase, a 5-yr. passive monitoring phase and a 10 yr. long-term monitoring phase. Revegetation costs were calculated for a surface are of 66.8 – 85.4 ha with a unit cost of approx. EUR 3200 /ha including fertiliser and seed. The costs for monitoring are based on the assumption that one full time staff be employed for a five year so called active care period monitoring phase. Other cost factors included are reclamation performance, agronomic performance assessment (examination of grazing cheep), wildlife monitoring, surface water quality, groundwater quality, dust monitoring, geotechnical monitoring (piezometers and visual inspections).

The decommissioning cost for Boliden the tailings pond are estimated to be EUR 1.5 million. This includes the arrangements for securing a permanent water cover, stabilisation of shallow bottoms, reconstruction of discharge devices and re-vegetation costs. At the last raise the dams are built to their final long term stable slope angle and required erosion protection is installed, costs that are not included in the decommissioning costs are given above [50, Au group, 2002].

At Pyhäsalmi EUR 3.6 million and at Hitura EUR 0.6 million have been reserved in the accounts for closure and after care. The total closure and aftercare costs for Pyhäsalmi tailings area is estimated to EUR 5.4 million.

Río Narcea has posed a bond of approximately EUR 3 million which corresponds to the Spanish norm (PTS 2 M/ha).

3.3 Industrial minerals: Mineral processing, tailings and waste-rock management

The term "industrial minerals" covers a wide range of different materials. Their common denominator is that they are all used as functional fillers or as production aids by industry. They are generally milled into a very fine powder before use. The main categories included in this family are talc, calcium carbonate (ground and precipitated), feldspar, kaolin, ball clays, perlite, bentonite, sepiolite, silica, borates, etc. The mineralogical and chemical characteristics, as well as the particle-size distribution of the final product, determine the possible end-uses. Quality requirements are usually very precise. The end-uses of these minerals are extremely diversified. The geological availability of industrial minerals depends on the categories considered: talc, for instance, is less common than silica sand. However, even for the categories which seem more common, the physico-chemical requirements are so high and precise that only a limited number of ore bodies can be worked.
[48, Bennett, 2002]

3.3.1 Mineralogy and mining techniques

3.3.1.1 Barytes

Barytes is the naturally-occurring mineral form of barium sulphate ($BaSO_4$).

Barytes deposits worldwide occur in ore bodies as residual, vein-type and bedded formats. Extraction is by both surface and underground techniques dependent on the geology and economics of the region. Each deposit and the most suitable extraction and processing route are very site-specific. Mine overburden and waste-rock would generally remain in situ, or be sold as a construction product or be used in general reclaim/restoration.

Out of the 8 major mines (without the Portuguese operation) within the EU15 55 % of the Barytes is produced by underground mining.

3.3.1.2 Borates

Boron minerals coming from open pit or underground mines are crushed into appropriate sizes and later fed to the mineral processing plant [92, EBA, 2002].

3.3.1.3 Feldspar

The mineralogical composition of most feldspars can be expressed in terms of the ternary system orthoclase ($KAlSi_3O_8$), albite ($NaAlSi_3O_8$) and anorthite ($CaAl_2Si_2O_8$). The minerals, the

composition of which is comprised between albite and anorthite are known as plagioclase feldspars, while those comprised between albite and orthoclase are called alkali feldspars. This latter category is of particular interest in terms of industrial use of feldspars.
[39, IMA, 2002]

3.3.1.4 Fluorspar

The mineralogy of the flourspar/lead sulphide operation can be described as follows:

- fluorspar with a grade of 26 – 38 %
- lead sulphide with a grade of 1.5 – 8 %
- barium sulphate
- zinc sulphide
- iron sulphide as pyrites and marcasite
- calcium carbonate as calcite
- quartz
- silicates.

Of the above only the two first are of economic interest; the liberation size of 6 mm makes the comminution and separation relatively simple, to pre concentrate the mineral in a static Sink/Float process [44, Italy, 2002].

Mining is done both underground and open pit.

In one operation the underground mining method is, applied in a vein, cut and fill mining [44, Italy, 2002].

3.3.1.5 Kaolin

Kaolinite can be formed as a residual weathering product, by hydrothermal alteration, and as a sedimentary mineral. The residual and hydrothermal occurrences are classed as primary and the sedimentary occurrences as secondary.

Primary kaolins are those that have formed in situ, usually by the alteration of crystalline rocks like granite or gneiss. The alteration results from surface weathering, groundwater movement below the surface, or action of hydrothermal fluids. Secondary kaolins are sedimentary minerals which were eroded, transported and deposited as beds or lenses associated with other sedimentary rocks. Most of the secondary deposits were formed by the deposition of kaolinite which had been constituted elsewhere. A type of kaolin deposits which can be considered as either primary or secondary, depending on the point of view, are arkosic sediments which were altered after deposition, primarily by groundwater

3.3.1.6 Limestone

No data has been supplied for this section. Please provide information.

3.3.1.7 Phosphate

No data has been supplied for this section. Please provide information.

3.3.1.8 Strontianite

No data has been supplied for this section. Please provide information.

3.3.1.9 Talc

No data has been supplied for this section. Please provide information.

3.3.2 Mineral processing

3.3.2.1 Barytes

There are no standard flowsheets for the industry due primarily to the wide range of products. Mineral processing varies from a simple crush-only aggregate-type operation through to heavy-media processing, jigging, fine grinding and flotation. At some operations small quantities of finished product are subsequently and separately acid-washed for special sale applications [29, Barytes, 2002]. Optical separation is also used in at least one operation.

The prime requirement for oil-well applications and for several of the filler applications (e.g. sound deadening, nuclear shielding) is specific gravity (SG) and often a 80 – 90 % $BaSO_4$ content is sufficient to meet this. These operations generally only require crushing the run-of-mine material to produce a finished product with no processing waste.

Several other operations only require simple gravity methods to enhance the quality for the finished product, generally jigging or heavy-media separation.

Mineral processing may be necessary:

- for more complex ore bodies
- where the barytes is associated with other minerals (e.g. fluorspar, iron ore)
- where the barytes is finely disseminated in the host-rock (flotation)
- for the chemical industry where grades greater than 97 % $BaSO_4$ are required.

The following flowsheet shows a site using gravity separation using jigs and flotation.

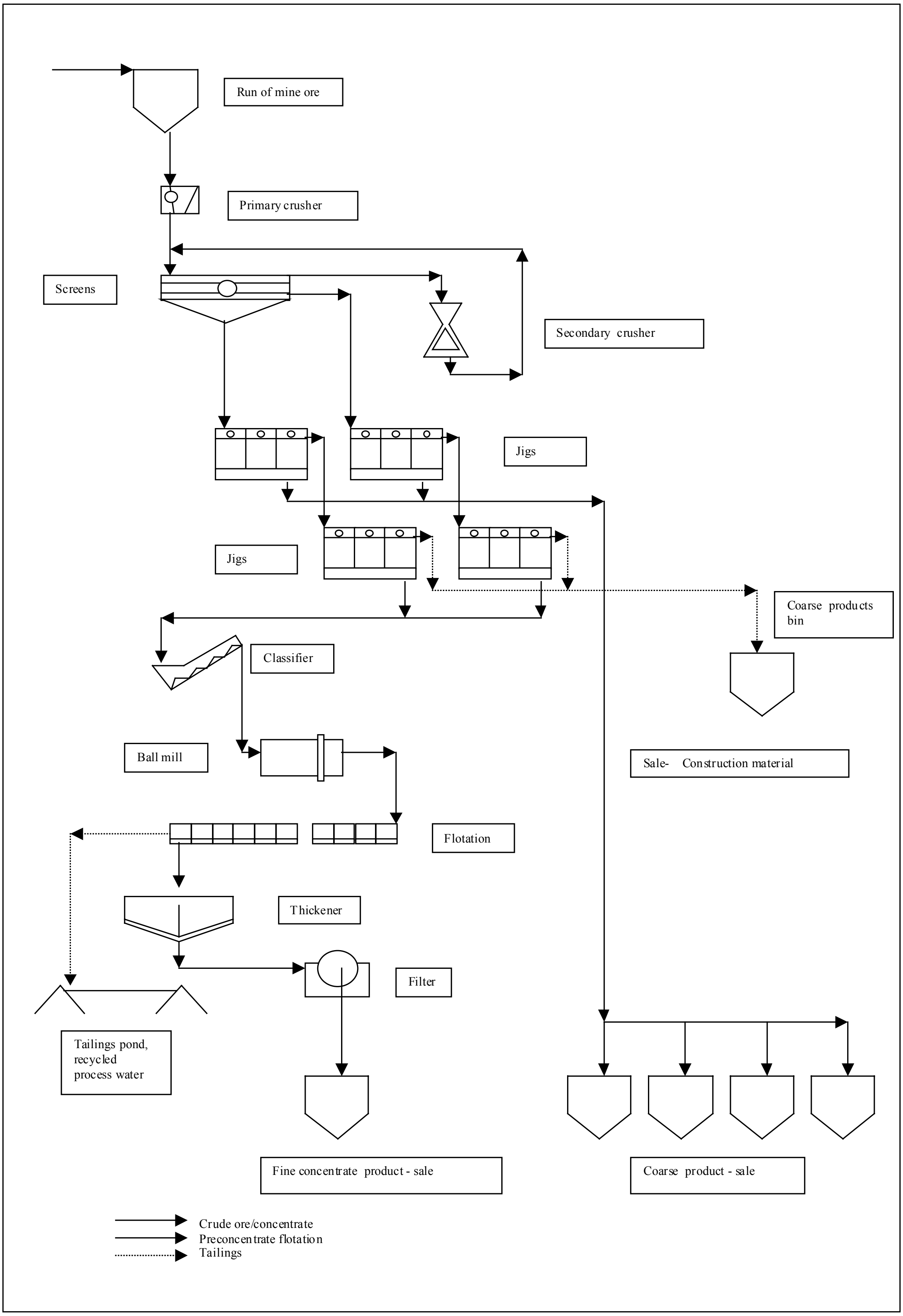

Figure 3.51: Flowsheet of barytes mineral processing plant using jigs and flotation [29, Barytes, 2002]

Sites with flotation operations use standard reagents for processing – alkyl sulphonates as collectors and all or some of sodium silicate, quebracho (tannin) and citric acid as pulp modifiers [29, Barytes, 2002].

3.3.2.2 Borates

No data has been supplied for this section. Please provide information.

3.3.2.2.1 Comminution

3.3.2.2.2 Separation

3.3.2.2.3 Others

3.3.2.3 Feldspar

3.3.2.3.1 Comminution

No data has been supplied for this section. Please provide information.

3.3.2.3.2 Separation

The essential use of the flotation process may be explained by the following figure:

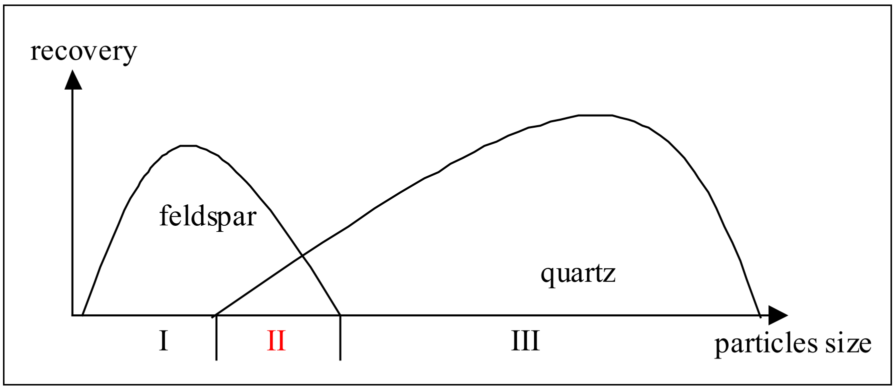

Figure 3.52: Feldspar particle vs. recovery graph
[39, IMA, 2002]

In Sections I and III a primary mechanical separation (hydrocycloning, centrifugation) can be achieved. In Section II, either optical, flotation or electrostatic separation can be used to separate feldspar from quartz, depending on both the intrinsic characteristics of the raw material, and the final product requirements.

The following flowsheet shows the steps involved in the recovery of feldspar.

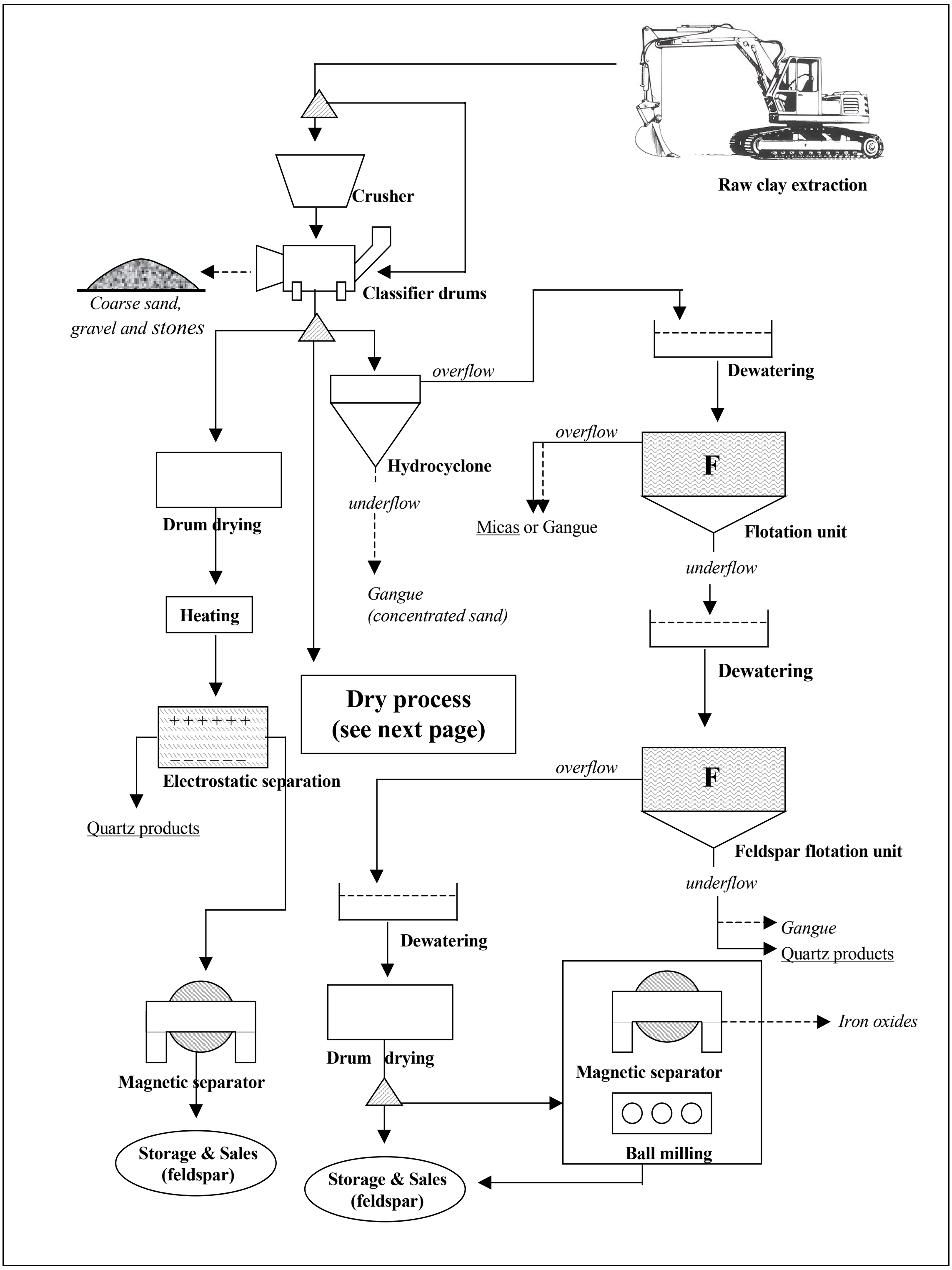

Figure 3.53: Flowsheet for Feldspar recovery using flotation
[39, IMA, 2002]

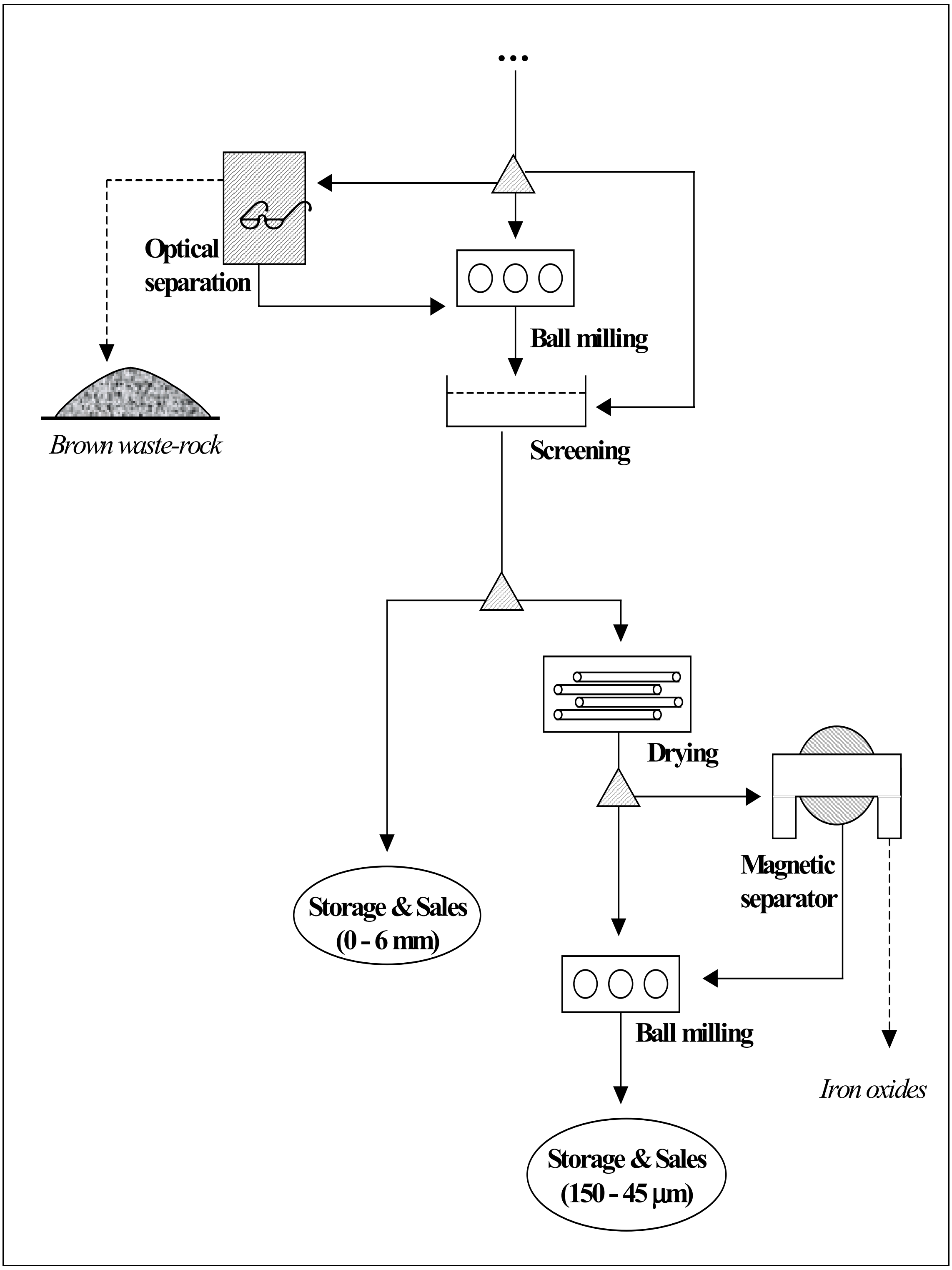

Figure 3.54: Dry processing step in the recovery of feldspar
[39, IMA, 2002]

The following table shows the inputs and outputs from the main steps of the feldspar process.

Process's step	INPUTS	OUTPUTS
1. Milling & classifying	▪ raw material ▪ water	▪ slurry mixture (containing feldspar) ▪ coarse sand, gravel, and stones
2. Hydrocycloning	▪ slurry mixture ▪ water	2.1 Overflow ▪ feldspar, fine sand and micas 2.2 Underflow ▪ gangue: concentrated sand ▪ process water
3. Dewatering by screens or vacuum filters	▪ feldspar, fine sand and micas	▪ feldspar, fine sand and micas ▪ process water
4. Micas or oxides flotation	▪ feldspar, fine sand and micas ▪ foam inhibitor ▪ acids (H_2SO_4) ▪ surfactants	2.1 Overflow ▪ micas or oxides 2.2 Underflow ▪ feldspar, fine sand, quartz ▪ process water
5. Dewatering by screens or vacuum filters	▪ output from the underflow of the previous step	▪ feldspar, fine sand, quartz ▪ process water
6. Feldspar flotation	▪ feldspar, fine sand, quartz ▪ foam inhibitor ▪ acids (HF) ▪ surfactants	2.1 Overflow (reverse flotation possible) ▪ feldspar 2.2 Underflow ▪ fine sand and quartz ▪ process water
7. Dewatering by filters	▪ output from the overflow of the previous step ▪ feldspar (moisture < 25 %)	▪ feldspar (moisture < 25 %) ▪ process water
8. Drying	▪ feldspar (moisture < 25 %)	▪ feldspar (moisture < 1 %)
9. Magnetic separation	▪ feldspar (moisture < 1 %)	▪ feldspar (moisture < 1 %) ▪ iron oxides

Table 3.54: Inputs and outputs from feldspar mineral processing steps
[39, IMA, 2002]

In the feldspar process, one may distinguish three different flotation steps, namely the micas flotation, the oxides flotation, and the feldspar flotation. Although each of them is based on the same physico-chemical principle, they differ by their experimental parameters (acids, pH, surfactants), and by the goal pursued.

3.3.2.3.3 Others

3.3.2.4 Fluorspar

3.3.2.4.1 Flotation

After crushing and grinding the ore into particles under 1 mm and dispersion into water in large cells, the fluorite grains are rendered hydrophobic by surface action of natural fatty acids (oleic acid for example). The "fatty" particles mix with the injected air bubbles to form a froth that is then mechanically skimmed off at the surface of the cells.

This froth, containing mainly calcium fluoride, is washed several times with water. Filtration of the slurry gives a filter-cake with around 10 % moisture and over 97 % of CaF_2.

The flotation process achieves an upgrade from 40 % to 98 % CaF_2.
[43, Sogerem, 2002]

3.3.2.4.2 Gravimetric separation

After crushing the ore into pieces or lumps of around 30 mm, the different components of the ore are separated by a sink-float process using a heavy medium (liquid of density above or under the density of fluorite). This process is not perfect (most grains are not 100 % pure CaF_2 or not pure non-CaF_2 mineral); it is a compromise between size and purity. It capable of upgrading the ore from 30 – 60 % CaF_2 to around 90 % CaF_2.

The gravimetric ("sink-float") concentration, a continuous process, is done in water environment at ambient temperature in closed system (cyclones or drums) with automated regulation. The water is recirculated in a closed circuit. The washed material is sorted by size (2 mm, 5 mm, 25 mm) and stored outside on concrete surface.

All tailings are recycled through the flotation plant described above to increase recovery. The finished product can be sold in wet form and the delivery to the customers is done in covered dump trucks. If it is delivered dried, transportation is done in covered dump-trucks or in silo-trucks.
[43, Sogerem, 2002]

3.3.2.4.3 The fluorspar/lead sulphide process

The process at the flourspar/lead sulphide mine is as follows:

The ore is pre-concentrated at the mine site using a sink/float process. The pre-concentrate with a fluorspar grade of 43 – 50 % is transported via trucks to the mineral processing plant 57 km away from the mine, the reason of this being the availability of large amounts of water, not available at the mine.

The mineral is ground in ball mills to 100 % passing 0.5 mm. The first mineral recovered is the lead sulphide in a 3-stage flotation unit. The reject of this stage is then processed in a 4-stage fluorspar flotation unit. The commercial products are filtered in drum filters.
[44, Italy, 2002]

3.3.2.5 Kaolin

The processing of kaolin varies greatly from company to company; each kaolin producer uses different equipment and methods. Also, companies that use identical methods may use them at different stages of the processing.

Kaolin "ore", generally composed of kaolinite, quartz, micas, feldspar residues, etc., is commonly wet processed to eliminate the unwanted minerals. The various steps in the processing are:

- placing the "ore" in suspension with water
- recovery of the kaolin fraction through sieving and cycloning
- concentration of the suspension through decantation in basins followed by passage through filter-presses.

The kaolin properties (brightness, rheology, purity, grain size distribution) can be improved during the treatment, by using magnetic separation, bleaching or centrifugation.

3.3.2.5.1 Comminution

No data has been supplied for this section. Please provide information.

3.3.2.5.2 Separation

Coarse clay may be used as a low grade filler or a ceramic clay. Alternatively, it can be upgraded by further processing. The flotation process is used to refine coarse clay and to maximise the recovery of kaolin. It can increase the kaolin recovery yield up to 15 %, which is a significant improvement in the management of this natural resource. Not all producers use flotation, which depends on the product requirements and the characteristics of the deposit.

The essential use of the flotation process can be explained by the following figure:

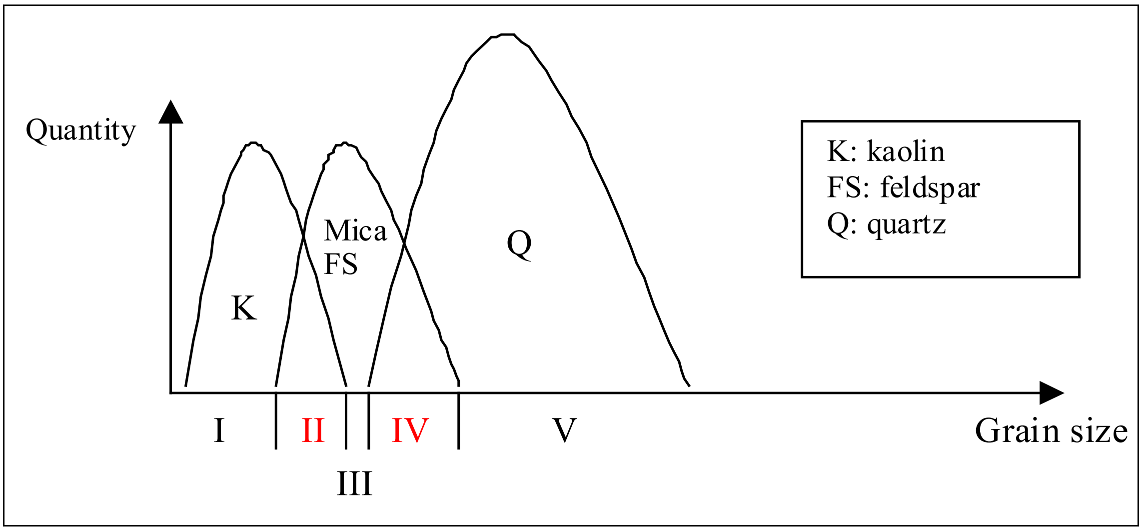

Figure 3.55: Kaolin grain size vs. quantity graph

In Sections I, III, and V, a primary mechanical separation (cycloning, centrifugation) can be achieved.

In Sections II and IV the grain size of different minerals is equal. If there is only a minor difference in specific weight, mechanical separation is not possible. Other differences have to be used. At smaller grain sizes (Section II) the only possible way is by flotation. At larger grain sizes, Section IV, other methods, like electrostatic separation of feldspar, is possible.

The following table shows the inputs and outputs from the main steps of kaolin processing.

Process step	INPUTS	OUTPUTS
1. Classifying	Raw material Water	Coarse sand, gravel and stones Slurry mixture (containing kaolin)
2. Hydrocycloning	Slurry mixture Water	2.1 Overflow Kaolin + fine sand, micas, (and feldspar) 2.2 Underflow Kaolin + fine sand, micas, (and feldspar) Process water
3. Flotation	Underflow from the hydrocycloning step, or kaolin concentrate Acid (H_2SO_4, H_3PO_4) Surfactants Anti-foam chemicals Alkaline solution (NaOH)	2.1 Overflow Kaolin mixture (after acid neutralisation) 2.2 Underflow Very fine sand, micas, (and feldspar) Process water
4. Thickening	Overflow from the hydrocycloning step or flotation Flocculent	Kaolin concentrate (15 – 30 % solid content)
5. Product separation and benefication	Kaolin concentrate, or kaolin mixture 5.1 Magnetic separation 5.2 Bleaching Sodium hydrosulphite Ozone gas 5.3 Centrifugation	Kaolin Iron oxides (very small amount) Very fine sand and micas
6. Filtering	Kaolin, kaolin concentrate	Kaolin (moisture < 18 %) Process water
7. Drying	Kaolin (moisture < 18 %)	Kaolin products

Table 3.55: Inputs and outputs in the processing of Kaolin

3.3.2.5.3 Others

3.3.2.6 Limestone

The vast majority of the mine production is marketable, as can be seen in the following tables.

Quarry

Ore (natural calcium carbonate)	16655000 t	100.0 %

Material in plant

Stock for sale	16100000 t	96.7 %
Tailings released to the outside	75000 t	0.4 %
Dust stored on-site	111000 t	0.7 %
Tailings stored on-site for recultivation of the quarries	369000 t	2.2 %

Tailings released to the outside:
These tailings include the flotation residues with the mica (such as phlogobit, biotit, muscovit) and graphite impurities. They are sometimes settled in ponds or direcly released to the recipient.

Dust stored on-site:
This dust includes all the tailings coming from the various dust collectors and cleaning systems in the plant bagging stations, etc.

Tailings stored on-site for the recultivation of the quarries:
This kind of material consists mainly of off-colour production or ground fillers and pigments outside of the product specification.

The production of ground calcium carbonate (GCC) starts with its extraction. Identifying the right orebody in terms of composition, homogeneity, etc. is essential to the whole production process that will follow; a pure calcium carbonate source needs to be identified. Generally, the processing includes washing, sorting of undesirable by-minerals, grinding, size classification of particles and possibly drying. Depending on the circumstances and intended uses, the order and necessity of those different steps vary. At the outlet of the process, the material is delivered in bags or in bulk (trains, boats, trucks) when dry, or as bulk container from slurries. GCC results directly from the exploitation of pure calcium carbonate ore bodies (ore grade > 96 %). The production process maintains the calcium carbonate very close to its original state, resulting in a finely ground product delivered either in dry or slurry form. Blasted raw marble is pre-crushed, and depending on the geology washed and sometimes screened. The fines are normally sold for different applications, such as road making, cement mills etc.

In the dry process, calcium carbonate is ground in ball mills, classified and stored in silos, or bags, before shipped by railway wagons or trucks. The products are mainly used in paint and plastics industries, minor applications are in the chemical industry, for fertilising and desulphurisation. Fillers and pigments for the paper industry are produced as slurries, which are finely dispersed calcium carbonate in water. Crushed material is ground with water in rod mills, or ball mills in open or closed circuit, classified and stored in silos before loaded onto railway wagons or trucks.

Due to the geology and mineralogy some calcium carbonate deposits contain unwanted minerals such as graphite, mica, or schist. To remove these natural impurities, selective mining and optical separation are developed together with other mineral processing steps in order to meet the requirements of the customers. Such mineral processing systems can be flotation or magnetic separation.

When magnetic minerals are bound to the marble, magnetic separation is a successful method to separate those "impurities".

Gangue minerals like mica (such as phlogobite, biotite, muscovite) lead to abrasion in the paper producing machines, while graphite leads to a grey colour in the pigments. Therefore, product requirements impose to separate these minerals during the production process of the aqueous dispersion by means of flotation. The thickened concentrate is normally dewatered in filter presses and sold to cement mills or deposited to a waste dump

3.3.2.7 Phosphate

No data has been supplied for this section. Please provide information.

3.3.2.7.1 Comminution

3.3.2.7.2 Separation

3.3.2.7.3 Others

3.3.2.8 Strontianite

No data has been supplied for this section. Please provide information.

3.3.2.8.1 Comminution

3.3.2.8.2 Separation

3.3.2.8.3 Others

3.3.2.9 Talc

No data has been supplied for this section. Please provide information.

3.3.2.9.1 Comminution

3.3.2.9.2 Separation

3.3.2.9.3 Others

3.3.3 Tailings management

3.3.3.1 Barytes

No data has been supplied for this section. Please provide information.

3.3.3.1.1 Characteristics of tailings

3.3.3.1.2 Applied management methods

The following table shows the tailings management methods that are applied to different mineral processing schemes.

Type of mineral processing	No of sites	% total output	Tailings management
Crushing Only	2	15 %	Nil
Crushing + Jigs only	4	23 %	Nil
Crush + Grind + Flotation	2	22 %	Dry tailings
Crush + Grind + Flotation	5	40 %	Wet tailings

Table 3.56: Tailings management methods applied to Barytes mines in EU15 and Candidate Countries
[29, Barytes, 2002]

It can be seen that 5 sites, which together produce 40 % of the Barytes, use wet tailings management. Two out of these five sites together discard only 12500 tonnes of tailings into small ponds and nearly half of this tonnage is regularly dredged out as a product for land use.

3.3.3.1.3 Safety of tailings facility and accident prevention

3.3.3.2 Borates

No data has been supplied for this section. Please provide information.

3.3.3.2.1 Characteristics of tailings

3.3.3.2.2 Applied management methods

3.3.3.2.3 Safety of tailings facility and accident prevention

3.3.3.3 Feldspar

3.3.3.3.1 Characteristics of tailings

The following table shows the characteristics of the materials released from the process.

Process step	Material released from the process	Destination
1. Comminution & classifying	▪ coarse sand, gravel, and stones	▪ by-product or tailings heap
2. Hydrocycloning	▪ concentrated sand ▪ process water	▪ by-product or tailings pond
4. Dewatering by screens or vacuum filters	▪ clear water overflow is directly recycled or used to hold reserves of water.	
4.1 Micas flotation	▪ micas ▪ process water	▪ by-product or tailings pond
4.2 Oxides flotation	▪ oxides ▪ process water	▪ tailings pond
5. Dewatering by screening or with vacuum filters	▪ clear water overflow is directly recycled or used to hold reserves of water.	
6. Feldspar flotation	▪ fine sand, quartz, and micas ▪ process water	▪ by-product or tailings pond
7. Dewatering in filters	▪ clear water overflow is directly recycled or used to hold reserves of water ▪ process water, tailings pond	
8. Drying	▪ n/a	▪ n/a
9. Magnetic separation	▪ iron oxides	▪ by-product or tailings heap

Table 3.57: Products and tailings from the mineral processing of feldspar
[39, IMA, 2002]

Besides the tailings heaps consisting of coarse sand, gravel and stones, there are tailings ponds which contain:

Solid materials:

- fine sand and micas (more than …%)
- some iron oxides (less than …%)
- flocculents (in the ppm range)
- *to be completed…*

Liquid (process water)

- water at a pH value of about 4.5
- foam inhibitor.

Please provide more detailed information.

3.3.3.3.2 Applied management methods

No data has been supplied for this section. Please provide information.

3.3.3.3.3 Safety of tailings facility and accident prevention

No data has been supplied for this section. Please provide information.

3.3.3.4 Fluorspar

3.3.3.4.1 Characteristics of tailings

No data has been supplied for this section. Please provide information.

3.3.3.4.2 Applied management methods

In one operation the tailings, containing 1 to 5 % CaF_2, are backfilled into the mine after dewatering with filter-presses, located inside the plant itself. The water is entirely recycled. The coarseness of the tailings is close to the one of the finished concentrated fluorspar, that is less than 350 µm.

The constituents are silica and shale (80 - 90 % SiO_2), and on a smaller scale iron derivates (5 - 10 % Fe_2O_3: shales, iron hydroxides, iron carbonate), other oxides (1 – 2 % Al_2O_3), iron/copper sulphides, and of course left CaF_2 (1 – 5 %).
[43, Sogerem, 2002]

In another case the tailings are cycloned in a dense medium to separate the sands from the muds; the sands are settled in “sand ponds”. The muds are pumped into “settlements ponds”.

The process water is cleaned in a three ponds. The clean water from the third pond is partially recycled and partially discarded into the river. The total volume of the tailings ponds is about 1300000 m^3.

The dried sands are stocked in heaps and are sold for civil construction works; the muds are under evaluation for new uses such as for tiles, cement.

The further development is to eliminate the settlement pond introducing filter press sections.

The tailings facilities are located near the plant very close to the river. The ground where the facilities are located is an alternation of sands and clay layers, so that no seepage occurs.

A conventional dam with a clay nucleus of the classical trapezoid shape contains the tailings. The dam slope is 1:1.5. The dams are raised every 3 –4 years.

A characterisation of the site is in progress in order to evaluate the chemical situation, the leaching behaviour, and so on. The alternative solution to the present management will take after the results of the study. An important factor to be considered in these conditions are related to the heavy metal contents and the systems to avoid that those metals can migrate in water and surrounding proprieties.
[44, Italy, 2002]

3.3.3.4.3 Safety of tailings facility and accident prevention

In the case of the fluorspar/lead sulphide operation the dam flanks and water spillage system are checked visually on a day by day basis. The water coming from the overflow of the ponds is chemically checked weekly before the discharge in the river. The phreatic surface is controlled by means of piezometers. For safety reasons the dam height is limited to 7 - 10 m.

There are no specific emergency plans because the risk of a heavy accident is considered "basically zero".
[44, Italy, 2002]

3.3.3.4.4 Closure and aftercare

The closure and aftercare plan for fluorspar/lead sulphide operation is currently in progress. The costs of closure are expected to be in the order of several million of Euro. The monitoring of the site after the end of the operational life must be carried out for several years (currently about 10 years are foreseen) in order to establish if any migration of heavy metal occurs. There are no arrangements for financial assurance to cover the long term risk of pollution, but a special fund is established by the company in the annual balance to finance the closure operations [44, Italy, 2002].

3.3.3.5 Kaolin

3.3.3.5.1 Characteristics of tailings

Characterisation of the materials released from the process

Process's step	Material released from the process	Destination
1. Classifying	▪ coarse sand, gravel and stones	▪ heap or saleable products (if local market available)
2. Hydrocycloning	▪ fine sand, micas, (and feldspar) ▪ process water	▪ If it contains feldspar, it is further refined in the feldspar process ▪ micas is a commercial product ▪ fine sand: heap or saleable products (if local market available) ▪ tailings pond
3. Flotation	▪ very fine sand, micas, (and feldspar) ▪ process water	▪ tailings pond ▪ if it contains feldspar, it is further refined in the feldspar process
4. Thickening	Clear water overflow is directly recycled or used to hold reserves of water.	
5. Product separation and benefication	▪ very fine sand and micas ▪ iron oxides	▪ tailings pond or ▪ heap (compared to the other outputs, the amount is here negligible - several orders of magnitude less)
6. Filtering	▪ process water	▪ tailings pond ▪ the filtrate ("process water") can also be recycled (depends on applied flocculants)
7. Drying		

Table 3.58: Tailings and products from Kaolin mineral processing
[40, IMA, 2002]

Beside the heap of coarse sand, gravel and stones, there are tailing lagoons which contain:

Solid materials:

- fine sand and micas (more than 95 %)
- some iron oxides (less than 1 %)
- flocculents (in the ppm range).

Liquid (process water)

- water at a pH value of about 4.5
- some phosphates
- some sulphates
- foam inhibitor.

3.3.3.5.2 Applied management methods

No data has been supplied for this section. Please provide information.

3.3.3.5.3 Safety of tailings facility and accident prevention

No data has been supplied for this section. Please provide information.

3.3.3.6 Limestone

3.3.3.6.1 Characteristics of tailings

No data has been supplied for this section. Please provide information.

3.3.3.6.2 Applied management methods

The lime industry uses tailings ponds from which the water is recirculated to the mineral processing plant. The tailings are a saleable byproduct. As far as possible waste-rock and dry tailings are also sold for other applications such as road making, cement and concrete manufacturing, but when there is a lack of customers, those aggregates have to be brought to heaps.

Prior to discarding, the ground is investigated in order to check whether the geology, hydrology, environmental issues and stability fit the requirements set up by the competent authorities. These studies are essential to get the permission for a heap from the competent authorities. The waste-rock and tailings are discharged together in horizontal layers. The end benches are immediately covered with soil and reclaimed with grass, and trees according to long-term recreation plans. The evolution of the heap is monitored as well as water quality, ground water level, and slope stability if relevant or required by the authorities.

Slurried tailings are either

- dried (thickener and filter press) and discarded on a tailings heap or
- discharged to the outside water system (effluent) under conditions controlled by the competent authorities or
- stored in a tailings pond (one case in Europe).

In the latter case the quality of the mineral deposit is such that about one third of the quarried stone is not suitable to the mineral processing plant and was used to construct the 16 m wide starter dam after removal of the huminous material. The slope of the starter dam was 1:1 and the impermeable core is protected against erosion by a layer of 1 - 2 m of 0 - 20 mm material. The impermeable core consists of 2 - 3 m of clay surrounded by a membrane.

Eventually the dam was raised. The starter dam was broadened (+ 12 m) and its height increased (+ 5 m).

Today the total area of the clarification pond is about 45 hectares. All tailings are pumped at the same point into the pond (single-point discharge). The seepage water through the dam is gathered and is pumped back into the pond or is discharged into water dam.

When the level of the flotation sand has risen to a certain level, the discharge is moved and the dry flotation sand is excavated and sold. According to analyses of flotation sand (NEN 7341, NEN 7343 and ISO 11466), the contents of heavy metals are negligible. Also the concentration of flotation reagents is very low and they are very tightly fixed on the mineral particles but easily decompose if liberated.
[42, IMA, 2002]

3.3.3.6.3 Safety of tailings facility and accident prevention

The control and monitoring of the tailings facilities is done by both industry and the competent authorities. All the constructions (plans, design, etc.) must receive prior approval by the competent authority. The dams are checked every day and all possible changes in the constructions are marked in the control diary. If any leak is noticed, it will be instantly repaired and the information will be sent to the authority. An in-depth inspection is done yearly, and the authority audits the constructions and the record-keeping every five years.
[42, IMA, 2002]

3.3.3.7 Phosphate

No data has been supplied for this section. Please provide information.

3.3.3.7.1 Characteristics of tailings

3.3.3.7.2 Applied management methods

3.3.3.7.3 Safety of tailings facility and accident prevention

3.3.3.8 Strontianite

No data has been supplied for this section. Please provide information.

3.3.3.8.1 Characteristics of tailings

3.3.3.8.2 Applied management methods

3.3.3.8.3 Safety of tailings facility and accident prevention

3.3.3.9 Talc

No data has been supplied for this section. Please provide information.

3.3.3.9.1 Characteristics of tailings

3.3.3.9.2 Applied management methods

3.3.3.9.3 Safety of tailings facility and accident prevention

3.3.4 Waste-rock management

3.3.4.1 Barytes

No data has been supplied for this section. Please provide information.

3.3.4.2 Borates

No data has been supplied for this section. Please provide information.

3.3.4.3 Feldspar

No data has been supplied for this section. Please provide information.

3.3.4.4 Fluorspar

One producer backfills all waste-rock along with the tailings in the underground operation. The waste-rock comes from the excavation of galleries in rock mass outside of the ore body. The waste-rock is used as back fill, so that the heaps in surface are reduced to minimum and are only used as temporary deposit [44, Italy, 2002].

3.3.4.5 Kaolin

No data has been supplied for this section. Please provide information.

3.3.4.6 Limestone

No data has been supplied for this section. Please provide information.

3.3.4.7 Phosphate

No data has been supplied for this section. Please provide information.

3.3.4.8 Strontianite

No data has been supplied for this section. Please provide information.

3.3.4.9 Talc

No data has been supplied for this section. Please provide information.

3.3.5 Current emission and consumption levels

3.3.5.1 Barytes

No data has been supplied for this section. Please provide information.

3.3.5.1.1 Management of water and reagents

3.3.5.1.2 Emissions to air

3.3.5.1.3 Emissions to water

3.3.5.1.4 Soil contamination

3.3.5.1.5 Energy consumption

3.3.5.2 Borates

No data has been supplied for this section. Please provide information.

3.3.5.2.1 Management of water and reagents

3.3.5.2.2 Emissions to air

3.3.5.2.3 Emissions to water

3.3.5.2.4 Soil contamination

3.3.5.2.5 Energy consumption

3.3.5.3 Feldspar

3.3.5.3.1 Management of water and reagents

1. Micas flotation:

Chemicals used in the process:

Chemicals	Average concentration
Acid (H_2SO_4)	pH about 3
Surfactant	10 - 100 ppm
Foam inhibitors	10 - 100 ppm

Details to be provided by the EUROFEL TC

2. Oxides flotation:

Chemicals used in the process:

Chemicals	Average concentration
Acid (H_2SO_4)	pH about 3
Surfactant	10 - 500 ppm
Foam inhibitors	10 - 100 ppm

Details to be provided by the EUROFEL TC

3. Feldspar flotation:

Chemicals used in the process:

Chemicals	Average concentration
Acid (HF)	pH < 3
Surfactant	10 - 500 ppm
Foam inhibitors	10 - 100 ppm
Alkaline solution (CaO, $Ca(OH)_2$, NaOH)	To neutralise to a pH value of about 4.5

3.3.5.3.2 Emissions to air

In the case of the fluorspar/lead sulphide mine the tailings are humid so that no dust is emitted.

3.3.5.3.3 Emissions to water

No data has been supplied for this section. Please provide information.

3.3.5.3.4 Soil contamination

No data has been supplied for this section. Please provide information.

3.3.5.3.5 Energy consumption

No data has been supplied for this section. Please provide information.

3.3.5.4 Fluorspar

3.3.5.4.1 Management of water and reagents

In one case the clean water from the last clarification pond is partially recycled and partially discarded into the river. The total volume of the tailings ponds is about 1300000 m^3 [44, Italy, 2002].

The water is cleaned before the discharge; the reagents used in mineral processing are basically of vegetal origin (oleins from olive oil, soaps as pine oil and so on); potentially dangerous reagents are chemically treated before discharge.
The water consumption is in average 8000 m^3 per day.
[44, Italy, 2002]

3.3.5.4.2 Emissions to air

Emissions are not given. They are considered to be negligible [44, Italy, 2002].

3.3.5.4.3 Emissions to water

3.3.5.4.4 Soil contamination

In the case of the fluorspar/lead sulphide operation the nature of the material processed could lead to heavy metals contamination. The metals contained are lead, zinc, iron and fluor. However, the concentrations are low and the emissions are monitored periodically.

3.3.5.4.5 Energy consumption

No data has been supplied for this section. Please provide information.

3.3.5.5 Kaolin

3.3.5.5.1 Management of water and reagents

The reagents used in the flotation of Kaolin are listed in the following table.

Reagent	Average concentration
Acid (H_2SO_4, H_3PO_4)	To reach a pH value of about 2.5
Surfactant	10 - 100 ppm
Foam inhibitors	10 - 100 ppm
Alkaline solution	To neutralise to a pH value of about 4.5

Table 3.59: Reagents used in the flotation of Kaolin

3.3.5.5.2 Emissions to air

No data has been supplied for this section. Please provide information.

3.3.5.5.3 Emissions to water

No data has been supplied for this section. Please provide information.

3.3.5.5.4 Soil contamination

No data has been supplied for this section. Please provide information.

3.3.5.5.5 Energy consumption

No data has been supplied for this section. Please provide information.

3.3.5.6 Limestone

No data has been supplied for this section. Please provide information.

3.3.5.6.1 Management of water and reagents

3.3.5.6.2 Emissions to air

3.3.5.6.3 Emissions to water

3.3.5.6.4 Soil contamination

3.3.5.6.5 Energy consumption

3.3.5.7 Phosphate

No data has been supplied for this section. Please provide information.

3.3.5.7.1 Management of water and reagents

3.3.5.7.2 Emissions to air

3.3.5.7.3 Emissions to water

3.3.5.7.4 Soil contamination

3.3.5.7.5 Energy consumption

3.3.5.8 Strontianite

No data has been supplied for this section. Please provide information.

3.3.5.8.1 Management of water and reagents

3.3.5.8.2 Emissions to air

3.3.5.8.3 Emissions to water

3.3.5.8.4 Soil contamination

3.3.5.8.5 Energy consumption

3.3.6 Costs

For the flourspar/lead zinc operation mentioned in Section 3.3.3.4 the overall cost for tailings management in several ponds, 1300000 m^3 in total volume, is around EUR 210000 /yr; this includes energy consumption and maintenance of the section.

Please provide further information

3.4 Potash: Mineral processing, tailings and waste-rock management

The TWG recognised that the applied techniques for potash are very much different than all other industrial minerals, hence a separate chapter. Unless otherwise mentioned the following information was submitted by the potash BREF subgroup [19, K&S, 2002].

3.4.1 Mineralogy and mining techniques

Potash salt deposits were formed by the evaporation of seawater. Their composition is often affected by secondary changes in the primary mineral deposits. More than 40 salt minerals are known, which contain some or all of the small number cations Na^+, K^+, Mg^{2+}, and Ca^{2+}, the anions Cl^- and SO_4^{2-}; and occasionally Fe^{2+} and Br^-, as well. The most common minerals are listed in Table 3.60.

Anhydrite	$CaSO_4$
Carnallite	$KCl \times MgCl_2 \times 6H_2O$
Gypsum	$CaSO_4 \times 2H_2O$
Halite	$NaCl$
Kainite	$KCl \times MgSO_4 \times 11H_2O$
Kieserite	$MgSO_4 \times H_2O$
Langbeinite	$K_2SO_4 \times 2MgSO_4$
Leonite	$K_2SO_4 \times MgSO_4 \times 4H_2O$
Polyhalite	$K_2SO_4 \times MgSO_4 \times 2CaSO_4 \times 2H_2O$
Sylvite	KCl

Table 3.60: Most common salt minerals in potash deposits

The most important salt minerals are halite, anhydride, sylvinite, carnallite, kieserite, polyhalite, langbeinite and kainite. Gypsum and/or anhydride occur at the edges of salt deposits and in the overlying strata.

Other minerals, not described in detail here, are useful in elucidating difficult questions with regard to the origin of salt deposits. The individual minerals can be identified microscopically (grains or thin sections) and by x-ray analysis. For quantitative detection of the elements the emission spectroscopy (ICP = Inductively Coupled Plasma) is the most important technique.

Potash salt deposits always consist of a combination of several minerals (Table 3.61). The German term "Hartsalz" (hard salt) refers to the greater hardness of sulphate and magnesium - containing potash minerals.

Marine salt minerals	Main compounds
Sylvinite	Sylvite, halite
Carnallitite	Carnallite, halite
Hard salt	Sylvite, halite, kieserite and/or anhydrite
Kainitite	Kainite, halite

Table 3.61: Marine salt minerals

In the following, to avoid confusion, the term sylvinite will be used for KCl.

Salt deposits in Central Europe are the result of intensive evaporation of marine water more than 250 million years ago. Over millions of years, the original salt deposits were covered with other sediments, such as clay, limestone and anhydride. Tectonic influences left them as flat layers (sub-horizontal deposits:) or deformed them into steeply dipping deposits (see Figure 3.56 and Figure 3.57 respectively).

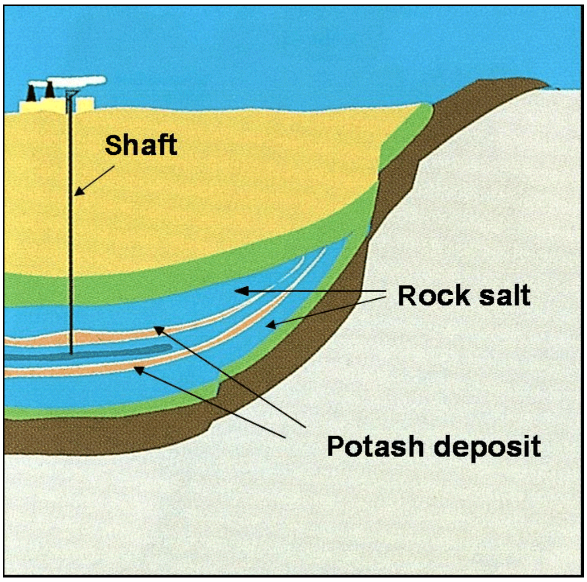

Figure 3.56: Sub-horizontal potash deposit

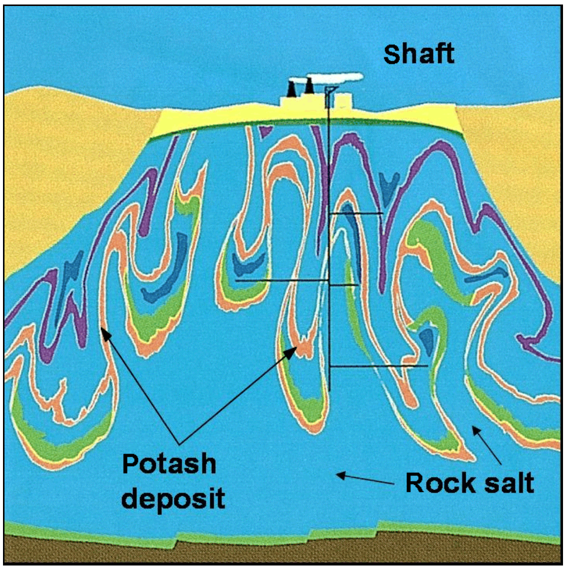

Figure 3.57: Steeply dipping potash deposit

Potash is usually extracted by room and pillar and sometimes longwall mining. Sometimes the solution mining method is also applied. However, today solution mining is only of minor local importance in Europe. Open pit mining is not an option, due to the water solubility of potash.

Room and pillar mining

In Europe, the room and pillar technique with drill and blast mining is usually applied. The height of stopes is about 2 - 3 m. An extraction rate of 25 - 60 % is usual. Two ways of applying this method are currently practised:

- *drilling and blasting*: Drilling machines are used to cut small diameter boreholes over a distance of 7 m to 30 m in the face, either horizontally (sub-horizontal/flat deposit) or vertically (steep deposit). The holes are filled with explosives (prills of Ammoniumnitrate with 3 % mineral oil) and the rock is blasted. The fractured salt is hauled by loaders to underground pre-crushing stations where it is crushed to a size which can be transported by conveyor-belts

- *continuous mining*: An excavation machine with a rotating head, so-called “continuous miner”, is used to mine the ore in a size which can be transported directly by conveyor belts. The following surface operations are similar to the drill and blast mining method. Bolts are placed in the roof of the underground galleries for support and to protect the workers and the equipment.

At present, potash mining in Germany is carried out in depths between 400 and 1200 metres. The ore is always transported in pre-crushed form by conveyor-belts to intermediate underground storage prior to hoisting with skips.

Longwall mining

This is the same method commonly used to mine coal deposits in Europe.

Sublevel stoping

In steeply dipping deposits in Northern Germany, sublevel stoping (also called “funnel mining”) is carried out. Entry drifts are driven one above the other at intervals of 15 - 20 m, and the remaining potash salt is mined by drilling vertical boreholes and blasting. The blasted ore falls into the main level underneath. The mined-out room, 100 – 250 m in height, is usually backfilled with salt tailings.

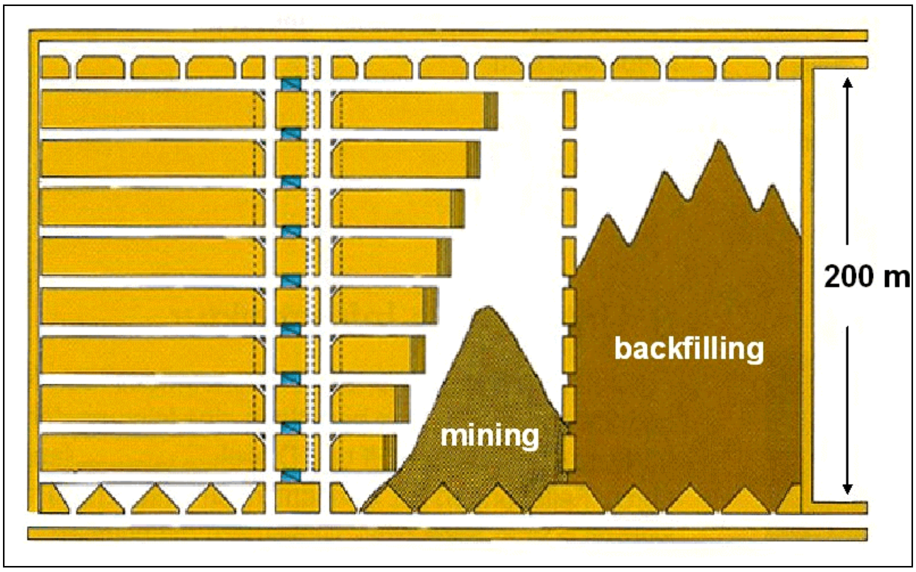

Figure 3.58: Sublevel stoping with backfill in steep potash deposits

Solution mining

KCl-unsaturated brine is injected in a borehole into the salt deposit to dissolve potassium chloride. The KCl-saturated brine is pumped back to the surface. The saturated solution crystallises and precipitates by evaporation of the brine in huge evaporator-vessels. A second separation process - e.g. flotation or recrystallisation - follows to purify potassium chloride and sodium chloride as marketable products.

Solution mining of potash ore is of minor importance in Europe.

Exploited potash deposits in Europe

The exploited potash deposits in Europe were mainly formed in the Permian period, which took place in a vast evaporite basin, called the Central European Basin. This basin extends from North-East England to Central Poland and Lithuania, and from Central Germany to the northern part of the North Sea. The Alsacian and Spanish deposits were formed in the Tertiary period and are isolated basins.

France

The deposit in Alsace contains two sylvinite seams in a marl-rock salt series. The upper layer has a thickness of up to 2 m and contains 19 - 25 % K_2O; the lower, up to 5.5-m-thick layer, with 15 – 23 % K_2O, also contains 15 % insolubles (clay, anhydrite, and dolomite). Mining is carried out at comparatively high rock temperatures at a depth of 500 - 1000 m in flat or slightly inclined seams that have been disturbed by faults. The last producing mine is expected to be closed in 2003.

Germany

In the Werra and Fulda areas, the Hessen and Thuringia potash seams of the Werra series are mined (hard salt and carnallitite in level deposits at a depth of 400 - 1000 m with a thickness of 2 - 5 m, K_2O: 9 – 12 %, $MgSO_4$: 4 – 20 %). The Stassfurt potash seam of the Stassfurt series was mined in the Harz-Unstrut-Saale area (hard salt and carnallitite at a depth of 500 - 1000 m and a thickness of 5 m, K_2O: 20 %). The last potash mines, extracting hard salts of the Stassfurt series closed in 1991 for economic reasons. The potash seams Ronnenberg and Riedel of the Leine series are mined in the Hanover area in salt diapirs (sylvinite in inclined deposits, depth: 350 - 1400 m, thickness: 2 - 40 m, K_2O: 12 – 30 %). Finally, potash is mined on the Massif of Calvörde near Zielitz (depth: 350 - 1200 m, Ronnenberg sylvinite inclined at <18 - 25°, thickness: up to 10 m, K_2O: 14 – 20 %).

Spain
Deposits are located in two areas of the Ebro basin. In Catalonia and Navarra, potash salts lie above the rock salt. These deposits are up to 10 m thick in Catalonia and up to 15 m in Navarra. Above this occurs an interbedded deposit of rock salt, carnallitite, marl, and anhydrite. Only the sylvinite seams A and B are mined. These are up to 4 m in total thickness at a depth of 1020 m, some deposits are level and some inclined. The crude salt contains 12.5 – 14 % K_2O.

United Kingdom
In Yorkshire, a level deposit of sylvinite and carnallitite is extracted, which correlates with the German Riedel seam both petrographically and stratigraphically (average thickness: 7 m, K_2O: ca. 25 %, depth: 800 - 1300 m).

3.4.2 Mineral processing

Processing of potash generally involves a series of steps including size reduction (crushing/grinding), separation (hot leaching-crystallisation, flotation, electrostatic separation), de-brining. These steps are described below.

3.4.2.1 Comminution

The salt minerals in run-of-mine potash ore are intergrown to varying extents. Before the minerals can be separated and the useful components recovered, the raw salt must be sufficiently reduced in size to liberate the desired mineral from the gangue.

For the hot leaching process, a maximum grain size limit of 4 - 5 mm is adequate. For mechanical processing (e.g. flotation), the potash minerals must be ground to a degree of liberation > 75 %. For sylvinite minerals and hard salts, this is achieved by grinding to a maximum size of 0.8 - 1.0 mm.

Various grain size fractions are produced in mills and different types of screens. In the first stage impact- or hammer mills produce generally about 4 - 12 mm particles, depending on the raw material and the processing method used. The final fine grinding stage works with rod mills (when wet) or under dry conditions with roller mills or impact crushers (see figure Figure 3.59). The selection of the used equipment aims at minimising the generation of fines and ultrafines which have a negative influence on the subsequent separation, e.g. in flotation the reagent consumption increases significantly with the amount of fines due to the larger specific surface.

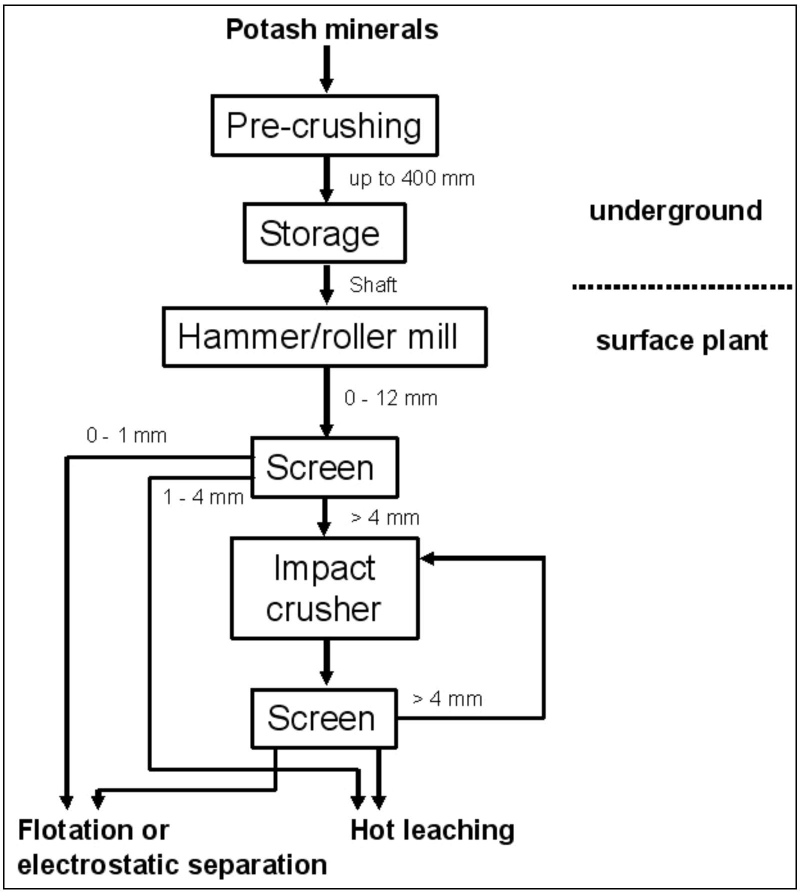

Figure 3.59: Dry grinding and screening (schematic) of potash ore
[19, K&S, 2002]

3.4.2.2 Separation

If potash is mined "mechanically", i.e. not by solution mining, four methods can be applied for separating the desired salts from the gangue:

1. hot leaching
2. flotation
3. electrostatic separation
4. heavy-media separation.

For all wet processes (1,2,4) de-brining is necessary.

The following sub-sections describe these process steps.

3.4.2.2.1 Hot leaching process

Two different processes are used, depending on the composition of the salt minerals. In the **sylvinite hot leaching process** the other salts present besides KCl and NaCl play only a minor role in process solutions. The **hard salt leaching process** solutions contain appreciable amounts of $MgSO_4$ and $MgCl_2$. In the case of carnallite-containing hard salts or unique carnallitite, preliminary carnallite decomposition must be carried out, if the amount of carnallite present exceeds a critical level of about 20 - 30 %.

The potash minerals, ground to a fineness of < 4 - 5 mm, are stirred in a continuous dissolver with leaching brine heated to just below its boiling point. The leaching brine (with a temperature of about 110°C) is the preheated mother liquor from the crystallisation stage of a

previous cycle of the process. The potassium chloride should be extracted from the minerals as completely as possible, and the resulting product solution should be nearly saturated. The tailings consist of two fractions of different particle size. The coarse fraction is removed from the dissolver and de-brined. The fine fraction (e.g. slime) is removed from the dissolver along with the crude solution. After separation in a clarifier, the fine fraction is filtered off.
The tailings are washed with water or plant brines low in potassium chloride to remove the adhering crude solution, which has a high potassium chloride content. The tailings are then discarded by stacking or backfilling in the mine. If kieserite must be separated, the tailings are transported to further mineral processing (e.g. flotation). The filtrate of tailings-dewatering is recycled to the re-circulating brine.
The hot, clarified, solution is cooled by evaporation in the vacuum station. Evaporated water must be replaced to avoid crystallisation of undesired sodium chloride. The desired potassium chloride crystals, formed by cooling the crude solution stage by stage (down to about 25°C), are separated from the mother liquor and processed further. The mother liquor (saturated with KCl and NaCl at 25°C, see Section 2.3.2) is heated and recycled to the dissolver as leaching brine. The layout of a leaching plant including crystallisation is shown in Figure 3.60.

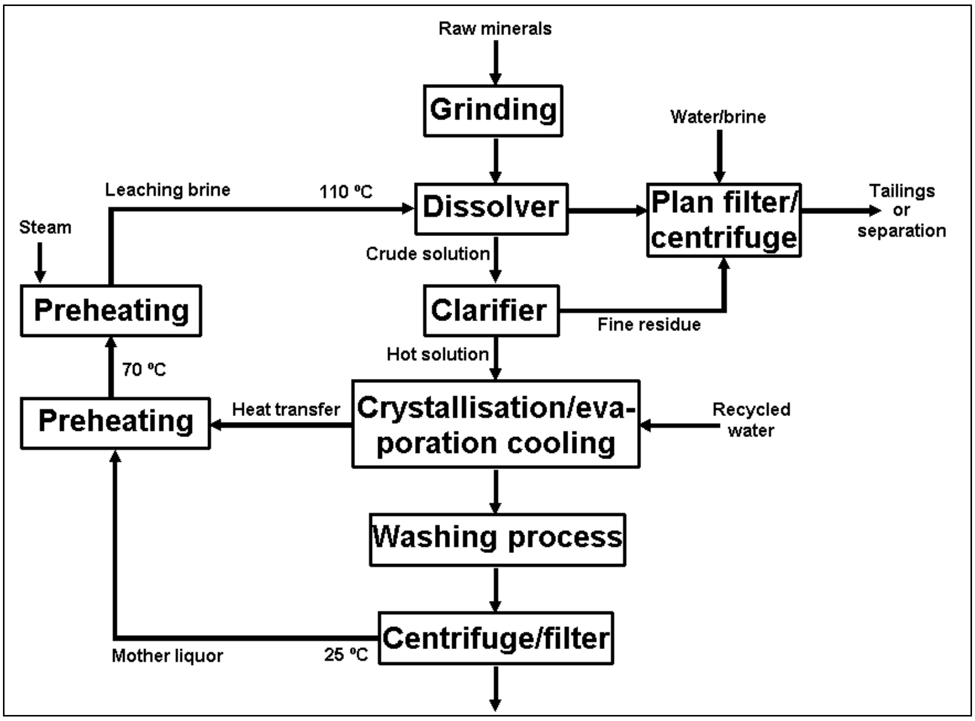

Figure 3.60: Flow diagram hot leaching-crystallisation process for production of KCl from potash minerals (schematic)

This simple process is used for the treatment of sylvinite minerals only. The mineral processing of hard salt minerals is more complicated. With higher magnesium salt contents, the temperature dependence of the solubility of NaCl becomes undesirable and the yield of potassium chloride decreases.

In many plants, especially in Canada, where flotation is the main production process, small hot leaching plants are also operated, in which the product "fines" (< 0.2 mm) are re-crystallised, or potassium chloride is separated from flotation tailings or thickened clay slurries. These procedures give a considerable improvement in total yield and result in a very pure, completely water-soluble product. The hot leaching process is necessary to generate pure potassium chloride products for chemical or pharmaceutical uses.

3.4.2.2.2 Flotation

In the German potash industry potash flotation as well as the kieserite flotation is used. After grinding or previous separation-processes the fine size fraction (0 - 1 mm) is added to an aqueous, saturated potassium/kieserite and sodium chloride solution (Flotation liquor, see Section 2.3.2). Flotation essentially relies on the phenomenon that, fresh mineral surfaces, conditioned with specific surface active chemicals, can be induced to adopt either a hydrophobic or hydrophilic attitude in solution. If air bubbles are then introduced into the solution, mineral particles (if hydrophobic) will preferentially attach themselves to air bubbles at the air/water interface and float on the top. A frothing agent such as pine oil is usually added and rotating paddles scrape the potassium chloride or kieserite bearing froth from the surface for further treatment.

The most satisfactory collecting agents are long chain alkylammoniumchlorides. Figure 3.61 shows the mineral processing of the raw minerals or intermediates in rougher and cleaner flotation cells to a de-brined, filtered product in an schematic illustration.

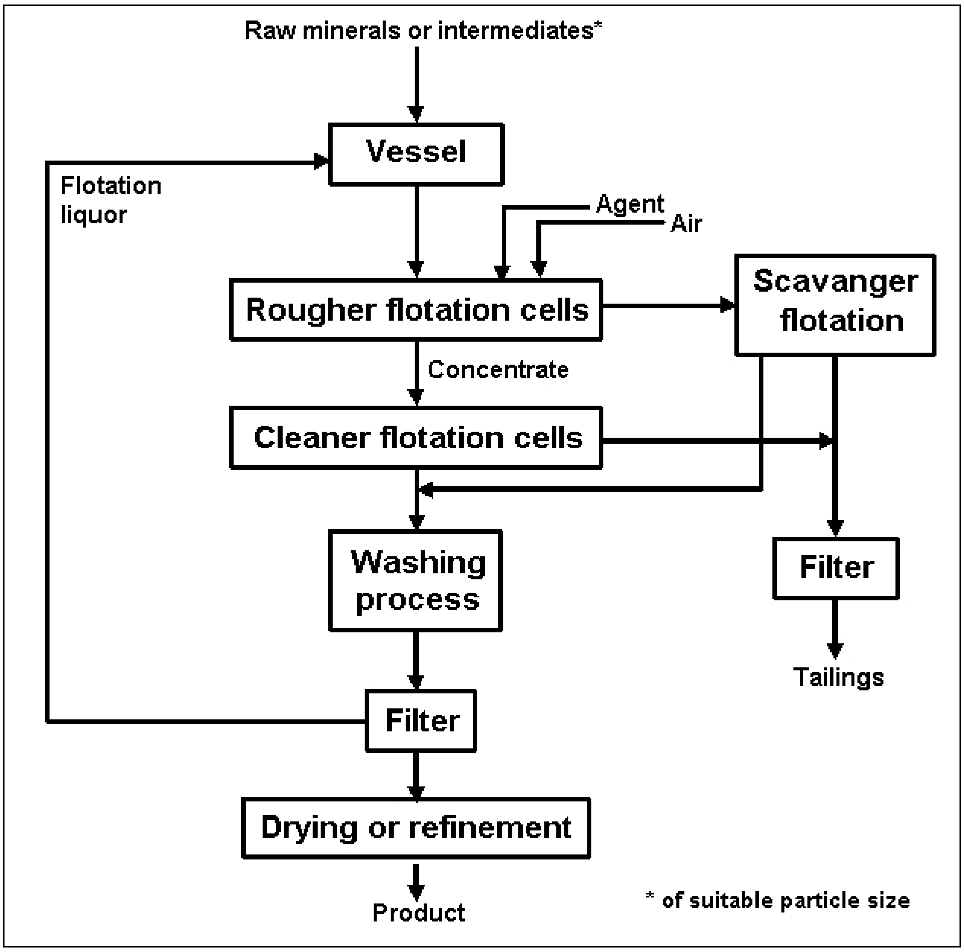

Figure 3.61: Flow diagram flotation plant (schematic)

3.4.2.2.3 Electrostatic separation

Crushed and ground raw salt is conditioned to achieve greater retention of the electrostatic charge by heating to less than 100 °C. The crystals are coated with an organic agent such as a primary fatty acid, a derived salt, ester or amine. Depending on the aim of separation 20 to 100 g of conditioning agents per tonne of raw salt are applied.

The ground mineral is electrostatically charged - under defined relative humidity - by friction in a heated fluidised bed (Figure 3.62). Separation of the halite minerals occurs when the charged crystals fall under gravity through an electric field of about 120000 volts in a free fall separator.

The separation process is controlled by adjustable flaps, that are placed in the bottom of the separator (see Section 2.3.2). The middlings are reconditioned and recycled.

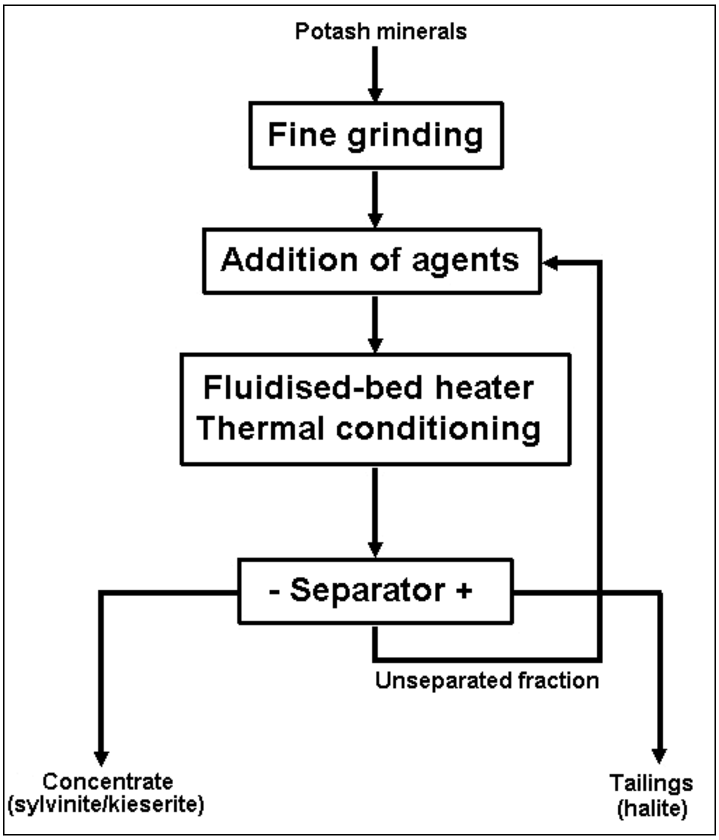

Figure 3.62: Flow diagram electrostatic separation

In most cases, a multi-stage separation or treatment is used. The solid tailings (sodium chloride/halite) are stacked directly on the tailings heap. Other options, e. g. first the separation of sylvinite and carnallite from kieserite are also possible and applied at other plants.

3.4.2.2.4 Heavy-media separation

Halite has a higher density than sylvinite (specific gravity 2.13 g/cm^3 versus 1.9 g/cm^3 for sylvinite). Commercial heavy media operations use a very finely divided weighting agent, typically ferrosilicon or magnetite of fine grade, which is slurried to create an artificial heavy medium at the specific gravity required for separation. After separation, the magnetite or ferrosilicon is recovered by magnetic separation and re-circulated to the system.

International Minerals & Chemical Corporation (IMC) operates a plant of this type in Canada, and for separation of langbeinite (specific gravity 2.83 g/cm^3) from sylvinite/halite a plant in New Mexico, United States. At present, this technique is not used in Europe.

3.4.2.3 De-brining

The products and tailings from all treatment processes except for the dry electrostatic process are obtained as suspensions/slurries with various solid contents and must be de-brined - after being thickened in circular thickeners. The equipment used are centrifuges, plan filters, drum filters and belt filters, especially for de-brining fine tailings (moisture about 9 - 14 %) and when washing the filter cake is necessary. The choice of equipment is determined mainly by the particle size of the material to be treated and the content of other minerals like clay.
For coarse products and tailings, vibrating screens and screw screen centrifuges are commonly used.

3.4.3 Tailings management

The mineral processing of potash minerals leads to over 78 % solid or liquid tailings, which must be managed without damage to the environment (see Figure 3.63).

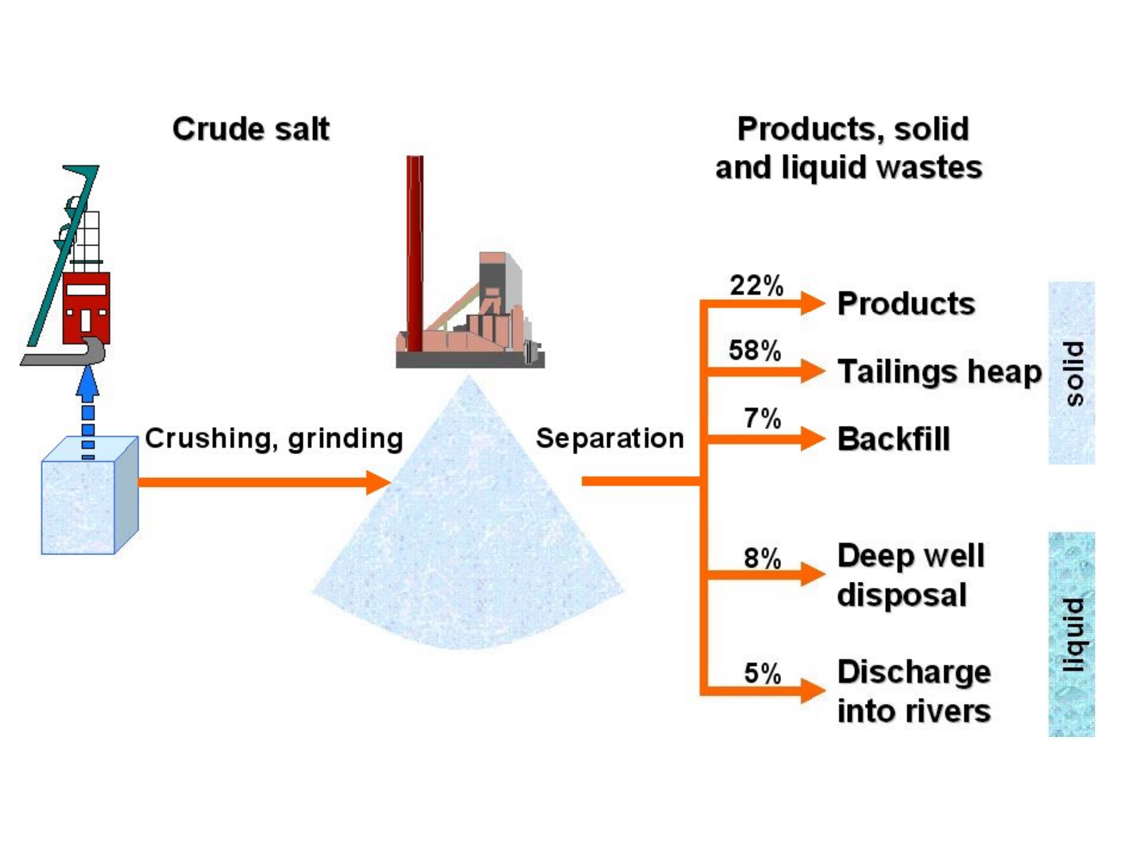

Figure 3.63: Distribution of products, solid and liquid tailings after mineral processing

Six methods for managing of process water and/or tailings are applied:

- storing solid tailings on heaps
- backfill solid tailings into mined out rooms of underground works
- storing slurried tailings on tails piles (only in Canadian/US-Potash Mines)
- marine tailings management of solid and liquid tailings
- pumping liquid tailings into the ground (deep well tailings management)
- discharging liquid tailings into rivers.

3.4.3.1 Characteristics of tailings

Solid potash tailings consist of sodium chloride with a few percent of other salts and insoluble materials like clay and anhydrite (Figure 3.64, "sylvinite tailings"). Hard salt tailings additionally contain about 5 % of kieserite (Figure 3.64, "hard salt tailings").

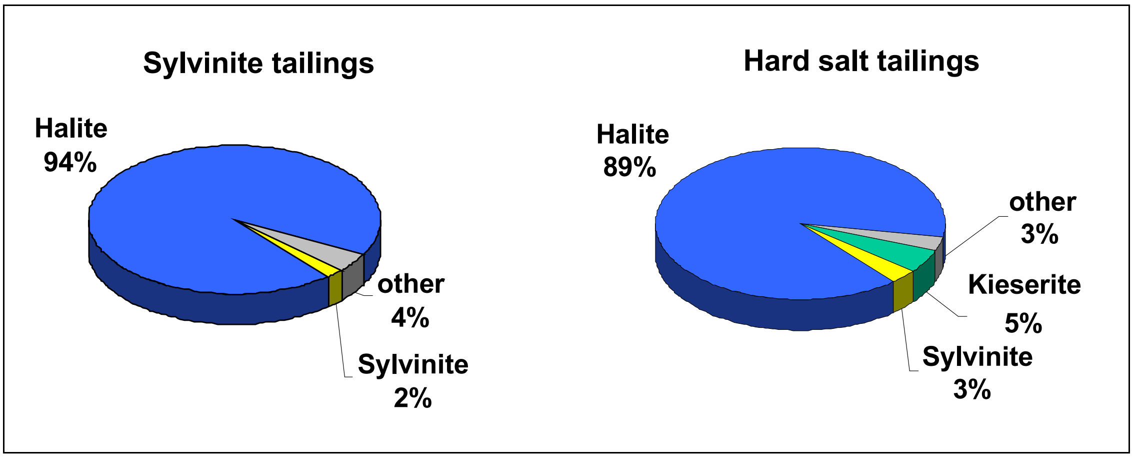

Figure 3.64: Mineral composition of sylvinite and hard salt tailings

The stacked tailings harden immediately, and the density of the tailings increases to nearly the same density as underground due to compaction. This was shown by measurements from borehole-samples of tailings heaps. Heaps are stacked with an angle of repose of about 37 ° (natural soil angle: 25 degrees). Therefore no problems with the slope stability of the heap occur, if the underlying ground is stable. There is a wide experience in stacking potash tailings, the first heaps going up to 200 m in height where started about 30 years ago. Smaller heaps with tailings from potash mining exist from the beginning of potash-mining in about 1890.

Precipitation dissolves the tailings heaps slowly and over a long period of time. As a result of compaction and hardening, the interior of potash tailings heaps is impermeable to water. Water and generated brines flow down in an outer sphere around the inner impermeable core. To protect soil and groundwater, especially the outer seam of heaps outside the impermeable core zone is carefully sealed and the brines are collected in sealed ditches around the heap. Water and generated brine is collected and discharged into surface waters. The slope of the heap consists of hardened rock salt without any erosion after compaction and re-crystallisation.

The highly soluble NaCl needs careful management to reduce the impact on the local environment. However, the tailings usually do not contain significant amounts of heavy metals or other substances, besides NaCl, harmful to the environment.

Liquid potash tailings consist of

Please provide further information.

3.4.3.2 Applied management methods

The key roles for the total amount of salt tailings generated by a mine are primarily the potash seam configuration, rock stability and mineral composition. In fact, these are all natural conditions that vary vastly between mines and deposit and sometimes even within one deposit. As a result, there is no standard model of mines in terms of processing and generation of products and salt tailings. Each mine has its own specific conditions affecting tailings salt or brine generation. Therefore, the managing of tailings has to be looked after individually. Also, these specific conditions can change over the lifetime of a mine. Any operator, however, will try to minimise the amount of non-used associated minerals moved because of economic reasons.

For **solid tailings** the storing on tailings heaps and backfill into mined out rooms of underground works is applied. The tailings from the hot leaching and flotation process with sodium chloride as the main compound is dewatered by centrifuges, filters and then transported by conveyor-belts to the tailings heap. In addition, in Germany, the dry electrostatic separation process allows dry management of tailings on tailings heaps.

For **liquid tailings** deep well discharge (under specified geological conditions) and discharging into rivers is used. Under special geographical conditions, marine discharge of solid and liquid tailings is applied (see Section 3.4.3.2.6).

3.4.3.2.1 Tailings heaps

About 21 million tonnes potash tailings are stacked in the German potash industry every year. As a consequence of this, large tailings heaps are built with quantities of 25 to 130 million tonnes, altitudes of 90 to 240 m with a land consumption of 47 to 110 ha.

The largest tailings heaps, their location, altitude, size, the quantity of tailings and the main compounds are shown in Table 3.62.

Plant/facilities	Location	Altitude (m)	Size (ha)	Quantity (million tonnes)	Main compound	Remarks
Hattorf	Werra-area	160	47	59	Halite	
Wintershall	Werra-area	240	55	99	Halite	
Unterbreizbach	Werra-area	42	4.6	<1	Kieserite	Currently reprocessed
Neuhof-Ellers	Fulda- area	180	70	80	Halite	
Sigmundshall	Hannover area	150	26	25	Halite	
Zielitz	Zielitz Zielitz	50 90	53 110	 130	Halite Halite	

Table 3.62: Tailings heaps of the German potash mines

The following figure shows a picture of a typical salt tailings heap in Germany.

Figure 3.65: Aerial view of a salt tailings heap

Environmental impact studies including baseline studies are part of the design of these heaps.

The base line studies include the research for different aspects e.g.

- stability of the heap
- stability of the supporting strata
- water protection (ground- and surface water, water quality and supply)
- dust emissions
- technical operations
- wildlife habitat
- rehabilitation and after-care
- controlling and monitoring systems.

It is necessary to ensure the **stability of the heap** to avoid movements of parts of the heap. The rock salt hardens immediately because the moisture of the stacked material is sufficiently low. Therefore no significant erosion effects and support around the heap is not necessary.

The safety of the tailings heap results from the application of fundamental civil engineering rules.

The **stability of the supporting strata** is controlled regularly by seismic monitoring (see monitoring and control systems, below), which search for and determine seismic, seismic-acoustic and geo-mechanic facts. Survey of pillars and the determination of the mineral compounds are used to calculate and observe the stability of the mined out rooms.

For **water protection** the following must be considered:

- water balance (groundwater and surface water)
- detected aquifer stratum
- watersheds
- water impermeable clay layer
- water evaporation
- water re-utilisation
- water supply and distribution management
- quantity and minimisation of accumulated drainage water
- determination of salt quantity
- determination of used land for stacking.

The water permeability of the soil is determined (baseline conditions). Mostly the found soil components are sufficiently impermeable to prevent the contamination of groundwater. If not, the ground under potash tailings heaps is sealed by improving the natural soil with up to 4 %of clay. The clay is milled into the natural soil and the mixture is distributed and compressed to reach impermeable conditions. After treatment the permeability coefficient is controlled and if insufficient the procedure is applied again.

The interior of potash tailings heaps is impermeable to water. Water and generated saline solutions only flow down in an outer sphere around the inner impermeable core (Figure 3.66). The toe of the heaps outside the impermeable core zone is carefully sealed and the solutions are collected.

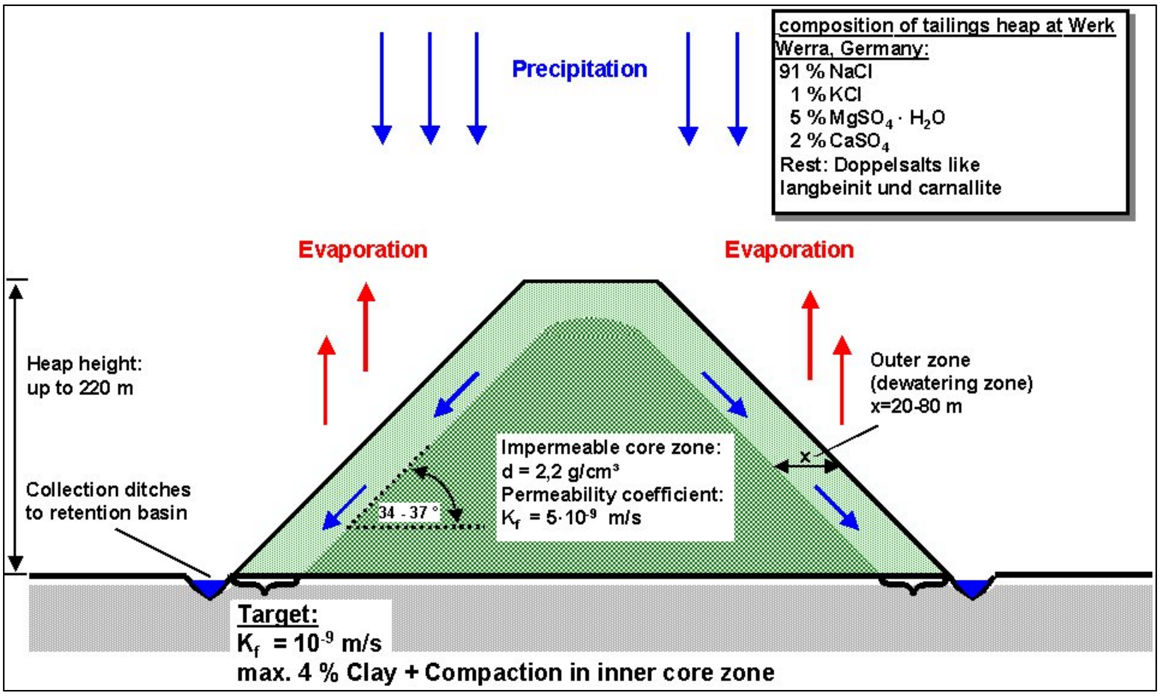

Figure 3.66: Schematic drawing of a tailings heap in German potash mining

After collecting the brine in the retention basin for intermediate storage and depending on the received water quality, the liquid is pumped to the river or into the ground (deep well discharge). In some cases the collected brines are re-used for processing (e.g. granulation, recycled processing brine). In general only small amounts of collected brine are re-used.

Since the water flow from precipitation runs down the heap underneath the surface (see blue arrows in figure above) erosion at the surface does not occur. If possible, saline drainage from the heaps is kept separately from surface run-off. This is one way to minimise salt water contamination of soil and groundwater.

Another objective is to reduce land use by stacking the tailings to a maximum of height. In this operational technique (see below), the design used (conical/longitudinal heap) and the natural angle of repose are critical to obtain this.

The commonly applied technique uses conveyor-belts, continuously stacking the tailings on a heap, which is located near the processing plant. After the addition of a small amount of processing brine to the dry tailings from electrostatic separation the moisture of the stacked combined tailings results to the aimed 5 - 6 %. The stacked salt hardens immediately because of compaction and re-crystallisation

The **technical operations** for stacking have been applied and optimised over more than 30 years.

The salt tailings are stacked using conveyor-belts and spreader systems, this allows steeper, higher stacking than wet stacking. Up to 1200 t per hour solid tailings are stacked on one heap. These enormous amounts of material are piled near the processing plant, to minimise material transport over long distances or through communities.

The distribution of tailings on the heap is performed by combination of several conveyor-belts. Depending on the type of construction chosen, the discharging belt can be slewed, adjusted in height and, if necessary, be telescoped. A low discharge height is preferred. A last short underlying conveyor-belt, arranged below the main conveyor-belt is reversible (see figure 18), which is particularly useful in windy conditions

Figure 3.67: Photo of a conveyor-belt with an underlying reverse belt

Processing and therefore also tailings discharge is continuous day and night. The employees usually work in rotating shifts. Continuous working systems create less dust and noise and material transport over long distances is not required.

The **wildlife habitat**, the actual state and the future development are examined and considered.

The **controlling and monitoring regime** examines seismic events or subsidence of the surface as a result from mining activities. The stability of the supporting strata and underground mined rooms can be controlled by seismic monitoring.

At the surface different controlling and monitoring systems are applied e.g. for groundwater protection, determination and control discharging brine to the river and the mineral processing process, dust emissions, energy consumption, water supply etc.

Several locations with slope inclinometer instrumentation are used to study the deformation and the stability of the tailings heap.

3.4.3.2.2 Tailings piles

Commonly the tailings in **Canadian/US** plants are pumped as a slurry with 20 – 35 % solids to the top of the tails piles in the tailings management area. The slurry flows down the gentle back slope of the tails pile with the slimes settling out at the toe of the pile. Low containment dykes are built to confine the discharge of brine to the surrounded area. At present, the tails piles are generally in the order of 50 m in height. As a result, in Canada large areas are occupied by tails piles

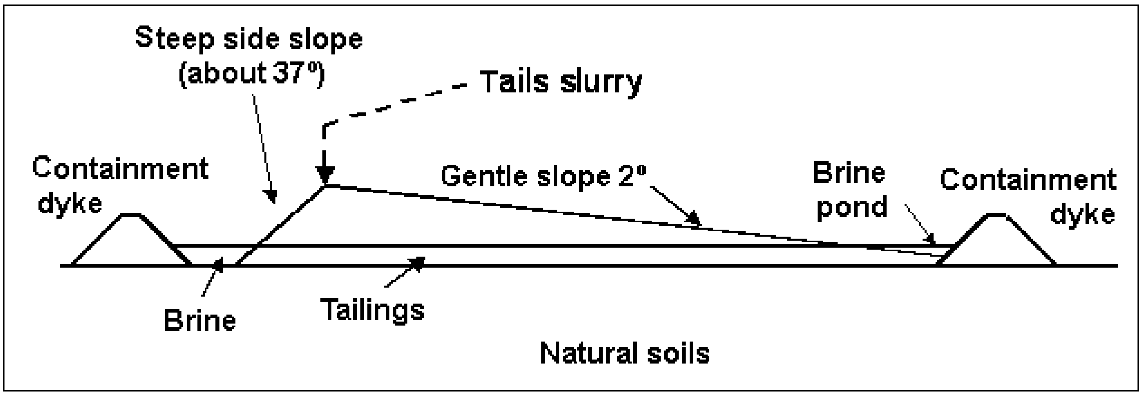

Figure 3.68: Typical cross-section of Canadian tails piles (schematic)

3.4.3.2.3 Backfill

The second method of tailings management for solid tailings is the **backfill into mined out rooms of underground works**. This method is applied in steeply dipping deposits in Northern Germany as well as in the potash mines of New Brunswick in Canada. Since the bulk density of the tailings is much lower than that of the original potash ore, only a part of the tailings can be accommodated by the space left after extraction of crude salt.

In most potash plants, where the mineral is mined from flat deposits, backfill is not carried out for economic reasons.

A similar method - but less important for active European mines - is backfilling of tailings as a slurry. The tailings slurry is returned underground to fill up the potash cut-and-fill stopes, which are shaped as "domes". But the applicability of this option, amongst other reasons, depends on the existence of suitable geological formations (local steeply dipping deposit).

At one plant, Unterbreizbach in the Werra-region, brine is added to the tailings and the resulting slurry is pumped for backfill.

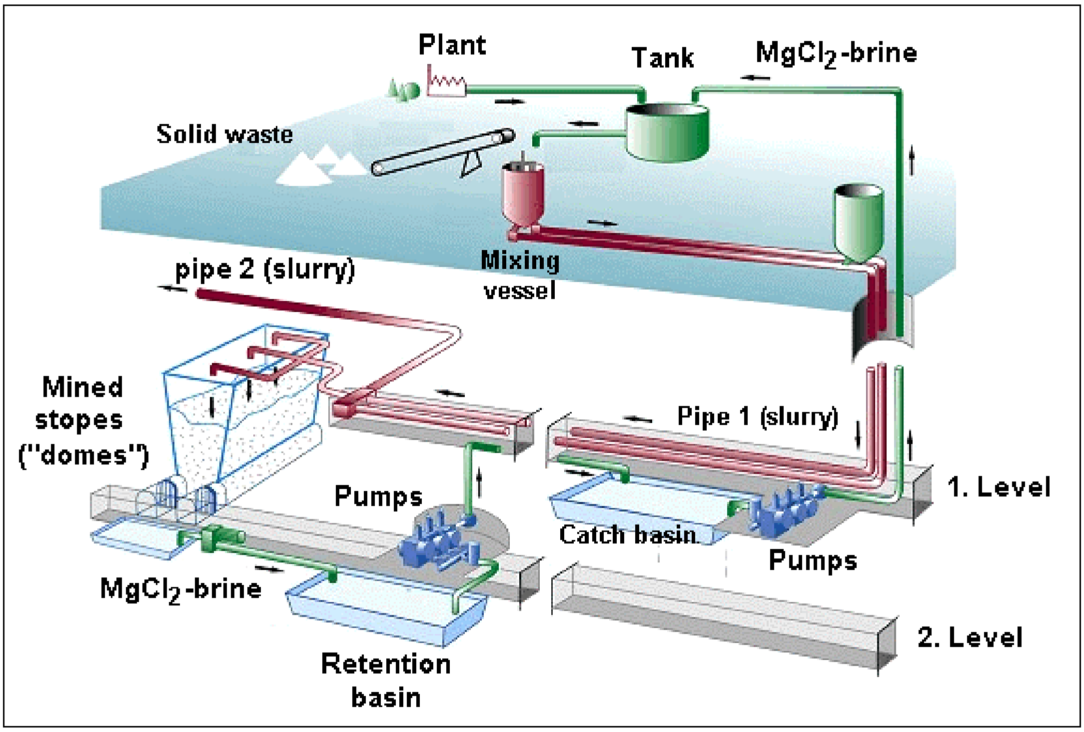

Figure 3.69: Backfill system of solid tailings (sodium chloride) at the plant Unterbreizbach, Germany

The plant Unterbreizbach differs from the other potash plants with flat deposits in various aspects:

Geology:
- the exploited seam Thuringia contains a very thick layer of carnallite above the hard salt seam. When the carnallite is mined, a series of empty "domes" are left.

Mineral processing:
- combination of thermal dissolution process and the flotation of kieserite.

Tailings management:
- salt tailings (solid sodium chloride) from the flotation of kieserite are slurried with $MgCl_2$-brine (salt-saturated) from the thermal dissolution process and pumped underground for backfill. The efficiency of the backfill system could be increased with a second pipe. The brine is recovered underground and pumped back to the surface for re-use.

In short, backfill is only applied under certain geological conditions where mining creates large stopes.

3.4.3.2.4 Surface water discharge

Brine from production, sometimes mixed with small amount of salt process water from the tailings heap, is collected in retention basins from where the brine is discharged into surface water (e.g. river). Figure 3.70 shows one of these basins.

Figure 3.70: Water retention basin of german potash mine

3.4.3.2.5 Deep well discharge

Pumping salt solutions back into the ground is possible if certain geological requirements are met. The formation used for this purpose must possess sufficient porosity and permeability and, must have no contact with formations that can be used for water supply. Most of this brine is comes from carnallite processing and production of potassium sulphate.

In the German potash industry a combination of river and deep well discharge are used. As much water as possible is discarded into the river system. This is determined by the set threshold for chloride in the river taking into account the total discharge of all potash mines (see figure below). All excess water is pumped into the deep wells.

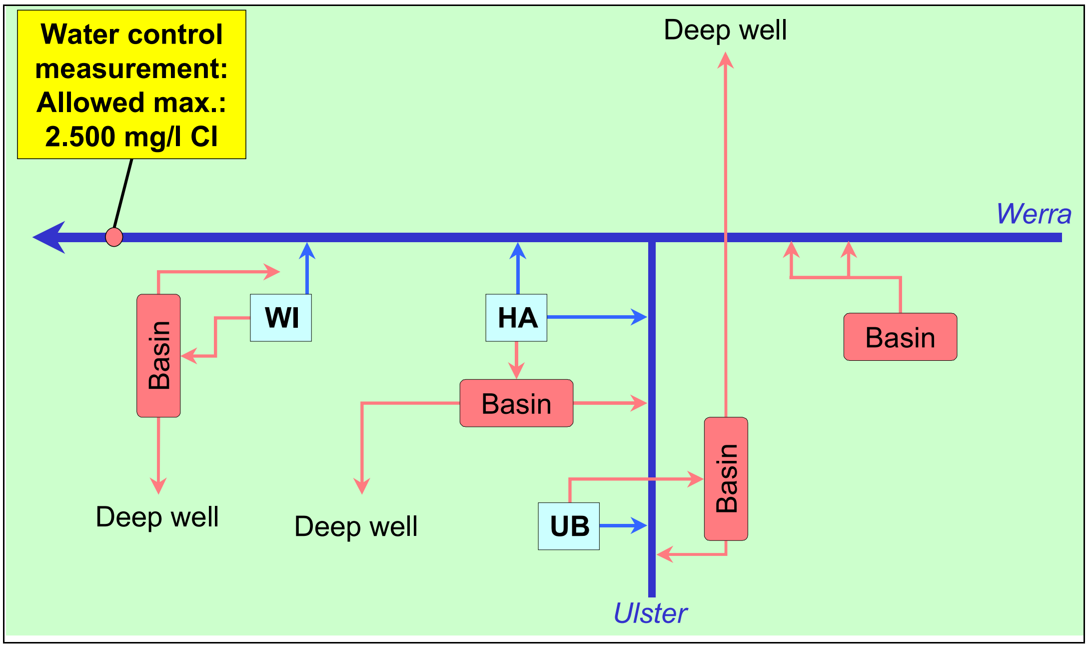

Figure 3.71: Management of three potash mines (WI, HA, UB) in the Werra area, Germany

3.4.3.2.6 Marine tailings management

At the refinery process of the Cleveland Potash Plant the raw minerals are crushed and separated into the potash and the tailings fraction. The tailings consist primarily sodium chloride, less calcium sulphate and clay. These naturally occuring components are mixed with seawater and discharged into the North Sea through a long outfall pipeline.

3.4.3.3 Safety of tailings facility and accident prevention

In the design of the TMFs the following factors are considered:

- examination of ground stability
- examination of heap stability
- reduction of permeability of supporting strata if the average permeability coefficient exceeds e.g. 1 x 10^{-9} m/s, but site-specific and depending on the findings of the environmental impact assessment
- avoidance of artificial sealing layers with low shearing strength (has a negative effect on heap stability)
- application of moist tailings but with a moisture content below about 10 %.

Inspections of tailings heaps are routinely carried out by the operator. These include yearly surveillance of the heaps and observation of ditches and basins.

3.4.3.4 Closure and aftercare

The **rehabilitation and after-care**, the description of the actual state and future development of plant-location including the tailings management area, and the closure of the mining operation is compiled into a detailed plan.

After permission of the monitoring and surveillance plan for closure, the operation facilities from the plant must be removed. However, the tailings heaps remain unchanged for a long period of time. A fund for future maintenance cost is financed from operational costs before closure.

3.4.4 Waste-rock management

Since potash mining is only carried out underground, the amounts of waste-rock are relatively small. The waste-rock remains underground in mined out areas of the mine.

3.4.5 Current emission and consumption levels

The quantities of emissions and effluents vary from mine to mine. They are also in some respect a function of natural conditions - the components of the exploited deposit and the mined minerals. Site-specific contributions - the form of mineralisation, the grade and liberation of the material, the mixture of mineral constituents in the mined deposit - are always unique. Depending on the mined ore and the desired products a process is chosen with solid and liquid tailings in varying proportions. Emissions and effluents are also a function of management and processing method.

3.4.5.1 Management of water and reagents

In general it is possible to dissolve all solid tailings and discharge the resulting solution including insolubles into natural water systems (e.g. marine tailings management in UK).

Tailings heaps generate saline solutions when atmospheric precipitation dissolves the salt. This drainage water is collected in sealed ditches around the tailings heap and pumped into sealed retention basins. From these retention basins the saline water is discharged into natural flowing waters (e.g. rivers) or pumped into the ground (deep well tailings management).

The sealings of ditches and retention basins are inspected to avoid soil and groundwater salinisation. Furthermore the water of groundwater wells in the surrounding of a tailings heap is periodical analysed to verify its quality.

No addition of water is applied for backfilling. In the case of the backfill of slurries at the plant Unterbreizbach, processing brine is combined with solid tailings. The brine is used as a transportation medium only and recycled. Processing brine is re-used for different applications in mineral processing to minimise the consumption of water.

In solid tailings no significant amounts of reagents are detectable. The only reagents used result from the electrostatic separation or the flotation process. These processing methods work with a low content of organic compounds (salicylic acid, fatty amines).

The main components of the liquid brine are inorganic salts, while the presence of organics (TOC) and heavy metals is negligible - a consequence of the deposit-formation by the evaporation of seawater about 250 million years ago.

3.4.5.2 Emissions to air

Dust
There are several approaches to dust reduction:
Pre-primary approaches, which start in the processing to reduce the amount of dust, are the detailed selection of the used processing equipment, e. g. for crushing and grinding.

Furthermore, de-dusted crude salt is separated at the electrostatic separation facility into the dry solid tailings and intermediate fractions.

Primary approaches are all ways of reducing emissions during stacking and can be divided into organisational and technical approaches (Table 3.63). A secondary approach is to stop dumping at stormy weather.

Primary approaches	Organisational	Continuous processing Reduction of transport distances Maintenance Logistic of stacking areas
	Technical	Use of wind protection (e.g. covering of conveyor-belt) Transverse/reverse conveyor-belt Moistening of the solid tailings
Secondary approaches		Stop dumping if wind speed exceeds the pre-determined limit

Table 3.63: Primary and secondary dust reduction approaches

Dry solid tailings from electrostatic separation are moistened indoors. The tailings are transported on conveyor belts and stacked with a moisture content of about 5 – 6 %. This leads to low dust emissions are small because of re-crystallisation of the surface layer.

The only atmospheric pollution is salt dust from stacking tailings on the top of the tailings heap, especially when discharging from a conveyor-belt onto a heap at very high wind. Therefore stacking is stopped automatically, if the wind speed exceeds a pre-determined limit. During recent years the maximum dust detected by several immission measuring stations (dust monitoring and control system) around the tailings heaps showed less than 60 mg/m^2/day. All available immission data show, that no harmful effects on human beings (employees/inhabitants) and the environment could be detected.

Noise

Noise is commonly produced by machinery (e.g. conveyor belts) and vehicles. Belt drives are commonly encapsuled. The use of vehicles in the heap area is low, because of the use of conveyor-belts and the short distance between the heap and the processing plant. Sometimes one or two caterpillars/dozers work on the heap. Traffic for staff and equipment transportation is very limited. During the compilation of data for a permit application for storing solid residue baseline studies of noise impacts were generated, including on-site measurement

3.4.5.3 Emissions to water

The combination of the tailings management methods is site-specific, therefore emissions to water vary from plant to plant.

One approach to reduce emissions to water is to re-use the process brine, as applied successfully at several plants. Only the surplus, which cannot be re-used, e.g. because of saturation with magnesium containing salt, is either pumped into deep wells or discharged into the river system.

Surface run-off collected from the heaps is generally not re-used in the mineral processing.

No noticeable amounts of trace elements, heavy metals or organic substances can be detected in the surface run-off from the heaps. The main components of surface run-off are salts such as sodium, magnesium, potassium and calcium chlorides and sulphates. The volume of surface run-off from the heap depends on land consumption, precipitation (per year) and the components of the salt tailings.

If the mineral kieserite ($MgSO_4 \cdot H_2O$) is one component of the mined salt, some kieserite will be in the tailings, too. Upon contact with rainwater kieserite is hydrated and thus binds some of the rainfall. In consequence the water binding capacity of a tailings heap from potash mining is strongly dependent on the specific minerals content.

A second important factor influencing the amount of surface run-off is the evaporation of water, which depends on several factors like temperature, humidity, wind speed, colour of the tailings, sunshine intensity etc.

3.4.5.4 Soil contamination

No data has been supplied for this section. Please provide information.

3.4.5.5 Energy consumption

No data has been supplied for this section. Please provide information.

3.4.6 Costs

No data has been supplied for this section. Please provide information.

3.5 Coal: Mineral processing, tailings and waste-rock management

3.5.1 Mineralogy and mining techniques

All of Germany's hard coal resources are carboniferous in age. While the Saar and Ibbenbüren basins represent remnants of larger coalfields, the Ruhr contains massive resources that dip towards the North Sea. Lying at a depth of around 1200m in the north of the active coalfield, the strata have been fragmented by major regional folding and faulting. Conditions in the Saar basin are more complex than in the Ruhr.

The high-quality coking, gas and steam coals typically contain 6 – 9 % ash, and less than 1 % sulphur, although some seams require extensive washing before sale. The Niederberg mine and the Ibbenbüren deposit contains anthracite.

Longwall faces of up to 400 m are now in service. Seams worked range in thickness from 1.0–4.8 m, with ploughs being used in the thinner seams and shearers in thicker applications.

Hard coal in the Czech Republic mainly occurs in the Upper Silesian Basin. The major fault, called the Orlova fault, divides the Czech part of the Upper Silesian Basin into the western section (the Ostrava part), which is older and of paralic character of sediments and coal seams, and the eastern section (the Karviná part), which exhibits limnic character of the sediments as well as of coal. The western part consists of several tens of thin coal seams of high grade coking coal, whereas the eastern part is characterised by abundant thick seams containing mixed coking coal and highly volatile steam coal. Some of the characteristics of hard coal include a carbon content of more than 73.4 %, less than 50 % volatile matter, and a dry (ash free) calorific value that exceeds 24 MJ/kg.

Mining in the Ostrava part of the basin has reached depths of about 1000 m, which together with complex and unfavourable mining and geological conditions makes economic mining extremely difficult. Consequently, the Ostrava mines were gradually abandoned. The majority of mines in the eastern part have enough reserves which can be extracted with much lower costs. However, this coal is of low grade, as far as coking properties are concerned.

Relatively large reserves of coal were verified south of the original Upper Silesian basin, particularly near Frenštát pod Radhoštěm, where carboniferous sediments are buried under Miocene sediments and the Beskydy napes. Here, the coal would be extracted from the depths of 800 to 1300 m under difficult geological and mining conditions. As the deposit is situated on the border of protected landscape area, there can arise conflicts of interests with Beskydy protection in the case of mining.
[83, Kribek, 2002]

Most operations in EU15 and Candidate Countries are based around longwall mining, using both shearers and ploughs for production. Most mines operate in several seams, with each unit operating several faces. An increasing number of longwalls are controlled remotely from surface, high levels of automation allowing saleable output of up to 20000 t/d per face [79, DSK, 2002], [83, Kribek, 2002].

In Spain coal is also mined in open pits [84, IGME, 2002]

3.5.2 Mineral processing

In general after the extraction step, paricle size ranges from pieces of more than one metre in diameter to ultrafine grains (< 5µm). In the three German coalfields of Ruhr, Saar and Ibbenbbüren a wide range of coal qualities are mined, from anthracite at the Ibbenbüren colliery with 6 % of volatile matters (VM) up to the high volatile bituminous coals of the Ensdorf underground mine with more than 36 % VM. In 2000, 12 coal processing plants with feed rates between 950 and 1700 t/h were in operation in these coalfields. [79, DSK, 2002].

Typically the coarse fraction (> 30 - 60 mm) is separated from the heavier gangue by dense media separation (e.g. drewboy) and the fine coal (> 0.5 mm) is separated by flotation. The mid-size coal fraction is recovered in jigs.

A typical flowsheet can be seen in the following figure.

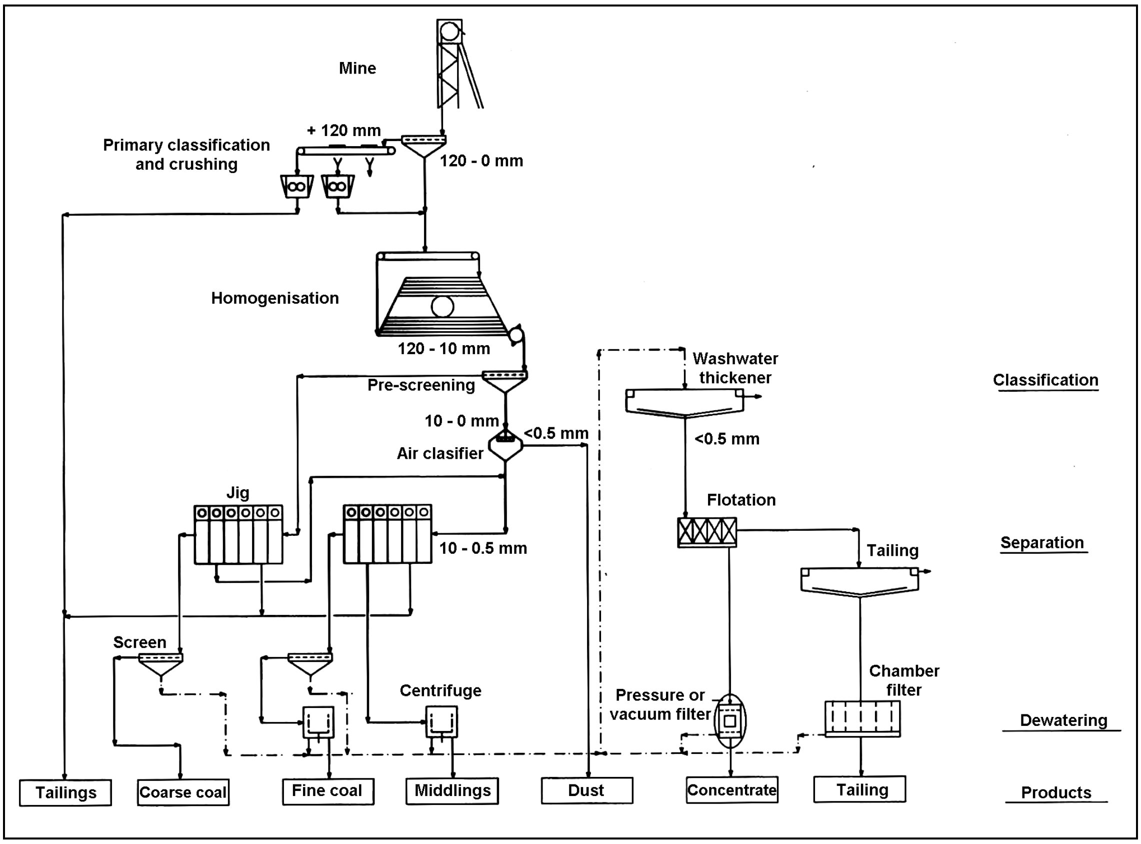

Figure 3.72: Standard flowsheet for coal mineral processing
[79, DSK, 2002]

There is also one site that uses hydrocyclones instead of flotation of the fines [83, Kribek, 2002].

3.5.3 Tailings management

3.5.3.1 Characteristics of tailings

Typically tailings from the Ruhr, Saar and Ibbenbüren areas consist of 55 - 60 % clay shale, 30 - 40 % of sandy clay shale and 5 to 15 % sandstone (Prosper-Haniel mine) [79, DSK, 2002].

The fine flotation tailings from Ruhr, Saar and Ibbenbüren coal mines < 0.5 mm with a > 77 % solids and a homogeneous mineralogical composition were tested in detail. In physical and chemical tests with long-term considerations including environmental impact assessment it has been proven that flotation tailings can be used for the construction of surface liners even achieving the stringent requirements of the German Technical Standard for the construction of liners for landfills [80, DSK, 2002]. In laboratory tests pure flotation slurries from hard coal processing can reach k_f –coefficients of around 5 x 10^{-9} m/s. In-situ tests resulted in k_f – coefficients of ~ 2 x 10^{-7} m/s. These k_f - coefficients do not reach values required by TASi/LAGA standards for mineral liners (k_f = 5 x 10^{-10} m/s) and surface seals for landfill category I (k_f = 5 x 10^{-9} m/s). [79, DSK, 2002].

In the Ostrava and Karviná areas the coarse tailings are handled on heaps and the fines from flotation are send to basins or ponds. In one case a level of radioactivity of 75.5 ± 6.9 Bq/kg was measured in the tailings [83, Kribek, 2002].

In addition to fine coal, here is a partial list of typical products used in coal mineral processing plants:

- anionic flocculent - work with negatively charged ions
- cationic flocculent - work with positively charged ions
- reagent flocculent
- lime
- natural and modified starches
- caustic starch
- sulphuric acid - pH adjuster
- alum (aluminium sulphate) - pH adjuster
- anhydrous ammonia.

[81, MSHA, 2002]

3.5.3.2 Applied management methods

In the Ruhr, Saar and Ibbenbüren areas a total of 23 tailings heaps and 7 tailings ponds are currently in operation [79, DSK, 2002].

Considerable amounts of tailings from coal mining have to be handled, since they can amount up to around 50 % of raw production. Principally, three management options are available:

- internal application, i. e. for underground backfill or construction projects linked to mining operations (e. g. compensation measures for mining-induced ground subsidence such as heightening of bridges or embankments)
- external application, i. e. commercial products, such as bulk mass material or base material in construction sector and civil engineering
- management on dumps and in ponds.

As a rough guide, around one quarter of all rock and tailings in the Ruhr, Saar and Ibbenbüren area is being used for internal and external purposes, whilst the remainder has to be deposited (see figure below).

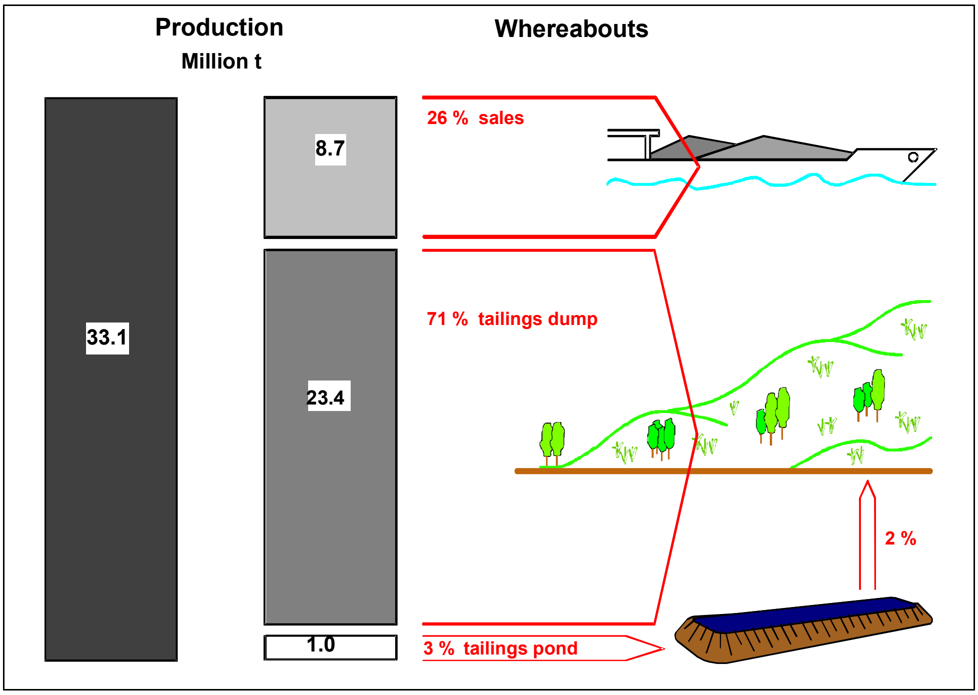

Figure 3.73: Tailings production and applied management methods in the Ruhr, Saar and Ibbenbüren area
[79, DSK, 2002]

Flotation tailings, which are around 13 to 18 % of total tailings, are transported with trucks on public roads [79, DSK, 2002].

Fine tailings < 0.5 mm from flotation are thickened to 40 - 51 %. In order to make them suitable for deposition, however, they have to be further dewatered. This is done predominantly in chamber filter presses with more than 1000 m² of filter area. Occasionally, also screen bowl centrifuges are in use for draining of flotation tailings. Water content of the finest tailings drained in this way, however, is approximately twice as high as at tailings dewatered in chamber filter presses.

In Spanish coal mines the coarse material in discarded onto heaps, used as backfill or as filling material in other areas. Flotation slurries are either
- filtered and sold or
- filtered and discarded with the coarse tailings or
- discharged as slurries into tailings ponds.

[84, IGME, 2002]

3.5.3.2.1 Backfilling

One of the obvious options for tailings management is backfilling, i. e. transporting tailings back into underground working areas and filling the previously created cavities, called the "gob" or "goaf". In coal mining backfilling is dependent on a series of geological and technical conditions in order to be applied successfully in economical terms. Since clay content in tailings from hard coal can cause blockages in pipelines when pumped with water, in the Ruhr, Saar and Ibbenbüren areas pneumatic backfill methods have been favoured in the past.

The currently unfavourable economic situation of European coal mining (highly subsidised) in combination with a reduction of coal extraction from steeply dipping seams has lead to a decrease of backfill operations since late 1980's in the Ruhr, Saar and Ibbenbüren area. In the 1970's, backfill methods for flat dipping seams had been developed allowing integration of backfill technique into extraction, conveying and face support technology. Limits of application were identified for pneumatic backfill operations with low seam dips and seam thickness of less than 1.9 m. Several approaches, aiming at applying backfill methods in smaller coal seams have failed.

Investment costs for an adequate backfill infrastructure in Ruhr, Saar and Ibbenbüren collieries have been calculated at up to EUR 40 million. Additional investigations showed, that operational costs amount to EUR 20 per tonne of coal produced, split equally to staff and material costs.

The application of backfill technique results in a considerable burden in economic terms owing to the large investment and increased operational costs leading to performance losses in extraction operations. Backfill operations, therefore, are considered for those cases, in which they are economically tolerable and necessary for ecological reasons with regard to ground surface situation. Backfilling is currently not practised in the Ruhr, Saar and Ibbenbüren area.

Below, pro and cons of backfill method in hard coal mining are summarised.

Potential advantages of pneumatic backfill technique, such as

- reduction in surface subsidence of approximately 50 % as compared to caving and therefore reduction of internal and external mining damages on the surface
- reduction of tailings volumes to be managed on surface
- extension of operational lifetime of existing or planned dump sites
- cost savings at surface management of tailings
- better handling of rock strata pressure
- advantages for mine ventilation system, improving climatic conditions underground
- under certain circumstances reduction of underground water intake

have to be considered in the light of a series of disadvantages:

- usually, subsidence movements last longer as compared to caving method (can cause delay in surface rehabilitation works or repeated damages at already repaired objects)
- idle times at coal production owing to disruptions in backfill operations (e. g. damages at back-fill pipeline); this can cause unfavourable extraction dynamics, i. e. change of load (delay/acceleration of the movement processes covering rock strata and surface
- backfill panels adjacent to caving panels create effects of a pillar edge, equivalent to an elongation peak on ground surface
- increased rock burst danger as compared to caving method
- installation of backfill system at an existing colliery is very difficult and expensive (dimensions of underground roadways and entries)
- need of a second conveyor system for tailings transport in opposite directing to coal transport requires large investment
- exact synchronisation of tailings supply with coal production necessary
- backfill method limits face operations in terms of advance speed and panel production capacity, requires another panel at times
- additional hazard potential through tailings in shafts with personnel transport
- increase of production costs by at least EUR 20 per tonne of coal through backfill operations.

[79, DSK, 2002]

3.5.3.2.2 Tailings heaps

As shown in Figure 3.73 some 23.4 M tonnes of tailings – out of a total of 33.1 M tonnes – from the Ruhr, Saar and Ibbenbüren area were discarded onto tailings heaps.

The development of tailings heap design in the Ruhr, Saar and Ibbenbüren areas is shown in the following figure.

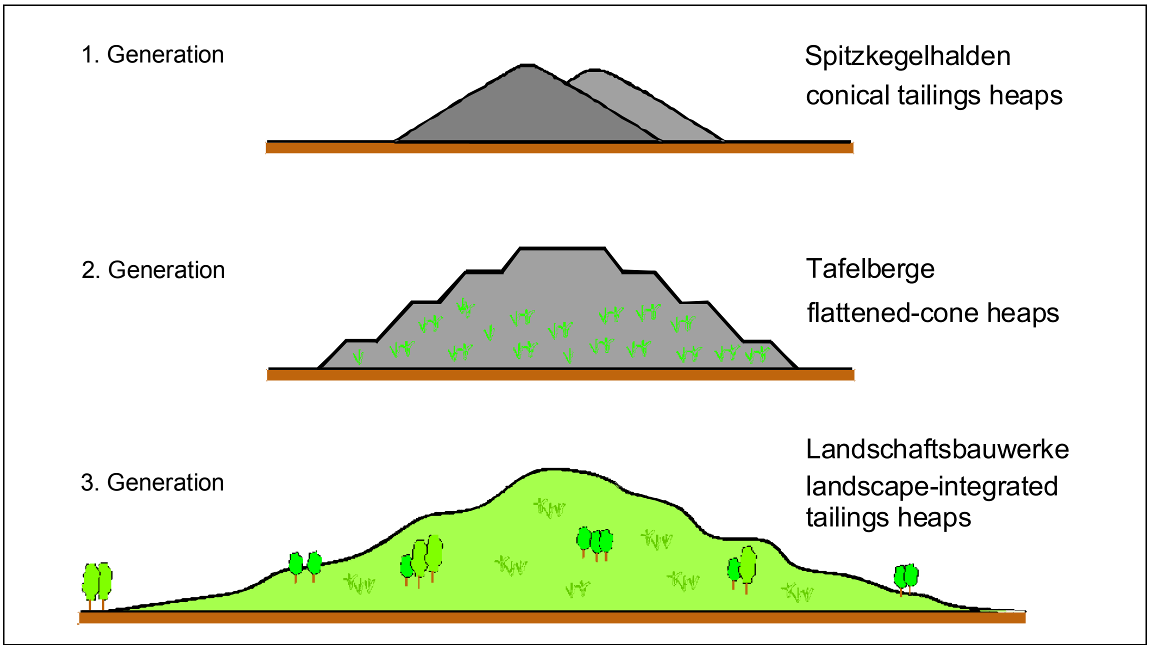

Figure 3.74: Development of tailings heap design in the Ruhr, Saar and Ibbenbüren areas [79, DSK, 2002]

Since the 1970's, the third generation of tailings dumps - so-called landscape-integrated earth constructions – has been established. Since then these heaps have been accepted as essential landscape elements in the densely populated industrial regions of Ruhr and Saar owing to their high recreational and ecological value.

Principally, tailings are dumped onto the heaps in layers. Thickness of layers ranges from 0.5 to 2.0 m, in special cases up to 4.0 m. Compaction is achieved by way of the trucks' rolling wheels and via vibration crawlers. By doing so, a so high compaction is achieved that no oxygen or precipitation can penetrate into the dump body and, thus, the hazard of acidification by pyrite oxidation is minimised.

The principle of erecting a tailings heap is shown in Figure 3.75, displaying four construction steps within the spreading phases. The first step is the construction of an outer rim wall, which is immediately revegetated, serving as shield for subsequent deposition of tailings in the inner zone.

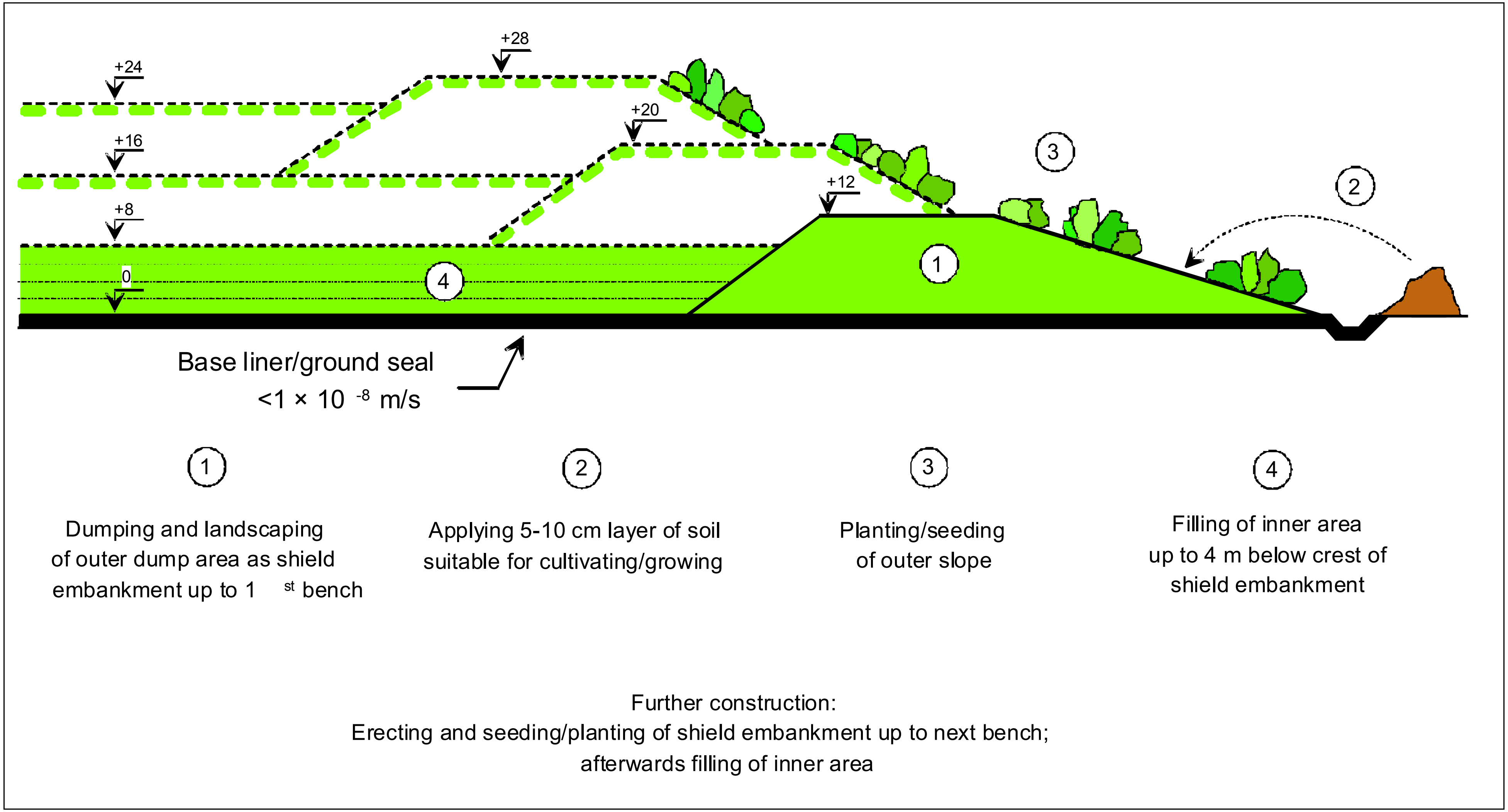

Figure 3.75: Schematic drawing of tailings heap construction in the Ruhr, Saar and Ibbenbüren areas [79, DSK, 2002]

It is known from investigations - by lysimeter tests -, that percolating water from coal tailings heaps can contain dissolved solids. Results from the above tests showed that chloride can be washed out off the dump and sulphate, calcium and magnesium can increase owing to pyrite oxidation. Through acids generation in that context, trace elements could be mobilized in case of decreasing pH-values as well as of falling buffer capacity of tailings material or aquifer, respectively.

As a consequence, groundwater protection is the main environmental concern when constructing and operating a heap. Nowadays, four main measures are used to protect ground water from possible heap effluents. In Figure 3.76 these principal options for protecting ground and surface water systems in the context of tailings dump design, that are available are shown.

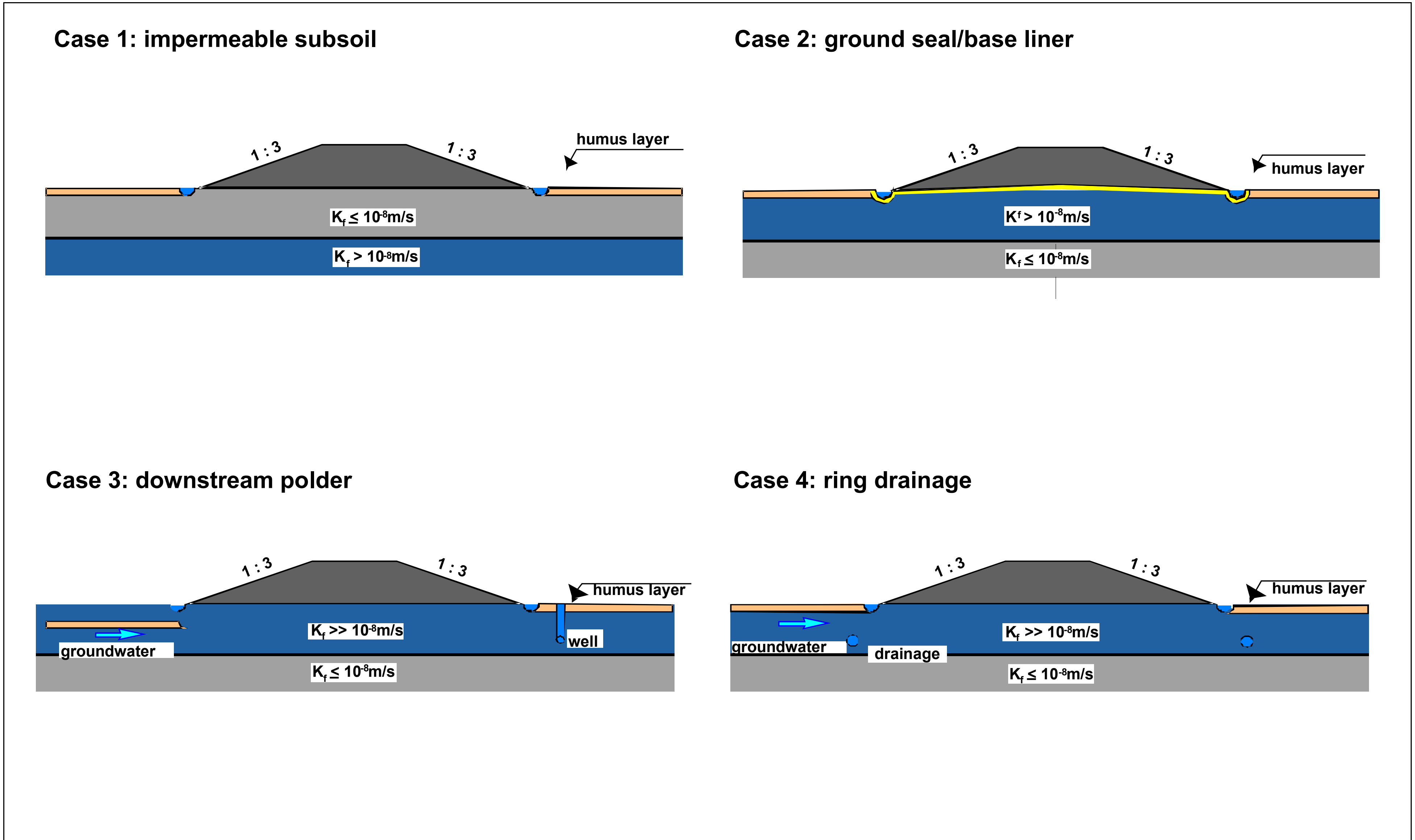

Figure 3.76: Tailings heap design – options for avoiding negative effects on ground and surface water system [79, DSK, 2002]

Depending on site-specific circumstances, specific solutions are chosen, i. e. single measures or a combination of different measures.

Recent findings obtained during grounding works for wind power stations on tailings heaps demonstrated the effectiveness of these operational measures. The inner body of the heap was strongly compacted and absolutely dry.
[79, DSK, 2002]

The coarse tailings, typically several hundred thousand tonnes per year, from coal mines in the Ostrava and Karviná areas are transported to the heap on conveyor belts or with trucks. In other cases they are used in the reclamation of old tailings basins or for landscaping of subsidence areas.
[83, Kribek, 2002]

3.5.3.2.3 Tailings basins/ponds

Often the fine slurry from flotation of coal mines is pumped to sedimentation basins (e.g from subsidence) or engineered ponds. The settling of tailings is carried out in several ponds/basins in series. The settled tailings are excavated periodically and refloated or sold. The clarified overflows are mostly recycled to the mineral processing plant [83, Kribek, 2002], [84, IGME, 2002].

3.5.3.3 Safety of tailings facility and accident prevention

The Ostrava and Karviná area has a high seismic risk. Therefore seismic events are monitored [83, Kribek, 2002].)

3.5.3.4 Site closure and aftercare

Land availability is very limited in the densely populated areas of the Ruhr and Saar coalfields. Areas under use for industrial purpose such as tailings management have to be reintegrated into the landscape as rapidly as possible.

The dumped tailings are sampled immediately after dumping, after the two years and after three years as far as required. Per each 2500 m² dump area, three samples from depths between 0 and 20 cm are taken and blended for a representative mixed sample. One sample is taken from a depth between 40 to 50 cm. Investigation of sample material includes pH-value determination to identify the acidification grade, total sulphur content (1st sample) and total alkalinity content. For the second samples, the contents of P_2O_5, potassium, calcium and magnesium accessible to plants are determined. These results are taken into account for the soil cover and the revegetation.
[79, DSK, 2002]

Revegetation can be accelerated by different measures:

- loosely tipping tailings to two metres depth in the outer area in order to accommodate good root formation
- blending with materials such as fly ash from power plants, lime and dolomite rock buffering capacity can be increased, water retention ability is improved and nutrient capacity is increased
- applying a 5 to 10 cm thick layer of arable soil. For a quick and lasting vegetation, either a thick (around 1.8 m, when tailings properties require that option) or a thin earth layer (5 to 10 cm) are favoured options. In most cases, such soils are available in sufficient quantities, herbs find enough potential for root formation and shrubs are planted directly into the tailings. This has the advantage, that the young plant can accustom itself to soil conditions available in tailings material and leads to a natural root formation, which can provide enough moisture to the plant also in dry seasons

- applying mineral fertilising lack of nutrients can be compensated. Organic fertilisers contain nutrients, which are organically bound; they are released by microbial degradation. Additionally, they improve soil structure, activate soil organisms and enhance water retention capacity
- applying surface mulching to enhance protection against adverse climatic conditions as well as for improvement of humus enrichment and water retention capacity, especially in the early stages of vegetation. Mulching materials can be straw or hay, but also wood chaff
- irrigating in extremely dry seasons at night time only.

After completion of slope areas the dump surface is sown with herbs seed. The herbs layer assists the heap's integration into the landscape, prevents erosion to a major extent and contributes to humus formation in the uppermost soil layer. Sizing and composition of seed mixture is dependent on local situation at individual dumps, on ground structure and on climatic influences. In the case of wet sowing, water is used as carrier. Apart from the seed, fertiliser, soil amelioration agents and mulch, mixed with water, can also be applied.

In a next step shrubs and trees are chosen only after evaluation of soil investigations. Selecting the plants and designing the planting scheme is done in close co-operation with forestry authorities. Plant material, in most cases, is taken from tree nurseries after a growing period of three years and planted with a narrow spacing of 1 x 1 m.

Apart from the vegetation measures described above, by landscaping wet and dry bi-otopes, small water courses as well as by creating areas left to natural succession, reclamation in the Ruhr, Saar and Ibbenbüren areas aims at creating the basis for a variety of fauna and flora habitats.
[79, DSK, 2002]

A regional closure plan for the landscaping of mines and tailings management facilities in the Ostrava and Karviná area has been developed [83, Kribek, 2002].

3.5.4 Waste-rock management

The small amounts of waste-rock from underground operations are managed with the coarse tailings on the heaps.

Please provide information for waste-rock information of open pit coal mines.

3.5.5 Current emission and consumption levels

3.5.5.1 Management of water and reagents

The clarified water from basins/ponds in the Ostrava and Karviná area are re-used in the mineral processing plant. Surplus water is discharged is discharged to surface water.

In flotation the agent Flotalex, a mixture of alcohols and mineral oil, is used in concentrations of 0.25 - 0.35 kg/t. As a flocculant an agent called Sokoflok 22 is added.
[83, Kribek, 2002].

3.5.5.2 Emissions to air

To minimise dust and noise immissions from dumping tailings transport and spreading operations, ramps and working benches are transferred into the heap's inner area as far as possible and are shielded by embankments or hollows [79, DSK, 2002].

3.5.5.3 Emissions to water

Fine tailings from flotation are often stored in ponds and basins (e.g. Ostrava and Karviná area). Most of the clarified water is re-used in the mineral processing plants. However, in some cases surplus water is discharged to surface water. The amounts of discharge per year and the concentrations of emissions to surface water are shown in the following table.

Parameter	Unit	Site					
		Paskov	CSA	Lazy	Dukla	CSM	
Discharge	Mm^3	0.2	2.0	1.6	4.0	0.27	
$ChSK_{Cr}$	mg/l	22208		19.19	50.91	1920.2	
$ChSK_C$	mg/l		16985				
BSK5	mg/l		2333	4.34	6.54	20.65	
RL-V	mg/l		1310				
RAS	mg/l	687833					
NEL	mg/l					2.5	
NL	mg/l	131667	7166	9.88	20.58	285.4	
$P_{celk.}$	mg/l	0.04					
$N\text{-}NH_4$	mg/l	0.06	0.33	0.2	1.48		
Cl	mg/l		382.5				
Cl_2	mg/l	156167					
SO_4	mg/l	204.5	290.5				
PO_4	mg/l		0.055				
FN	mg/l		0.1				
Fe	mg/l			0.17	0.22		
Mn	mg/l			0.09	0.14		
Hg	µg/l	0.9					
Cd	µg/l	0.5		<0.005	<0.005		
CN	µg/l		6				
FN	mg/l		0.1				
pH			8	8	7.61		

Table 3.64: Amount of discharge and concentrations of emissions from tailings ponds/basins in the Ostrava and Karviná area in 2000
[83, Kribek, 2002]

Information from other coal mining regions to be added to above table. Please provide information.

3.5.5.4 Soil contamination

No data has been supplied for this section. Please provide information.

3.5.5.5 Energy consumption

No data has been supplied for this section. Please provide information.

3.5.6 Costs

No data has been supplied for this section. Please provide information.

3.6 Oil shale: Mineral processing, tailings and waste-rock management

No data has been supplied for this section. Please provide information.

3.6.1 Mining techniques

3.6.2 Mineral processing

3.6.3 Tailings management

3.6.3.1 Safety of tailings facility and accident prevention

3.6.4 Waste-rock management

3.6.5 Current emission and consumption levels

3.6.5.1 Management of water and reagents

3.6.5.2 Emissions to air

3.6.5.3 Emissions to water

3.6.5.4 Soil contamination

3.6.5.5 Energy consumption

3.6.6 Costs

4 TECHNIQUES TO CONSIDER IN THE DETERMINATION OF BAT

This section presents a number of techniques for the prevention or reduction of emission and techniques to prevent or mitigate accidents in accordance with Section 6.3 of the Communication (COM (2000) 664). They are all currently available and applied. Examples are given in order to demonstrate techniques, which illustrates a high environmental or safety performance. The techniques that are given as examples depend on information provided by the industry, European Member States and the valuation of the European IPPC Bureau.

The following list shows how techniques to consider in the determination of BAT can be described:

EXAMPLE
- description
- main achieved environmental or safety benefits
- applicability
- cross-media effects
- operational data
- economics
- driving force for implementation
- reference literature.

Under Section 4.2.1 an example discussion of BAT candidate techniques is included as guidance.

4.1 Techniques to reduce emissions

4.1.1 Techniques to prevent or reduce ARD generation of tailings and waste-rock

e.g. selective management of ARD and non-ARD waste-rock, de-pyritisation of tailings, test methods

Please provide information/opinion on this subject.

4.1.2 Techniques to reduce reagent consumption

e.g. use of dry separation methods instead of flotation, improved on-line analysis and automatic reagent adjustment

Please provide information/opinion on this subject.

4.1.3 Techniques to reduce dusting

e.g. changing spigotting points, progressive revegetation of heaps, irrigation

Please provide information/opinion on this subject.

4.1.4 Techniques to reduce noise emissions

technical (e.g. housing) and organisational

Please provide information/opinion on this subject.

4.1.5 Techniques to reduce emissions to water

e.g. collecting and treating seepage water, re-use of reagents and process water, groundwater monitoring

Please provide information/opinion on this subject.

4.1.5.1 Cyanide

e.g.
- techniques to minimise use of cyanide (on-line analysis combined with automatic dosing instrumentation
- alternative leaching techniques
- techniques to destroy/recycle CN.

Please provide information/opinion on this subject.

4.1.5.2 Xanthates

Please provide information/opinion on this subject.

4.1.5.3 Thiosalts

Please provide information/opinion on this subject.

4.1.5.4 Arsenic removal

Please provide information/opinion on this subject.

4.1.6 Techniques to reduce seepage into ground

e.g. liners

Please provide information/opinion on this subject.

4.1.7 Techniques to reduce footprint of TMF

e.g. backfill, pre-separation in the mine

Please provide information/opinion on this subject.

4.1.8 Techniques to maximise backfill

e.g. using paste backfill instead of hydraulic backfill

Please provide information/opinion on this subject.

4.1.9 …

Please suggest further techniques

4.2 Techniques to prevent accidents

4.2.1 Techniques to construct and raise dams

Possible options for dam construction have been discussed in Section 2.4.1.1.2. The following table summarises the different ways of constructing tailings dams.

Dam type	Applicability	Discharge suitability	Water storage suitability	Raising rate restrictions	Construction material	Seismic resistance	Relative dam cost
Conventional dam	Suitable for any type of tailings	Any discharge procedure suitable	Good	Entire embankment constructed initially	Natural soil borrow	Good	High
Upstream	At least 40 - 60 % sand in whole tailings. Low pulp density desirable to promote grain size segre-gation	Peripheral discharge and well controlled beach necessary	Not suitable for significant water storage	Less than 5m/yr most desirable	Natural soil, sand tailings or waste-rock	Poor in high seismic areas	Low
Downstream	Suitable for any type of tailings	Varies according to design details	Good	None	Sand tailings or mine wastes if production rates are sufficient. Otherwise natural soil.	Good	High
Centreline	Sands or low plasticity fines	Peripheral discharge of at least a normal beach necessary	Not recommended for permanent storage. Temporary flood storage acceptable with proper design details	Height restrictions for individual raises may apply	Sand tailings or waste-rock if production rates are sufficient, otherwise natural soil	Acceptable	Medium

Table 4.1: Comparison of dam construction techniques
[11, EPA, 1995]

The upstream method is the cheapest method, because tailings can be used and the least amount of material is necessary for a given raise. The main disadvantage of this method has in the past been the physical stability and its susceptibility to liquefaction. Care must be taken in the design in order to control the phreatic surface, which can be achieved by correct drainage. Also, the exposed tailings, used to build the dam should not have ARD potential.

The finer tailings discharged as cyclone overflow are often intrinsically weaker than the coarser fraction and less dense since it is either deposited under water or may not be able to drain, consolidate or desiccate. Their less permeable nature may also result in a high phreatic surface indicating the development of pore pressures. The material can form a weak zone with respect to downstream slope stability and the upstream method ins consequently only used where it can be demonstrated that the finer material is rendering strong enough by drainage and/or desiccation to provide adequate support to the slope. Stability is a major concern in areas of high seismicity.

If the upstream method is applicable it can even be favourable over the other methods, especially the downstream method, because the phreatic surface will tend to remain low. The following figure illustrates this by comparing an upstream dam constructed out of cycloned tailings and a downstream dam of the water retention type using an impermeable core.

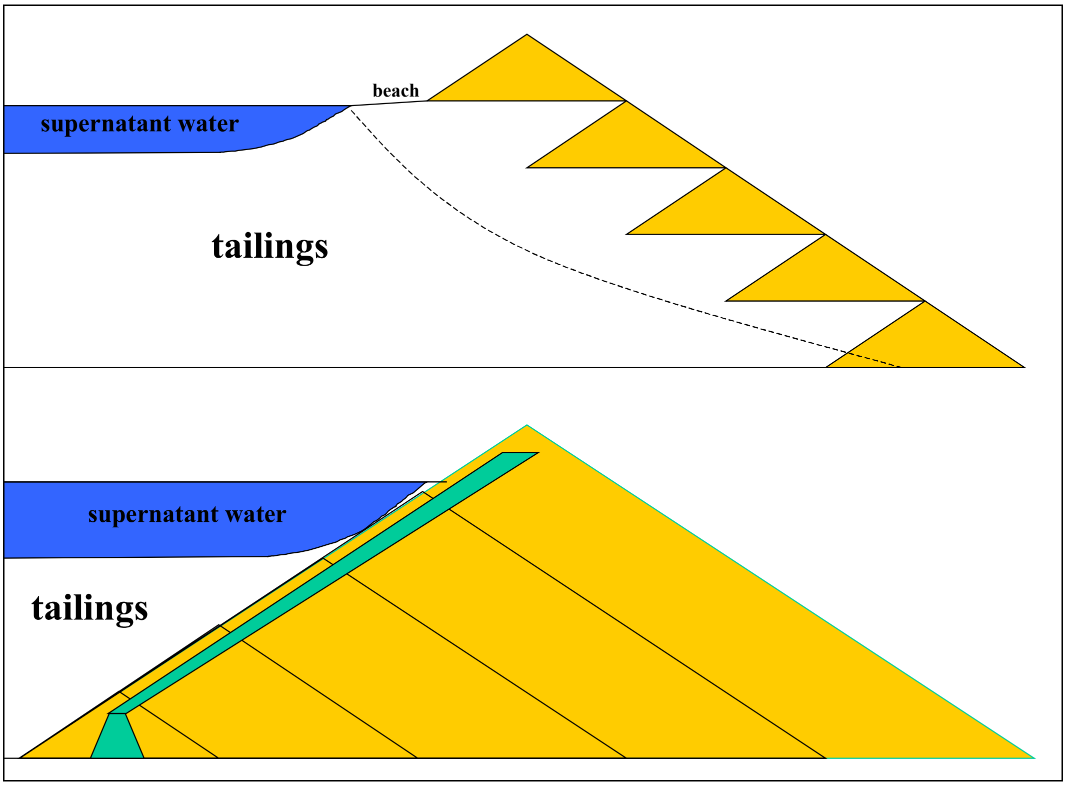

Figure 4.1: Comparison of upstream and downstream method of tailings dam construction

It can be seen that the in the case of the downstream method the impermeable core keeps the supernatant pond in place. In case of increased leakage through the core the stability of the dam would be jeopardised.

The upstream method is not suitable for wet covers, at least as illustrated above, because it is designed and constructed to dewater rapidly. This contradicts the objective to store water on top of the tailings.

The benefit of using the high cost option of building water retention type dam is that the dam is constructed initially so that quality control of the material is easy to realise. Also any further raises are built in on construction project. This is different in methods were the dams are constantly raised with cycloned tailings. In these case the appropriate characteristics have to be constantly monitored and evaluated.

In many cases the centreline method seems to be a good compromise between seismic risk and the costs.

4.2.2 Materials to raise dam

e.g. borrow material, tailings or waste-rock

Please provide information/opinion on this subject.

4.2.3 Outlet arrangements

e.g. culvert, barge, overflow

Please provide information/opinion on this subject.

4.2.4 Techniques to monitor seepage through dam

e.g. visual monitoring of seepage water, dam surveillance, piezometers (manual, automatic)

Please provide information/opinion on this subject.

4.2.5 Techniques to monitor stability of dams and heaps

Surveillance, piezometer, lysimeters, inclinometers

Please provide information/opinion on this subject.

4.2.6 …

Please suggest further techniques

Please provide information/opinion on this subject.

4.3 Techniques to mitigate accidents

e.g. emergency storage ponds, second dam wall

Please provide information/opinion on this subject.

5 BEST AVAILABLE TECHNIQUES FOR THE MANAGEMENT OF TAILINGS AND WASTE-ROCK IN MINING ACTIVITIES

6 EMERGING TECHNIQUES FOR THE MANAGEMENT OF TAILINGS AND WASTE-ROCK IN MINING ACTIVITIES

7 CONCLUDING REMARKS

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GLOSSARY

1. GENERAL TERMS, ABBREVIATIONS, ACRONYMS AND SUBSTANCES

ENGLISH TERM	MEANING
AAAAAAAAA	
Acid generation	Production of acidity irrespective of its effect on the adjacent pore water or whether the material is net acid producing or neutralising.
Acid mine drainage (AMD), Acid rock drainage (ARD)	Acidic Drainage stemming from open pit, underground mining operations, waste-rock or tailings facilities that contains free sulphuric acid and dissolved metals sulphate salts, resulting from the oxidation of contained sulphide minerals or additives to the process. The acid dissolves minerals in the rocks, further changing the quality of the drainage water.
Acid Potential (AP)	Maximum potential acid generation from a sample. The calculation of AP (or MPA) is an integral part of acid/base accounting.
Acidity	Measure of the capacity of a solution to neutralise a strong base.
Air classifier	Machine Equipment to separate dust (<0.05 mm)fine particles from the dry input material (<10 mm) or equipment to remove fine and coarse fractions from an air stream.
Alkali	Proton acceptor. A substance that, more or less readily, takes up hydrogen ions in a water solution.
Alkalinity	Measure of the capacity of a solution to neutralise a strong acid.
Anaerobic	A biological process which occurs in the absence of oxygen.
Appurtenant works, Auxiliary works, Appurtenances	All structures, components and facilities functionally pertaining to the tailings dam, including, but not limited to, spillways, decant towers and pipelines, reclaim pumps, water conduits, diversion structures, etc.
Aquifer	A water-bearing layer of rock (including gravel and sand) that will yield water in usable quantity to a well or spring.
BBBBBBBBBB	
Backfill	Reinsertion of materials in extracted part(s) of the ore body. Materials used for backfilling can be waste-rock or tailings from the mineral processing plant. In most cases backfill is used to refill mined-out areas for in order to ▪ assure ground stability ▪ prevent or reduce underground and surface subsidence ▪ provide roof support so that further parts of the ore body can be extracted and to increase safety ▪ provide an alternative to surface disposal ▪ improve ventilation.
Bio-availability	Property of a substance which makes it accessible and potentially able to affect an organism's health. Depends on site-specific conditions.
Bio-leaching	Process in which minerals are dissolved with the aid of bacteria.
Blending	Mixing of the raw material to get input material with a steady quality for subsequent processes.
BOD	Biochemical oxygen demand: the quantity of dissolved oxygen required by micro-organisms in order to decompose organic matter. The unit of measurement is mg O_2/l. In Europe, BOD is usually measured after 3 (BOD_3), 5 (BOD_5) or 7 (BOD_7) days.
BREF	BAT reference document

CCCCCCCCCC	
Chamber filter press	Equipment to de-water the fine particles in a slurry.
COD	Chemical oxygen demand: the amount of potassium dichromate, expressed as oxygen, required to chemically oxidise at ca. 150 °C substances contained in waste water.
Comminution	Size reduction of an ore by crushing and/or grinding to such a particle size that the product is a mixture of relatively clean particles of mineral and gangue. In order to produce a relatively pure concentrate, it is necessary to grind the ore fine enough to liberate the desired minerals.
Compaction	Process resulting in a reduction in volume. The change typically results from externally applied loads, creating tighter packing of the solid particles. In fine soils in particular, this requires an egress of pore water. Greater compaction often results in increased consolidation.
Concentrate	Marketable product after separation in a mineral processing plant with increased grade of the valuable mineral.
Cross-media effects	the calculation of the environmental impacts of water/air/soil emissions, energy use, consumption of raw materials, noise and water extraction (i.e. everything required by the IPPC Directive)
Crushing	Comminution process that reduces the particle size of run-of-mine ore to such a level that grinding can be carried out. This is accomplished by compression of ore against rigid surfaces, or by impact against surfaces in rigidly constrained motion path.
Cyanidation	Method of extracting gold or silver from crushed or ground ore by dissolution in a weak solution of typically sodium but also potassium or calcium cyanide. Also known as cyanide leaching. The precious metals are then recovered from the pregnant solution : ▪ either by precipitation on zinc dust (Merril-Crowe process), ▪ or by adsorption on activated carbon inside a column (carbon in leach, (CIL)) or within the pulp (carbon in pulp, (CIP)).
DDDDDDDDDD	
Decant lines	Pipelines that carry water decanted from the tailings pond through, above or around the tailings dam to a downstream collection point.
Decant tower	Intake structure that is raised as the tailings pond rises. The decant tower skims off the clear water from the surface of the tailings pond and carries it away using decant lines.
Decommissioning	Process by which a mining operation is shut down.
De-watering	Process of removing water from an underground mine or open pit, or from the surrounding rock or non-lithified area. The term is also commonly used for the reduction of water content in concentrates, tailings and treatment sludges.
Diffuse emission	Emissions arising from direct contact of volatile or light dusty substances with the environment (atmosphere, under normal operating circumstances). These can result from: ▪ inherent design of the equipment (e.g. filters, dyers…) ▪ operating conditions (e.g. during transfer of material between containers) ▪ type of operation (e.g. maintenance activities) ▪ or from a gradual release to other media (e.g. to cooling water or waste water). Fugitive emissions are a subset of diffuse emissions.
Diffuse sources	Sources of similar diffuse or direct emissions which are multiple and distributed inside a defined area

Diversions	For tailings ponds, diversions are usually relatively small interceptor ditches that collect runoff from the contributing watershed and divert it downstream beyond the tailings pond and dam.
Drainage	Manner in which the waters of an area exist and move, including surface streams and groundwater pathways. A collective term for all concentrated and diffuse water flow.
Drainage chemistry	Concentrations of dissolved components in drainage, including element concentrations, chemical species and other aqueous chemical parameters.
Drowning the beach	Rapid rising of the free water in the tailings pond which covers or floods the semi-pervious upstream beach of the tailings dam and results in a free water surface against the tailings dam.
EEEEEEEEEE	
"End-of-pipe" technique	Technique that reduces final emissions or consumptions by some additional process but does not change the fundamental operation of the core process. Synonyms: "secondary technique", "abatement technique". Antonyms: "process-integrated technique", "primary technique" (a technique that in some way changes the way in which the core process operates thereby reducing raw emissions or consumptions)
EC50	Effect concentration 50. The concentration at which effects are observed in 50 % of the test population after administering a single dose. Effects include the immobilisation of daphnia, inhibition of growth, cell division or biomass production, or the production of chlorophyll by algae.
Ecosystem	Community of organisms and their immediate physical, chemical and biological environment.
Effective neutralisation potential (ENP)	The fraction of the NP, which will neutralise acid generation and acidity inputs maintaining a drainage pH of 6.0 or above.
Effluent	Controlled water discharge into the environment from a man-made structure. For example, the drainage products from a water treatment plant. or liquid produced after processing the mineral, which passes to a water clarification circuit for treatment
EIPPCB	European IPPC Bureau
Emerging techniques	Name of a standard chapter in BREFs
Emission	The direct or indirect release of substances, vibrations, heat or noise from individual or diffuse sources in the installation into the air, water or land
Emission limit values	The mass, expressed in terms of certain specific parameters, concentration and/or level of an emission, which may not be exceeded during one or more periods of time
Environment	Interrelated physical, chemical, biological, social, spiritual and cultural components that affect the growth and development of living organisms.
EOP	End-of-pipe
Erosion	Detachment and subsequent removal of either rock or surface material by wind, rain, wave action, freezing, thawing and other processes.
Europe	Current EU Member States (EU15) and EU Enlargement Candidate Countries (see Section 2 of this glossary)
Evaporation	Physical process by which a liquid is changed into a gas.

Existing installation	An installation in operation or, in accordance with legislation existing before the date on which this Directive is brought into effect, an installation authorised or in the view of the competent authority the subject of a full request for authorisation, provided that that installation is put into operation no later than one year after the date on which this Directive is brought into effect
Extraction methods	There are basically four methods of extracting ore: ▪ open pit (open cast) mining ▪ underground mining ▪ solution mining ▪ quarrying.
FFFFFFFFFFF	
Financial guarantee	Funds provided through various financial instruments, which may be used by a regulatory authority to offset closure costs.
Flocculant	Substance that causes suspended particles to aggregate or clump. The larger apparent particle size causes the aggregated clumps to settle. Flocculants are used to aggregate small particles whose slow settling rate makes them otherwise very difficult to remove.
Flotation	A form of separation of minerals from gangue based on their different surface reaction to certain reagents (or alternatively based on the interfacial chemistry of mineral particles in solution). Reagents are used to adhere to the target mineral, and render its surface hydrophobic. The target mineral which then rises to the top of the flotation cell with the injected air, where it can be collected as a froth. When the aim is to float the gangue this process is called reverse flotation.
Free CN	The cyanide not combined in complex ions, both the molecular HCN and the cyanide ion [24, BC CN guide, 1992]
Freeboard	Vertical distance (height) between the normal maximum operating level of a pond and the crest of the dam ,the purpose of which is to provide attenuation capacity in times of flood or sudden ingress of water
GGGGGGGGG	
Gangue	That part of an ore that is not economically desirable but cannot be avoided in mining (see Figure G1).
Geochemistry	Study of the distribution and abundance of elements in minerals, rocks, soils, water and the atmosphere.
Geology	Study of the earth, its history and the changes that have occurred or are occurring, and the rocks and non-lithified materials of which it is composed and their mode of formation and transformation.
Gossum	ore within the upper part of a sulphidic ore body that has been weathered to an oxide ore.
Grade	Dimensionless proportion of any constituent in an ore, expressed often as a percentage, grams per tonne (g/t) or parts per million (ppm).
Grinding	Comminution process yielding a fine product (< 1 mm), where size reduction is accomplished by abrasion and impact and sometimes supported by the free motion of unconnected media such as rods, balls and pebbles.
Groundwater	Part of subsurface water in the zone of saturation. Distinct from surface water.

HHHHHHHHH	
Humidity cell test	Kinetic test procedure used primarily to measure rates of acid generation and neutralisation in sulphide-bearing rock.
Hydrogeology	Study of groundwater. A branch of hydrology.
Hydrology	Study of all waters in and upon the earth, including ground water, surface water and precipitation. When used in conjunction with the term hydrogeology, hydrology is more restrictively defined as the study of precipitation and surface waters.
IIIIIIIIIII	
IEF	Information Exchange Forum (informal consultation body in the framework of the IPPC Directive)
Immission	Occurrence and level of polluting substance, odour or noise in the environment
Industrial minerals	Non-metallic ore, non-fuel or non-gemstone rock, mineral or non-lithified material of economic value. Industrial minerals are primarily used for construction or in chemical and manufacturing industries. Examples include barytes, borate, feldspar, fluorspar, kaolin, feldspar, limestone, phosphate, potash, strontianite, and talc..asbestos, gypsum, salt, limestone, baryte, garnet, potash, graphite, mica and talc.
Infiltration	Entry of water into a porous substance.
Installation	Stationary technical unit where one or more activities listed in Annex I of the IPPC Directive are carried out, and any other directly associated activities which have a technical connection with the activities carried out on that site and which could have an effect on emissions and pollution
IPPC	Integrated pollution prevention and control
JJJJJJJJJJJJJ	
Jig	Equipment in which materials are separated in a continuos flow according to their different densities.
KKKKKKKKK	
LLLLLLLLL	
LD50	Median lethal dose (abbreviated MLD or LD50) is the dose required to kill half of the individuals in a group similarly exposed within a specified period of time
Leachate	Solution obtained by leaching; e.g. water that has percolated through soil containing soluble substances and that contains certain amounts of these substances in solution.
Leaching	Passage of a solvent through porous or crushed material in order to extract components from the liquid phase. For example, gold can be extracted by heap leaching of a porous ore, or pulverised tailings. Other methods are tank leaching of ore, concentrates or tailings and in-situ leaching.
Liberation	Release of the valuable mineral(s) from the gangue.
Life-cycle	Design, construction, operation, monitoring, closure, restoration, after-care of a facility
Liquefaction	Phenomenon that occurs in loose saturated soils usually when the excess pore water pressure (e.g. caused by an earthquake) becomes equal to the original confining pressure, and the soil behaves like a dense fluid, unable to resist significant shear stresses.
Lithology	Composition of rocks, including physical and chemical characteristics such as colour, mineralogical composition, hardness and grain size.

Long-term phase	Period of time required, after the end of the rehabilitation phase, for the tailings to become sufficiently inert that they no longer pose any problems to the environment.
Lysimeter	Device for collecting water from the pore spaces of soils and for determining the soluble constituents removed in the drainage
MMMMMMM	
Maximum credible earthquake (MCE)	Hypothetical earthquake that could be expected from the regional and local potential sources for seismic events and that would produce the severest vibratory ground motion at the site.
Mine production	In the case of metals, the amount of metal in the concentrate after production, in all other cases, unless stated otherwise, the amount of concentrate by weight after mineral processing
Mine production	In the case of metals: amount (by weight) of metal in concentrate. In case of industrial minerals: amount of concentrate
Mine stone	Tailings which are used as a product, e.g. as building material for dike construction, road making etc.
Mineral processing (benefication, ore dressing, mineral dressing, milling)	Processes to produce marketable mineral products (concentrates) from ore. This is usually carried out on the mine site, the plant being referred to as mineral processing plant (mill or concentrator). The essential purpose is to reduce the bulk of the ore, which must be transported to and processed by subsequent processes (e.g. smelting), by using methods to separate the valuable (desired) mineral(s) from the gangue. The marketable product of this is called concentrate, the remaining material is called tailings. Mineral processing includes various procedures that rely on the mineral's physical characteristics (i.e. particle size, density, magnetic properties, colour) or physicochemical properties (surface tension, hydrophobicity, wetability).
Mineral processing plant (mill, concentrator)	Facility, where mineral processing is carried out.
Mineral resource	Concentration or occurrence of natural, solid, inorganic or fossilised organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.
Mineral/mineral resource	Inorganic or organic compound occurring naturally in the earth's crust, with a distinctive set of physical properties, and a definite chemical composition (see Figure G1).
Mining	Methods and techniques to extract ore from the ground, including support facilities (e.g. stockpiles, workshops, transport, ventilation) and supporting activities in the mine itself or in the vicinity.
Mining operation	Any extraction of ore from which mineral substances are taken, where the corporate intent is to make an operating profit or build continuously toward a profitable enterprise.
Mitigation	Activity aimed at avoiding, controlling or reducing the severity of adverse physical, chemical, biological and/or socio-economic impacts of an activity.

Monitoring	Process intended to assess or to determine the actual value and the variations of an emission or another parameter, based on procedures of systematic, periodic or spot surveillance, inspection, sampling and measurement or another assessment methods intended to provide information about emitted quantities and/or trends for emitted pollutants
Multi-media effects	See *cross-media effects*
NNNNNNNNNN	
n/a	not applicable OR not available (depending on the context)
n/d	no data
Neutralisation	Raising the pH of acidic solutions or lowering the pH of alkaline solutions to near-neutral pH (about pH 7) values through a reaction in which the hydrogen ion of an acid and the hydroxyl ion of a base combine to form water.
Neutralisation potential (NP)	General term for a sample's or a material's capacity to neutralise acidity.
OOOOOOOOO	
Open pit (open cast) mining	Mining operation takes place on the surface. Mining operation and environment are in contact over an extended area.
Operator	Any natural or legal person that is responsible for the control, operation, and maintenance of the mine, mineral processing plant, tailings dam and/or related facilities including the the after-closure phases.
Ore	Mineral or variety of accumulated minerals (including coal) of sufficient value as to quality and quantity that it/they may be mined at a profit. Most ores are mixtures of extractable minerals and extraneous rocky material described as gangue (see Figure G1).
Orebody (mineral deposit)	Naturally occurring geological structure consisting of an accumulation of a desired mineral and waste-rock, from which the mineral can be extracted, at a profit, or with a reasonable expectation thereof (see Figure G1).
Overburden	Layer of natural grown soil or massive rock on top of an orebody. In case of open pit mining operations it has to be removed prior to extraction of the ore (see Figure G1).
PPPPPPPPPPP	
Percentage extraction	Proportion of ore mined from the orebody expressed as a percentage of the original in-situ amount of ore.
Permeability	Capacity of a rock or non-lithified material to transmit fluid.
Phreatic	Pertaining to ground water
Phreatic surface	The surface between the zone of saturation and the zone of aeration; that surface of a body of unconfined ground water at which the pressure is equal to that of the atmosphere
Piping	Mostly subterranean erosion of non-lithified materials caused by flowing water. Results in the formation of conduits due to the removal of particles.
Pollutant	Individual substance or group of substances which can harm or affect the environment
Primary crushing	Process of reducing ore into smaller fragments to prepare it for further processing and/or so that it can be transported to the processing plant. In underground mines, the primary crusher is often located underground, or at the entrance to the processing plant.

Primary measure / technique	Technique that in some way changes the way in which the core process operates thereby reducing raw emissions or consumptions (see *end-of-pipe technique*)
Probable maximum earthquake (PME)	A geotechnical engineering parameter determined by the maximum recorded earthquake at the site, the maximum recorded earthquake for a site in a similar location for which historic data is available or the one-in-10000-year earthquake predicted statistically from previous earthquakes in the region.
Probable maximum flood (PMF)	The most severe precipitation and/or snowmelt event considered reasonably possible at a particular geographic location. A site-specific determination based on the possible range in meteorological and hydrological events and conditions. Variables include the duration, the area and the time of year. Usually defined as the 1:10000 year flood or two or three times the 1:200 year flood.
Pump barge	Barge that floats in the tailings pond and supports the pumps that are used to reclaim the fee water in the pond for re-use in the mineral processing plant.
QQQQQQQQQ	
Quarry	Whole area under the control of an operator carrying out any activity involved in the prospecting, extraction, treatment and storage of minerals, including common related infrastructures and waste management activities, being not a mine. It is distinguished from a mine because it is usually open at the top and front, and used for the extraction of building stone, such as slate, limestone, gravel and sand
RRRRRRRRRR	
Reclaim lines	Pipelines that are used to convey the reclaimed water from the tailings pond to the mineral processing plant.
Reclaim system	Several components comprising the system constructed to reclaim water from the tailings pond and deliver it to the mineral processing plant. May include such items as: pump barges, reclaim lines, decant towers, and decant lines.
Reclaim water	Water recovered from the TMF, water treatment plant or mineral processing plant for re-use in the mineral processing plant.
Reclamation (rehabilitation, recultivation)	Restoration of land and environmental values of a mine site after the ore is extracted. Reclamation operations are usually underway as soon as the ore has been removed from a mine site. The process includes restoring the land to its approximate original appearance by restoring topsoil and planting native grasses and ground covers.
Recovery	Proportion, expressed as a percentage, of a constituent pertaining to the concentrate (or in the case of coal final tonnage) as compared to the total amount of that mineral initially present in the feed prior to mineral processing. A measure of mining, extraction and processing efficiency.
ROM	Run of mine ore. Unprocessed conveyed material from mining operation.
Runoff	Part of precipitation and snowmelt that does not infiltrate but moves as overland flow.
SSSSSSSSSSSS	
Sample	Representative amount of solids, which are drawn from orebodies or processes to perform analytical testwork. The amount of solids and the number of samples drawn from the orebody or the process stream has to be statistically accurate.
Screening	Separating material into size fractions

Seam	Flat lying or near flat lying stratum or bed of desirable material (typically: coal, potash, salt, lignite or bitumen).
Secondary measure / technique	See *end-of-pipe technique*
Seepage recovery dam	Small, water retention dam located downstream of the tailings dam, whose purpose is to intercept, collect, and return to the tailings pond all surface and subsurface seepage flows that by-pass the main tailings dam.
Separation	Processing methods to separate ore into concentrate and tailings.
Shaft	Primary vertical or inclined opening through mine strata used for ventilation or drainage and/or for hoisting of personnel or materials (e.g. ore, waste-rock); connects the surface with underground workings.
Slurry	A suspension of liquids and solids
SME	Small and medium enterprise(s)
Solubility	Quantity of solute that dissolves in a given volume and type of solvent, at given temperature and pressure, to form a saturated solution. The degree to which compounds are soluble depends on their ability, and that of the other dissolved species, to form ions and aqueous complexes in a particular drainage chemistry.
Spigotting	Procedure whereby the tailings are discharged into the tailings pond through a large number of small outlets or spigots. Spigotting produces a fairly even distribution of tailings over the tailings beach that forms the upstream semi-impervious zone of the tailings dam.
Starter dam	Initial tailings dam, which is constructed before the mining operation starts and provides the starting point for construction of the ultimate tailings dam.
Sub-aerial method of deposition	Name commonly used in North America for a method of spigotting which uses spray bars to place thin layers of tailings on a previously deposited beach.
Supernatant pond	The area of water held on a tailings pond above the settled tailings, normally removed by pumping or decanting
TTTTTTTTTT	
Tailings	Ore from which as much as feasible of the desired minerals have been removed. Tailings consist mainly of gangue and may include process water, process chemicals and portions of the unrecovered minerals.
Tailings beach	The area of "dry" tailings against the upstream face of the bund and normally above any pond level retained on the dam
Tailings dam	Structure designed to settle and store tailings and process water. Solids settle in the pond. The process water is usually recycled.
Tailings heap	Engineered facility for the storage of tailings on the surface Dry disposal of tailings on the surface.
Tailings line	Pipeline used to carry the tailings from the mineral processing plant to the tailings pond.
Tailings management facilities (TMF)	Refers to the fact that tailings from mineral processing steps have to be discarded/stored or recovered. The chosen method depends amongst many other factors on the physical characteristics (coarse or fine) and on the treatment of the ore (dry or wet). Typical tailings management facilities or methods are: ▪ tailings dam/pond ▪ tailings heap ▪ backfill ▪ recycling (construction material) ▪ reprocessing (extract content of ore by new better processing methods).

Tailings pond	Engineered facility for storing tailings resulting from ore processing and for clearing and recycling of process water, most of the times formed by a dam construction. Mainly contains tailings along with varying amounts of free water.
Tailings sand	Sand obtained from the total tailings for use in construction of the tailings dam. Often produced by cycloning the total tailings.
Thickening	Liquid-solid separation process to increase the concentration of a suspension by sedimentation, accompanied by the formation of a clear solid.
Top soil	Natural huminous layer on top of the orebody, which has to be stript prior to start-up of extraction (see Figure G1).
Total CN	The total of all cyanide existing in the various compounds in aqueous solution [24, BC CN guide, 1992]
TWG	Technical working group
UUUUUUUUUU	
Ultramafic	Igneous rock composed chiefly of mafic minerals, e.g. monomineralic rocks composed of hypersthene, augite, or olivine
Underground mining	Extraction of the ore takes place under the surface. The orebody is accessed by shafts, ramps and galleries.
VVVVVVVVVV	
Vein	Thin complex structure of ore accumulations surrounded by gangue.
VOC	volatile organic compounds
WWWWWWW	
WAD CN	Weak acid dissociable cyanide represents cyanides that are dissociated under reflux with a weak acid, usually at pH 4.5 [24, BC CN guide, 1992]
Waste-rock	Part of the orebody, without or with low grades of ore, which can not be mined and processed profitably (see Figure G1).
Waste-rock management facility (WRMF)	Facility where waste-rock is discarded, stored and in some cases treated, including waste-rock heap leaches.Part of the orebody, without or with low grades of ore, which can not be mined and processed profitably (see Figure G1).
Water balance	Process whereby all water entering the pond, all water leaving the pond, and all water leaving the pond, and all water losses, are defined such that the net gain or loss of water into the tailings pond can be determined.
Water table	Elevation at which the fluid pressure is equal to atmospheric pressure. The surface separating the vadose zone (where water is held under tension) from the saturated zone (where fluid pressures are greater than zero). The level to which water will rise in a well just penetrating the saturated zone.
Weathering	Processes by which particles, rocks and minerals are altered on exposure to surface temperature and pressure, and atmospheric agents such as air, water and biological activity.
XXXXXXXXXX	
YYYYYYYYYY	
Yield	Mass ratio of concentrate to feed, calculated on dry basis and expressed as a percentage.
ZZZZZZZZZZ	

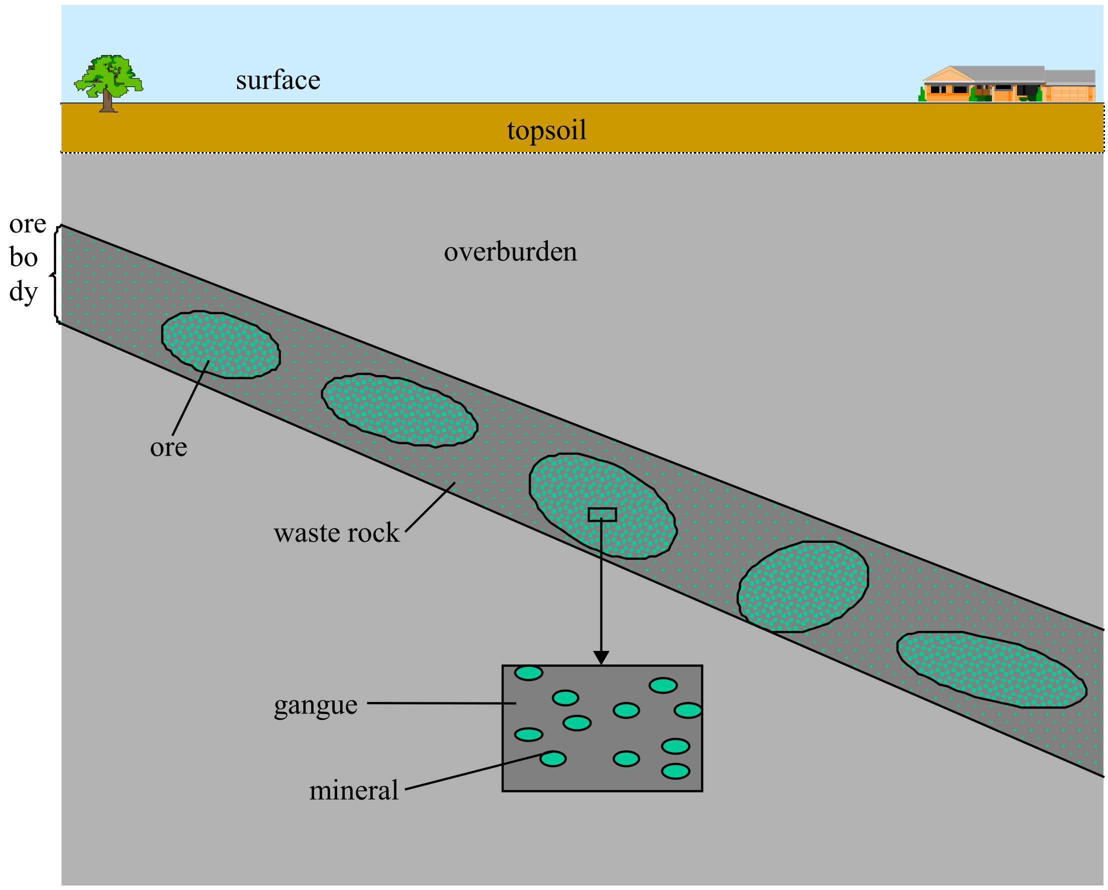

Figure G1: Schematic drawing of an orebody

Sources:

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2. CURRENT EU MEMBER STATES LIST (EU15)

Short Name	Full Name	Abbreviation[1]	Currency[2]	Currency ISO Code[3]
Austria	Republic of Austria	A	Euro	EUR
Belgium	Kingdom of Belgium	B	Euro	EUR
Germany	Federal Republic of Germany	D	Euro	EUR
Denmark	Kingdom of Denmark	DK	Danish krone (pl. kroner)	DKK
Spain	Kingdom of Spain	E	Euro	EUR
Greece	Hellenic Republic	EL	Euro	EUR
France	French Republic	F	Euro	EUR
Finland	Republic of Finland	FIN	Euro	EUR
Italy	Italian Republic	I	Euro	EUR
Ireland	Ireland	IRL	Euro	EUR
Luxembourg	Grand Duchy of Luxembourg	L	Euro	EUR
Netherlands	Kingdom of the Netherlands	NL	Dutch Guilder; Euro	NLG; EUR
Portugal	Portuguese Republic	P	Euro	EUR
Sweden	Kingdom of Sweden	S	Swedish krona (pl. kronor)	SEK
United Kingdom	United Kingdom of Great Britain and Northern Ireland	UK	pounds sterling	GBP

1. In BREFs, list Member States in English alphabetical order, using these abbreviations decided by the Permanent Representations
2. Former Currencies (pre-euro)
   - Austria - Austrian schilling (ATS)
   - Belgium - Belgian franc (BEF)
   - Germany - German mark (DEM)
   - Spain - Spanish peseta (ESP)
   - Greece - Greek drachma, pl drachmae (GRD)
   - France - France franc (FRF)
   - Finland - Finnish markka, pl. markkaa (FIM)
   - Italy - Italian lira, pl. lire (ITL)
   - Ireland - Irish pound (punt) (IEP)
   - Luxembourg - Luxembourg franc (LUF)
   - Portugal - Portuguese escudo (PTE)
3. ISO 4217, as recommended by Secretariat-General (SEC(96) 1820).
4. List of countries (Situation at 26.6.2002)

3. EU ENLARGEMENT CANDIDATE COUNTRIES

Short Name	Full Name	Country ISO Code[1]	Currency	Currency ISO Code[2]
Bulgaria	Republic of Bulgaria	BG	lev (pl. leva)	BGN
Cyprus	Republic of Cyprus	CY	Cyprus pound	CYP
Czech Republic	Czech Republic	CZ	Czech koruna (pl. koruny)	CZK
Estonia	Republic of Estonia	EE	Estonian kroon (pl. krooni)	EEK
Hungary	Republic of Hungary	HU	forint (inv.)	HUF
Latvia	Republic of Latvia	LV	lats (pl. lati)	LVL
Lithuania	Republic of Lithuania	LT	litas (pl. litai)	LTL
Malta	Republic of Malta	MT	Maltese lira	MTL
Poland	Republic of Poland	PL	Zloty	PLN
Romania	Romania	RO	Romanian leu (pl. lei)	ROL
Slovakia	Slovak Republic	SK	Slovak koruna (pl. koruny)	SKK
Slovenia	Republic of Slovenia	SI	Tolar	SIT
Turkey	Republic of Turkey	TR	Turkish lira	TRL

4. SOME OTHER COUNTRIES

Short Name	Full Name	Country ISO Code[1]	Currency	Currency ISO Code[2]
Australia	Commonwealth of Australia	AU	Australian dollar	AUD
Canada	Canada	CA	Canadian dollar	CAD
Iceland	Republic of Iceland	IS	Icelandic krona (pl. kronur)	ISK
Japan	Japan	JP	yen (inv.)	JPY
New Zealand	New Zealand	NZ	New Zealand dollar	NZD
Norway	Kingdom of Norway	NO	Norwegian krone (pl. kroner)	NOK
Russia	Russian Federation	RU	new rouble; Russian rouble	RUB; RUR
Switzerland	Swiss Confederation	CH	Swiss franc	CHF
United States	United States of America	US	US dollar	USD
1. ISO 3166 2. ISO 4217				

5. COMMON UNITS, MEASUREMENT AND SYMBOLS

TERM	MEANING
ACkWh	kilowatt-hours (alternating current)
atm	normal atmosphere (1 atm = 101325 N/m^2)
bar	bar (1.013 bar = 1 atm)
barg	bar gauge (bar + 1 atm)
billion	thousand million (10^9)
°C	degree Celsius
cgs	centimetre gram second. A system of measurements now largely replaced by SI.
cm	Centimetre

TERM	MEANING
cSt	centistokes = 10^{-2} stokes
d	Day
g	Gram
GJ	Gigajoule
h	Hour
ha	hectare (10^4 m^2) (=2.47105 acres)
J	Joule
K	kelvin (0 ºC = 273.15 K)
kA	kiloamp(ere)
kcal	kilocalorie (1 kcal = 4.19 kJ)
kg	kilogram (1 kg = 1000 g)
kJ	kilojoule (1 kJ = 0.24 kcal)
kPa	Kilopascal
kt	Kilotonne
kWh	kilowatt-hour (1 kWh = 3600 kJ = 3.6 MJ)
l	Litre
m	Metre
m^2	square metre
m^3	cubic metre
mg	milligram (1 mg = 10^{-3} gram)
MJ	megajoule (1 MJ = 1000 kJ = 10^6 joule)
mm	millimetre (1 mm = 10^{-3} m)
m/min	metres per minute
mmWG	millimetre water gauge
Mt	megatonne (1 Mt = 10^6 tonne)
Mt/yr	megatonnes per year
mV	millivolts
MW_e	megawatts electric (energy)
MW_{th}	megawatts thermal (energy)
ng	nanogram (1 ng = 10^{-9} gram)
Nm^3	normal cubic metre (101.325 kPa, 273 K)
Pa	pascal
ppb	parts per billion
ppm	parts per million (by weight)
ppmv	parts per million(by volume)
s	second
sq ft	square foot (= 0.092 m^2)
St	stokes. An old, cgs unit of kinematic viscosity. 1 St = 10^{-6} m^2/s
t	metric tonne (1000 kg or 10^6 gram)
t/d	tonnes per day
trillion	million million (10^{12})
t/yr	tonne(s) per year
V	volt
vol-%	percentage by volume. (Also % v/v)
W	watt (1 W = 1 J/s)
wt-%	percentage by weight. (Also % w/w)
yr	year
~	around; more or less
ΔT	increase of temperature
µm	micrometre (1 µm = 10^{-6} m)
Ω	ohm, unit of electrical resistance
Ω cm	ohm centimetre, unit of specific resistance

TERM	MEANING
% v/v	percentage by volume. (Also vol-%)
% w/w	percentage by weight. (Also wt-%)

6. LIST OF CHEMICAL ELEMENTS

NAME	SYMBOL	NAME	SYMBOL
Actinium	Ac	Mercury	Hg
Aluminum	Al	Molybdenum	Mo
Americium	Am	Neodymium	Nd
Antimony	Sb	Neon	Ne
Argon	Ar	Neptunium	Np
Arsenic	As	Nickel	Ni
Astatine	At	Niobium	Nb
Barium	Ba	Nitrogen	N
Berkelium	Bk	Nobelium	No
Beryllium	Be	Osmium	Os
Bismuth	Bi	Oxygen	O
Boron	B	Palladium	Pd
Bromine	Br	Phosphorus	P
Cadmium	Cd	Platinum	Pt
Calcium	Ca	Plutonium	Pu
Californium	Cf	Polonium	Po
Carbon	C	Potassium	K
Cerium	Ce	Praseodymium	Pr
Cesium	Cs	Promethium	Pm
Chlorine	Cl	Protactinium	Pa
Chromium	Cr	Radium	Ra
Cobalt	Co	Radon	Rn
Copper	Cu	Rhenium	Re
Curium	Cm	Rhodium	Rh
Dysprosium	Dy	Rubidium	Rb
Einsteinium	Es	Ruthenium	Ru
Erbium	Er	Rutherfordium	Rf
Europium	Eu	Samarium	Sm
Fermium	Fm	Scandium	Sc
Fluorine	F	Selenium	Se
Francium	Fr	Silicon	Si
Gadolinium	Gd	Silver	Ag
Gallium	Ga	Sodium	Na
Germanium	Ge	Strontium	Sr
Gold	Au	Sulphur	S
Hafnium	Hf	Tantalum	Ta
Helium	He	Technetium	Tc
Holmium	Ho	Tellurium	Te
Hydrogen	H	Terbium	Tb
Indium	In	Thallium	Tl
Iodine	I	Thorium	Th
Iridium	Ir	Thulium	Tm
Iron	Fe	Tin	Sn
Krypton	Kr	Titanium	Ti
Lanthanum	La	Tungsten	W
Lawrencium	Lr	Uranium	U
Lead	Pb	Vanadium	V
Lithium	Li	Xenon	Xe
Lutetium	Lu	Ytterbium	Yb
Magnesium	Mg	Yttrium	Y
Manganese	Mn	Zinc	Zn
Mendlevium	Md	Zirconium	Zr

7. SI UNIT PREFIXES

Symbol	Prefix	Term	Number
Y	yotta	10^{24}	1 000 000 000 000 000 000 000 000
Z	zeta	10^{21}	1 000 000 000 000 000 000 000
E	exa	10^{18}	1 000 000 000 000 000 000
P	peta	10^{15}	1 000 000 000 000 000
T	tera	10^{12}	1 000 000 000 000
G	giga	10^{9}	1 000 000 000
M	mega	10^{6}	1 000 000
k	kilo	10^{3}	1000
h	hecto	10^{2}	100
da	deca	10^{1}	10
-----	-----	1 unit	1
d	deci	10^{-1}	0.1
c	centi	10^{-2}	0.01
m	milli	10^{-3}	0. 001
µ	micro	10^{-6}	0.000 001
n	nano	10^{-9}	0.000 000 001
p	pico	10^{-12}	0.000 000 000 001
f	femto	10^{-15}	0.000 000 000 000 001
a	atto	10^{-18}	0.000 000 000 000 000 001
z	zepto	10^{-21}	0.000 000 000 000 000 000 001
y	yocto	10^{-24}	0.000 000 000 000 000 000 000 001

ANNEXES

ANNEX 1

Sampling and analytical methods from CN Code to be included

ANNEX 2

"Methods for the characterisation of mining waste" will be inserted later.

ANNEX 3

The report MiMi (1998) State-of-the-art-report on "Prevention and control of pollution from tailings and waste-rock products will be inserted here at a later stage.

The report can be downloaded from the following web site:

http://mimi.kiruna.se/PDF/Prev&C_98_2.pdf