



# Recovery of metals from waste incinerator bottom ash

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## ABBREVIATIONS

BA	Bottom Ash
ECS	Eddy Current Separator
EPA	Environmental Protection Agency
FE	magnetic Iron/Steel
GDP	Gross Domestic Product
HIMS	High Intensity Magnetic Separator (with flux density >0.2T)
HNF	Heavy Non Ferrous metal (density >5g/cm <sup>3</sup> : mainly copper, brass, zinc, stainless steel, lead, silver, gold)
LDS	Laser Diffraction Spectroscopy for particle size analysis
LIMS	Low Intensity Magnetic Separator (with flux density ≤0.2T)
LME	London Metal Exchange
LNF	Light Non Ferrous metal (density <5g/cm <sup>3</sup> : mainly aluminum, magnesium, titanium)
MS	Magnetic Separator
MSM	Minimum Sample Mass
MW	Municipal Waste
MWI	Municipal Waste Incinerator/Incineration
NF	Non Ferrous metals
UMTEC	Institute for Environmental and Process Engineering, University of Applied Sciences, Rapperswil/Switzerland
SS	Stainless Steel
WEEE	Waste Electrical and Electronic Equipment
XRF	X-ray fluorescence spectroscopy

## DEFINITIONS

### bottom ash BA

primary BA (=dry BA)	BA after dry extraction
wet BA	BA until 1 week after wet extraction
secondary BA	wet extracted BA after more than 1 week of curing

### metals

steel (FE)	ferromagnetic ferrous metal
stainless steel (SS)	paramagnetic ferrous metal
non ferrous metal (NF)	metal that is <u>not ferromagnetic</u> (including SS!)*
light NF (LNF)	NF with density $\leq 5\text{g/cm}^3$ (e.g. Al, Mg, Ti...)
heavy NF (HNF)	NF with density $> 5\text{g/cm}^3$ (e.g. Cu, Zn, Pb, Sn, SS, Au, Ag...) plus alloys such as brass and bronze
native metal	metal pieces free of mineral adhesions
metal total	total metal assay including the amount present in chemical compounds such as oxides
metal potential	metal that can be potentially recovered from BA by conventional dry processing (i.e. native metal pieces $> 2\text{mm}$ )
ferromagnetic particle	the particle is lifted by a magnet with surface flux density $\leq 0.2\text{T}$ (LIMS)
paramagnetic particle	the particle is lifted by a magnet with surface flux density $0.2 \dots 0.6\text{T}$ (HIMS)
non magnetic	the particle is <u>not</u> lifted by a magnet with surface flux density $0.6\text{T}$ (HIMS)

\* If SS is recovered as coarse metal scrap (e.g. by hand picking) it is sometimes not included into the NF fraction but listed separately.

## 1. Introduction

Metal extraction from the incineration residue of municipal waste is not only environmentally beneficial but also profitable. The literature on the topic is unfortunately rather fragmented. While many publications deal with the leaching behavior of Municipal Waste Incinerator Bottom Ash (MWI-BA), only few address the technical and economic aspects of metal extraction. This document fills the gap.

Although the recovery of scrap metal from BA has a long history, the dawn of modern BA processing may be placed in the early 1990ies. Three factors have greatly contributed to its proliferation since:

- A significant increase in the amount of waste incinerated
- New technologies for the recovery of nonferrous metals
- A sharp increase in metal prices

As the title suggests, focus is on the extraction of metals from MWI-BA. Neither the potential recovery of glass and ceramics or other mineral matrix material from MWI-BA, nor the potential uses of MWI-BA as building material will be discussed, the latter depending mostly on the local environmental legislation anyway. The perspective is limited to BA produced from grate incinerators (“stoker type”), although analogies may be drawn to BA produced from fluidized bed incinerators.

Because the composition of BA is highly variable, it is difficult to derive meaningful conclusions by comparing published results from different sources. In order to present a coherent picture, the data in this document will almost exclusively be drawn from work that has been carried out by UMTEC over the course of approximately two decades. The focus is on MWI-BA from plants in central and northern Europe.

Reviews on BA processing are provided by [20, 21, 22]. For digging deeper into the scientific literature the reader is advised to start with the publications by P.C. Rem and his collaborators at the Technical University of Delft (particularly recommended is [4]), as well as those by Th. Pretz and his group at RWTH Aachen. For an introductory textbook on separation theory and practice the reader is referred to [1].



*Fig. 1.1: Bottom ash from a Municipal Waste Incinerator is excavated for processing. Exothermal chemical reactions lead to a marked increase in temperature which evaporates a significant amount of moisture.*

## 2. Basics of Municipal Waste Incineration

### 2.1 Global aspects

Waste incineration is expensive, but has distinct environmental advantages over landfilling. Through incineration, the organic content of the waste is converted into thermal energy which can be used for electrical power generation or district heating. The incineration residue consists essentially of inorganic materials and metals. Thus, the complex chemical reactions of organic compounds, e.g. acids and chelating agents, with metals are prevented. In contrast, the inorganic chemistry of mineralized incineration residues is well understood and can be fairly easily controlled.

Incineration is globally on the rise. Currently (2016) an estimated 650 MWI grate furnace units are in production worldwide, 450 of them in Europe. Assuming that that each unit processes some 500t/day, and that 23% of the mass of waste is discharged as bottom ash, one arrives at 75'000t of bottom ash being produced daily. These contain potentially recoverable metals worth US\$ 4 Mio.

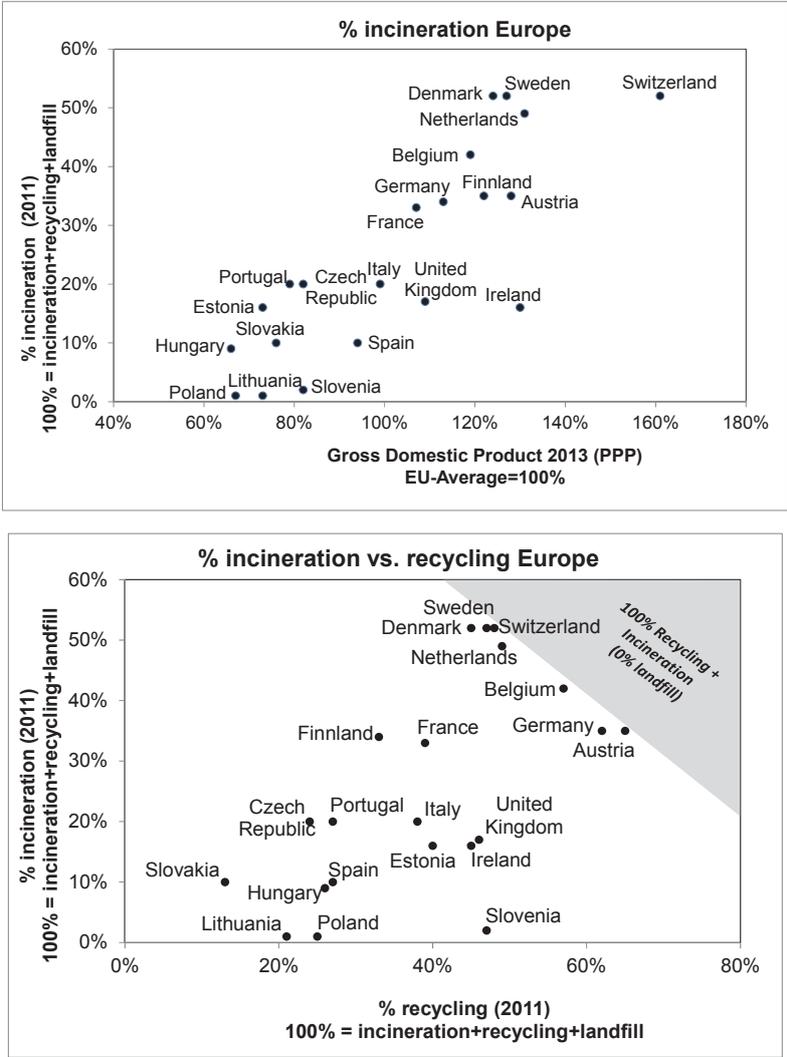


Fig. 2.1: Incineration quota increases with Gross Domestic Product (top). In contrast to a common misconception, incineration does not compete with recycling, but with landfilling (bottom). Rich countries recycle and incinerate their wastes, while poor countries landfill (data EUROSTAT, IMF).

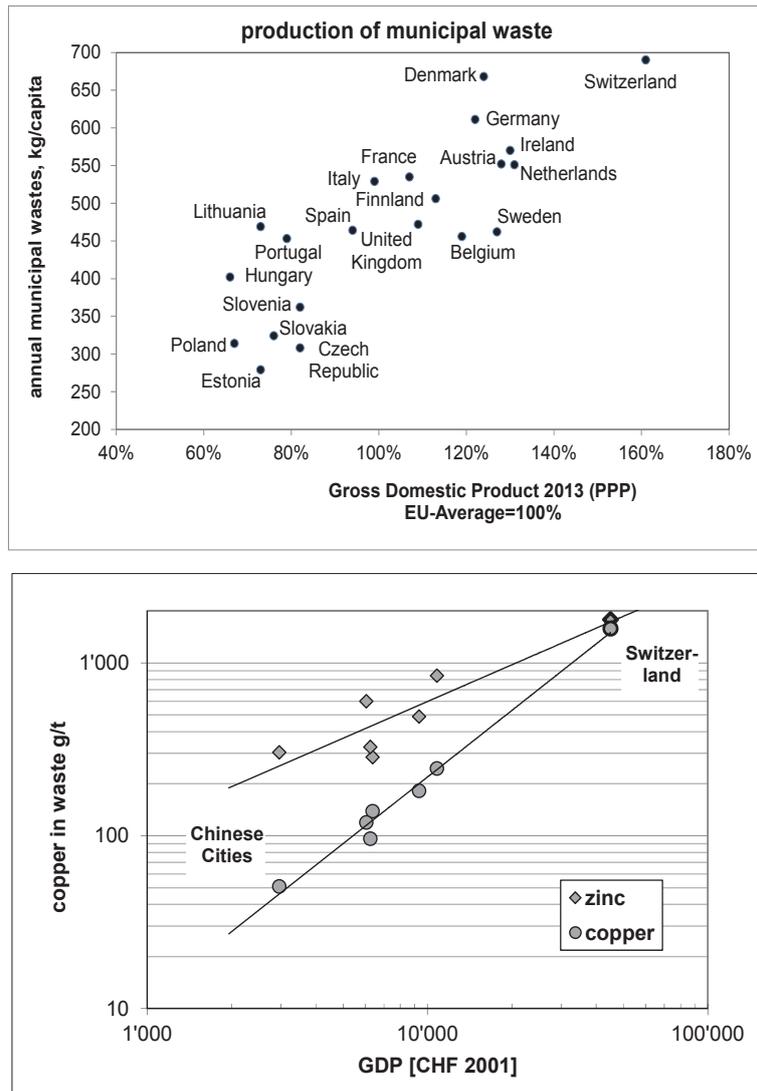


Fig. 2.2: The total amount of municipal waste generated increases with GDP.

Fig. 2.3: Zinc and copper content of wastes from 6 cities in China (balanced from the chemical analyses of BA and filter ash) plus data from Switzerland. As GDP increases, so does metal content.

As shown in Fig. 2.1, MWI is correlated with GDP. Rich countries not only incinerate but also recycle their wastes. Waste incineration and direct recycling of separately collected wastes (recycling containers for metals, WEEE, batteries...) do not compete but they are supplementary. More than half of the metals present in consumer goods are small metal pieces in intricate composite with other materials. This prevents them from being recovered through the established separate collection channels (for example: the metal clip, the metal spring and the metal refill of a plastic ball point pen). In Switzerland approximately more than half of the metals present in consumer wastes do not find their way into the channels of separate collections but end up in the residual waste which is incinerated.

Fig. 2.2 shows that the total amount of MW produced per capita increases with GDP. In Fig. 2.3 it is shown that also metal content in the waste rises with increasing GDP. This effect is particularly pronounced for copper. The concentration of copper in municipal waste seems a good indicator for the degree of development of the region the MWI operates in. This notion is backed by data we have obtained from 6 MWI in China.

In summary we conclude that, as global GDP increases:

- the total amount of MW produced will increase
- the metal concentration in the MW will increase
- a relative increase in MW incineration over landfilling will occur

As the prices of metals will – regardless of substantial volatility – also increase in the long run (Fig 7.1), metals extraction from MWI-BA will play an increasingly important role in the medium-term future.

## **2.2 Incineration Technology**

While there are various different classes of MWI, the predominant type is the grate incinerator as shown in Fig. 2.4. The waste is introduced into the incinerator and is conveyed through the incineration zone towards the discharge by means of moving grate elements. Within the incineration zone, the metal pieces that have been present in composite with plastics or other organic material are being liberated as the organics burn off. The mineralized non-combustible residue from the incineration is called ash and is essentially a mix of minerals and metals.

The ash is discharged from the grate and passes through the extractor duct into the extractor, which is a syphon filled with water. The syphon ensures that no air can leak into the combustion chamber which may otherwise upset the combustion process. The ash is conveyed out of the water bath most commonly by way of a reciprocating piston extractor. The BA then contains approximately 20% by weight of water. A few incinerators use a “dry extraction” i.e. an extractor that is not filled with water and therefore prevents the bottom ash from being wetted (chapter 5.1).

The oxygen necessary for combustion is provided by air blown from below through passages in the grate elements and carries away very fine ash particles (“fly ash”). After heat recovery in the boiler, the fly ash and other pollutants are removed from the flue gas in a series of filters and/or washers. The cleaned flue gas is then released through the stack. In some plants in the US the fly ash is mixed with the wet BA on site of the MWI, which reduces the potential of metals recovery from this mixture significantly.

While it is well known that the mean temperature on the surface of the firebed is around 900°C, little is known about the temperature distribution within the firebed. Our own research suggests that within a given incinerator at a given time, there may be large variations of temperature the waste is actually exposed to. Our experiments with a multitude of temperature probes that were simultaneously introduced into an incinerator suggests that 80% of the probes were subjected to temperatures between 700°C and 1100°C which, in turn means, that 20% were exposed to temperatures below and above this range.

When, for example, high calorific shredder residue is co-incinerated in MWI one may expect “hot spots” due to the fact that, despite all mixing efforts, this material cannot be evenly dispersed into the municipal waste. There is some evidence that bales of such material may lead to localized temperature peaks that result in the formation of large lumps of sintered ash which entrap the metal pieces previously present in the shredder residue. Vice versa, one will occasionally find lumps of wet wastes, such as dewatered sludge from waste water treatment, to be discharged virtually unscathed.

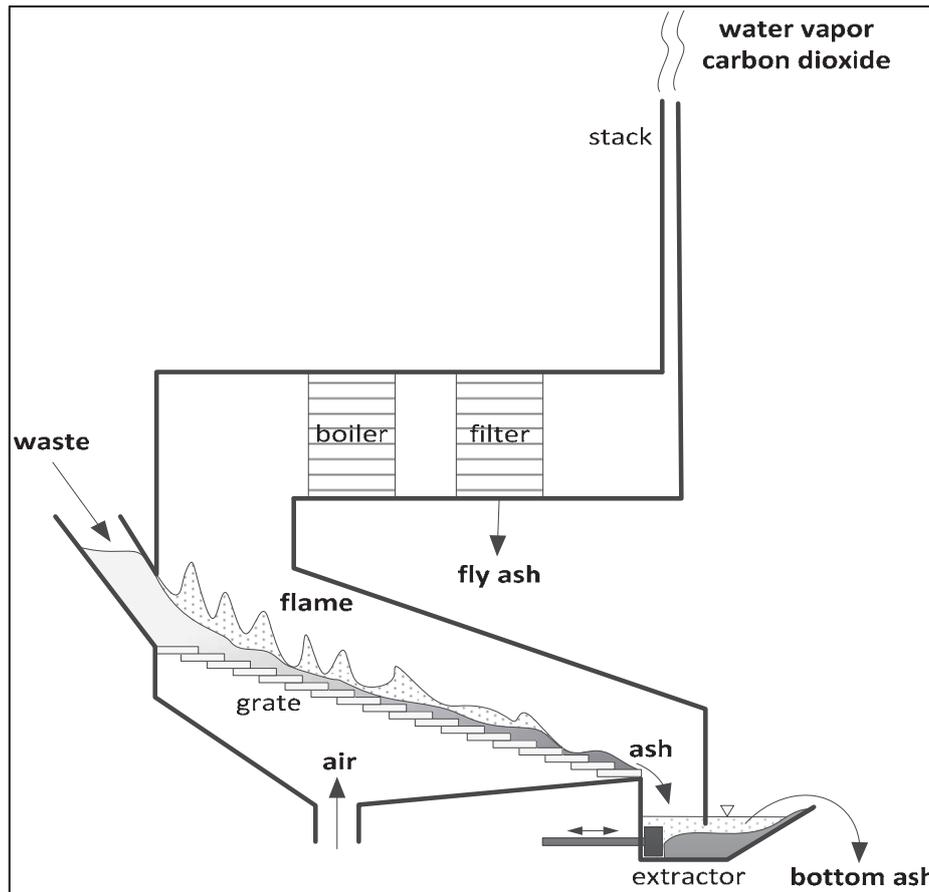


Fig. 2.4: Simplified schematic of a “stoker type” Municipal Waste Incinerator (MWI).

Since occasionally thermal damage to the grate elements occurs, it is rumored that thermite reactions lead to temperatures exceeding the melting point of steel. Such reactions of aluminum with iron oxides – both in plenty supply in MWI - are strongly exothermic. Other sources speculate that the damage to the grate, which seems to correlate with the input of increased amounts of aluminum, may in truth be the consequence of magnesium present in industrial wastes which was erroneously identified as aluminum.

### 2.3 Composition of “primary” Bottom Ash

As will be discussed in the chapter 3.3.1, the BA is subject to significant transformations once it is wetted in the extractor. In this document, the BA before it is wetted will be referred to as “primary BA” and after wetting as “wet BA”.

The primary BA is a free flowing bulk solid with a bulk density of approximately  $1.1 \text{ t/m}^3$ , which is increased to  $1.25 \text{ t/m}^3$  after compaction. The primary BA consists of the following major components (Fig. 2.5):

- Mineral Fraction  $>2 \text{ mm}$  (individual particles of glass, porcelain, tiles, pottery, cement, sintered grit...)
- Slag  $>2 \text{ mm}$  (solidified partially molten material)
- Native Metals  $>2 \text{ mm}$  (ferrous FE and NF)
- Unburnt Organic Matter  $>2 \text{ mm}$  (e.g. books, leather, wood...)
- Grit (fine grained particles  $<2 \text{ mm}$  comprising all of the above plus metal oxides, filler material from paper and plastics, wood ash...)

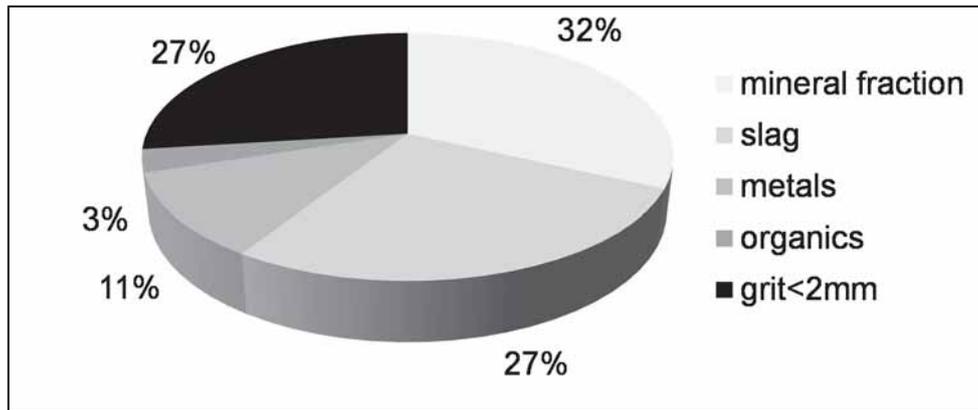


Fig. 2.5: Composition of the primary bottom ash

The “mineral fraction” comprises all mineral material >2 mm that is not slag. It consists of approximately 1/3 glass and porcelain of sizes mostly 4-64 mm (Fig. 2.9, 2.10). The glass content can vary substantially depending on the intensity of the separate collection of glass for direct recycling. The slag is a mixture of solidified (partially) molten phases that are often attached to or enclosing foreign objects such as ceramics or pieces of metal (Fig. 2.8; Fig. 4.4). Slag particles are formed at locations in the firebed where temperatures are sufficiently high to exceed the softening temperature of glass. Their formation depends on the calorific value and composition of the waste as well as on incineration conditions, in particular the amount of air being introduced through the grate and the intensity of the stoking action. The slag particles are rather dense, often have a “glassy” appearance, and a blackish color (Fig. 2.8). They contain approximately 40% iron-oxides and are weakly magnetic. The “grit” is material <2mm which is, with respect to metals, slag and mineral content, of a similar composition as the BA in general. In contrast to the coarse size fractions, the grit consists of approximately 2% water soluble species. The alkalinity of BA is in the order of 1 mol/kg (endpoint pH 7) and 4 mol/kg (endpoint pH 4).

A typical particle size distribution of incinerator ash is shown in Fig. 2.6. Approximately 90% of the BA is present in sizes distributed over three orders of magnitude ranging from 0.1 mm to 100 mm. The mean particle size is typically around  $d_{50}=8\text{mm}$ .

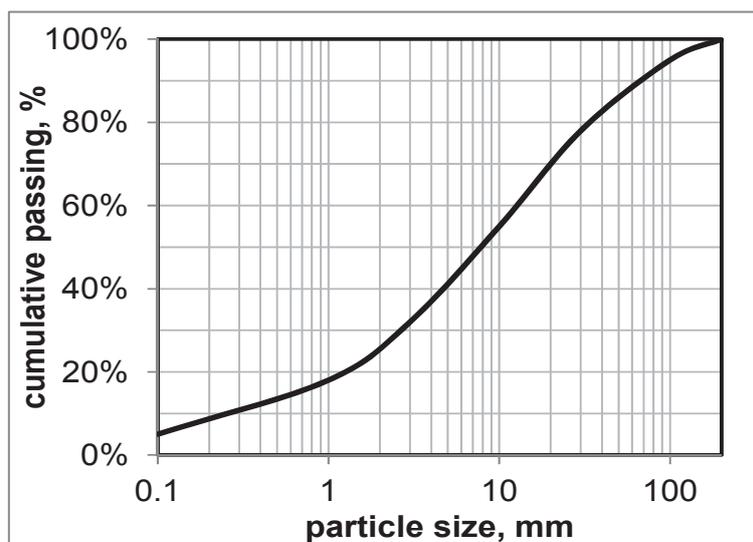


Fig. 2.6: Typical particle size distribution of primary BA.



Fig. 2.7: Primary BA <32 mm.



Fig. 2.8: Slag 8-32 mm.



Fig. 2.9: Glass in BA: 4-8 mm



Fig. 2.10: Glass and ceramics in BA: 8-32 mm.

#### 2.4 Solidification of Bottom Ash after wet extraction

Directly after wet extraction, the BA has a moisture content of approximately 18-20% (by weight) and a pH of 11-12. In Fig. 2.11 the water retention capacity of the BA is shown versus particle size.

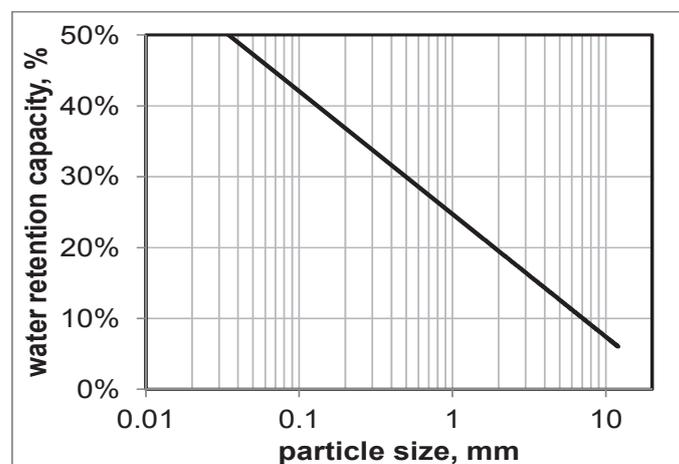


Fig. 2.11: The water retention capacity for fine size fractions of BA is much higher than for coarse size fractions.

These data were determined with sieved size fractions that were submerged in water, then left to drip for 5 minutes. Afterwards their moisture content was determined by drying at 105°C. Caution: By combining the information of Fig. 2.6 and Fig. 2.11, i.e. multiplying the products of size fractions (Fig. 2.6) and respective water contents, one arrives at a mean water content of only 13 % (instead of the 20 % mentioned above). This is due to the fact that the procedure for determining water retention capacity involved individual narrowly sized BA fractions which can accommodate less interstitial water than a size distributed bulk solid.

As soon as the BA enters the water flooded extractor, a large number of chemical reactions are triggered as primary “wet BA” converts into secondary BA. In this chapter we deal only with the transformations the mineral matter is subjected to. The processes involving metals is discussed in chapter 3.3.1.

On contact with water, soluble and semi-soluble salts, which make up 2% of the “grit” fraction, dissolve. These are, for example,  $\text{CaCl}_2$ ,  $\text{Na}_2\text{SO}_4$ ,  $\text{KOH}$ ,  $\text{Ca}(\text{OH})_2$ . As a consequence, the pH of the interstitial water in the BA increases to around pH12.3. This, in turn, triggers (pozzolanic) reactions with silicates. These are also finely dispersed within the grit fraction leading to the formation of new minerals. Minerals are also formed by precipitation (e.g. gypsum) or absorption of  $\text{CO}_2$  from air (limestone).

The formation of new mineral phases leads to a progressive solidification of the BA. Over the course of a few months of curing, the BA will solidify into a solid block. Thus the slag, the mineral fractions and the metals are being progressively encapsulated by a matrix consisting of solidifying grit that has slightly cementitious properties (Fig. 2.12, 2.13, 2.14). Obviously, this is undesirable with respect to the extraction of the metals, as these must be later liberated from this matrix before they can be separated. In this document we use the term “wet BA” for primary BA directly after wet extraction and the term “secondary BA” for solidified BA. We somewhat arbitrarily draw the line between “wet BA” and “secondary BA” at 1 week of curing after wet extraction.

Landfilled BA develops temperatures of 60-80°C due to exothermal processes. Major contributions to this effect are made by the hydrolysis of the oxides of alkaline and earth alkaline metals as well as by the oxidation and corrosion of steel and aluminum.

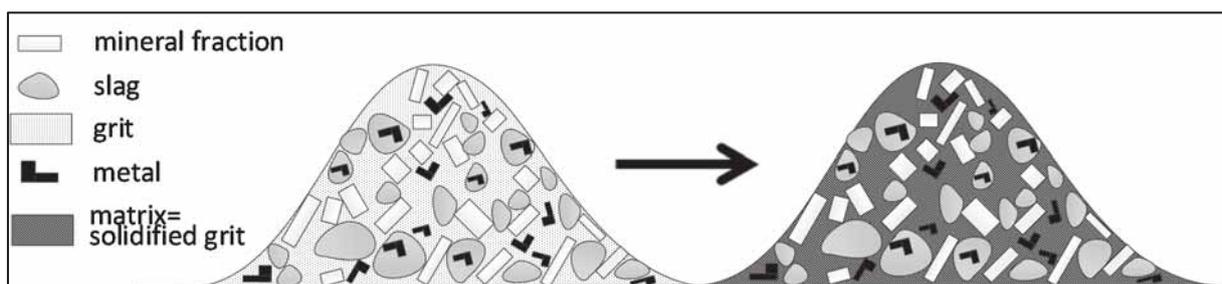


Fig. 2.12: Solidification of free flowing bulk solid after wet extraction.

Fig. 13 (left) shows the results of  $\text{BA} < 2 \text{ mm}$ , when separated in a heavy liquid at 3.0 kg/L after given time intervals. Initially just about 50% of the BA fragments float. When the same procedure is repeated periodically during the curing process, the fraction of BA with density  $< 3 \text{ kg/L}$  increases as particles of heavy metals (FE copper, brass), that had previously reported to the “sinks”, have meanwhile been embedded in sufficiently large lumps of solidifying (light) mineral material so the whole lump reports to the “floats”.

Fig. 13 (right) shows the irreversible water uptake of four size fractions of BA cured over the course of 8 weeks. Samples of each 100g dry-extracted BA were moistened and then stored in

a water-saturated atmosphere. After a curing period of 8 weeks, they were dried at 105°C. The difference between the initial dry weight (100g) and the weight after drying at 105°C is the “irreversible water uptake”. This “missing” water was apparently consumed by the formation of minerals, such as hydroxides.

Fig. 2.14 (left) supports this data: the compressive strength of test-blocks (BA<4mm with 20% moisture) increased with curing time. Fig. 14 (right) points into the same direction. Immediately after extraction, secondary BA was screened at 0.063 mm and then divided into samples which were set aside for curing. After curing for time periods starting at one day and extending to 90 days, the samples were analyzed for particle size by way of LDS (laser diffraction spectrometry) and the amount <0.063 mm was thus determined. Apparently, some of the material <0.063mm “disappears” with time – a clear indication for the progressive solidification of the material. From this data it is concluded that the solidification of wet BA commences immediately after extraction, thus progressively encapsulating fine small metal pieces in a newly formed mineral matrix. While fine metal particles are embedded into the mineral matrix within a few days, larger particles are encapsulated within weeks.

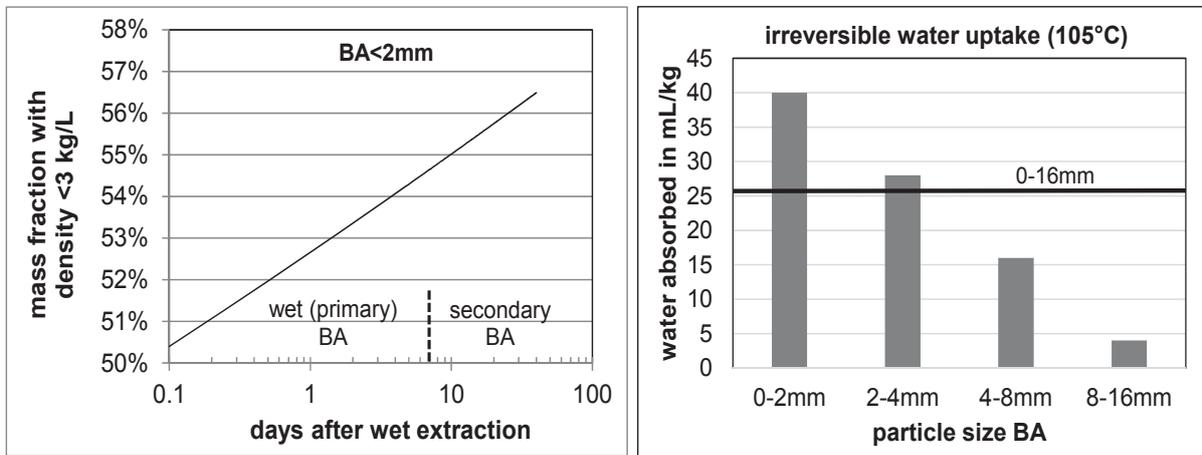


Fig. 2.13: Progressive solidification of BA being indicated by (left) an increasing amount of material with density <3.0 kg/L and (right) by the irreversible uptake of water.

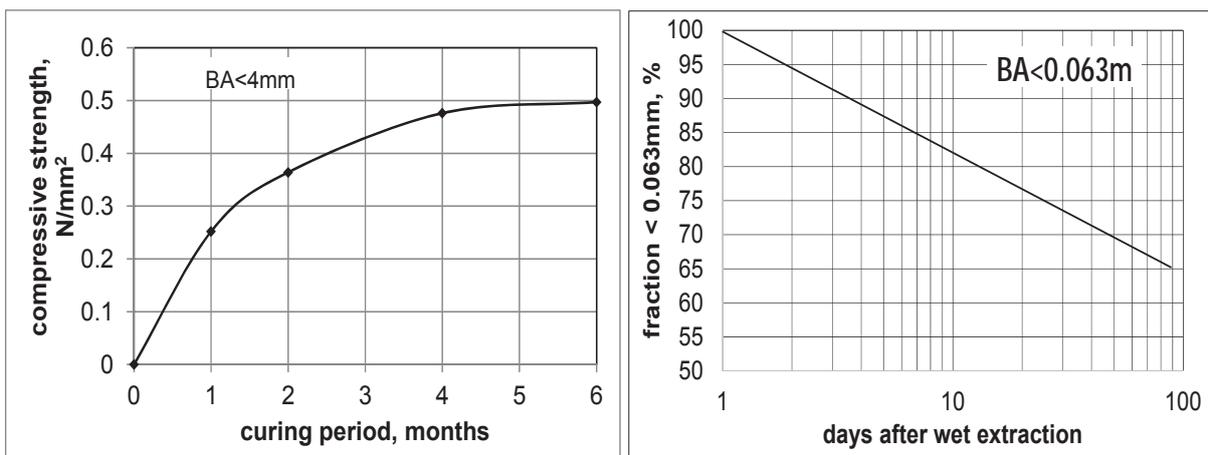


Fig. 2.14: Progressive solidification of secondary BA can be inferred from (left) the increase of compressive strength in Proctor-tests over the course of 6 months and (right) the continuously decreasing amount of material <0.063 mm due to the formation of agglomerates.

In effect, the material <2 mm, the grit, acts upon solidification as a binder for the coarser material, which is virtually inert with respect to hydraulic setting processes (Fig. 2.12). This is in analogy to concrete, which also consists of hydraulically active fine-grained cement and hydraulically inert coarser components i.e. sand and gravel.

With respect to metals extraction from BA it is very advantageous to remove the material <2 mm from the BA immediately after wet extraction. The material >2mm does not solidify readily which is very advantageous for the later recovery of metals, for example with eddy current separators.

While most of the approximately 20% moisture contained in the wet BA evaporates, approximately 15% (of the above 20% moisture) is being consumed by chemical reactions, for example by the oxidation of iron and aluminum (forming the respective hydroxides) and the formation gypsum hydrates. The bulk density of compacted secondary BA (after removal of metal scrap) in a landfill is about 1.35 t/m<sup>3</sup> (dry mass).

### 3. Metals in Municipal Waste Incineration

#### 3.1 Origin of metals in municipal waste

Metals occur in waste either in their “native” form (including alloys), or chemically bonded (mainly as oxides). In the context of metals recovery from BA, only native metals are of commercial interest.

Although in virtually all countries that employ waste incineration, the separate collection of metals is also implemented, a surprising large amount of metals ends up in the incinerators and hence in the BA. It is estimated that, despite diligent separate collection, more than 50% of all metals passing through Swiss households end up in the municipal waste which is incinerated. The reason being, that most native metals found in consumer products are small in size and usually in composite with other materials such as plastics, ceramics or textiles. Only metal pieces exceeding 100 mm in size and essentially free of composite material are successfully recovered through separate collection (e.g. cans, silverware...). Smaller pieces of metals and some small pieces of electronic wastes, such as batteries and small entertainment devices, are disposed of into the household waste.

The following table gives an overview of the aluminum alloys encountered in bottom ash. It is the result of analyses of 41 aluminum particles randomly grabbed from a Swiss bottom ash. The aluminum recovered from BA is of remarkably good quality. This result ties in with those produced by other researchers.

*Table 3.1: Matching pieces of aluminum found in BA with aluminum products.*

alloy	typical application	
1XXX	high purity Al >99%	28%
8XXX	screw tops (bottles)	22%
3XXX	cans: mantle	17%
5XXX	cans: lid	11%
6XXX	div. industrial applications	9%
cast Al	pots&pans	9%
--	other	4%

Also, a quite considerable amount of metal is introduced through co-incineration of industrial or commercial wastes. One example are the shredder residues from automotive and electronics recycling which contain a fair amount of nonferrous metals, albeit in rather small sizes.

#### 3.2 Effects of the incineration process on metals

In order to describe the behavior of metals in the incinerator, macroscopic transformations (melting and bulk chemical reactions) and microscopic transformations (chemical reactions involving the surface) are distinguished.

##### 3.2.1 Macroscopic transformations

Once in the incinerator, the overall fate of a piece of metal depends largely on its physical and chemical properties. With respect to transfer either into the bottom ash or into the fly ash, one can distinguish four groups of metals:

1. Iron, copper, brass, nickel, chromium and gold end up almost entirely in the bottom ash. As they have melting temperatures above those normally encountered in MWI, the bulk of these metals will not engage in chemical reactions except on their surfaces, which may become oxidized or corroded. Since these metals are prevalent in the waste in their native form, the stoking action of the grate will cause a downward segregation from the surrounding waste because of their high density.
2. Aluminum (and its alloys melt at around  $660^{\circ}\text{C}$  but are protected from progressive oxidation by a tenacious aluminum oxide skin that encloses the melt. Molten aluminum droplets trickle through the firebed onto the grate which is cooled by the inflowing air. Here the aluminum melt solidifies, resulting in odd shaped “nuggets” (Fig 3.3). In addition to oxidation, some aluminum is lost by the formation of aluminum nitride, resulting from the reaction of nitrogen from air with aluminum metal at around  $900^{\circ}\text{C}$ .
3. Zinc and lead are transferred as more or less equal fractions into the bottom ash and the fly ash. Most of the “native” zinc that ends up in the in the bottom ash is present in brass, which has a higher melting temperature than zinc itself. Much of the lead is present not as native metal but in chemical compounds such as lead crystal which end up in the BA. Since zinc and lead melt at incinerator temperatures, they engage in thermochemical corrosion reactions, for example by chloride formation, and are hence partially transferred into the filter ash.
4. All of the mercury and most of the cadmium are transferred to the flue gas and either end up in the fly ash (cadmium) or in dedicated washers and filters (mercury). Some cadmium is trapped in rechargeable batteries which, by virtue of their high density, segregate through of the firebed onto the grate and remain largely intact (Fig. 3.4). Some cadmium is also encountered as anti-corrosion paint or colored glazing on pottery and consequently ends up in the bottom ash.

As a rule of thumb, pieces of native metal or alloys will mostly end up in the bottom ash, if their melting temperature is either above  $1000^{\circ}\text{C}$ , or they are protected by special mechanisms from bulk chemical reactions (aluminum, cadmium in batteries). It is important to note that the fate of metal pieces depends to a large extent on their success in working their way down onto the grate where they are cooled by the inflowing air and thus protected from oxidation, melting and thermal corrosion. To which extent this process depends on parameters like particle density, particle shape and the stoking action has not been established yet and calls for further research.



*Fig. 3.1: Copper is often present in the form of wire which is oxidized on the surface (left).*



*Fig. 3.2: Brass is mostly encountered in the form of mountings and fittings (right).*



Fig. 3.3: Aluminum melt solidified as “nugget”.

Fig. 3.4: Approximately 2/3 of the batteries pass the incinerator intact. This sample was taken from magnetic scrap recovered by BA-processing.

### 3.2.2 Microscopic transformations

Although metals with high melting temperatures will not engage in bulk chemical reactions, their surfaces may be significantly altered, mostly by oxidation and thermal corrosion.

Superficial oxidation is particularly pronounced with copper and steel even much below their respective melting points (Fig. 3.5). These data as well as the ones shown in Fig. 3.6 and Fig. 3.7 were obtained by treating metal cubes of 8cm<sup>3</sup> in a muffle furnace (with no inert gas atmosphere). As shown in Fig. 3.6 and 3.7, the extent of oxidation depends not only on temperature but also on exposure time.

The surprisingly fast oxidation of copper and steel is due to the fact that the oxide scales of these metals are brittle and flake off easily, thereby exposing fresh metal surface. In contrast, SS is hardly oxidized at all, which is due to a protective layer of chromium oxide preventing further oxidation. Also, aluminum and brass are much less prone to oxidation than copper and steel (Fig. 3.5). We were able to verify these laboratory data under plant conditions, albeit with much more scatter.

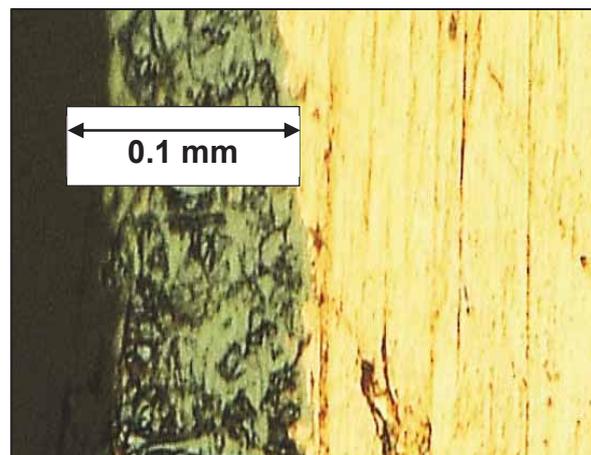
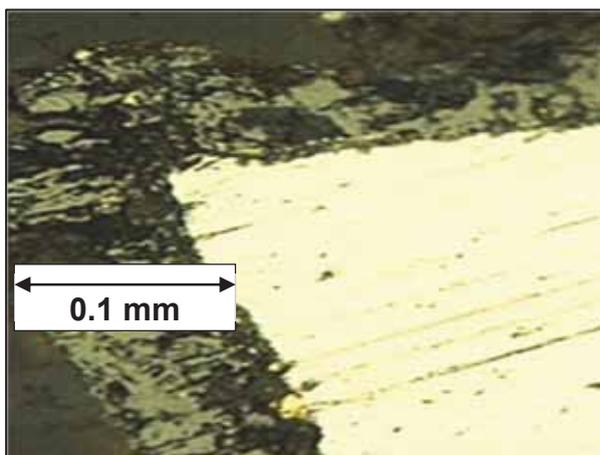


Fig. 3.5: After 1h exposure to 900°C: steel (left) and copper (right) have developed an oxidation layer of approximately 0.08 mm thickness.

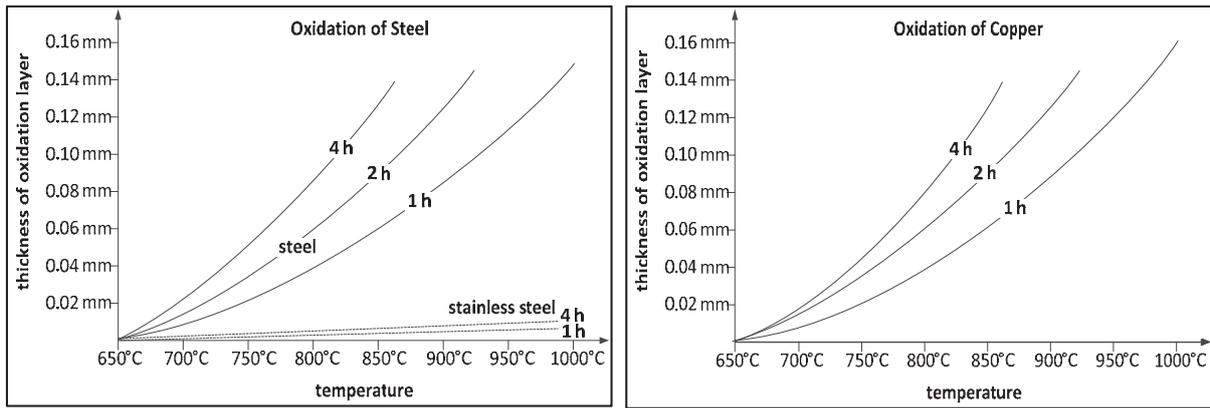


Fig. 3.6: Oxidation progress on steel (left) and copper (right) as functions of temperature and time.

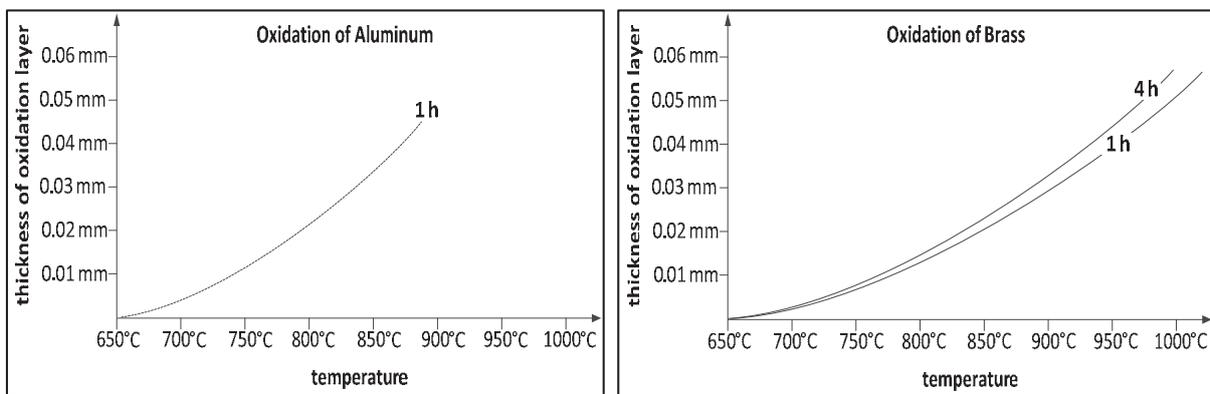


Fig. 3.7: Oxidation progress on aluminum (left) and brass (right) as functions of temperature and time.

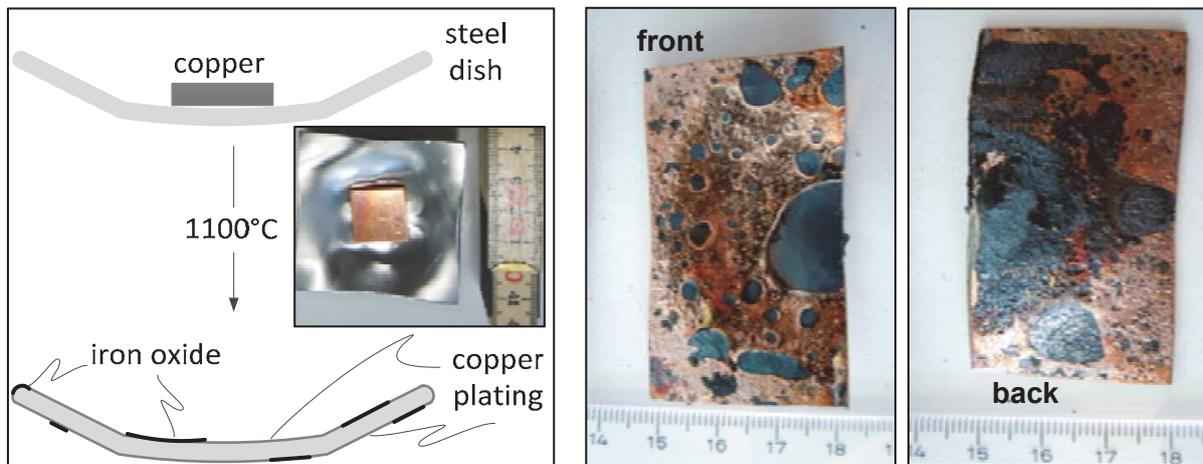


Fig. 3.8: Copper plated pieces of steel recovered from BA (after removal of the oxide scales by shredding).

Depending on the thickness of the metal objects introduced into the incinerator, the relative loss by oxidation can be expected between negligible (massive pieces) and substantial (thin metal sheets). It is estimated that approximately a third of the iron and aluminum in the waste are lost by thermal oxidation [3]. Experiments with aluminum cans have shown that in MWI approximately one third of the aluminum mass is lost due to thermal oxidation.

Another phenomenon is the plating of steel with copper under incineration conditions. When steel is recovered from BA by magnetic separation, the steel surfaces are typically covered with

a layer of black or grey oxide scales. After processing the material through a shredder, the adhesions are removed and it becomes apparent, that many pieces of steel are coated with thin films of copper of variable thickness (0.02-0.2 mm), giving them a copper colored tint (Fig. 3.8).



*Fig. 3.9: As temperature increases, the steel surface oxidizes and the liquid copper penetrates the boundary between steel and iron oxide thus plating the steel (left). Front and back after cutting the steel dish in half (right).*

In order to further investigate, tests we carried out tests in a muffle furnace. As delineated in Fig. 3.9 left, a piece of copper 4x16x16 mm was placed on a flat steel dish (60x60 mm) and then exposed to a temperature of 1100°C for 30 minutes (the melting point of copper is 1083°C). Afterwards, the surface of the dish was covered by a layer of black iron oxide scales, which was cautiously removed by tapping with a hammer.

The result is shown in Fig. 3.9 right. The entire dish was plated with a thin layer of copper not only on the inside but, surprisingly, also on the outside. Apparently the liquid copper was able to not only penetrate the small gap between steel and iron oxide scales on the inside of the dish. Driven by capillary forces, the copper melt rose against gravity over the rim of the dish and also plated the outside. The dish was thus completely copper plated save some areas where the oxide layer was so strongly attached to the steel that it did not provide a gap for the liquid copper to intrude. The iron oxide scales that chipped off after tapping with a hammer are ferromagnetic and also strongly contaminated with copper oxide. By covering the samples with sand extracted from processed BA and then exposing them to temperatures around 1100°C it was shown that some of the copper-oxide had been taken up by in the slag phases.

The copper balance of a MWI is shown in Fig. 3.10. Of the initial 2 kg Cu per ton of waste, approximately 98% report to the BA (the balance being carried into the filter ash). Around two thirds of this amount are typically recovered as native metal in conventional BA processing plants. Some 17% of the copper present in the BA is lost into the processed BA (going, for example, to landfill). The copper in the processed BA is present in the form of oxides (50%). The other 50% is native metal in the form of particles too small (<4mm) to be recovered by means of conventional dry processing.

Some 14% of the copper is in composite with steel and follows the steel through the various BA processing stages. Some 6% of the copper initially present in the waste ends up being coated onto steel by the mechanisms discussed above. An additional 7% of the copper in the MW is present in composite with iron in the form of solenoids, e.g. as part of transformers, loudspeakers and electrical drives.

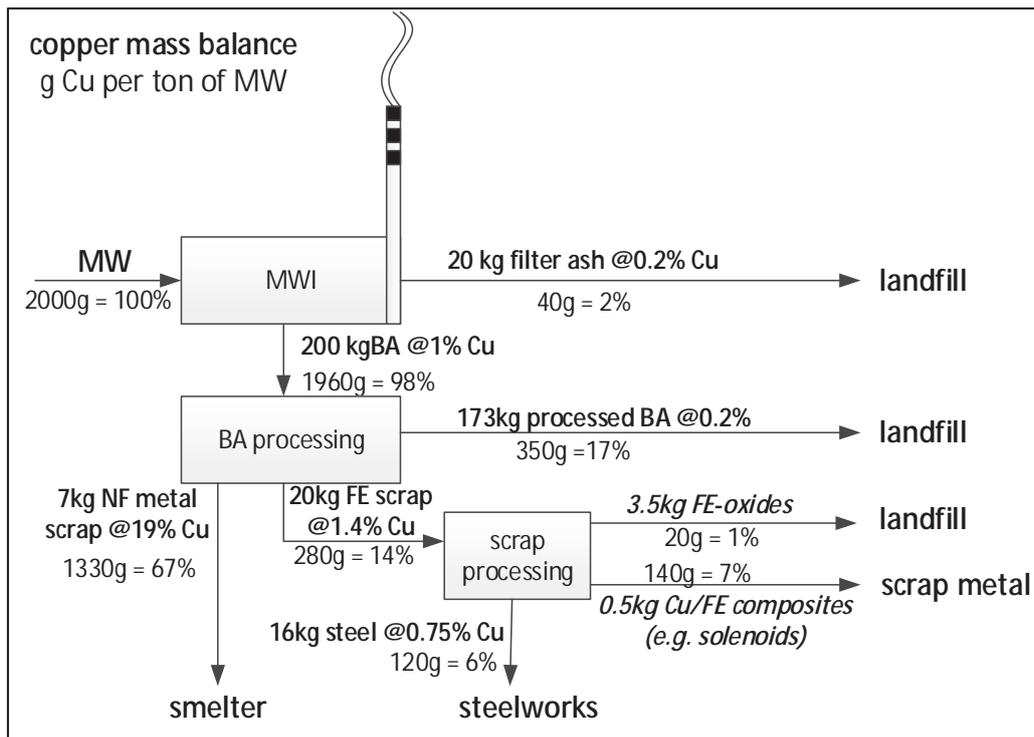


Fig. 3.10: The copper-balance of a MWI.

### 3.3 Metals in bottom ash

#### 3.3.1 Alterations after wet extraction

As soon as the primary BA is discharged into the wet extractor, a multitude of chemical reactions affecting the metals therein are triggered. For example, heavy metals that had been present in the BA as water soluble salts (e.g. chlorides and sulfates), are instantaneously dissolved. Some of the heavy metal ions will then adsorb onto the iron oxides, the rest remaining in solution.

If the wet BA is transferred to a landfill, it must be kept in mind that its 20% moisture content has the same composition as the brine in the extractor, i.e. pH 10.5-12 and high concentrations of chlorides and sulfates. After landfilling, the native metals in the BA are, over the course of weeks and months, subject to various alterations such as corrosion and progressive encapsulation into the solidifying mineral matrix. Corrosion is an issue in particular with the more ignoble metals iron and aluminum.

Iron is aggressively corroded by chloride and sulfate ions and also by transformation to iron hydroxides at high pH [3]. Under typical landfill conditions it is estimated that one third of the iron present directly after wet extraction is lost by corrosion within 1-2 years. From observations made during the excavation of old landfills we estimate that within 15 years two thirds of the native FE originally present the landfilled BA is corroded.

Aluminum is corroded in an aqueous alkaline environment by reactions that lead to the release of hydrogen. This process may, under unfavorable conditions, lead to deflagrations on landfills and may therefore become a safety issue. For reference: each kg of corroded aluminum produces approximately 1.35 m<sup>3</sup> of hydrogen. The release rate of hydrogen depends strongly on pH and is – per ton of BA - typically around 50 L per day within the first 10 days of disposal.

In total approximately 2.4 m<sup>3</sup> hydrogen, the equivalent of 1.8 kg of aluminum, is released per ton of BA within 3 months and very little thereafter. When excavating a landfill, that contained BA which had been deposited more than 10 years ago, we observed numerous aluminum nuggets embedded in a white powder of aluminum corrosion products. Only an estimated 10-15% of the original mass of the aluminum nuggets 8-20 mm in size had corroded within a decade.

Copper and brass are hardly affected by corrosion in landfilled BA. At pH 12 most heavy metals are immobilized as hydroxides. Under anaerobic conditions, the formation of sulfides may also contribute to the immobilization of heavy metals. However, copper may be mobilized by chelating agents and can cause problems by exceeding the legal limits for landfill leachates. Possible mobilization mechanisms involve the chelation of copper by organic compounds produced by low temperature carbonization processes in the incinerator. Our chemical analysis of leachate from BA has shown that typical carbonization products, such as aldehydes and alcohols are present. Also, a mobilization by ammonia may play a role. Problems with high copper concentrations in the landfill leachate are usually limited to newly established landfills and are less pronounced if the leachate needs to trickle through a thick layer of previously filled BA. We speculate that this may be due to adsorption of dissolved copper species on iron oxides produced by corrosion processes within the landfill and also due to the precipitation as sulfide, or carbonate.

Wet BA has a distinct odor one component of which is ammonia. It is not fully understood yet how ammonia is produced in wet BA, but a major contribution appears to stem from the hydrolysis of aluminum and magnesium nitrides which are produced by the respective metals under incineration conditions.

With respect to later processing, the encapsulation of metals into the solidifying mineral matrix is important, because the metals can only be recovered by later processing, if they are first liberated from the surrounding mineral matrix.

### 3.3.2 Properties relevant for bottom ash processing

For assessing the economy of BA-processing, the most important factors are the amount of native metals present in the BA and their size. The metal content depends, as pointed out in chapter 2.1, strongly on the local conditions, in particular GDP. The following data can be considered “typical” for MWI in central Europe, for example in Germany, France, the Benelux and the Nordic countries. The data presented apply to a primary bottom ash (or a freshly discharged wet BA). If the BA had been landfilled before processing (secondary BA), corrections must be made for loss of metal through oxidation and corrosion.

Table 3.2 was generated from data published in [2], which is a very thorough analysis of the metal inventory in the waste and the residues of a particular MWI in Switzerland. Of 200'000t waste annually processed in this incinerator, 44'000t solid incineration wastes are produced, of which 90% are bottom ash. The table shows the concentrations of selected metals in the waste. It also shows the transfer coefficients of these metals from the waste into the bottom ash and their respective concentrations in the BA. While the data tie in quite well with those derived from other studies, the feed to this particular MWI comprised a relatively large amount of industrial wastes, in particular co-incinerated shredder residues. The concentrations of some metals are therefore at the upper end of the usual spectrum. In particular the value determined for gold is, without co-incinerated residues from WEEE shredders, typically around 0.5g/t.

**Table 3.2: Data from a Swiss MWI [2]**

		conc. in waste	transfer to BA	conc. in BA			conc. in waste	transfer to BA	conc. in BA
		mg/kg	%	mg/kg			mg/kg	%	mg/kg
<b>Aluminum</b>	<b>Al</b>	<b>17'000</b>	<b>94.7%</b>	<b>81'389</b>	Niobium	Nb	2.50	90.8%	11.5
Barium	Ba	749	94.4%	3'575	Platinum	Pt	0.06	91.7%	0.27
Beryllium	Be	0.28	100.0%	1.42	Praseod.	Pr	1.90	96.7%	9.29
Bismuth	Bi	2.80	38.9%	5.51	Rhodium	Rh	0.000092	100.0%	0.00047
Cadmium	Cd	8.90	13.2%	5.94	Rubidium	Rb	8.31	69.1%	29.0
Chromium	Cr	180	90.6%	824	Ruthenium	Ru	0.00050	100.0%	0.0025
Cobalt	Co	11.0	87.3%	48.5	Scandium	Sc	0.96	89.2%	4.33
<b>Copper</b>	<b>Cu</b>	<b>2'230</b>	<b>97.8%</b>	<b>11'026</b>	Selenium	Se	0.45	45.3%	1.03
Gadolinium	Gd	0.75	94.7%	3.59	<b>Silver</b>	<b>Ag</b>	<b>5.30</b>	<b>75.0%</b>	<b>20.1</b>
Gallium	Ga	2.20	86.2%	9.59	Strontium	Sr	130	91.7%	603
Germanium	Ge	0.21	78.8%	0.84	Tantalum	Ta	1.20	89.0%	5.40
<b>Gold</b>	<b>Au</b>	<b>0.40</b>	<b>95.3%</b>	<b>1.93</b>	Tellurium	Te	0.085	48.5%	0.208
Hafnium	Hf	2.60	64.3%	8.45	Thallium	Tl	0.079	51.4%	0.205
Indium	In	0.29	57.5%	0.84	Tin	Sn	74.0	58.1%	217
<b>Iron</b>	<b>Fe</b>	<b>32'000</b>	<b>98.1%</b>	<b>158'704</b>	Tungsten	W	56.0	98.1%	278
Lead	Pb	540	58.0%	1'583	Vanadium	V	11.0	91.9%	51.1
Lithium	Li	9.00	88.5%	40.3	Yttrium	Y	7.85	53.9%	21.4
Molybdenum	Mo	8.60	89.7%	39.0	<b>Zinc</b>	<b>Zn</b>	<b>1'600</b>	<b>57.7%</b>	<b>4'667</b>
Neodymium	Nd	7.26	96.5%	35.4	Zirconium	Zr	116	64.3%	377
<b>Nickel</b>	<b>Ni</b>	<b>120</b>	<b>93.3%</b>	<b>566</b>					

The metals in BA are either present as native metal or chemically bonded. Only the native metals are of economic value, of which only metals >2 mm are potentially recoverable by means of conventional technology. Consequently, the metals can be divided into two classes:

1. Metals >2 mm: these are native metal pieces of economic value that are potentially recoverable.
2. Metals <2 mm: these are either chemically bonded metals with no economic value or they are native metal pieces that are not potentially recoverable by conventional means.

**In order to evaluate a process for recovering metals from BA one should always use the content of native metal pieces >2 mm as the 100% benchmark.**

Typical metal recoveries by conventional means are in the order of 40-65% of nonferrous (NF) metals and around 85-95% for steel (FE).

As shown in Fig. 3.11, less than half of the total content of most metals is present in potentially recoverable particles >2 mm. One exception is copper of which 70% is >2 mm. Fig. 3.12 is another representation of the same data but based on the native metals >2 mm only. The total content of iron (FE plus SS) of 16% matches the concentration given in Table 3.2 (15.8%), likewise zinc with 0.50% (0.47%). Copper 0.81% (1.1%) and aluminum 5.8% (8.1%) are approximately 30% below the respective data in Table 3.2.

Most of the above native metals and alloys are present in the BA as separate pieces. One notable exception is copper which is often found in coils around iron, for example in electric motors, solenoids and transformers. Metals may also occur embedded into their original mineral matrix, for example in combination with porcelain as in spark plugs.

Although the major constituent of SS is iron, SS is by convention designated a NF metal for two reasons. First, in the separation processes employed for BA processing, it behaves like a nonferrous metal as it is not amenable to magnetic separation (unlike FE). Secondly, SS scrap fetches, just as the proper NF metals, much higher prices than normal steel. It is also common to subdivide the NF into the “light non-ferrous” LNF fraction ( $\rho < 5 \text{ t/m}^3$ : aluminum, magnesium, titanium) and the “heavy non-ferrous” HNF fraction ( $\rho > 5 \text{ t/m}^3$ : copper, brass, zinc, lead, SS, gold, silver).

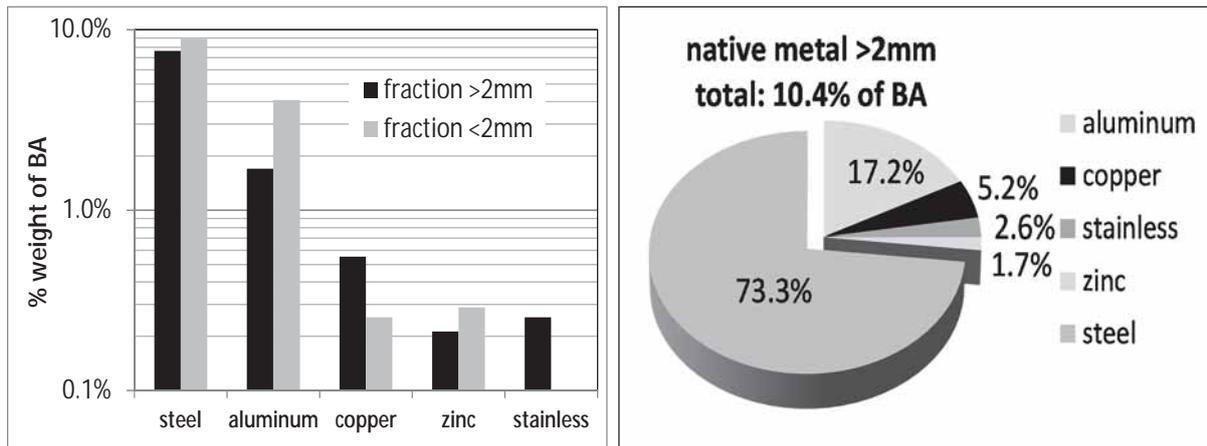


Fig. 3.11: Total mass fractions of metals in BA. Potentially recoverable with conventional technology are only metals >2 mm.

Fig. 3.12: The total mass of potentially recoverable metals >2mm in BA is 10.4% of which approximately a quarter is NF.

Another group of metals not yet discussed is precious metals, namely gold and silver. Potentially recoverable coarse pieces (>2 mm) of these metals will either come as separate particles (jewelry, coins...) or as composites, e.g. gold plated stainless steel wristbands of watches. Sizable amounts of native silver and gold particles are also present in the fraction <2 mm. These originate from electronic devices (silver solder, gold plated contacts) and are not recoverable by means of conventional dry BA processing. From published data and from our own research it appears reasonable to assume a gold content in the order of 0.5g/t of BA (if derived exclusively from MW). If up to 10% shredder residues are co-incinerated, the gold content may increase as high as 2g/t (Table 3.2). Much of the silver comes in the form of eating utensils (“silverware”) which are accumulated in the coarse fraction.

In addition to the metals already discussed, one will also find a surprising number of coins in BA, approximately half of them in foreign currency. Although the number and value of coins in BA varies considerably, one may assume that approximately 1 coin of 5g, having a nominal value of 0.3€, is found per each 100 kg of BA, i.e. a potential value of 3€/t of BA. The coins are usually transferred into in the NF scrap by BA processing. From the NF scrap they can be extracted and further processed to be sold at their nominal value.

A property of great relevance for metal recovery from BA is the particle size of the metal pieces. Coarse particles can be extracted much more efficiently than small particles. Obviously, the particle size distribution of the metals present in BA can vary considerably, depending on the composition of the waste. Fig. 3.13 gives an impression of the particle sizes of metal pieces in BA. While the mean size of FE and SS is around 40 mm, aluminum and brass have mean sizes of approximately 13 mm and copper has a mean size of only 5 mm. It should be noted however, that a sizable amount of the copper is present as wire (Fig. 3.1) which may slip through the apertures of the sieves used for particle size analysis. The nominal particle size of copper particles may therefore be considerably smaller than the geometry of the copper pieces would suggest. As particle size increases so does the ratio of LNF:HNF which is of economic importance as discussed in chapter 7.

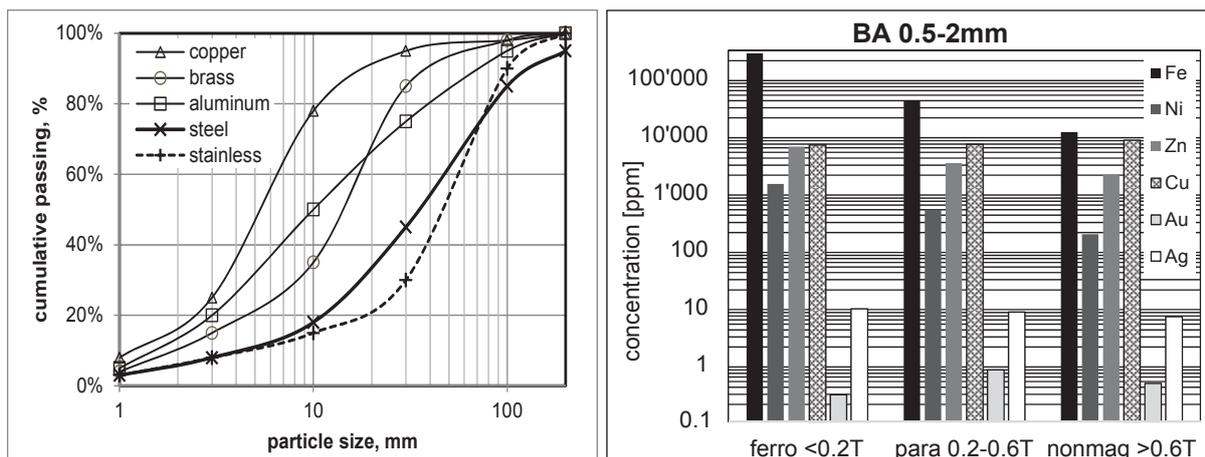


Fig. 3.13: Size distribution of metal particles in BA. SS and FE occur as much bigger pieces than aluminum, brass and copper.

Fig. 3.14: Concentrations of selected metals in BA 0.5-2mm. The relative masses of the fractions were:  $M_{ferromag}=20\%$ ,  $M_{paramag}=20\%$ ,  $M_{nonmag}=60\%$ .

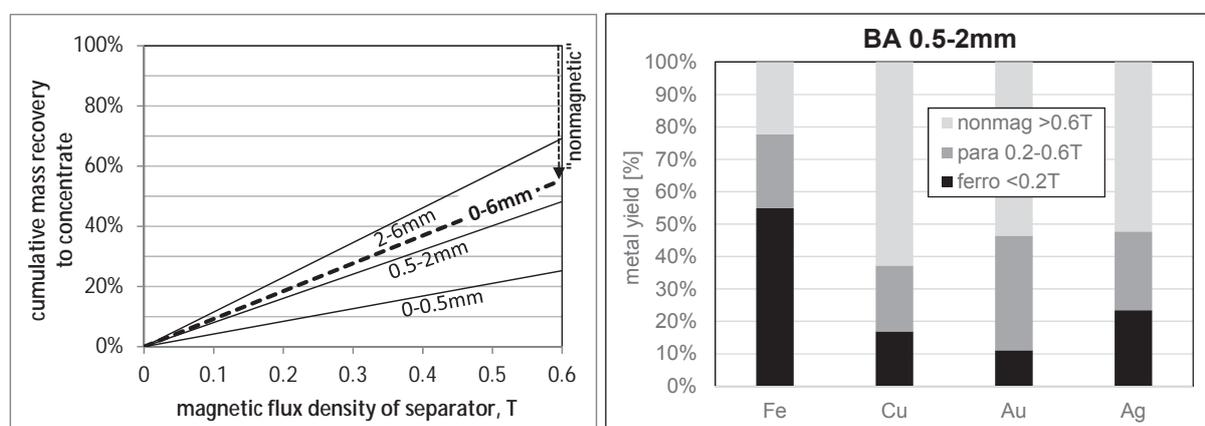


Fig. 3.15: BA <6mm was separated into classes of magnetic susceptibility by increasing the magnetic flux density of the separator. (left) With strong magnets, up to half of the BA <6mm is attracted. (right) Surprisingly, only around 55-65% of the metals Cu, Au, Ag are recovered in the nonmagnetic fraction.

Fig. 3.15 shows the results of magnetic separations of three size fractions of BA. Secondary BA <16mm was comminuted to <6mm and then sieved into fractions 0-0.5 mm, 0.5-2 mm and 2-6 mm (all unbreakable material >6mm, in particular metals, was discarded). These size fractions were then separated into classes of magnetic susceptibility by magnets of surface flux densities 0.07...0.6T. As shown in Fig. 3.15 (left), with strong magnets (0.6T) approximately 50% of the total mass of BA 0-6mm is recovered to the magnetic concentrate. This figure also shows that coarser size fractions (2-6mm) contain more ferro- or paramagnetic material than smaller size fractions (0-0.5mm). Apparently, the coarse mineral particles contain ferromagnetic particles which render the mineral particles ferro- or paramagnetic. If those coarse mineral particles are comminuted to finer sizes, the previously encapsulated magnetic particles are liberated and can then be removed by magnetic separation.

The size fraction 0.5-2mm was separated into three magnetic susceptibility classes and these analyzed for total metal content. Our definitions of ferromagnetic, paramagnetic and nonmagnetic particles are illustrated by Fig. 3.16. Ferromags adhere to a magnet with a surface flux density of 0.2Tesla (=2'000Gauss), paramags to a magnet with flux density of 0.6T, and nonmags are considered all particles that are neither ferro- nor paramagnetic. The data on metal

yield shown in Fig. 3.15 (right) were derived from the data set in Fig. 3.14. Unsurprisingly, iron is highly concentrated in the “ferromagnetic fraction”, less in the paramagnetic fraction and depleted in the nonmagnetic fraction. Surprisingly, however, only slightly more than half of the nonmagnetic metals copper, gold, and silver are recovered in the nonmagnetic fraction. Apparently around half of these metals are, even in the size fraction 0.5-2mm, still encapsulated in mineral particles that also contain magnetic particles.

Fig. 3.17 is a sketch of copper and iron species encountered in the BA. The figure also shows where these species would end up when subjected to (i) gravity concentration followed by magnetic separation or (ii) magnetic separation followed by gravity concentration. If the native copper present in the composite particles minerals+FE+Cu is to be recovered, it must first be liberated. This can be achieved by means of fine grinding. The copper can then be recovered by means of wet processing. These mechanisms most probably also apply to the noble metals Au and Ag.

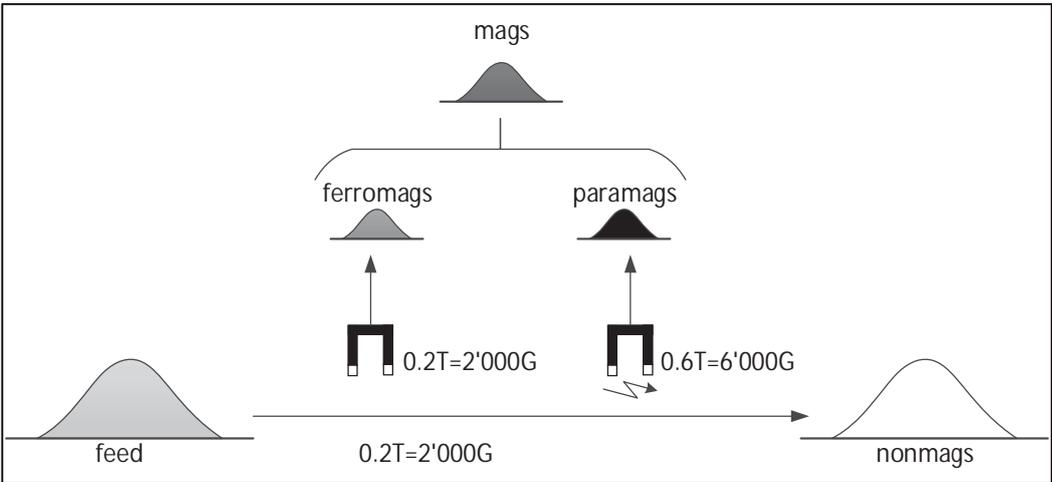


Fig. 3.16: Illustration of our definitions for magnetic particles.

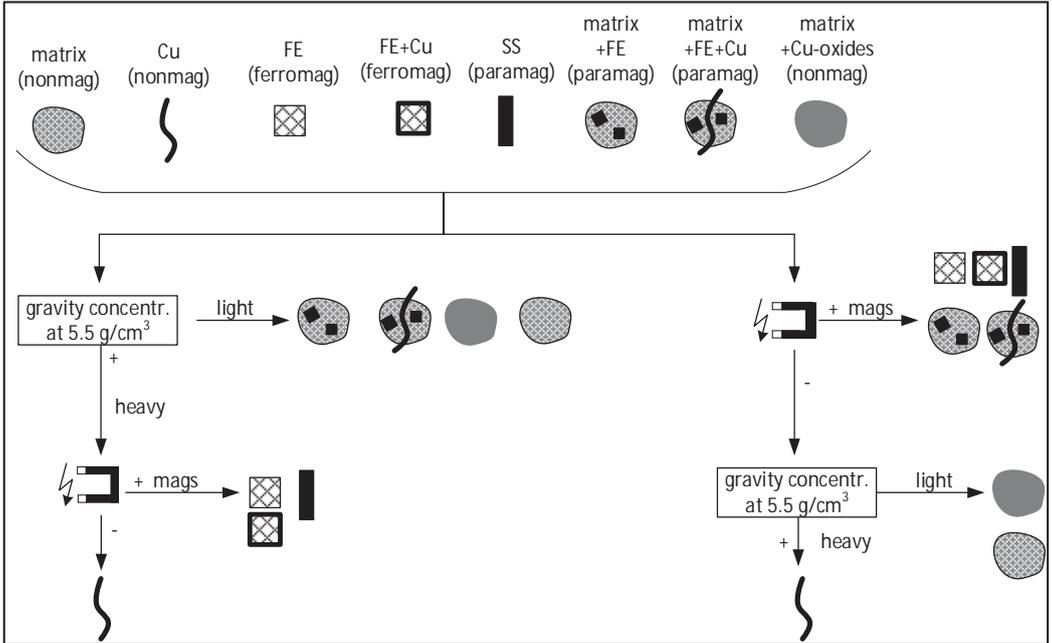


Fig. 3.17: Copper and iron species in BA and their fate when subjected to magnetic and gravity separation processes.

## 4. Equipment for Processing Bottom Ash

### 4.1 Introduction to BA processing

In many aspects, BA processing is similar to processing a primary ore, for example native gold embedded in quartz. By means of mechanical processing, the metal is first separated from the bulk of the matrix material. This metal-concentrate is then passed on to a smelter or a foundry.

For separating metals from mineral matter by means of mechanical processing, physical forces that preferentially act on the metal particles are employed (Fig. 4.1). As a prerequisite one needs to identify a physical property that is more pronounced in the metal than in the mineral matrix material. Properties typically employed for the extraction of metals from BA are:

- Magnetic susceptibility
- Electrical conductivity
- Density

In the separator, the particle stream is subjected to a force acting perpendicular to the general direction of transport. The metal particles are thus forced onto trajectories different from those of the mineral particles and are – after overcoming the edge of a splitter - recovered in the concentrate.

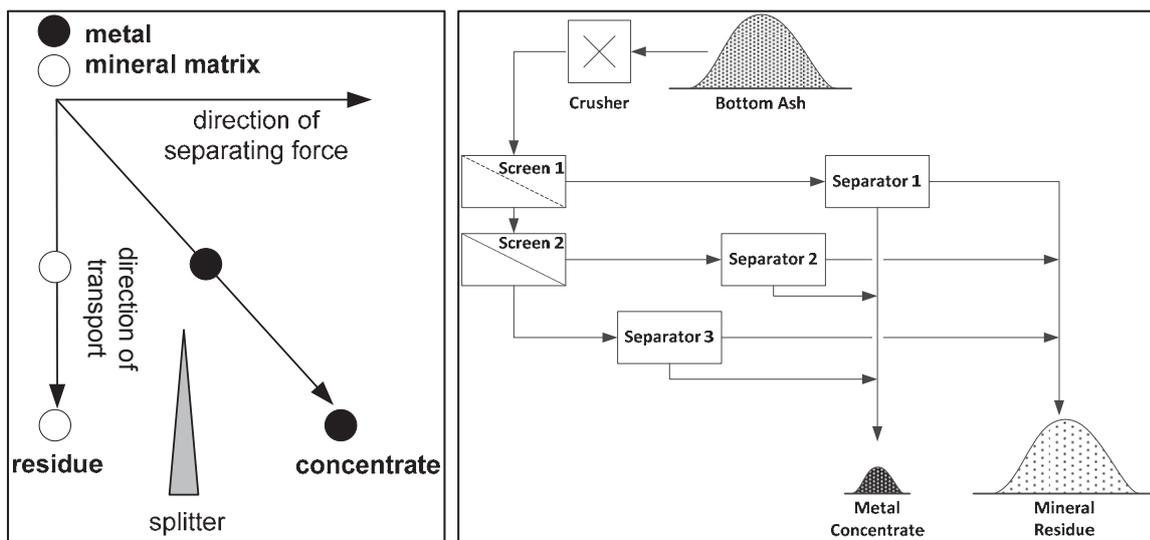


Fig. 4.1: Principle of mechanical separation processes.

Fig. 4.2: Simplified flow sheet of a BA processing plant.

Another prerequisite for the successful separation of a metal from the mineral matrix is that the metal is present as a separate particle and not embedded in the mineral matrix. As the BA can often only be processed after the solidification process has already progressed, the metals need to first be liberated from the mineral matrix. This liberation is effectuated by comminution, i.e. crushing and grinding.

In order to obtain reasonable separation efficiencies, it is not advisable to run the full size range of a comminuted BA through one single separator. Instead one processes narrow size fractions through individual size-dedicated separators. This requires preliminary classification of the BA by screening processes.

In a processing plant, metals extraction is carried out in three stages:

1. Comminution (for liberation)
2. Classification (for better separation performance)
3. Separation (for transferring the metal into the concentrate)

This is illustrated by Fig. 4.2 where the BA is first processed through a crusher and then classified on two screens into three size fractions which are individually processed on three separators.

Some attempts have been made at recovering the metals from BA by hydrometallurgical means, i.e. through dissolution, separation of the pregnant solution from the BA, and then precipitation of the metals out of this solution. This is, however, not feasible with BA for various reasons. An acid extraction, for example the mobilization of copper by sulfuric acid, is not an option due to the high alkaline potential of the BA. Much of acid would be wasted by dissolving the mineral matrix of BA (instead of the copper), rendering the process economically not viable. Environmental concerns also restrict the use of chemicals. Gold, for example, could be leached in an alkaline environment by cyanide; copper by ammonia or organic chelating agents. This is however – considering the stringent environmental legislation in countries that employ MWI – not feasible. In short, hydrometallurgical extraction is technically possible but economically, so far, not viable for extracting metals from BA.

In contrast, the acid leaching of filter ash from MWI may be economically attractive and is, in fact, carried out at a number of Swiss plants. This is, however, only economically feasible if the disposal fees for the filter ash are significantly lower after the hazardous heavy metals have been removed. This may be particularly attractive if the plant uses flue gas washers (instead of dry or semi-dry flue gas cleaning). After the absorption of SO<sub>2</sub> and HCl from the flue gas, the washing solution is a mixture of dilute acids and can be used on site for heavy metals extraction from filter ash. After extraction, the filter ash is disposed of in a low level landfill at a much reduced cost. The extracted heavy metals are either recovered from solution through electrolysis or precipitated as hydroxides and then recycled in dedicated smelters.

## 4.2 Comminution

In BA processing, comminution is carried out with the purpose of liberating the metal particles locked into the mineral matrix as shown in Fig. 4.3 and Fig. 4.4.

The comminution devices most commonly used for BA processing are impact crushers, as shown in Fig. 4.5. Although the conventional impact crushers with a horizontal shaft are commonly used, so called “vertical shaft impact crushers” may be better suited in particular with respect to reduced wear. The implementation of impact crushers may have been inspired by their widespread use in construction waste processing. They are, however, not ideally suited for BA crushing. Better results may be obtained by crushers that exert compressive forces instead of impact. Along this line of thought jaw crushers have been used but they may cause problems when substantial amounts of unbreakable oversize material, such as big pieces of FE, are processed. Such oversize pieces of metal must be removed beforehand. Also, jaw crushers tend to become clogged when processing wet BA that includes unburnt materials such as pieces of wood or books.

One would expect high compression roller crushers to be ideally suited for BA processing. In contrast to conventional roller mills or yaw crushers, these devices do not crush single particles but subject a thick bed of material to high compression forces. The fractures through the particles then propagate preferentially along metal/matrix boundaries. These devices may offer

advantages if the feed contains unbreakable oversize pieces, as long as their diameter is less than the gap width of the high pressure roller crusher. Our own tests with high compression roller mills have however not been satisfactory. The desired selective breakage was achieved, but the BA formed, during the comminution process, rather solid press-agglomerates which required a secondary treatment for dispersing those agglomerates. The process was therefore rated as not economically viable.

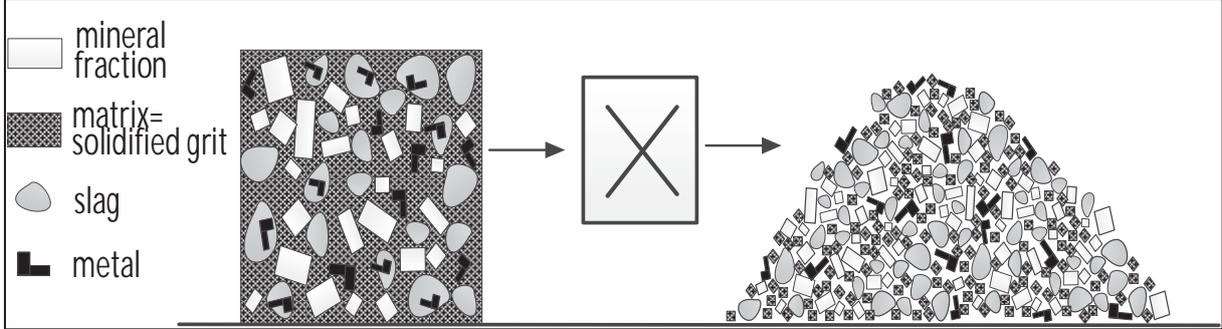


Fig. 4.3: After comminution the metals are fully liberated. Note that the particle size of the all mineral matter has been reduced while the size of the metal pieces remains unaltered.

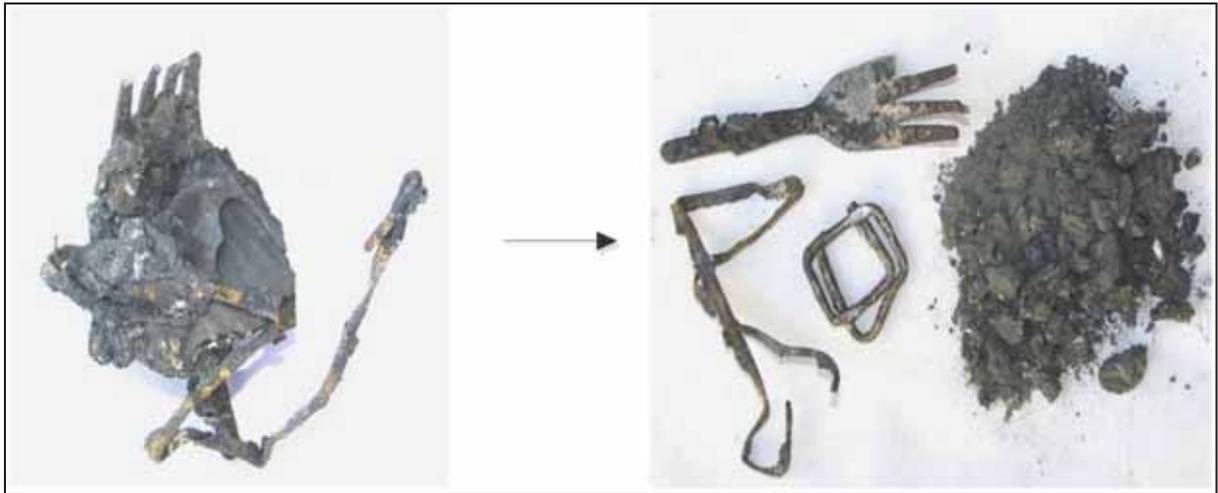
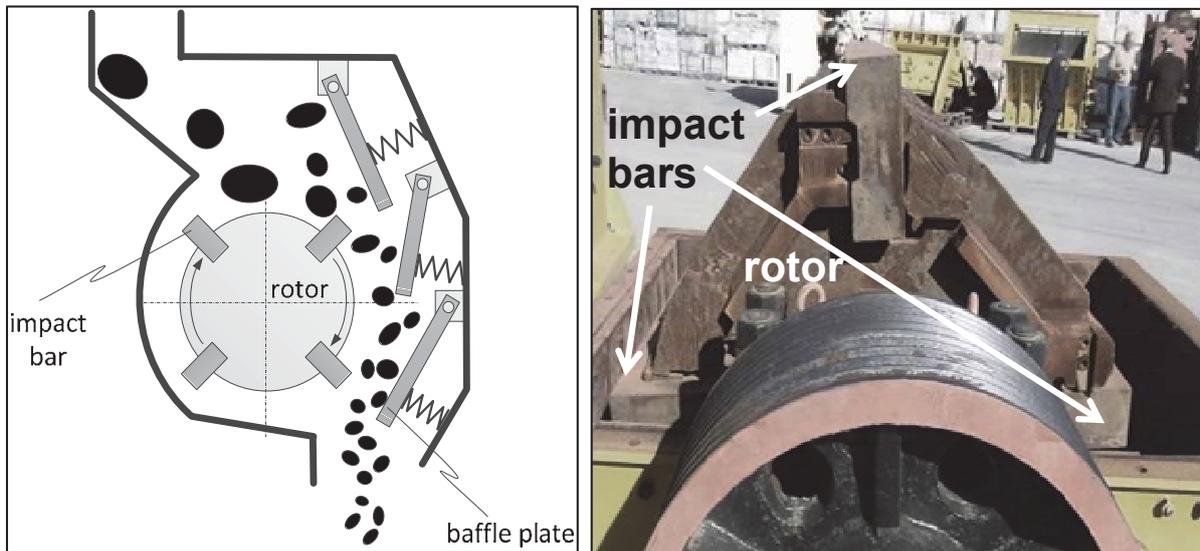


Fig. 4.4: Liberation of pieces of FE and SS embedded in slag: (left) before comminution and (right) after.

When crushing BA, “selective comminution” occurs, as all brittle material, i.e. the mineral material, is reduced in size but not the metals. The metal pieces may be flattened and warped, but will essentially remain intact. If taken to the extreme, one could run the BA through a series of crushers and mills thus pulverizing the mineral matrix. If the material is then processed over a screen, the pulverized mineral matrix would pass the screen while the metals would accumulate in the coarse fraction. While this process is being used in the laboratory in order to determine the metal content (chapter 6.2.3), it is not feasible on industrial scale for a number of reasons.

Obviously, the comminution equipment will be subjected to substantial wear. More important is the loss of metal contained in the bottom ash. Although this loss is insignificant when crushing coarse material, it may become substantial when grinding to finer sizes – in particular, when impact crushers or related devices are used, as these will cause losses by abrasion of soft metals such as aluminum and copper. In countries where the processed BA is used to substitute

for gravel in road construction, excessive comminution is not an option as pulverized mineral material cannot be used for construction purposes. While energy consumption is a consideration, it is not a major factor in economic terms when processing BA. Grinding all BA to, for example, 1 mm size would require only some 3 kWh/t.



*Fig. 4.5: Impact crusher: The rotor with the impact bars revolves at approximately 1000 rpm. The material is crushed by the impact bars and also by impact on the baffle plates. The photo shows the interior after opening the lid with the baffle plates.*

The BA is generally only comminuted to a size that brings about the desired degree of metal liberation. As a rule of thumb one may assume that, in order to liberate a nonferrous piece of metal with diameter  $x$  mm the BA must be comminuted to a size of approximately  $2x$  mm. If, for example, the recovery of NF metal pieces of 4 mm diameter is desired, crushing of the BA to a mean particle size of 8 mm will be necessary. As can be read off Fig. 3.13, around half of the copper contained in BA is smaller than 4 mm and will not be liberated, if the material is not crushed to sizes below 8 mm. Since liberation is the prerequisite for separation, half the potentially recoverable copper (and also 20% of brass and aluminum) would therefore be lost if no comminution to  $<8$ mm is employed.

Only few operators have so far recognized this and implement two stage crushing. The flipside of fine crushing is, as already pointed out, that the pulverized mineral material cannot be later used for construction purposes. If this is an option, one faces an optimization problem: fine grinding will lead on the one hand to increased revenue through increased metal recovery. It will on the other hand lead to decreased revenue because the mineral residue needs to be landfilled, which is more expensive than reuse as construction material.

One option for selectively liberating even fine metal particles without excessive comminution of the mineral matrix is electrofragmentation. In this process the BA is submerged in a water bath and subjected to a series of high voltage discharges. The pulses travel along the interfaces between metal and solidified mineral matrix which are consequently fissured. After application of moderate mechanical stress, the mineral matrix falls apart, setting the metal particles free without excessive fragmentation of the mineral material. This process is currently being tested at the site of a MWI in Switzerland.

### 4.3 Classification

In the context of mechanical process engineering, “classification” is a separation according to particle size whereas “concentration (=“sorting””) is a separation according to physical properties, such as magnetic susceptibility, density or electrical conductivity. For optimum metal extraction from BA in the concentration processes it is crucial, that the material is first classified and that the resulting size fractions are individually run through the separators (Fig. 4.2). This classification is usually performed on screens.

In order to remove large pieces of iron, the feed is often first passed over a bar-sizer (Fig. 4.6 left). As compared to conventional vibrating screens, the bar-sizer avoids the potential problem of blinding (i.e. pieces of material plugging the screen apertures). The oversize from the bar-sizer may then be passed over a heavy duty trommel screen, equipped with punched plates, as shown in Fig. 4.6 (right). An advantage of the trommel screen is that mineral matrix adhering to the metal pieces will chip off thus producing high quality scrap.

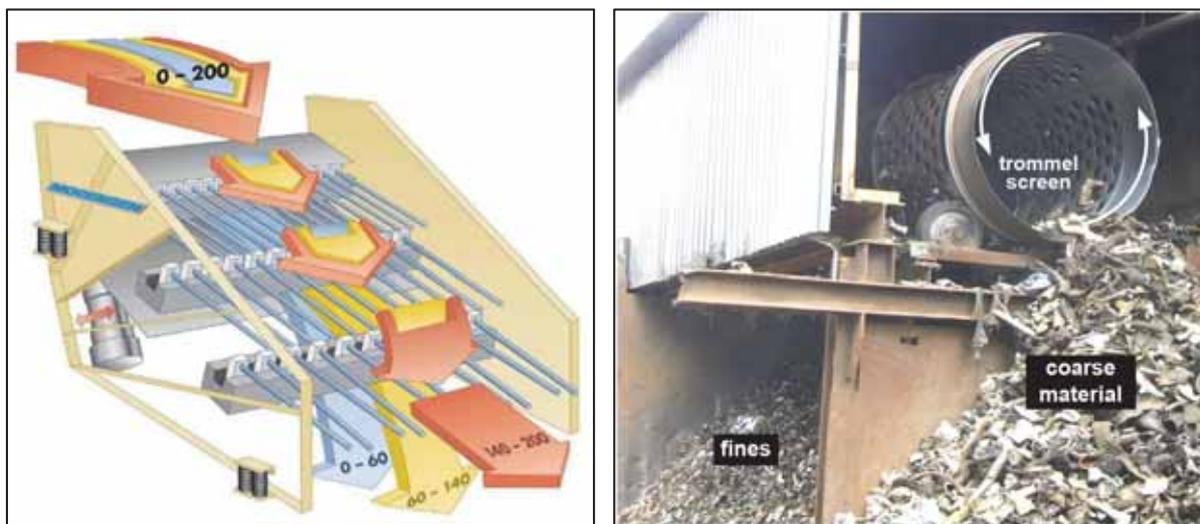


Fig. 4.6: Equipment for the coarse classification of BA: (left) a bar-sizer (drawing by Mogensen) and (right) a trommel screen.

The most common screen used in BA processing is the vibrating screen an example of which is shown in Fig. 4.7. The screen rests on springs and is set into circular motion by a rotating eccentric tappet. The deck provides a multitude of apertures of equal dimensions. The apertures may be round, rectangular or – as shown in Fig. 4.7 right, elongate. As the material hops over the deck of the inclined screen, the fines pass the apertures and collect at the bottom while the coarse particles stay on top of the deck. Problems may arise from particles getting stuck and blinding the screen apertures. This is a quite common effect when longish metal pieces dive into the apertures. When vibrating screens are used, multiple deck screens should be avoided, as the screens must be easily accessible for the periodic removal of pinched particles.

Screen blinding can be avoided with star screens as shown in Fig. 4.8. The “stars” are flexible polyurethane elements which are strung onto rotating spindles. The star elements on adjacent spindles are positioned in a staggered arrangement in order to enable mutual cleaning. While blinding is avoided, the screens must be occasionally checked for wires spooled onto the spindles.

Screening BA at sizes below 10 mm is difficult. In addition to sieve blinding, the BA may be sticky. This can be avoided when the BA, which is dripping wet after extraction, is first cured for a few weeks and then crushed for liberation of the metals. As already discussed the BA

“dries” during the solidification process, as the moisture content is evaporated or used up in chemical reactions.

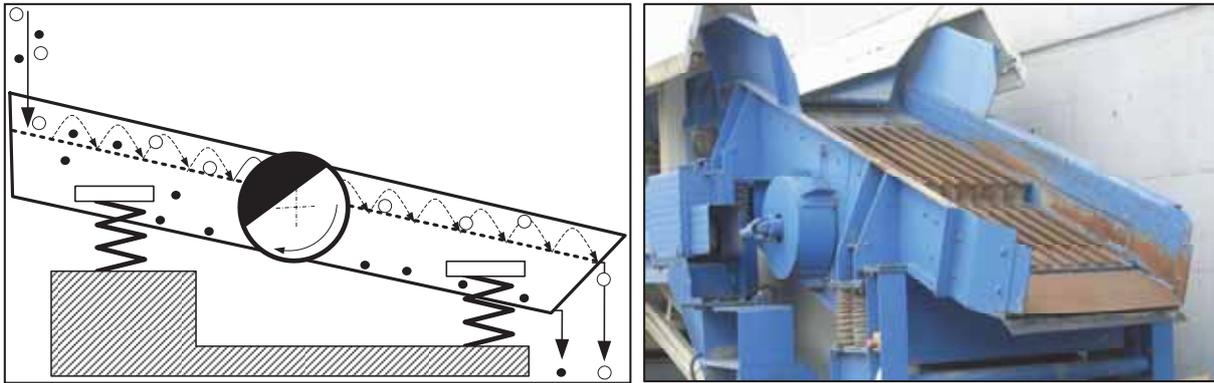


Fig. 4.7: Vibrating screen: in principle (left) and production scale (right).

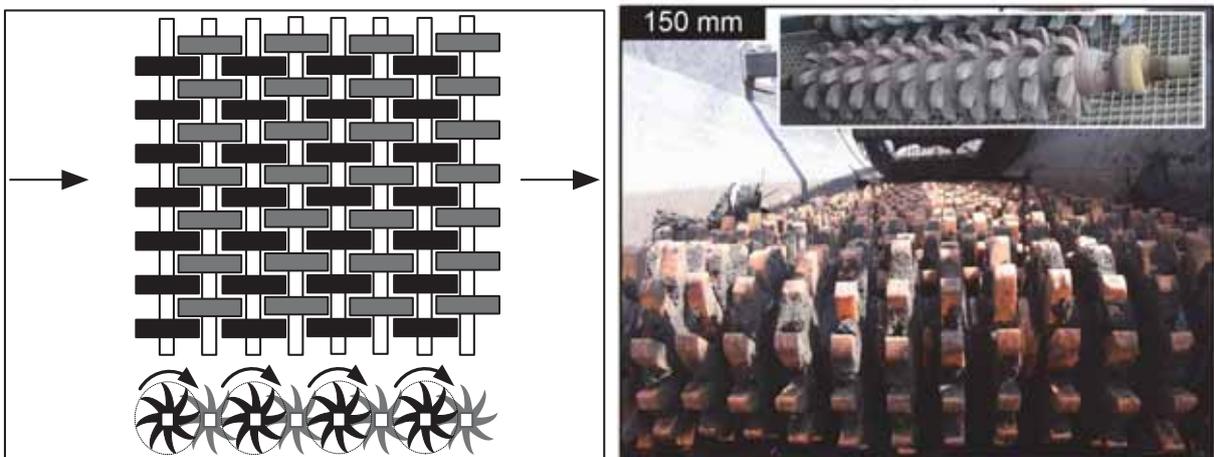


Fig. 4.8: In order to avoid blinding, sizes 10-100 mm are often processed on star-screens.

If the BA is processed right after extraction or the curing is not carried out under a roof, the BA may be wet and sticky which rules out fine screening on conventional vibratory screens. In this case “flip-flow” screens are used. As shown in Fig. 4.9a, the flexible screen mats exert a “trampoline effect” on the material which leads to shear forces strong enough to liberate fine particles sticking to the surfaces of coarse particles. Although BA contains very abrasive particles with sharp edges (e.g. broken porcelain), the flexible sieve mats are surprisingly resistant to wear. The minimum split size attained on such screens is in the order of 3 mm. The apertures are often not rectangular but slotted.

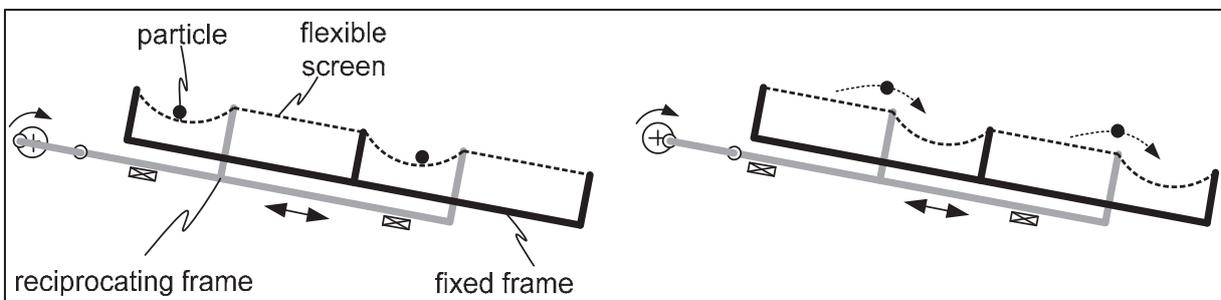


Fig. 4.9a: Schematic of “flip-flow” screen, used for the classification of moist and “sticky” BA.



Fig. 4.9b: (left) Installation of a “flip flow” screen. (right) BA on a 16 mm deck.

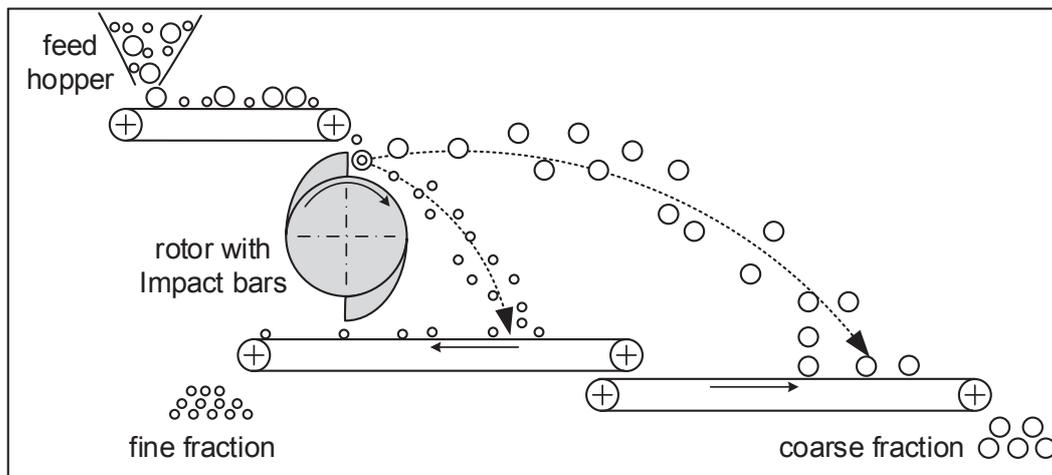


Fig. 4.10: “ADR” technology for separating moist BA.

The difficulty to separate wet “sticky” BA at small particle sizes on screens has brought about unconventional devices, such as the “ADR”, that classify particles according to their individual trajectory parabolas. One such device is a rotor with protruding impact bars, somewhat similar to the rotor of the impact crusher depicted in Fig. 4.5. The material is fed vertically onto the rotor, the bars capture the particles and accelerate them until they detach and fly along the flight trajectories determined by their size (the larger particles flying farther than the smaller ones). Other devices employ a throwing belt instead of a rotor for the same purpose. Mostly, these devices are used to remove the sticky fines (<2mm) before processing wet BA, as the fines may upset the separation processes, for example by forming sticky layers on conveyor belts or buildup in crushers or separation devices. The devices may be enhanced by “air-knife” technology where the material passes under an air blower to remove the fines.

If dry BA is processed, one may be able to classify at split sizes as low as 0.5-3 mm. As mentioned above, sieve blinding by elongate particles may become a major problem with conventional vibrating screens. An alternative may be gyratory screens (Fig. 4.11). As the screen is moved horizontally, the material does not “hop” but rather glides over the screen mat. This prevents longish particles from vertically diving into apertures too small for their passage. Gyratory screens have been found very effective in recovering small pieces of copper wire from BA when installed downstream of a vibrating screen with approximately the same screen aperture size. On the vibrating screen the wires slip through the apertures but they are then recovered on the gyratory screen of equal aperture size.

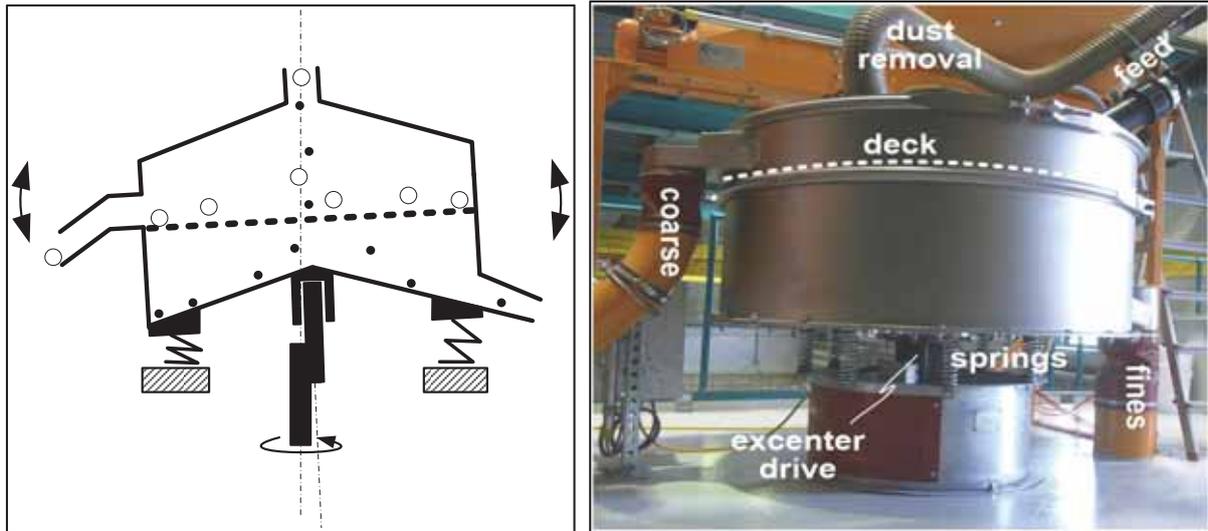


Fig. 4.11: Gyrotory screens are employed, if the feed contains fibrous and elongate particles.

#### 4.4 Sorting

Central to metal extraction from BA are mechanical sorting processes. The material is conveyed through the separator, where physical forces act preferentially on the target particles (metal) thus removing them from the particle stream. These physical forces can be either of mechanical or electromagnetic nature. The separators most common in BA processing plants are magnetic separators and eddy current separators which are therefore discussed here in some detail. Other devices being used in some BA processing plants, but not as commonly as the above, will be briefly introduced.

##### 4.4.1 Magnetic Separators (MS)

Magnetic separation recovers particles with a marked magnetic susceptibility. For the purpose of processing of BA one can distinguish 3 fractions (Fig. 3.16):

- Ferromags: strongly magnetic particles (FE)
- Pramags: weakly magnetic particles (SS, iron oxides)
- Nonmags: nonmagnetic particles (glass, nonferrous metals...)

While strongly magnetic particles can be recovered with low intensity magnets (LIMS), the recovery of the weakly magnetic fraction requires high intensity magnets (HIMS). The “magnetic flux density” is used for describing the “strength” of magnets and is measured in Tesla (1 T=10<sup>4</sup>000 Gauss). In the context of BA processing, LIMS are defined as MS with surface flux densities <0.2 T and as HIMS devices are defined with flux densities exceeding 0.2T, in particular 0.6 T. Particles that are not recovered by HIMS with 0.6T are considered “nonmagnetic”.

All magnetic separators employed in BA processing are of the “lateral pole” kind, where the magnetic poles are arranged next to each other as shown in Fig. 4.12. In contrast to the “opposing pole” design, where the material passes between facing magnetic poles (e.g. through the gap of a “horseshoe” magnet), the material cannot become jammed in the separation gap. In modern MS it is quite common to install permanent magnets (as opposed to electromagnets). For LIMS, ferrite magnets are often preferred (because cheaper), while for HIMS “rare earth” magnets based on Neodymium are used.

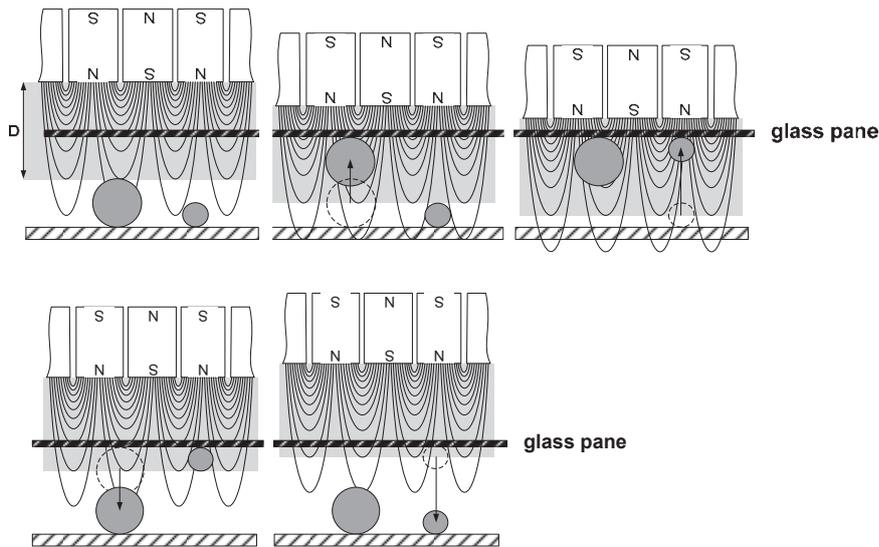


Fig. 4.12: Schematic of the “lateral pole” design of magnetic separators.  $D$  is the outer boundary of the zone of attraction for a particular material (e.g. steel). As the magnet approaches the steel spheres, the bigger sphere is attracted first (top). Upon withdrawing the magnet from the glass pane, the big sphere drops first (bottom).

It is important to note that the attraction decreases exponentially with distance from the surface of the magnetic poles. A given magnet has a defined “zone of attraction” for a given material, say FE – regardless of the size of the object. Only if the center of gravity of a given FE object lies within this zone of attraction, it will be attracted against gravity.

Fig. 4.12 (top) shows – in a very simplified manner - what happens, if a magnet with an outer limit of the zone of attraction  $D$  approaches two steel spheres positioned next to each other on a table from above.  $D$  depends on the material to be attracted (e.g. steel) and the composition of the magnet (e.g. NdFeB). Between the magnet and the spheres a glass pane is installed. The magnet approaches from above until the center of gravity of the large sphere lies within  $D$ . At this point the sphere is attracted and pinned to the glass pane. Only when the magnet is further approached to be close enough for the center of gravity of the small sphere to lie within  $D$ , will the small sphere also be attracted. On withdrawing the magnet again (Fig. 4.12 bottom), the large sphere will drop first and only if the distance of the magnet is further increased will the small sphere also drop. The first part of the experiment (approach) would suggest that large steel particles are “more strongly attracted” than small particles while the second part of the experiment (increasing distance) suggests just the opposite. Truth is that the zone of attraction  $D$  is (ideally) independent of particle size.

For recovering magnetic material of sizes  $<80$  mm, drum magnets are the equipment of choice. They comprise of a stationary magnet around which a drum of nonmagnetic material revolves. The magnetic particles are pinned onto the drum, which rotates them out of the magnetic field. The deflection magnet (Fig. 4.13 left) preferentially attracts small particles while the extraction magnet (Fig. 4.13 right) favors the pickup of large particles. For BA processing, the deflection magnet is generally used. As the deflection magnet carries even pieces of FE with adhering thick crusts of mineral material into the concentrate, the recovery is good, but quality of the concentrate is poor. In contrast, the extraction magnet produces a good scrap quality – but at the expense of recovery: much of the FE is lost into the nonmagnetic residue. Extraction magnets are occasionally used (as well as overhead magnets) for removing tramp steel from the unsolidified BA immediately after wet extraction (on site the MWI). A discussion of grade/recovery curves follows in chapter 5.2.2.

BA processing plants recover FE from the BA for two reasons. First, FE can be sold for a profit and secondly bulky chunks of FE negatively affect the properties of the mineral residue. If the residue is used for road construction large metal pieces must be removed anyway and if the mineral residue is landfilled, bulky metal pieces prevent optimum compaction. Essentially, for BA-processing three types of MS are being used:

- drum magnet deflecting
- drum magnet extracting
- overhead magnet

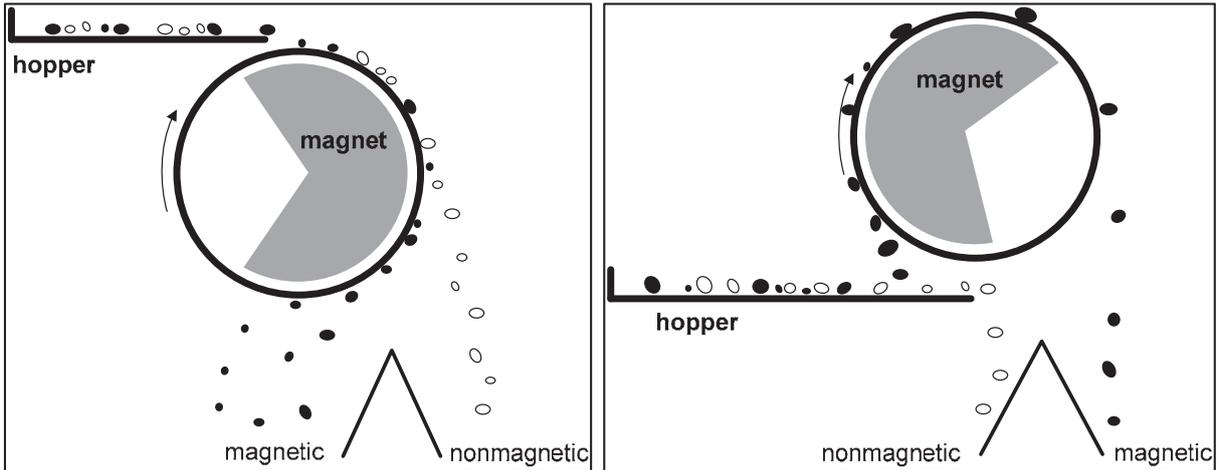


Fig. 4.13: Drum magnets: (left) deflection magnet, (right) extraction magnet.

Overhead magnets are either equipped with electromagnets or with permanent magnets. The design of an overhead magnet is shown in Fig. 4.14. The material travels on the conveyor belt and follows – after having passed the head pulley – the free fall trajectory parabola. FE is attracted by the magnet, gets pinned onto the belt that runs around the magnet, is then drawn by the belt out of the magnetic field and falls into the concentrate bin. The overhead magnet is usually installed as LIMS upstream of the crusher for removing oversize FE of size >80 mm.

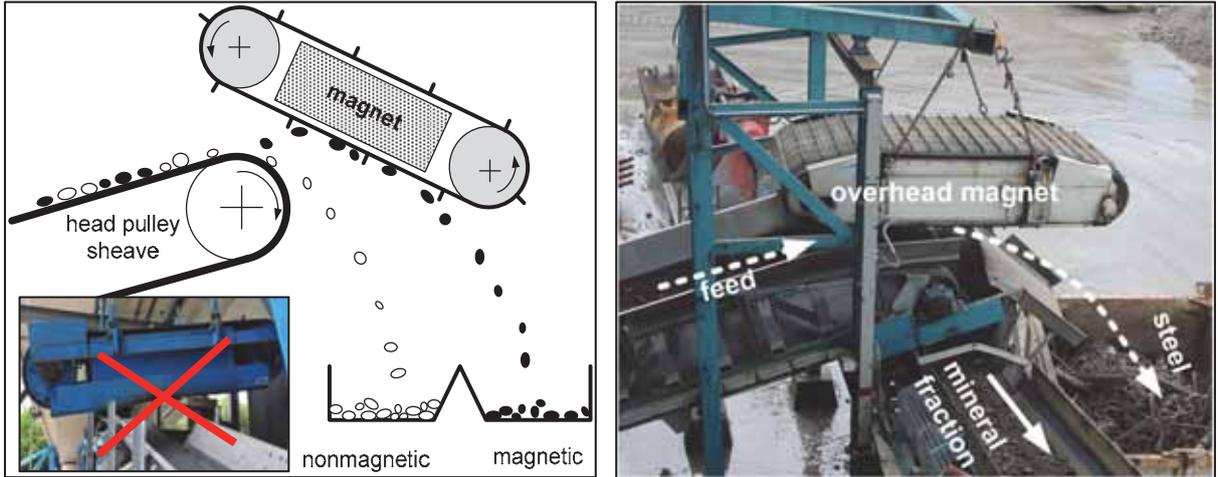


Fig. 4.14: Overhead magnet: schematic and photo. A “crossbelt design” (lower left corner on drawing) must be avoided if coarse BA is processed.

Frequently, the overhead magnet is installed as “crossbelt magnet” perpendicular to the belt. For processing coarse BA this design must be avoided. First, pieces of FE that are buried below

the overlying material layers, cannot be separated reliably. Much worse however is the following scenario. Occasionally, one end of an elongate piece of FE (e.g. a pole), that has wiggled itself through the primary screen, will be attracted by the magnet, then become jammed between conveyor belt and crossbelt, then pierce the conveyor belt and become jammed behind a pulley and eventually cause substantial damage to the belt. The author has yet to see a crossbelt magnet employed for processing coarse BA with no puncture holes in the belt underneath.

Our analyses of large amounts of ferromagnetic steel scrap extracted from BA support the strong affinity of copper and steel as discussed in chapter 3.2.2. The concentration of copper in the raw scrap (as extracted by the magnets) was in the order of 1.3% of the total mass. After processing the steel scrap through an automobile shredder and removing the adhering mineral matter and oxide coatings (through screening) as well as removing copper/iron-composites (electric motors, transformers...) by hand picking, the concentration of copper in the steel scrap was still 120g Cu / 80kg FE = 0.15% (see also Fig. 3.10).

#### 4.4.2 Eddy Current Separators (ECS)

In addition to magnetic separators, eddy current separators are “workhorses” of BA processing plants, as they are able to recover electrically conductive materials, such as nonferrous metals (NF).

Major parts of ECS are a conveyor belt which travels at 1-2m/s, a head pulley, a rotor with strong magnets attached to its surface, and a splitter (Fig. 4.16). The rotor is often positioned off center within the pulley sheave (STEINERT design) and rotates at much faster speeds than the sheave (e.g. 2'000 rpm). The magnets on the rotor surface generate an eddy current inside electrically conductive particles. This current, in turn, produces a magnetic field that causes the particles to be repelled by the rotor. Nonconductive particles remain unaffected and follow the trajectory parabola. A splitter is positioned between the trajectories of the nonconductive and the conductive particles.

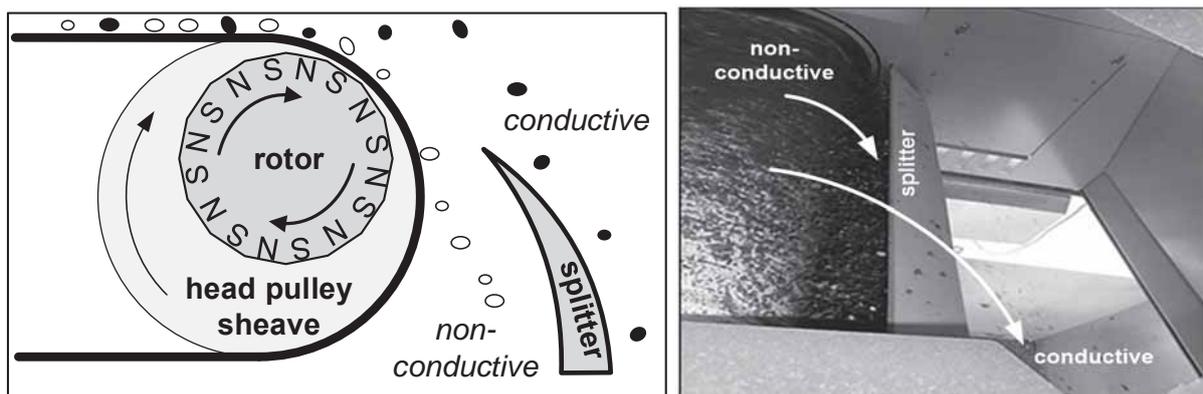
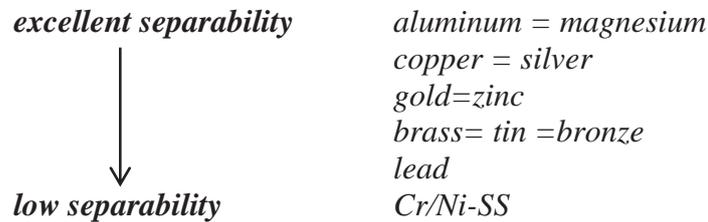


Fig. 4.16: Eddy Current Separator ECS: Schematic of principle (left). View from diagonally above into the separation chamber (right). On the photo one can see NF particles passing over the splitter while the mineral material dives under the splitter.

Since the material of the sheave must be nonconductive, reinforced plastic is used. Obviously, the belt cannot be reinforced with a conductive material either. Because the magnetic flux density, as pointed out above, decreases exponentially with distance, the gap between the rotor and the feed material must be minimized. Therefore, both belt and sheave are manufactured as thin as possible.

Although they are electrically conductive, the magnetic attraction of ferromagnetic particles exceeds the repelling force by far. The trajectories of SS, which has a much lower magnetic susceptibility than FE, depend much on the exact composition of the alloy. While chromium steels are lost into the nonconductive residue, Cr/Ni-steels like V2A and V4A will sometimes be recovered in the conductive concentrate.

The repulsive force depends on the conductivity of the material – the higher the conductivity, the stronger the repulsion. However, opposed to the direction of the repulsive force (vertical upward component) the gravitational force pulls vertically downward. As the gravitational force depends on particle density, the ratio of conductivity and density can be used as an indicator for the resulting force which determines the particle trajectory. For example silver, due to higher conductivity would be expected to be more repulsed and therefore separate better than aluminum. But since silver has a much higher density than aluminum the ratio of conductivity and density determines that aluminum is in fact more strongly repelled by the ECS. Using these ratios as indicators for potential separation efficiency leads to the following ranking from good to bad separability on ECS:



Because the repulsive forces are rather subtle they do not allow for the separation of metal pieces that are locked within a mineral matrix: only fully liberated metal pieces are amenable to separation on ECS. Consequently, comminution to fine particle sizes is the key to the recovery of fine grained metal particles.

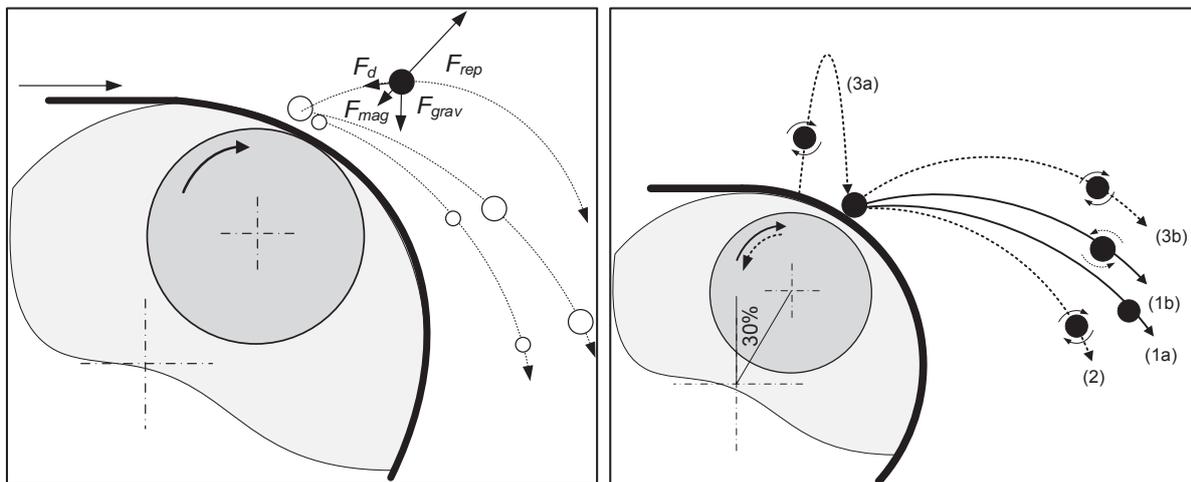


Fig. 4.17: (left) Forces acting on a conductive particle with magnetic properties. (right) The spin of the conductive particle is opposed to the direction of the rotor spin.

The trajectories of particles on ECS are determined mainly by four forces (Fig. 4.17 left):

- $F_{mag}$  Magnetic attraction (depending on magnetic susceptibility)
- $F_{rep}$  Eddy current repulsion (depending on electrical conductivity and shape)
- $F_{grav}$  Gravity (depending on particle density)
- $F_d$  Aerodynamic drag force

When metal particles  $>4$  mm are considered, the aerodynamic forces, in particular drag, can be neglected. Not so, if small particles are to be separated. At sizes  $<2$  mm air drag plays a major role and the above ranking does not hold any more. Provided equal size and shape, dense particles are relatively less affected by air drag. For example copper particles 2 mm in size follow approximately the same trajectory as 2 mm aluminum particles. At size 1 mm copper particles fly farther than aluminum particles of the same size and shape.

The aerodynamics of small particles is also influenced by their spin. When passing over the rotor, conductive particles are not only repelled, but they also start spinning in the opposite direction of the rotor. While this effect is – with respect to aerodynamics – negligible for large particles ( $>8$  mm), it becomes very pronounced at small particle sizes. Generally and as a default, the rotor rotates in the same direction as the sheave (co-rotational = clockwise in Fig. 4.17) and the particles spin counter-rotational. When rapidly spinning particles travel through air, a Magnus effect can be observed (also employed for the “banana kick” in soccer). Thus, the trajectory (1b) of the counter-rotationally spinning particle in Fig. 4.17 is flatter than the trajectory (1a) of a non-spinning particle, leading to a better separation.

Occasionally it is claimed that a counter-rotational rotor may lead to better separation with small particles. From theory one would expect just the opposite. The particle with a rotational spin should “dive” due to the Magnus effect and hence lead to a worse separation along trajectory (2). By using high speed cameras we have been able to establish that, with a counter-rotational rotor, small particles already begin to spin and climb vertically before reaching their proper  $30^\circ$  takeoff position (trajectory (3a) in Fig. 4.17). They then fall back onto the conveyor belt and, due to the rotational energy stored in their spin, bounce off horizontally as shown in Fig. 4.17 (3b). The effect is somewhat comparable to a tennis ball sliced with a top spin. This effect becomes particularly pronounced when setting the rotor at an angle less than the  $30^\circ$  position with respect to the axis of the head pulley sheave (as shown in Fig. 4.17 right). A counter-rotational rotor will therefore have two opposing effects on the particles. They will spin, rise and then bounce off horizontally (which is favorable) but, once airborne, they will tend to dive due to the Magnus effect (which is unfavorable). Our research has shown that small metal particles fly, on average, indeed farther with a counter-rotational rotor, but their trajectories display an extremely wide spread (between trajectories (2) and (3b)), overlapping substantially with the trajectories of the coarse nonconductive particles. In summary, we have not been able to establish that a counter-rotational rotor spin is advantageous if an ECS with an off-center rotor is used. This may be different with concentric rotor designs.

Among the many clever devices P.C. Rem and his group at TU Delft have invented for metals extraction from BA is a “wet ECS” using the spin of conductive particles suspended in water for separation, and also an ECS with counter-rotational rotor for separating moist particles sticking to the conveyor belt.

Particle shape is a major determinant with ECS. The best separation results are obtained with flat or compact pieces of metal. In elongate particles, like wires, the eddy current necessary for separation cannot be fully established and hence the repulsive force is low. This effect is also observed with particles that appear macroscopically homogenous and compact but include internal fissures which prevent the establishment of a vigorous eddy current.

Strongly magnetic particles must be removed from the feed, before the material passes the EC, for two reasons. First, they interfere with the magnetic field and thus prevent nonmagnetic conductive particles in their vicinity from establishing the eddy currents necessary for their separation. Second, in case of an emergency shutdown the belt would stop immediately, while the rotor, due to substantial inertia, will only slowly come to a halt. Meanwhile, magnetic

particles within the zone of magnetic attraction of the rotor would remain pinned onto the belt and the induced eddy current would cause their temperature to rise to red heat, eventually melting them into the belt.

While traditional ECS work well at particle sizes  $>6$  mm, recent developments are geared towards recovering ever smaller pieces of NF from BA. Currently, the recovery of 0.5 mm NF particles can be considered as state of the art. On cursory inspection, an ECS dedicated to fine particles processing differs little from ECS for the separation of coarse particles. On closer examination some differences are observed. For example, the rotor spins much faster and more magnetic poles are positioned on its surface with a much closer spacing thus leading to increased frequencies of the alternating magnetic field. Just as important as a powerful separation unit is an optimized periphery. For efficient separations, especially at small particle sizes, the following considerations are important:

- The material should be fed with a vibrating hopper onto the belt of the ECS. On vibrating hoppers, elongate particles align themselves in transport direction which is a prerequisite for recovering, for example pieces of copper wire, with an EC. If positioned not in transportation direction but perpendicular, elongate particles are not separated on ECS. One problem when conveying moist BA with vibrating hoppers is the buildup of fines on the hopper surface. Periodic removal of such scales is therefore of great importance.
- Not only strongly magnetic but also weakly magnetic particles should be removed before separation because they may interfere with the magnetic field necessary for inducing sufficiently strong eddy currents in small NF particles. This calls for a HIMS to be installed upstream of the ECS. As discussed in chapter 2.3 almost 50% of the feed material is magnetic and can potentially be separated by HIMS. Be aware however, that according to the mechanisms shown in Fig. 3.17, up to 50% of the copper, gold and silver can be trapped within these magnetic particles.
- Overloading must be avoided. The belt must not be covered with more than a monolayer of material. This imposes significant limitations to throughput.
- The belt (and the feed hopper) must be kept clean, as the buildup of deposits, in particular if they contain magnetic particles, is detrimental to separation efficiency. Consequently – when processing moist BA – scrapers and brushes must be installed for cleaning the belt.
- Although some ECS that use concentric rotors (with respect to the axis of the head pulley sheave), the dominant design is the off center arrangement which offers various advantages. The optimum angle of the axis of the rotor in relation to the axis of the head pulley sheave depends, among other things, on particle size. Generally, the rotor is set at approximately the “one o’clock position”, i.e. at  $30^\circ$ , as shown in Fig. 4.17 right. For coarse particles a setting of around  $15^\circ$  has been shown to be favorable while fine particles tend to better separate at a rotor setting of approximately  $45^\circ$ . In any case, the belt speed must be adjusted such that the particles do not slip on the belt while passing over the rotor.
- The splitter setting must be optimized and controlled in regular intervals. Some experts claim that the splitter must be set such that its edge is on level with the axis of the head pulley sheave (or even below), in order for the “clouds” of conductive and nonconductive particles to separate. Others argue that the splitter must be positioned

much higher to capture the deflected small conductive particles before their trajectory intersects that of coarse nonconductive particles. The optimum splitter setting depends on the separation issue at hand and is best determined experimentally.



Fig. 4.18: NF recovered by ECS: (left) mostly aluminum. (right) NF concentrate from a sensor sorter (after ECS): mostly SS.

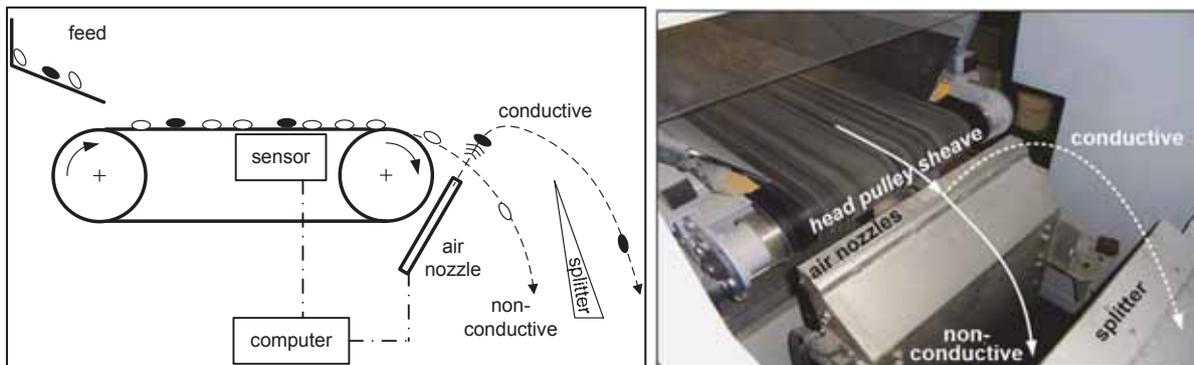


Fig. 4.19: (left) Functional principle of sensor sorting. (right) View into the separation chamber.

#### 4.4.3 Sensor Sorters

While all BA processing plants employ MS and ECS, sensor sorters are only used in very few plants. As they compete with hand picking they are employed mostly in countries with high labor costs.

The functional principle of sensor sorting is shown in Fig. 4.19. The device comprises a conveyor belt, a metal detector (sensor) and a bank of air nozzles that are linked to a pressurized air supply. The material travels on a conveyor belt with approximately 3 m/s and passes a metal detector, installed below the belt. If a piece of metal passing overhead is detected, the signal is passed on to the computer, which calculates the time until the particle is expected to be positioned on its parabola trajectory just above a particular air nozzle. At this time the nozzle is activated and the metal particle is driven by the air blast over the edge of the splitter.

There are two major advantages of sensor sorters over ECS:

- All metals can be recovered regardless of magnetic properties and shape.
- Metal particles that are still trapped within a mineral matrix can be recovered.

As can be seen from Fig. 4.18 right, the concentrate from the sensor sorter comprises a large number of longish pieces that are SS, such as pieces of silverware. In this fraction one will also find a fair amount of the coins contained in the BA. On the downside, sensor sorters are expensive in both investment and operation, require regular maintenance and may cause problems with dust. The operating cost of sensor sorters depends strongly on the consumption of pressurized air. In order to minimize operating cost, sensor sorters are installed downstream of the much cheaper MS and ECS (although all metal >6 mm that is recovered by magnets and ECS would also be recovered by sensor sorting).

Sensor sorters can be used for separating HNF and LNF from NF mixtures produced by ECS. In this case the attenuation of an X-ray beamed through the material is used, in combination with laser-supported shape analysis) as X-ray attenuation is correlated with the density of the object.

4.4.4 Hand Picking

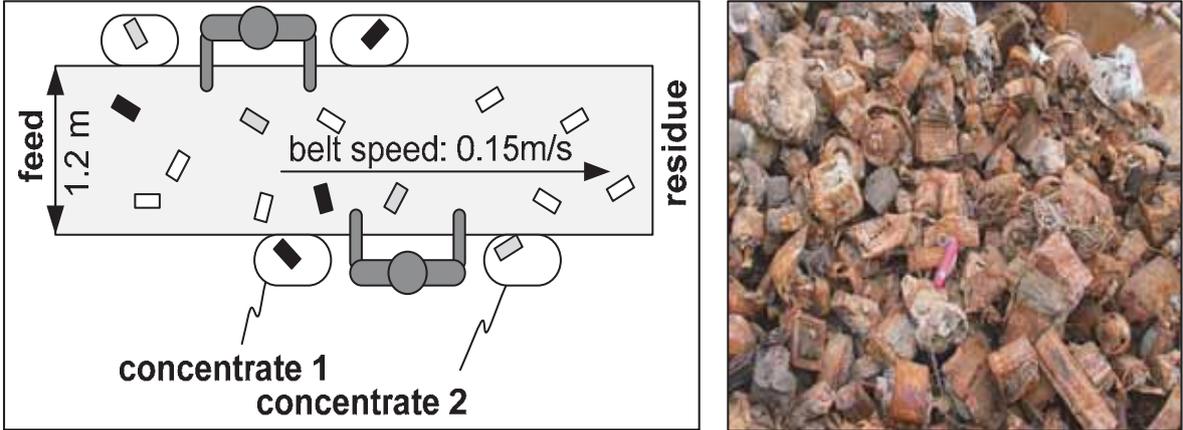


Fig. 4.20: Handpicking station.

Fig. 4.21: Transformers and electric motors handpicked from BA.

Even where sensor sorters are applied, hand picking is often used for sorting large pieces of metal into saleable scrap fractions. For example, coils of copper around an iron core (Fig. 4.21) are, unless removed by handpicking, transferred to the magnetic fraction of a BA-processing plant and eventually end up in the steel smelter. Not only would the potential profit for the sale of this copper be lost but steel foundries often impose stiff penalties on copper contaminated steel scrap.

The typical layout of a hand picking station is shown in Fig. 4.20. A belt of approximately 1.2 m width conveys the material at a speed of 0.15 m per second. The workers pick the target material off the belt and throw the pieces into designated bins. Typically 3-5 bins are provided, e.g. for SS, copper/iron composites, copper&brass, silver, and unburnt organics. The remainder is left on the belt and is transferred to the residue. Typically, the feed to the hand picking station is the coarse fraction of a classification at 100 mm. Hand picking stations must be equipped with adequate lighting and provisions against dust.

It is important to keep in mind that hand picking leads to good concentrate qualities, but recovery may be very poor if the workers are not properly supervised.

#### 4.4.5 Gravity Concentration

One way to separate the generally denser metals (e.g. copper with density  $8.9 \text{ g/cm}^3$ ) from the less dense mineral matter (e.g. glass with  $2.5 \text{ g/cm}^3$ ) is gravity concentration. Dry gravity concentration processes are generally limited to handling particle sizes 0.2-4 mm and could, at least in principle, be considered for processing BA. A prerequisite is that the metals are completely liberated and be present as free flowing bulk solid. These conditions are generally not met by BA because either the metals are liberated (right after extraction from the incinerator) but wet and sticky. Or - after the solidification process has advanced - the moisture content would be low enough to allow gravity concentration, but the metal pieces are locked into the solidified mineral matrix. In order to liberate these metal pieces one could grind the BA to a size of approximately 4 mm and then process them by means of gravity concentration. This is, however, economically not feasible and would also lead to massive problems with the generation of dust.

An option however, would be wet gravity concentration of BA before the solidification of the matrix has commenced. In this case the material is processed in an aqueous suspension which avoids both stickiness and potential dust problems. As a first step, the material  $<4 \text{ mm}$  is separated from the BA which can, for example, be accomplished on vibrating screens equipped with water sprays. The material  $<4 \text{ mm}$  would then be deslimed at, for example  $0.1 \text{ mm}$ . While the material  $>4 \text{ mm}$  is processed dry with the equipment discussed above (e.g. ECS), and the slimes are discarded, the material  $0.1\text{-}4 \text{ mm}$  can now be processed with devices were originally developed for recovering placer minerals or gold. One should note however, that some such devices do not work properly if “odd” particle shapes are processed. For example “spiral concentrators” are not able to successfully recover small pieces of copper wire despite the rather high density of copper. On the other hand, shaking tables, jigs, and centrifugal devices, have been shown to successfully extract heavy metals from BA. Due to the fact that aluminum has approximately the same density as the mineral material (glass, slag...), this metal can, as a matter of principle, not be separated by gravity concentration processes.

In gravity concentration processes involving BA  $<2 \text{ mm}$  the ratio of copper:silver:gold present in the BA of approximately  $(10'000):(20\dots50):(0.7\dots1.5)$  is maintained also in concentrates and residues. This is an indication of gold and silver being present either in native form or at least in composite with heavy minerals or metals that behave in gravity concentration processes similar to copper. Once the actual ratio Cu:Ag:Au has been determined for a particular BA, the analysis for copper can be used for estimating the concentrations of the precious metals in a given product. Direct analyses of gold and silver are difficult to determine directly due to the “nugget effect” (as discussed in chapter 6).

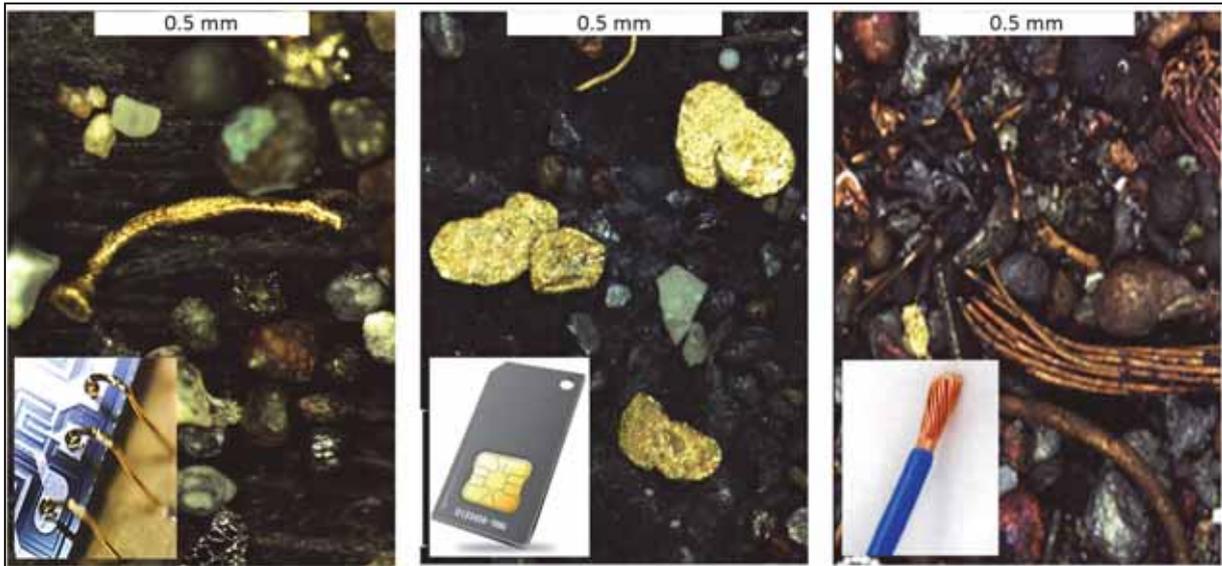


Fig. 4.22: Metals recovered from BA by gravity concentration: gold originating from ball bonded contacts (left), contact foils (middle) and copper from litz wire (right). Even the longest dimensions of these objects are approximately only 0.5 mm.

4.4.6 Unconventional Equipment

At least one company (in Germany) is specialized in extracting coins which are accumulated in size fractions 16-32 mm of the NF concentrates. First, the coins are further concentrated with specialized equipment that uses the disc-shape of coins as a separation criterion. The coins are then cleaned and polished and run through an automatic sorter. This device scans the front as well as the flipside of each coin, then compares the scan with images of coins stored in its data base, and accordingly transfers the coins into receptacles.



Fig. 4.23: Coins recovered from the NF fraction of BA: (left) after extraction from the NF concentrate; after polishing (Photos: Georg Schons/GS-GmbH).

What percentage of the coins, present in the BA, is actually recovered depends on the properties of the coins, the flowsheet of the plant and the equipment in use. If the BA is screened, for example, on devices with longitudinal apertures such as sizers, star screens and the like, the coins may slip into the residue and cannot be recovered. But if conventional vibrating screens with square openings are used, the coins have a good chance to end up in the NF fraction recovered by ECS or sensor sorter. Some coins are weakly magnetic, for example 1€ and 2€. If

upstream of the ECS a strong magnet is used, these coins may be transferred into the (weakly) magnetic fraction and can therefore not be recovered on the ECS but potentially on a downstream sensor sorter.

For recovering longish metal particles one will sometimes find specialized equipment installed. Usually, the above mentioned effect of longitudinal alignment of such particles on a hopper is used to convey the material over apertures that are overcome by long particles while more compact particles tip into these apertures. Such devices are typically used for recovering copper wires and stainless steel silverware.

## 5. Bottom Ash Processing

### 5.1 Processing Strategies

Given the fact that MWI bottom ashes have a surprisingly similar composition, even on a global scale, it is quite striking that no two BA processing plants are identical, but rather every plant is one of a kind. One might argue that the ashes may be similar but the boundary conditions, such as the legislation pertaining to the use of BA for road construction, may be different, and hence different processing techniques may prove to be optimal. And yet – plants operating in the same region under the same legislative umbrella often take entirely different approaches for solving the same problems. The great variety of technologies currently being applied to BA processing is an indication that this technology is still in its fledgling stages and that there is plenty of potential for optimization.

One strategic question the plant operator is confronted with is the extent of downstream integration into the metal processing chain. Most BA plants sell the metal concentrates as they are produced, i.e. including adhering mineral matrix material. Others accumulate their concentrates and run them batch wise through an impact crusher or shredder in order to achieve a higher grade and therefore an increased revenue per ton. In this manner the FE and SS concentrates are polished to a quality that generally allows for direct sale to steel plants. Very large BA treatment plants even go as far as processing their NF concentrates to yield material qualities good enough to be accepted directly by NF smelters or foundries. This is usually accomplished by sink/float separation to recover a LNF and a HNF product, the latter being further separated by sensor sorting and handpicking.

Major determinants for the selection of an optimized BA processing and disposal concept are the following (Table 5.1):

1. Annual BA throughput
2. Method of Extraction of the BA (wet, dry)
3. Location of the BA plant (on site MWI, on site landfill, elsewhere)
4. Technology for BA processing (dry, wet)
5. Fate of the residue (landfill, road construction)

**Table 5.1: Taxonomy of BA processing operations**

1 BA throughput t/a	2 extraction method	3 location of plant	4 proc. technology	5 fate of residue
1.1 <30'000	2.1 wet extraction	3.1 on site MWI	4.1 dry processing	5.1 landfill
1.2 30'000-150'000	2.2 dry extraction	3.2 on site landfill	4.2 wet processing	5.2 building material
1.3 >150'000		3.3 transfer site		
		3.4 mobile plant		

1. BA processing plants can be distinguished by throughput:
  - 1.1 low throughput <30'000t/a
  - 1.2 medium throughput 30'000-150'000t/a
  - 1.3 high throughput >150'000t/a

2. The bottom ash comes in the following two categories:
  - 2.1 Wet extraction of BA: This is by far the most common case
    - + cheap and robust extraction from the incinerator
    - solidification of the matrix and corrosion of the metals
  - 2.2 Dry extraction of BA: during the extraction of BA, contact with water is avoided
    - + dry processing of material 0.5-4 mm is possible
    - high investment cost, dust issues
  
3. As to their location, plants may be installed:
  - 3.1 On site of the MWI
    - + added value remains with the operator of the MWI
    - little space available, possible issues with dust, noise and odor
  - 3.2 On site of a landfill
    - + plenty of space; dust and noise no significant problem
    - limited water and electricity available, makes only sense if the residue is landfilled
  - 3.3 At a third location (transfer site)
    - + positioned at a logistically optimal location for cooperation with multiple customers for the mineral residue (construction companies, landfills)
    - multiple handling and transport
  - 3.4 Mobile plant
    - + great flexibility for cooperation with clients at distant locations
    - low tech process with reduced metal recovery; wet processing is not an option
  
4. With respect to technology, BA processing plants can be assigned to the following categories:
  - 4.1 Dry processing:
    - + simple and robust technology
    - poor recovery <4 mm
  - 4.2 Wet processing:
    - + good quality of the residue; recovery of metals 0.1...4 mm possible,
    - higher costs
  
5. Of great importance are the boundary conditions with respect to the fate of the mineral residue:
  - 5.1 The residue is landfilled
    - + robust solution, maximum metals recovery can be achieved after aggressive comminution
    - more expensive than sale as construction material
  - 5.2 The residue is used as a construction material
    - + cheaper than landfilling
    - must meet stringent environmental limits; some metal is lost because it is locked within coarse mineral particles

While many combinations of the above categories may be technically feasible, only few of them actually make economic sense. The advantages and disadvantages of some combinations are discussed below.

### Category 1.1/2.1/3.2/4.1/5.1

This is a standard case for MWI with low annual BA output operating in regions where the use of BA for road construction is either not permitted or not economically attractive. One example is Switzerland where the reuse of BA-residue for construction purposes is prohibited. A general advantage that comes with landfilling the BA-residue is the possibility of maximizing metals recovery through metals liberation by means of aggressive comminution. As discussed above, this is not an option when the residue is used as construction material because in this case the integrity of the coarse particles must be maintained.

Transferring the BA from a small MWI to a large processing plant located on a landfill (which also accepts BA from other MWI) offers various advantages, for example the economy of scale. If the MWI BA output is low, the installation of a BA processing plant on site is often not economically viable. This also pertains to the sale of the products. Small quantities of metal concentrates fetch low prices.

Another constraint is space. BA processing plants require plenty of space not only for the plant itself but also for the storage of the feed and products. If a MWI is located in a populated area, noise, dust and odor emissions may also be issues. On landfills this is generally not a problem.

Processing wet BA in a dry plant is usually not feasible because the sticky fines may build up deposits on conveyor belts, screens, in dead spots of the crusher and sorting machinery, which may eventually lead to blinding and plugging. Before dry processing, the BA is first left to dry for several weeks. As this process goes along with solidification, the secondary BA needs to be crushed before separation in order to liberate the locked metals from the surrounding mineral matrix.

A possible flowsheet of a dry processing plant is shown in Fig. 5.1. For initial classification a bar-sizer is installed. The purpose of the trommel screen that follows is not only classification but also cleaning of the coarse metal scrap of adhering mineral matrix. The hand pickers remove metal pieces >150 mm and Cu/Fe-composites >80 mm as well as unburnt organics. If FE 80-150 mm slips through hand picking, this can be tolerated as it will be recovered further downstream by the MS. The fines from the bar-sizer are passed under an overhead magnet for removing oversize FE that slipped through the bar-sizer. The material is then classified with a star screen and a flip-flow screen, the coarse fractions of which are crushed. The three size fractions (10-150 mm, 3-10 mm, 0-3 mm) are each passed over a train of MS and ECS. The size spread of 10-150 mm would be too wide for a good separation of NF on ECS but keep in mind that, according to Fig. 3.13, virtually all NF is <30 mm, i.e. in effect 10-30 mm instead of 10 – 150 mm.

In order to boost recovery, this flowsheet also includes a sensor sorter. The combined residues of the upstream processes are run through the sensor sorter in order to recover SS and metal pieces still locked into mineral matrix. The concentrate of the sensor sorter would be accumulated and periodically processed through the plant (e.g. once per month). Depending on the quality of the other metal concentrates these could also be re-processed for removing adhering mineral matrix.

If the flowsheet shown in Fig. 5.1 is used, one may expect a recovery of FE in the order of 95% and a NF recovery in the order of 85%. Reminder: according to the definition in chapter 3.3.2 the limit of 100% is given by the sum of all metal pieces >2 mm.

Depending on the quality of the BA it may be necessary to add to the flowsheet a wind-sifter for removing unburnt organic material (for example just upstream of the star screen). Also, dust abatement must be taken care of when processing dry BA.

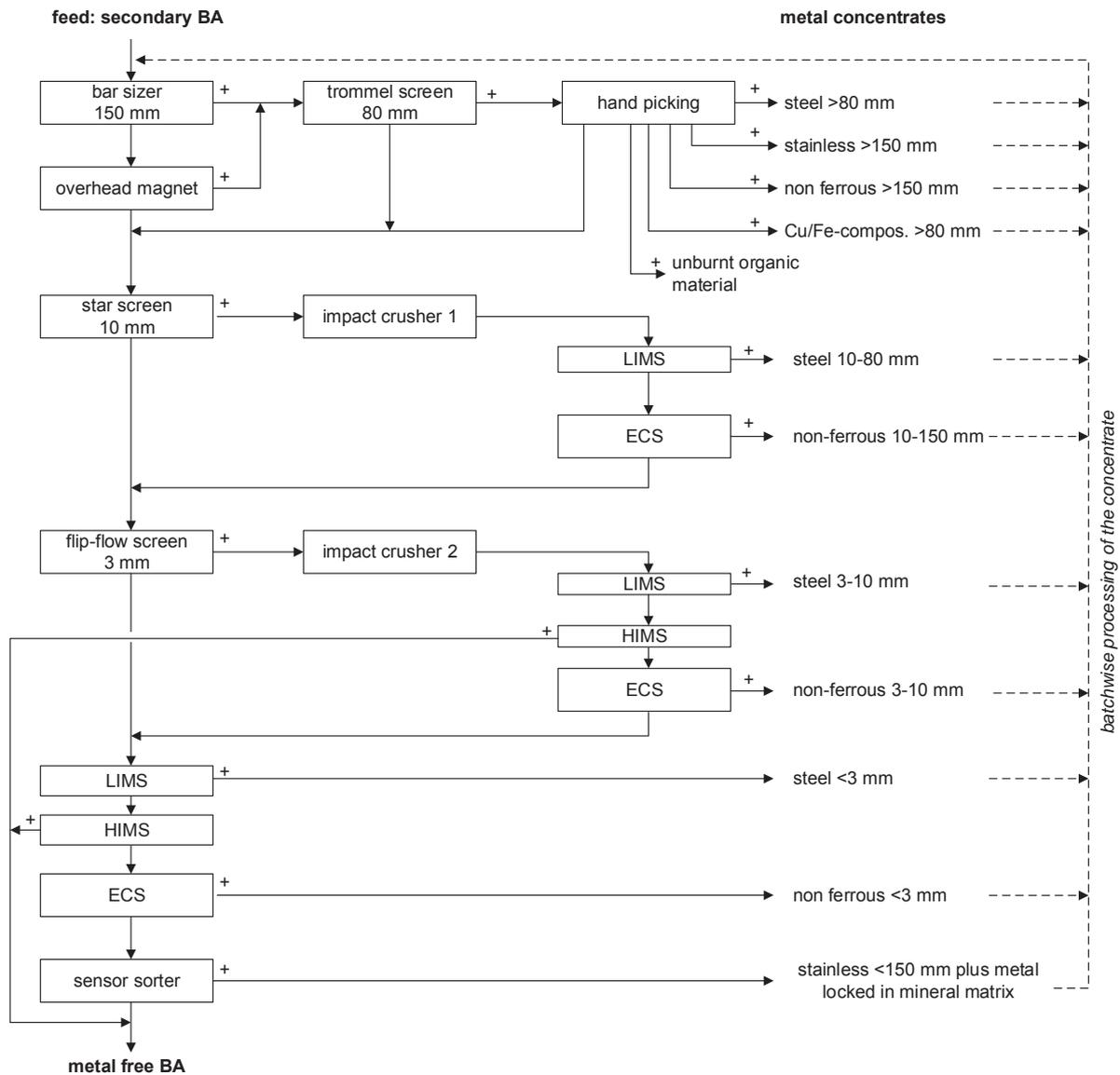


Fig. 5.1: Example for flowsheet of a dry BA processing plant “state of the art”.

Category 1.3/2.1/3.1/4.2/5.2

If the residue can potentially be used for road construction, it has to comply with stringent requirements concerning the release of water soluble pollutants. One way to meet such requirements is wet processing where the pollutants are essentially washed out of the BA.

An example of a possible flow sheet for a wet BA processing plant is shown in Fig. 5.2. After passing the bar sizer and the overhead magnet, the material is classified on two consecutive vibrating screens, the coarse fractions of which are run through magnetic and eddy current separators. Processing wet material on ECS is possible for particles >4 mm. On smaller

particles, capillary forces that lead to agglomerates and pin the particles to the belt, can be detrimental to separation efficiency.

The BA <4 mm is deslimed, for example on hydrocyclones or screw-type classifiers, run through a magnetic separator to remove ferrous particles (e.g. pieces of wire) that may interfere with the gravity concentration process. Depending on the specifications for recycled BA the LIMS concentrate is either reunited with the light sand fraction or landfilled. If this material contains sufficient copper (which may very well be the case) it may also be sold for further processing. The nonmagnetic fraction is then passed on into the gravity concentration process where the HNF 0.2-4 mm is recovered. The sand 0.2-4 mm is dewatered (with technology typically used in gravel processing), rinsed with fresh water and is then ready for use as construction material. If the sand cannot be reused, it goes to landfill, together with the filter cake.

The waste water from the desliming and the sand dewatering stages is flocculated for sedimentation in a thickener. The sludge, comprised of the particles <0.2 mm which contains much of the heavy metal contamination, is dewatered (e.g. on a filter press) and then landfilled.

According to the particle size distribution in Fig. 2.6 the amount of filter cake is approximately 10% of the feed material (on a dry basis). The cleaned water from the thickener, which contains soluble salts like chlorides and sulfates, is then recycled. The water balance is as follows:

Water in:

- with raw BA (20% moisture)
- fresh water used for rinsing 0.2-4 mm sand

Water out:

- with the residual moisture of washed BA >0.2 mm (estimated 14% moisture)
- with the residual moisture of filter cake BA <0.2 mm (estimated 30% moisture)
- with metal concentrates (negligible)
- brine

As both feed and the products are water saturated, basically the amount of fresh water needed to rinse the sand is the amount of brine that needs to be disposed of. This discharge stream can be either treated thermally (by evaporation, preferably using heat from the incinerator) or is led into a river, provided that the relevant water quality limits are met (which may require further cleaning, e.g. by way of ion exchange for removing dissolved heavy metals).

Generally, a wet BA processing plant would be positioned on site of the MWI as the infrastructure, in particular for the treatment of the brine, is already available. In contrast, landfills are typically in remote locations and do usually not have access to industrial infrastructure. Since the BA is processed right after wet extraction from the incinerator, the metals are still liberated and can therefore be recovered before the solidification process commences. This does not apply however to metal pieces locked into slag particles. In order to access these metals, the slag particles would need to be crushed which may cause problems as the fines that are generated from the wet material, may build up in the crusher eventually clogging this device.

The driving factor for wet processing is not metals extraction but the quality of the residue, which is, however, not the subject of this publication. With respect to metals extraction, wet processing offers some advantages over dry processing. Advantageous is the opportunity for using gravity concentration processes which makes the material <4 mm amenable to separation,

i.e. in particular copper, gold and silver mostly originating from electronic products. In economic terms, the additional cost of the wet process as opposed to a dry process, is covered by the revenue generated from the sale of the construction material (plus the saved disposal fee).

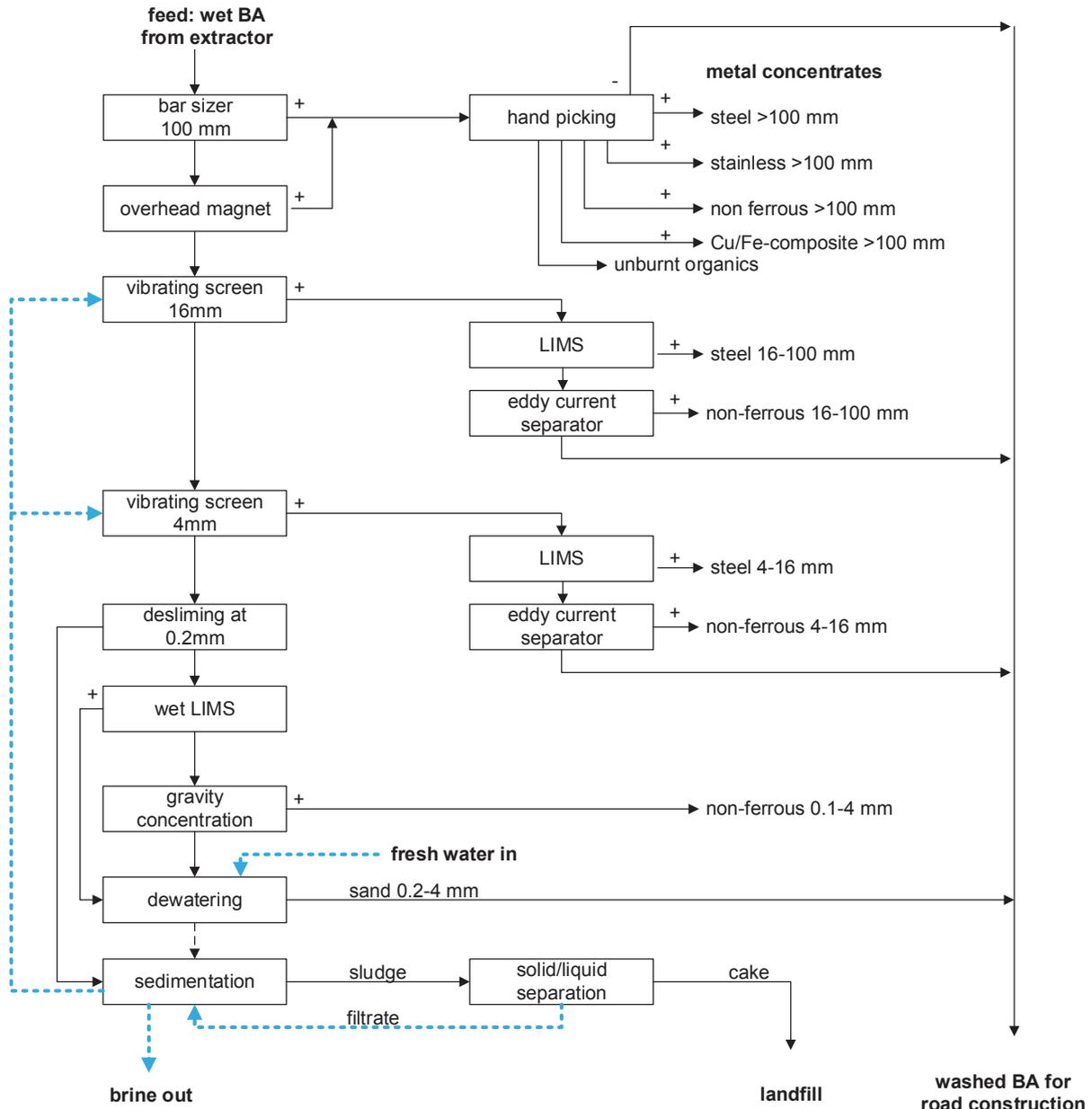


Fig. 5.2: Simplified flowsheet of a wet BA processing plant.

An interesting option is a hybrid of the two previously discussed concepts. This would encompass a wet separation of the material <4 mm on site of the MWI. According to Fig. 2.6 this would be approximately 35% of the BA. The coarse material would then be dry processed and the fines would be treated by wet processing techniques either on site or at a third location. In this manner the valuable heavy and precious metals <4 mm would be recovered by means of gravity concentration on the one hand. On the other hand the coarse metals could be recovered by means of MS and ECS. Another advantage would be that the wet coarse material takes much longer to solidify after the fines have been removed. This would enable processing of the material off site since the metal pieces remain liberated for a longer period of time.

Category 1.x/2.2/3.x/4.x/5.x

This category encompasses all processes that deal with BA after dry extraction. Currently, only very few grate incinerators use dry extraction - this method being more common with fluidized bed incinerators. By avoiding the contact with water many of the problems encountered in the processing of BA after wet extraction can be avoided. From the point of view of the BA processor striving for a maximum recovery of metals, dry extraction is the method of choice. From the viewpoint of the MWI operator, wet extraction is generally favored. When planning the engineering for a new plant one should seriously consider dry extraction. Retrofitting an existing plant with a dry extractor is, on the other hand, technically possible but economically often not feasible.

The material produced from dry extraction is the “primary BA”. As the material is not wetted, the detrimental corrosion and solidification processes characteristic for “secondary BA” are therefore suppressed and metal pieces that would otherwise be locked into the newly formed minerals remain liberated. One would therefore expect a comminution prior to the separation stages to be unnecessary. This is not the case however, because some metal pieces are contained in the slag (Fig. 2.12). In order to liberate these metals, the primary BA needs to be comminuted.



*Fig. 5.3: Metals 10-40 mm recovered from dry extracted BA: NF (left) and FE (right). Even thin foils of aluminum have been successfully recovered as a pure product with virtually no corrosion and adhesions of mineral material.*

The superior quality of the metal extracted from dry BA is quite obvious when the Fig. 5.3 is compared with the respective products recovered from wet BA as shown in Fig. 4.18 left and Fig. 4.15 left. From primary BA even particles of 0.25 mm can be potentially recovered with ECS. For comparison: from secondary BA one can, typically, expect a recovery of NF > 4 mm. The added amount of NF recoverable from primary BA is estimated to 15% if the BA stems from household waste and up to 25% if significant amounts (10%) of shredder residues are co-incinerated.

One drawback of processing primary BA is the potential for dust emissions. Keeping in mind that the dust from BA is loaded with heavy metals, rigorous provisions must be taken to avoid its generation and release.

## 5.2 Plant practice

### 5.2.1 Assessing separation efficiency

In order to optimize plant performance one first needs to assess separation efficiency. This is generally done by inspection of the grade/recovery curve:

- “Grade” is the concentration of the metal in the concentrate,  $C_c$
- “Recovery” is the relative amount of metal present in the feed that has been successfully transferred into the concentrate,  $R_c$

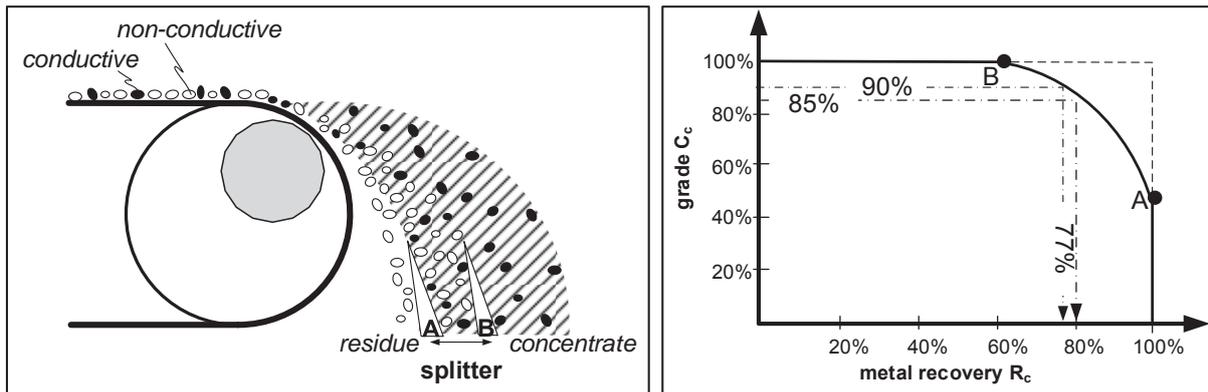


Fig. 5.4: The ECS splitter setting (A) results in 100% recovery but only 45% grade. Splitter setting (B) results in 100% grade but only 60% recovery.

Ideally, both grade and recovery are 100% as one would want to transfer all metal from the feed into the concentrate ( $R_c=100\%$ ) and at the same time produce a concentrate with no mineral contamination ( $C_c=100\%$ ). In reality, optimizing grade and recovery are conflicting objectives as becomes obvious when considering the setting of the splitter of an ECS. As pointed out in chapter 4.4.2, the particle trajectories of material processed through an ECS depend on a multitude of factors. Notably, the orientation of elongate particles plays a major role. If a particular elongate metal particle is aligned perpendicular to the direction of transport it will end up in the residue. If this residue is re-processed on the same ECS and the same metal particle is this time aligned with the direction of transport it can, on this second pass, be transferred to the concentrate.

In summary, the conductive and the non-conductive particles can be visualized as two segregating clouds of particles that take different mean trajectories but overlap due to considerable scatter, in particular of the of the conductive material trajectories, as shown in Fig. 5.4 left. Plotting grade and recovery as a function of the splitter setting generates the grade recovery curve in Fig 5.4 right.

If the customer specifies that the concentrate must have a grade of, for example, at least  $C=85\%$  one can read off the grade recovery curve that the splitter setting would result in a recovery of  $R=80\%$ . In other words, 20% of the metal is lost into the residue.

While BA plant operators have a fairly precise idea of the grade they produce, as this is relevant for the revenue generated by the sale of the concentrate, very few have any clue of the recovery their plant produces, i.e. the losses of potentially recoverable metal they incur. In order to optimize a plant it is crucial to not only control grade but also assess recovery. One example is the control of the handpicking station. If done sloppily, only the concentrate bins are inspected and the inspection is terminated if the grade is satisfactory. If, however, the residue is not

controlled for the amount of valuables lost, the recovery and therefore plant performance may be abysmal despite high grades in the concentrate bins. Since in handpicking stations grade will hardly ever be an issue, a thorough inspection of the residue for valuable metals lost may be even more important than examining the concentrates.

The example of the handpicking station can be expanded to the plant as a whole. One may safely assume that many plants produce higher concentrate grades than demanded by the minimum specifications set forth by the customer, because the plant personnel likes to be “on the safe side”. The consequences are obvious from Fig. 5.4 (right). If the concentrate grade is increased to 90% (instead of the 85% required) the recovery will drop from 80% to 77%. This may appear inconspicuous but could translate into a reduced overall profit of approximately 15% (as discussed in chapter 7.2).

When assessing plant performance, much confusion is caused by the lack of reference with respect to technical terms like “metal content”, “grade” or “metal recovery”. A case in point is the term “metal yield”. When plant operators refer to “metal yield” they usually mean the concentrate mass recovery because this is the product they sell by weight – mineral matrix included. Another source of confusion is the term “grade”. In scientific publications one will often find the “grade” being determined by a chemical analyses. This would include not only valuable native metals but also mineralized metals, such as metal oxides, which have no commercial value. We therefore suggest that the definitions in Table 5.1 be used.

**Table 5.2: definitions for grade, recovery and related terms**

metal potential $C_f$	$\frac{\text{mass metal} > 2\text{mm}}{\text{mass feed}}$
mass recovery to concentrate $R_{Mc}$	$\frac{\text{mass concentrate}}{\text{mass feed}}$
metal recovery to concentrate $R_c$	$\frac{\text{mass metal in concentrate}}{\text{mass metal} > 2\text{mm in feed}}$
concentrate grade $C_c$	$\frac{\text{mass metal in concentrate}}{\text{mass concentrate}}$
smelter yield $R_s$	$\frac{\text{mass bullion}}{\text{mass metal in concentrate}}$
bullion grade $C_s$	$\frac{\text{mass metal in bullion}}{\text{mass bullion}}$

*Note that in the above definitions the term „metal“ refers to „native metal“. And keep in mind that all data on metal content or yield should be given with respect to dry mass (i.e. the mass of the material after drying at 105°C).*

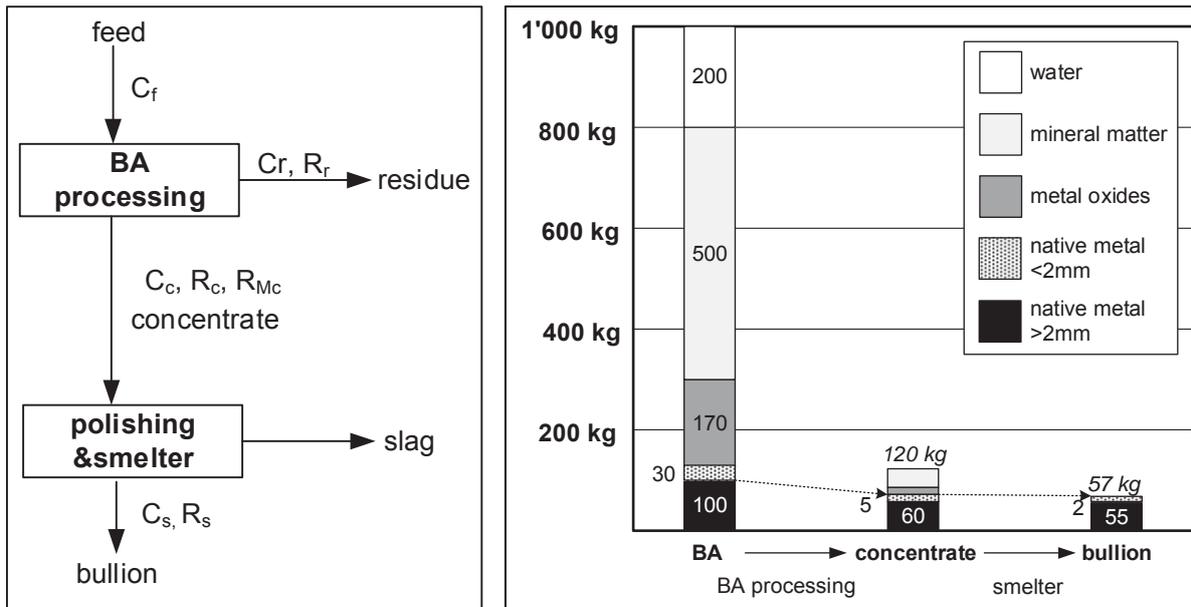


Fig. 5.5: (left) Simplified flow sheet of a BA processing operation plus downstream metal concentrate refining. (right) Example: Graphic representation of the composition of a BA, the metal concentrate and the product after smelting.

Fig 5.5 (right) illustrates the definitions from Table 5.2 by way of an example:

- total metal grade of the feed  $(100+30+170)/800 = 37.5\%$
- native metal grade of the feed  $(100+30)/800^* = 16.25\%$
- metal potential of the feed  $C_f = 100/800^* = 12.5\%$
- mass recovery to concentrate  $R_{Mc} = 120/800^* = 15\%$
- metal recovery to concentrate  $R_c = 65/100 = 65\%$
- concentrate grade  $C_c = 65/120 = 54.2\%$
- smelter yield  $R_s = 57/65 = 87.7\%$

\*reference: dry mass (105°C)

The “total metal grade of the feed” corresponds with the result of a chemical analysis. This includes (valuable) native metal as well as (valueless) mineralized metal.

When assessing the “metal potential” of a BA, only the pieces of native metal >2mm are of interest, as the pieces <2mm are only marginally recovered with conventional BA processing technology (chapter 3.3.2). One may argue that, with appropriate technology (e.g. wet processing), at least some of the metal <2mm can also be recovered and therefore the “metal recovery to the concentrate”  $R_c$  could, theoretically, exceed the “metal potential”. However, in practice,  $R_c$  is always significantly lower than the “metal potential”. Even if some metal <2mm had successfully been recovered, much more of the metal >2mm is lost into the residue. In practice, the “metal potential” being, according to our definition, native metal pieces >2mm, is never exceeded by  $R_c$ .

As mentioned above, the “mass recovery to concentrate”  $R_{Mc}$  includes mineral adhesions and is therefore useless for assessing the performance of a given plant, or for comparing different plants. In these cases, “metal recovery to concentrate”  $R_c$  should be used instead. Likewise, “concentrate grade”  $C_c$  refers to native metal content (without mineral adhesions). When estimating the commercial value of a concentrate by using LME-quotations for bullion, one needs to keep in mind that, even if the concentrate is pure metal, some of this metal is lost during processing in the smelter.

Many of the difficulties to be tackled when assessing plant performance arise from the lack of adequate sampling and sample preparation procedures. This topic is discussed in chapter 6.

### 5.2.2 Improving plant performance

Central to the optimization of BA processing plants with respect to metal recovery are narrow particle size distributions and full liberation of the metals.

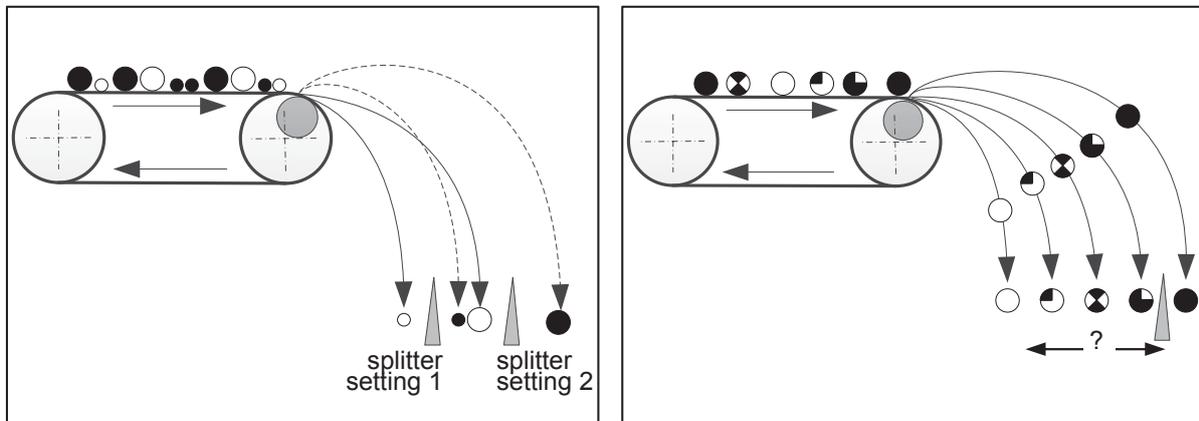


Fig. 5.6: (left) As the trajectories of coarse nonconductive particles and small conductive particles overlap, separation efficiency is low when processing wide particle size distributions. (right) If the metals were not sufficiently liberated, separation efficiency is low.

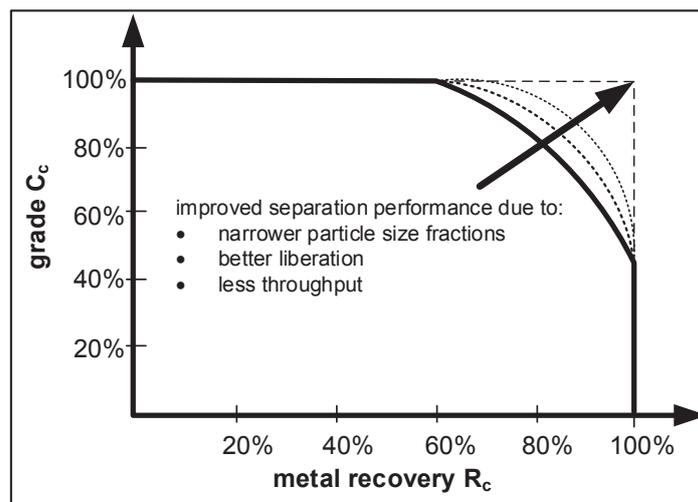


Fig. 5.7: Influence of various parameters on improving the grade/recovery curve. Ideally, the plant would operate at 100% metal recovery and 100% grade.

In practice, separators are often fed with a wide spectrum of particle sizes. This leads, for example on ECS, to overlapping trajectories of small conductive particles and coarse nonconductive material and hence to a grade/recovery optimization dilemma (Fig. 5.6 left; see also Fig. 6.2 left). To overcome this problem and achieve good grade and also good recovery, the material must first be classified into narrow size fractions. Then the fine particle fraction can be run through an ECS with splitter setting 1 and the coarse fraction can likewise be processed on a parallel ECS with splitter setting 2. As a rule of thumb, for sharp separations, the particle spectrum processed on an ECS should not be wider than a factor of 3 between the smallest and the largest particle in this size fraction (e.g. 2-6 mm, 6-18 mm, 18-54 mm).

The trajectory of a metal bearing particle also depends strongly on the liberation conditions. Depending on the amount of adhering mineral matrix, metal particles take different trajectories. It is a common mistake, when observing poor separation efficiency, to jump at the conclusion that the separator is to blame when, in fact, the separator works perfectly well but the metals are not liberated. The key to improved separation performance is, in this case, better liberation through finer grinding.

From an exclusively technical viewpoint, low throughput would be desirable as it increases separation efficiency (Fig. 5.7). From an economic viewpoint however, maximizing throughput is the most crucial parameter for maximizing profit. Assuming an ECS is to be economically optimized, the following sequence of events will take place as throughput increases.

Example (fictional): Plant Optimization

Initial condition: Throughput is 40 t/h, the splitter is set to achieve 80% grade (specification set forth by the customer) which leads to 4 t/h concentrate (10% of feed).

1. As throughput is increased to 47 t/h, separation efficiency is not affected and  $47 \cdot 0.1 = 4.7$  t/h concentrate is produced.
2. Further increase in throughput to 54 t/h leads to a decrease in grade. In order to compensate, the splitter is shifted (i.e. moved into the concentrate stream) until the target grade of 80% concentrate grade is reestablished. This leads to a reduced fraction of the feed being recovered as concentrate ( $R_{Mc} = 10\% \Rightarrow 9.4\%$ ). A total  $54 \cdot 0.094 = 5.1$  t/h of concentrate is produced.
3. Even more increase in throughput to 60 t/h calls for aggressive shifting of the splitter in order to maintain 80% grade. Now only 6% of the feed ends up in the concentrate, the mass of which is calculated as  $60 \cdot 0.06 = 3.6$  t/h.

Assuming – for simplicity – that the revenue for the concentrate be 20 €/t and operating cost be 60 €/h (independent of throughput), one arrives at the data shown in Fig. 5.8. While initially increased throughput brings about an increased amount of concentrate (at 80% grade), excessive increases in throughput decrease the amount of concentrate and therefore profit.

Keep in mind that the plant personnel will tend to operate at lower throughput than is economically advisable, as this requires less attention by the personnel than running the plant “near the tipping point”. Assume the plant in the example shown in Fig. 5.8 would be “normally” run at 47 t/h. An increase of merely 13% throughput ( $47 \Rightarrow 53$  t/h) would increase profit by 22% ( $34 \Rightarrow 41.5$  €/h).

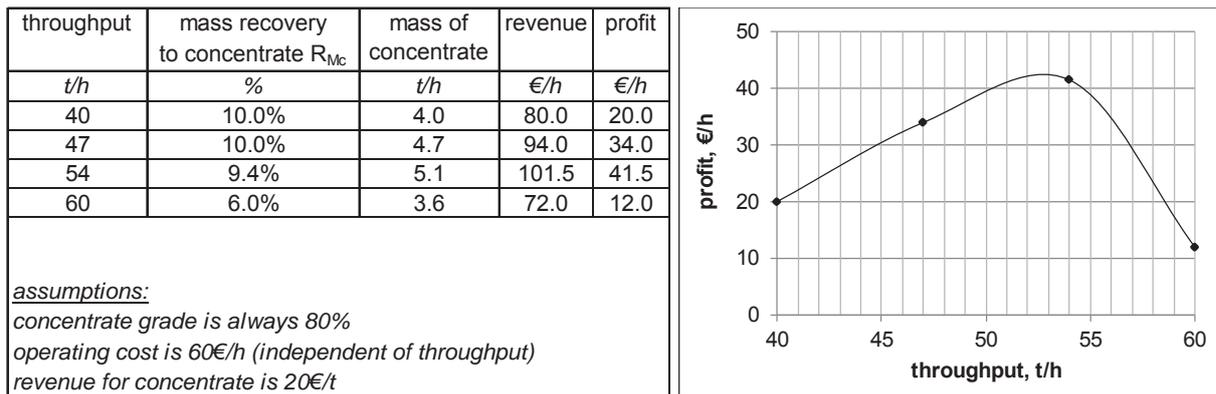


Fig. 5.8: Optimization of an ECS. Optimum throughput is approximately 53t/h.

One method for increasing plant performance is to run the material not through one separator only, but through two or even three identical separators in series. This may improve separation results considerably, in particular when odd shaped particles are to be separated by ECS. Particles that were not ideally positioned for separation on the first ECS get a “second chance” to find their way into the concentrate on the next ECS downstream. This philosophy may also be employed for improving low separation efficiency due to overloading.

As a rule of thumb one may assume that the recovery on the downstream device is, at the same setting parameters (for example splitter edge), approximately 65% of the recovery of the upstream device. The recovery decreases from stage to stage because the particles that are more easily separated, for example because they have a favorable shape, have already been recovered by the upstream devices.

Example: multiple ECS in series

The BA fed to the first ECS contained 2% (20 kg/t) fully liberated aluminum particles of which 60% were successfully separated, i.e. metal recovery  $ECS1=60\%$  (12 kg). On the second device  $ECS2$  one may expect a recovery of 65% of the recovery of  $ECS1$  i.e.  $0.65 \cdot 0.6=39\%$ . At the remaining head grade of 0.8% (=8kg/t) one may therefore expect another  $0.39 \cdot 8=3.1$  kg to be recovered to the concentrate. Running the material through  $ECS2$  would increase aluminum recovery from 12 to 15.1 kg per ton of BA processed. A third separation step would add another  $(20-15.1) \cdot 0.39 \cdot 0.65=1.24$  kg. Note, that this algorithm applies only if the metal particles are fully liberated – a small aluminum particle locked in a big slag particle will never be recovered into the concentrate, regardless of the number of ECS in series.

For improving plant performance it must be first established what mechanism is limiting the performance (wide particle size distribution, insufficient liberation, overloading, the efficiency of separator itself, particle shape, agglomeration due to moisture...). This requires a good understanding not only of the process but also of separation theory. Unfortunately, many plants do not employ qualified personnel with a sufficient theoretical background, e.g. in basic physics, to be able to identify the underlying reason of an observed separation problem and devise a proper ameliorative strategy. Instead, plant operators obtain their knowledge almost exclusively by “trial and error”, which may prove to be very costly to their employers. The often heard remark “Our plant is profitable – so why hire an (expensive...) engineer?” misses the point. That even fools are able to make a profit from BA processing only goes to show how much economic potential remains untapped.

## 6. Sampling and Analysis

For optimizing a BA processing plant it is necessary to determine mass flows and the associated concentrations of the metals correctly. Assessing overall mass flows is routinely done by weighing the feed and the products. Accurately determining metal content, however, requires in depth knowledge of sampling theory. It is beyond the scope of this paper to discuss the fundamentals of sampling and sample preparation but rather to provide “recipes” how one must proceed in order to arrive at meaningful results. The problems associated with sampling are:

- Sampling in the field: Obtaining a representative sample.
- Sample preparation and analysis: arriving at the correct result.

Sampling procedures for bulk solids have been devised mostly for the mining industry where precise results are of utmost importance. Taking into account the huge mass flows associated with mining, sampling is an extremely elaborate procedure which is carried out by highly specialized personnel. Such procedures are not practical for sampling the feed and products of a BA processing plant. For the considerations in this chapter it is assumed that the following boundary conditions apply:

- The sample is being drawn from a pile of bulk solid
- Only the content of native metal >2 mm is of interest
- The analyzed metal content should lie within 10% of the true value

The central objective of sampling bulk solids is to obtain a “representative” sample and to prepare the sample for later analysis in such manner that it remains representative. In the context of BA processing “representative” means that the sample must, within a standard deviation of  $\pm 10\%$ , contain as much of the metal in question as the whole stockpile that was sampled.

### 6.1 Sampling on site

When preparing a strategy for on-site sampling, the following two questions need to be addressed:

- How much sample should be obtained?
- In which way should the sample be drawn?

One of the most crucial factors for determining the precision of the analytical result is the “minimum sample mass” MSM. This is the mass of the sample to be drawn in order to be representative for the metal content of the stockpile as a whole (within the given margin of error of  $\pm 10\%$ ).

This MSM depends strongly on the following factors:

- Maximum size of the metal particles in the stockpile
- Mass fraction of the metal particles in the stockpile
- Density of the metal particles

While the density of the metal is known and the maximum size of the metal particles can often be estimated, determining the mass fraction (=concentration) of the metal is generally the objective of the sampling procedure to begin with. This leads to a recursive problem: the MSM is only known after analysis of the sample but in order to draw the sample the result of the analysis needs to be known. In practice, one will therefore have to estimate this parameter. Despite the equations for calculating MSM presented below the following rule of thumb applies: “Take as much sample as feasible. And then some more”.

When sampling BA for metal content at a precision of  $\pm 10\%$  one can derive from sampling theory the following equations (which is also plotted in Fig. 6.1):

$$MSM = \frac{25\rho}{\omega} d_{\max}^3 \quad (\text{eq. 5.1a})$$

$$d_{\max} = \sqrt[3]{MSM \frac{\omega}{25\rho}} \quad (\text{eq. 5.1b})$$

$MSM$  minimum sample mass [kg]       $d_{\max}$  maximum size of metal pieces [m]  
 $\rho$  density of the metal [ $\text{kg/m}^3$ ]       $\omega$  mass fraction of the metal [-]

The MSM depends very strongly on particle size ( $MSM \sim d_{\max}^3$ ). Large particles require much more MSM than small particles. Due to the “nugget effect” the MSM also increases with decreasing mass fraction of the metal in question: a sample of BA representative for (rare) gold coins is much larger than a representative sample for (fairly frequent) nickel coins.

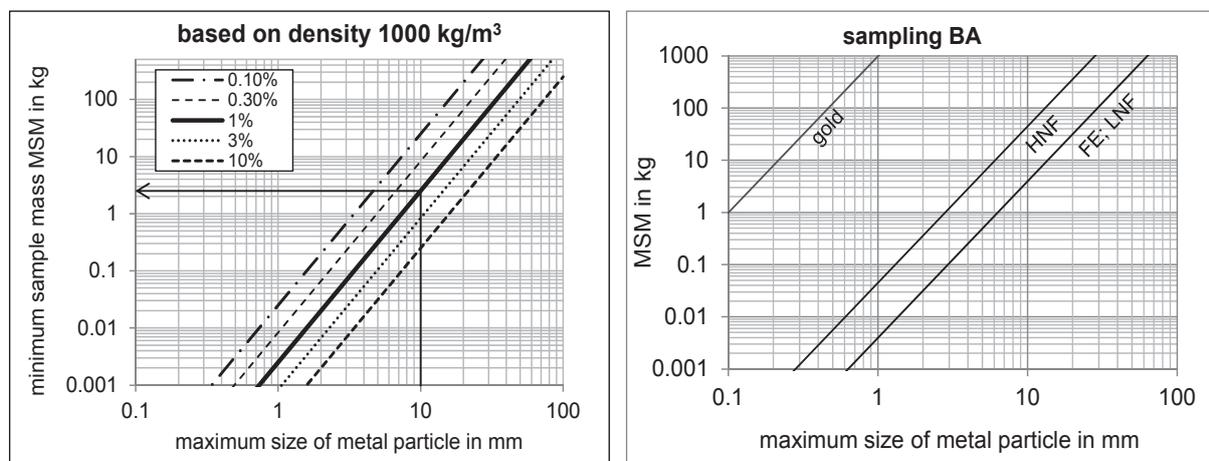


Fig. 6.1: The minimum sample mass MSM as a function of the maximum size of the metal particles at various mass fractions of metal  $\omega=0.1\% \dots 10\%$ . For a given metal the MSM read from the graph must be multiplied with the density of the metal in multiples of  $1'000 \text{ kg/m}^3$ . The MSM for copper (density= $9'000 \text{ kg/m}^3$ ) with maximum particle size 10 mm and a mass fraction of 1% the MSM would be  $2.5 \text{ kg} * 9'000 / 1000 = 22.5 \text{ kg}$  (left). Assuming typical concentrations of metals in BA, the required MSM can be estimated from the graph (right).

Example 1: A size fraction 0-16 mm of BA needs to be sampled for aluminum. It is estimated that the BA contains approximately  $\omega=2\%$  aluminum particles. The density of aluminum is  $\rho=2'700 \text{ kg/m}^3$ . The minimum sample mass MSM is calculated as  $MSM = 25 * 2'700 * 0.016^3 / 0.02 = 13.8 \text{ kg}$ .

Example 2: The residue of a processed BA 0-64 mm is to be sampled for residual copper potential (i.e. pieces  $>2 \text{ mm}$ ). The density of copper is  $9'000 \text{ kg/m}^3$ . Assuming that all copper pieces  $>6 \text{ mm}$  have been successfully recovered by the eddy current separators, the maximum size of copper particles in the residue is 6 mm (despite the maximum size of the mineral material being 64 mm). Further assuming that, of the 0.5% copper present in the feed an estimated 50% had been recovered in the NF concentrate, leads to approximately 0.25% native copper in the residue. The minimum sample mass is therefore  $MSM = 25 * 9'000 * 0.006^3 / 0.0025 = 19.4 \text{ kg}$ .

If the actual sample mass  $M^*$  differs from the MSM, the error to be expected is estimated as follows:

$$\varepsilon = \sqrt{MSM/M^*} * 0.1 \quad (\text{eq. 5.2})$$

$MSM$  minimum sample mass with implicit error  $\varepsilon=10\%$  [kg]

$M^*$  sample mass actually drawn [kg]

$\varepsilon$  error to be expected

Assuming that in the above Example 2, instead of  $MSM=46$  kg for example  $M^*=100$  kg had actually been drawn, the error would be reduced to  $\sqrt{46/100} * 0.1 = 6.78\%$ . Vice versa, precision decreases with the square root of the sample mass.

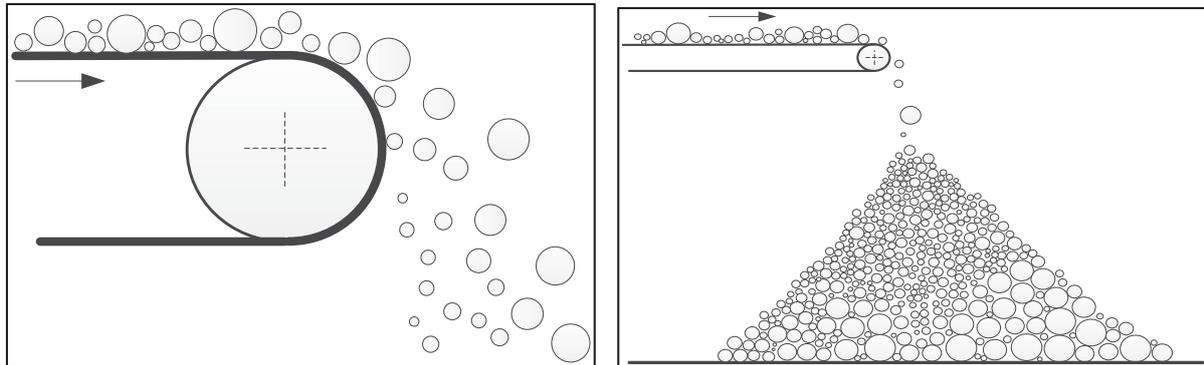


Fig. 6.2: Segregation according to particle size. Big particles fly farther than small particles of same initial horizontal velocity (left). Big particles accumulate at the foot of the pile and small ones in the center (right).

Once the MSM has been determined, the question arises how the sample should be obtained. Ideally, samples should be drawn while the material is in motion, e.g. at regular intervals from the head of a conveyor. This is usually impractical and rarely done in BA processing plants. Rather, the material is sampled from a stockpile.

When sampling bulk solids one must be aware of segregation according to particle size, in particular if the material had been stockpiled using a belt conveyor (Fig. 6.2). Ideally, the entire pile would be first homogenized (e.g. with an excavator or a front end loader) before sampling, but this is in practice usually not feasible. Instead, one will need to climb the pile and draw, with a shovel and bucket, approximately 20 individual samples at various locations of the pile, digging into the pile as good as possible. Each individual sample would comprise the “minimum sample mass” MSM.

This procedure is repeated periodically as the pile grows, for example in 10 intervals. The samples thus obtained are combined, thoroughly homogenized, and then graded to form a bed approximately 200 mm high. From this bed 20 samples with each 1/20 of the MSM are drawn at random to produce one representative sample of MSM. Obviously oversize metal pieces (with regard to the corresponding MSM) are removed either by screening or by handpicking and discarded. This sample is referred to as the “stockpile sample” which is further processed in the laboratory. If possible at all, a second sample of equal mass should be drawn as a retaining sample in case the results of the first sample are inconclusive or it is – after analysis – discovered that the sample mass was too low to meet the minimum precision demanded.

## 6.2 Analysis of the stockpile sample

Once a representative stockpile sample is drawn (for example 80 kg) it needs to be further processed in the lab for determining metal content. There are three different approaches:

- Chemical analysis (including fine grinding)
- Fire Assay (smelting)
- Selective comminution (“UMTEC” method)

### 6.2.1 Chemical analysis

In a modern lab for analytical chemistry, metal content is analyzed by, for example, XRF, ICP or AAS, which require approximately 1-10g material only. The problem in this case is “sample reduction” i.e. the method for arriving at a few grams of analysis sample that is still representative, with respect to metal content, for the many kilograms of the original stockpile sample. Usually, this problem is tackled in chemistry labs by comminution to reduce the size of the metal pieces and disperse them, which would allow for smaller MSM.

For illustration consider a stockpile sample with a copper content of an estimated 0.3% and a maximum size of the copper particles of 6.4 mm. According to eq. 5.1a the MSM of this sample is 19.7 kg. Assume from this amount of material a representative sample of 2 g needs to be provided for analysis. According to eq. 5.1b the MSM of a 2g sample corresponds with a size 0.3mm. Although it would be possible to grind 19.7 kg of material to <0.3 mm, this would be very tedious with conventional lab equipment. In practice, one would typically proceed in two stages. First the 19.7 kg stockpile sample is crushed to <2 mm (for example in a laboratory jaw crusher at minimum gap width). Taking eq. 5.1a one calculates for 2 mm copper particles the MSM as 600 g. The 19.7 kg material <2 mm can now be thoroughly mixed and a subsample of 600g taken therefrom. These 600 g are comminuted in the second stage to <0.3 mm, for example in a laboratory ball mill. After thorough mixing, the analysis sample of 2 g is now drawn and analyzed.

Although this is the standard lab procedure, the results will typically indicate – if applied to BA – a copper content that is only 25% of the true value, i.e. they would be completely off the mark. The reason being that copper – in the form of native metal – cannot be reduced in size by conventional lab equipment used for comminution (jaw crusher, ball mill, disc mill) - in particular if processed together with a large amount of mineral material. Copper, like other metals, is ductile and becomes flattened but not pulverized in conventional lab equipment. Consequently, even after pulverizing all brittle matrix material to <0.3 mm, the native copper particles are not comminuted and therefore the MSM of this material is still the same as in the original 20 kg stockpile sample. Analyzing a large number of samples of 2 g material will therefore yield wildly varying results. These will lie between the “background concentration” of copper, being present as oxide (in BA typically around 2’000 mg/kg=0.2% according to Fig. 3.11), and occasional extremely high values, if a piece of native copper of, for example, 6 mm in size was incorporated in the 2 g sample (e.g. 100’000 mg/kg=10%). Besides incompetent sampling, incompetent sample reduction is a reason why many published data report the copper content of BA with erroneously low values.

One way to get around this problem is shown in Fig. 6.3. 20 kg of material that was comminuted in the jaw crusher at 2 mm is sieved at 2 mm and the metals, that are accumulated in the fraction >2 mm, are ground to <2 mm. This can be accomplished in special vibratory disc mills, which few labs have however available in their equipment pool. The metal ground to <2 mm would then be mixed into the <2 mm material. From this mixture the sample of 600 g would be drawn and ground to <0.3 mm. Again one would sieve out the metals >0.3 mm after comminution,

grind them to  $<0.3$  mm and remix them into the material  $<0.3$  mm. The mix is then be used for drawing the required analysis sample of 2 g.

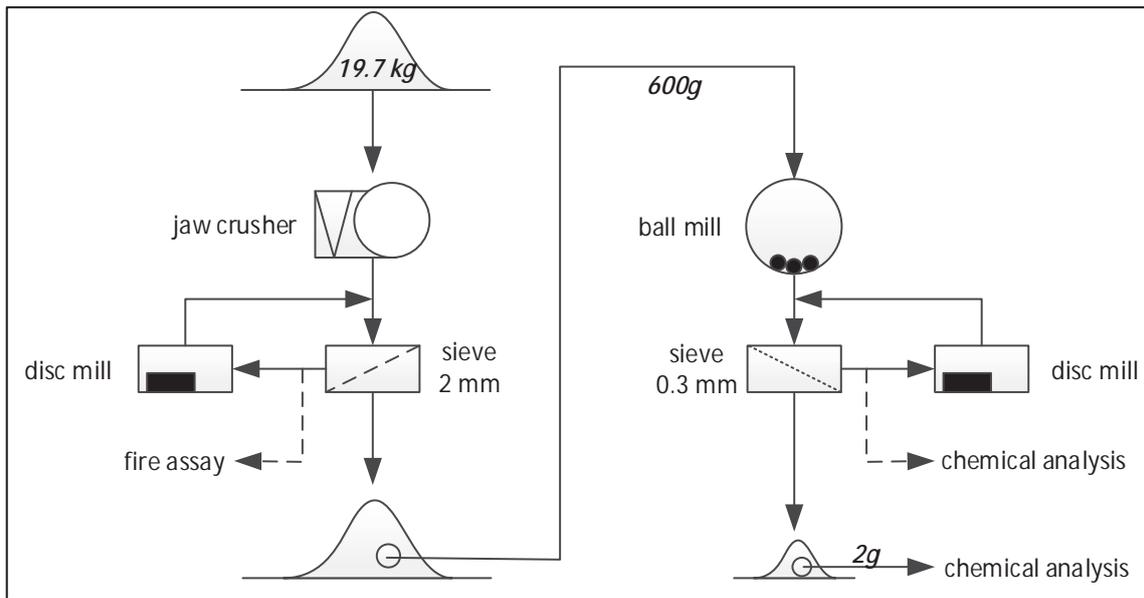


Fig. 6.3: Reduction of a 20 kg sample for chemical analysis. The coarse fractions from the sieves can be either ground in special mills and then remixed into the fine fraction or they can be analyzed as a whole, for example by fire assay or chemical analysis.

Alternatively, the coarse metal fraction ( $>0.3$  mm) could be digested as a whole in acids and then the solution be analyzed. This is generally feasible only for quantities less than 100g. For larger quantities, one may subject the sample to a fire assay as discussed below. This has the added advantage of measuring the content of metals in their native form (and not for example oxides).

In summary, conventional sample reduction procedures for providing analysis samples from BA are unsuitable and will produce wrong results. Moreover, a conventional chemical analysis cannot distinguish between “native metal” and “chemically bound metal (oxide)” but will yield a “total metal” result. As has already been pointed out (Fig. 5.5), only the native metal content is of interest. The total metal analysis is – from the technical/economic point of view – irrelevant.

### 6.2.2 Fire assay

If carried out correctly, fire assays are a robust method for determining the concentration of metals being present in their native form. However, fire assays are not practical with samples of – for example tens or even hundreds of kilograms of mineral matter containing a few percent of native metal.

Smelting is therefore limited to assays of concentrates where the metals are present in a highly enriched form. Assume 250 g of a concentrate of HNF is to be analyzed for the contents of copper, zinc, silver and gold. The material is first mixed with a flux (e.g. borax) and then heated in a crucible to approximately  $1'100^{\circ}\text{C}$ . The liquid metal collects at the bottom of the crucible while the slag collects on top of the molten metal (Fig. 6.4 left). After cooling, the brittle slag is mechanically removed from the metal regulus (by tapping with a hammer). There are various ways of analyzing the regulus for metal contents. Our preferred method is sawing, milling or

drilling vertically through the regulus and analyzing the shavings by conventional chemical methods. An alternative is sawing the regulus in half (Fig. 6.4 right) and measuring the metal concentrations directly on its surface along the cut, e.g. with an XRF or spark emission spectrometer.

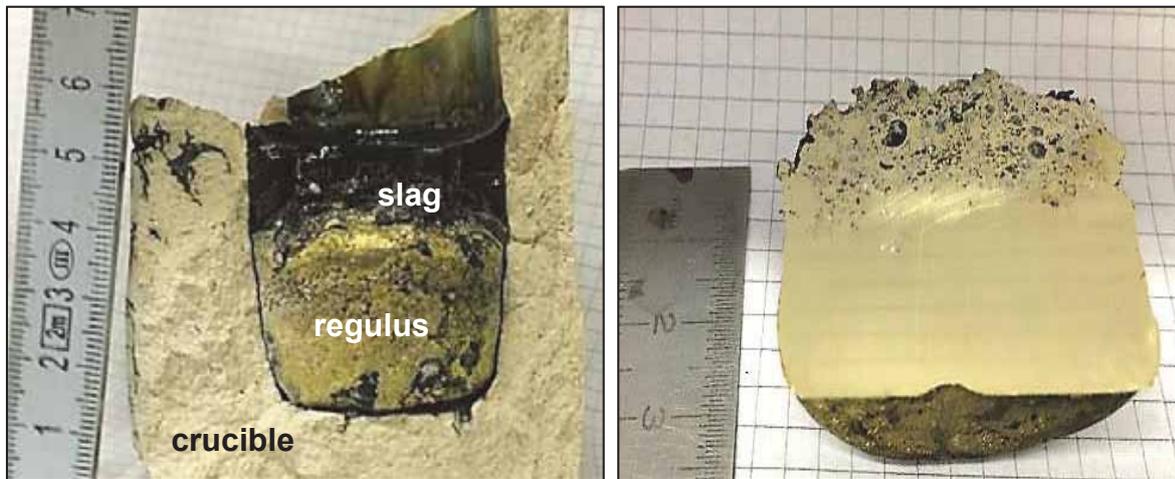


Fig. 6.4: Fire assay of 250g HNF concentrate from BA. The slag collects on top of the metal regulus (right). After the regulus has been sawed in half the surface can be analyzed for example by spark emission spectroscopy.

### 6.2.3 Selective comminution

This method – also known as the “UMTEC-method” - has been developed not only for analyzing metal content but to generate additional information useful for evaluating BA metal extraction processes. Not only the amount of the metals, but also their size distribution, is measured. This information can be used for forecasting the potential recoveries, if a given raw BA is processed on given separators. The method is also useful for evaluating plant performance by examining the residues. It is now part of a Swiss EPA directive for determining metal content in the residue from BA processing.

In essence, the method makes use of “selective comminution”. When subjected to compressive forces, as in jaw crushers or roll mills, the brittle mineral content of the BA is pulverized while the ductile metals are flattened but remain largely intact.

Fig. 6.5 presents an abbreviated schematic of the laboratory method applied to the residue of a BA residue after processing. As the coarse metals have already been removed from the BA, the MSM of the stockpile sample is, despite the material having a maximum particle size of 16 mm, only approximately 50 kg. First, the material is dried in order to interrupt corrosion processes that could reduce the content of native aluminum and steel. A multi stage comminution is followed by sieving of the broken material, until the native metals have been separated from the respective size fractions (only two of these stages are shown in Fig. 6.5).

The total metal contents (NF and FE) are calculated by weighing the masses of the metal concentrates with the respective mass fraction they were derived from. In case of the example shown in Fig. 6.5, assuming  $NF_{4-16}=0.28$  kg (of 40 kg) and  $NF_{1-4}=0.03$  kg (of 10kg), the total concentration of NF would be  $(0.28 \text{ kg} + 40 \text{ kg} / 10 \text{ kg} * 0.03 \text{ kg}) / 40 \text{ kg} = 1\%$  of dry mass (40 kg). As the water content of the 50 kg material drawn on site was 10 kg, the metal content of the bottom ash, as sampled, was 0.8%.

As the revenue for HNF is much higher than for LNF, not only the mass NF, but also the respective mass fractions of HNF and LNF need to be determined. The NF concentrates of sizes >16 mm can be sorted manually. For facilitated optical identification it may be advisable to clean the surfaces of the particles by briefly washing them in a dilute acid. As aluminum and SS pieces may look alike, a strong magnet may be helpful for unambiguous identification.

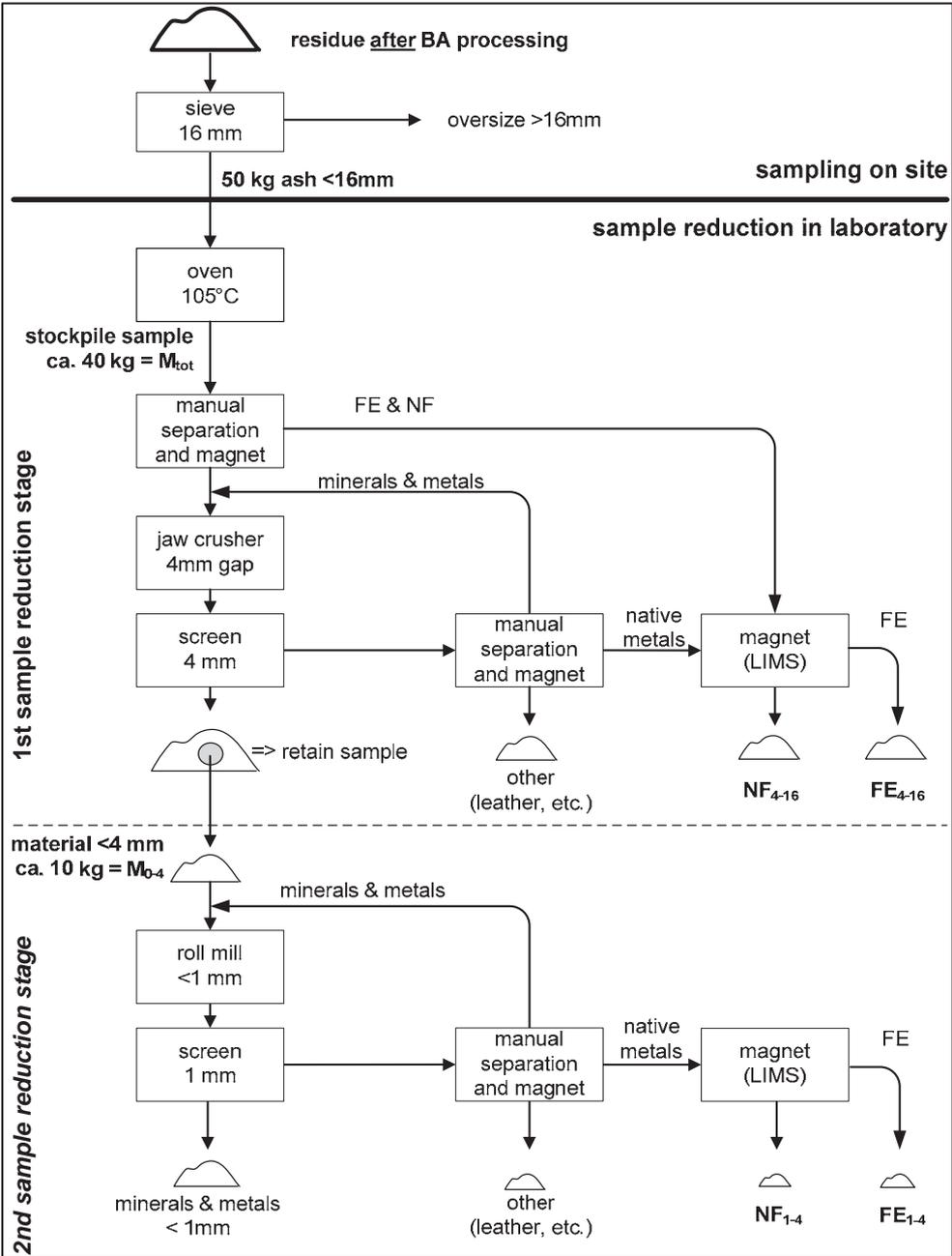


Fig. 6.5: Simplified schematic of metal analysis making use of selective comminution.

NF4-16 mm is separated with gravity concentration processes, for example on a jig. NF1-4 mm can be likewise separated by gravity concentration in a gold pan or by way of heavy media separation in a sodium tungstate solution at a density of 2.9 g/cm<sup>3</sup>. If the content of the individual metals in the HNF, in particular that of precious metals, needs to be determined, a representative sample of the HNF is subjected to a fire assay.

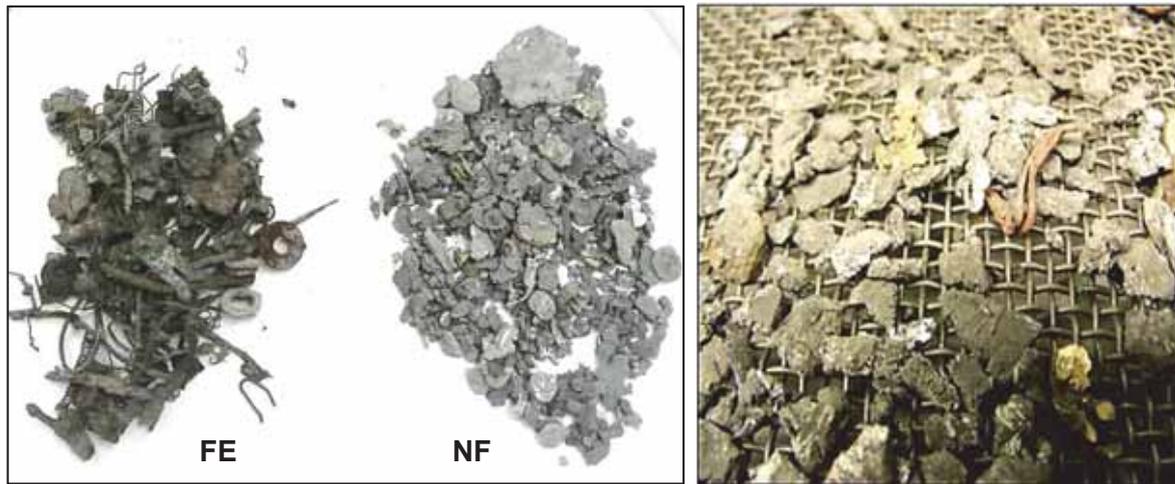


Fig. 6.6: Metals from selective comminution of BA: FE and NF 8-32 mm (left). NF on a 2 mm sieve (right). Note that the NF particles were flattened when passed through the jaw crusher (left) or roll mill (right).

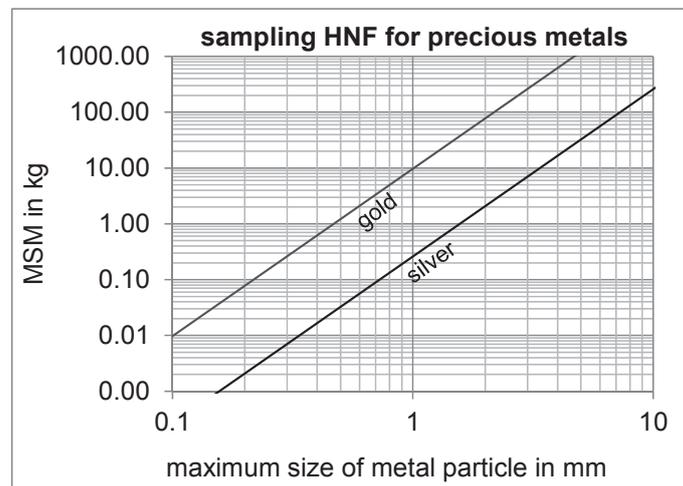


Fig. 6.7: The minimum masses required when sampling HNF concentrates for precious metals.

Due to the nugget effect, the MSM for determining precious metals in BA is much higher than that for the “mass metals” (see Fig. 6.1 right). An estimate of the MSM for a NF concentrate with respect to gold content can be derived from the following considerations. As stated previously, the ratio of copper:silver:gold in BA<2 mm is typically in the order of (10<sup>3</sup>000):(20...50):(0.7...1.5).

This ratio may be expanded also to larger size fractions, since copper is an indicator for GDP (as shown in Fig. 2.3) and one may assume that in regions with high GDP not only more electronics but also more jewelry are in circulation, and lost into the household waste, than elsewhere. Assuming that 50% of the NF concentrate be copper, the gold content of the concentrate would be 50g/t and that of silver 1 kg/t, the MSM for the precious metals are, according to eq. 5.1a, plotted in Fig. 6.7. Obviously it is quite impossible to determine the true gold content of a coarse HNF concentrate, for example 10 mm, with lab methods (because material containing 10mm gold particles would require ten tons of sample mass). In order to arrive at an order of magnitude estimate, one can however sample the HNF fines (e.g.<1 mm)

and analyze for copper and gold. The resulting copper/gold ratio can be used as an indicator for the copper/gold ratio in the coarse HNF (the copper content of which would need to be determined and is typically 35%-55%).

*Example: How much sample of HNF <1 mm (from BA) needs to be drawn if an error of  $\pm 20\%$  in the result for gold analysis by fire assay is tolerated? Taking eq. 5.2 leads to  $MSM/M^* = (0.2/0.1)^2 = 4$ . Reading from Fig. 6.7  $MSM = 10$  kg it is calculated that  $M^* = 2.5$  kg.*

If the NF metal content of a raw BA needs to be determined, the MSM is in the order of 300 kg or more. In this case it may be preferable to proceed with the first step of sample reduction on site. The material (e.g. 300 kg) is spread out and run over several times with a walk behind compactor (e.g. vibrating plate, tamper, or roller). Then it is classified at approximately 8 mm either with a sieve or with a rake and the coarse fraction is repeatedly run over again with the compactor until all mineral material is comminuted to <6 mm. The material >6 mm – comprised essentially of metals – is recovered for later separation into individual size fractions. The MSM of the material <6 mm is determined (e.g. 40 kg) and a representative sample drawn for further sample reduction in the lab.

Although the determination of NF content of the residue from a BA processing operation is crucial for assessing plant performance, this is rarely done because only few laboratories have both the know-how and the hardware to carry out such tests. At least the operators of high volume plants should consider developing their own “field procedures” along the lines of the discussed “selective comminution” method, possibly in cooperation with a local laboratory specialized in testing construction materials.

## 7. Economics of Metal Recovery from MWI Bottom Ash

The economics of metal recovery from BA depends on many variables, in particular on metal prices. Driven by rising metal prices, in Switzerland much of the BA that had been landfilled without prior processing within the past 3 decades has already been excavated and processed for metal recovery. Caution: the economic figures presented in this chapter are examples and serve exclusively for the purpose of illustrating the concepts. They must not be taken at face value for assessing the economics of a particular BA processing operation at a particular location at a particular point in time. For reference: currently (Feb. 2016) the currencies CHF, € and US\$ are approximately at parity.

### 7.1 Estimate of revenue for NF-concentrates

Table 7.1 is an overview of the metal contents and values assumed for the economic considerations discussed in this chapter. The overall value of the metals in BA is in the order of 100 €/t, of which 85% stems from the NF. While the HNF contribute only 10% by weight to the total metal content, approximately 60% of the total metal value contained in the BA lies in the HNF.

*Table 7.1: Contents and values of metals in BA (pieces >2mm)*

	metal	% BA	kg/t BA	€/kg	value €/t BA	% of total metal value
LNF	aluminum	1.65%	16.53	1.6	26.4	26.4%
HNF	copper	0.50%	5.00	5.6	28.0	27.9%
	stainless	0.25%	2.50	1.4	3.5	3.5%
	zinc	0.16%	1.63	1.9	3.1	3.1%
	lead	0.02%	0.21	1.7	0.4	0.4%
	gold	0.00005%	0.0005	37'000	18.5	18.4%
	silver	0.0010%	0.010	500	5.0	5.0%
	<i>total HNF</i>	<i>0.94%</i>	<i>9.35</i>		<i>58.4</i>	<i>58.3%</i>
NF	total NF	2.59%	25.88		84.9	84.7%
FE	steel	7.33%	73.3	0.21	15.4	15.3%
	<b>total metals</b>	<b>9.92%</b>	<b>99.18</b>		<b>100.28</b>	<b>100%</b>

The economic output of a BA processing operation depends to a large extent on the revenue gained from the sale of the NF concentrates. This revenue can be estimated in the following manner. According to Table 7.1 the BA contains 25.88 kg NF, of which 9.35 kg HNF and 16.53 kg LNF. The overall ratio of NF>2 mm in BA LNF:HNF is therefore approximately 1.8:1. However, since conventional BA processing recovers preferentially coarse particles, which are more easily separated than fine ones, and since in the coarse particle fraction the LNF:HNF ratio is considerably higher than overall (Fig. 3.13), the LNF:HNF ratio in the concentrates of BA processing is typically in the order of 3:1.

When estimating the prices for NF concentrates, one needs to first correct for the mass of the mineral matter present in the NF concentrate. This mass depends on the separator setting and the particle size of the NF and is typically on average 25% (being as high as 35% for concentrates of small particles <8 mm). To calculate the value of the LNF one may assume that all LNF is aluminum (which is indeed usually the case). The HNF is comprised of mostly of copper, stainless steel, zinc, lead, gold and silver. Reading from Table 7.1 that 9.35 kg of HNF has a value of 58.4 € gives the value of HNF as 6.24 €/kg. Note that this applies to the value of the HNF provided its composition is as shown in Table 7.1. The HNF concentrate actually derived from BA processing has a different composition. Because the higher value metals (Cu, Au) are somewhat depleted in the coarse size range while low value HNF like SS and brass are enriched, the value per kg of HNF recovered in the concentrate is lower than that per kg of HNF

originally present in the BA. Therefore one can use the market price of the copper (in Table 7.1 this is 5.6 €/kg) as a fairly precise estimator for the average metal value contained in a HNF concentrate.

The sales price of the NF is generally linked to the metal prices published by the LME (London Metal Exchange). But since the concentrate needs to be further processed (in sink/float plants and smelters) to deliver the metals in the form of bullion, the price actually paid by the sink/float plants or smelters for the metal content of the NF concentrate is only approximately 60% of the LME notation. The sales price to be expected per kg NF concentrate (assuming LNF:HNF=3:1) is therefore estimated as follows:

Mass balance: 1 kg NF = 0.25 kg minerals + 0.19 kg copper + 0.56 kg aluminum

LME prices:  $LME_{NF} = LME_{copper} * 0.19 + LME_{aluminum} * 0.56$

sales price:  $sale_{NF} = 60\% * LME_{NF} = 0.11 * LME_{copper} + 0.34 * LME_{aluminum}$

*Example: Estimate the sales price for 100t NF concentrates (LNF+HNF) from a BA processing operation on the basis of  $LME_{copper}=6'000$  €/t and  $LME_{aluminum}=2'000$ €/t. The estimate is  $0.11*6'000+0.34*2'000 = 134'000$  €.*

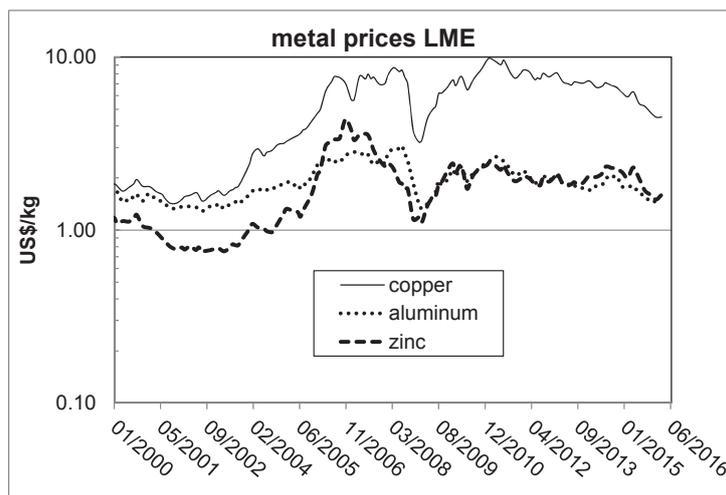


Fig. 7.1: The development of metal prices over the past 16 years.

## 7.2 Cost structures of dry and wet processing plants

An example for the cost structure of a dry BA processing plant is shown in Table 7.2 (upper table). It is assumed that the landfill fee is €28/t, regardless whether raw BA or processed BA is dumped.

Although it is often supposed that the investment (capex) of a plant increases linearly with size (throughput), this is not the case. A much better approximation is the assumption that investment is linked to the square root of throughput.

In table 7.2 the cost of labor is assumed to increase linearly with throughput while the cost of overhead is assumed to only increase with the square root of throughput. Revenue is not only generated from the sale of metals but also from the disposal fee saved by removing the FE and NF metals concentrates from the BA. These savings correspond with the mass of the concentrate which is, including 25% minerals, 9% ( $(7\%*80\%+2.6\%*45\%)/75\%=9\%$ ).

**Table 7.2: cost/revenue structure of BA processing operations**

<b>dry processing &amp; landfill</b>	
<b>base data</b>	<b>cost</b>
annual throughput	90'000 t/a
content FE metal >2mm in BA	7.0%
content NF metal >2mm in BA	2.6%
recovery FE metal >2mm in concentrate	80%
recovery NF metal >2mm in concentrate	45%
revenue FE metal	100 €/t
revenue NF metal*	1'800 €/t
landfill fee for raw BA (incl. transport)	28 €/t
disposal of processed BA (incl. transport)	28 €/t
capex: 10'000×throughput <sup>0.5</sup>	3'000'000 €
payoff period (linear)	6 years
interest rate	10%
cost labor; 6×throughput	540'000 €/year
cost overhead...:1500×throughput <sup>0.5</sup>	450'000 €/year
<i>*Based on "metal recovery" i.e. 1t pure metal (no minerals). This is equivalent to 1'800*0.75=1'350€/t of 1.33t concentrate incl. 25% minerals)</i>	
	payoff capex
	interest (0.5×capex×interest)
	maintenance (8% of capex)
	energy (10 kWh/t @ 0.15€/kWh)
	labor
	overhead, chem. analyses...
	2'015'000 €/year
	22.4 €/ton
	<b>revenue</b>
	saved disposal fees
	revenue FE metal
	revenue NE metal
	2'626'872 €/year
	29.2 €/ton
	<b>profit</b>
	611'872 €/year
	6.8 €/ton
<b>wet processing &amp; recycle</b>	
<b>base data</b>	<b>cost</b>
annual throughput	90'000 t/a
content FE metal >2mm in BA	7.0%
content NF metal >2mm in BA	2.6%
recovery FE metal >2mm in concentrate	80%
recovery NF metal >2mm in concentrate	45%
revenue FE metal	100 €/t
revenue NF metal	1'800 €/t
landfill fee for raw BA (incl. transport)	28 €/t
disposal of processed BA (incl. transport)	5 €/t
capex: 15'000×throughput <sup>0.5</sup>	4'500'000 €
payoff period (linear)	6 years
interest rate	10%
cost labor: 6×throughput	540'000 €/year
cost overhead...:1500×throughput <sup>0.5</sup>	450'000 €/year
additional operating cost wet processing	14.0 €/t
	payoff capex
	interest (0.5×capex×interest)
	maintenance (8% of capex)
	energy (10 kWh/t @ 0.15€/kWh)
	labor
	overhead, chem. analyses...
	disposal of 10% sludge
	add. operating cost wet proc.
	1'260'000 €/year
	3'972'000 €/year
	44.1 €/ton
	<b>revenue</b>
	saved disposal fees
	revenue FE metal
	revenue NE metal
	4'555'020 €/year
	50.6 €/ton
	<b>profit</b>
	583'020 €/year
	6.5 €/ton

The interrelation between throughput and profit is shown in Fig. 7.2: profit increases with throughput. At a throughput of 90'000t/a a profit of 6.8 €/t may be expected. A plant with double the throughput may expect the profit to increase by 65% to 11.2 €/t. While larger plants are more profitable than small ones, it must be kept in mind that, with increasing plant size, generally the catchment area for the acquisition of BA also needs to grow. This in turn increases transport costs. The benefits gained from a high capacity BA processing operation must therefore be balanced against the increased cost of transport.

The revenue for BA processing depends crucially on the amount of the NF concentrate produced. This depends, in turn, on the amount NF present in the feed (head grade), the fraction that is actually recovered, and the sales price (which is a linear function of the LME price).

In Table 7.1 these are:

- content NF in the feed: 2.6%
- NF metal recovery: 45%
- $LME_{NF}$ : 1'800 €/t

For a plant with an annual 90'000t throughput, the increase of a mere 10% in any of these parameters would boost profit by approximately 30%. If all three parameters are simultaneously increased by 10%, the profit would more than double. The treatment cost for the plant producing 90'000t/a is 24.4 €/t, of which approximately 50% is fixed costs. With fixed costs this high, the plant should be run in at least two shifts in order to maximize profit.

The lower part of table 7.1 is a breakdown of cost and revenue for a wet processing plant where most of the residue is used for road construction and only 10% needs to be landfilled (the filter cake). It is assumed that the BA residue (after processing) is delivered to the customer at no cost (5 €/t being the cost of the transport). For a wet processing plant both capital and operational expenditure are higher than for a dry processing plant. Consequently, the cost per ton of processed material is about double as high (44.1 € vs. 22.4 €).

Fig. 7.2 suggests that generally a wet processing operation is more favorable at high throughputs, while low throughput would favor a dry processing operation. Note that the wet processing operation recovers approximately half of the revenue from avoided disposal cost. Wet processing may therefore be favorable where landfilling is expensive.

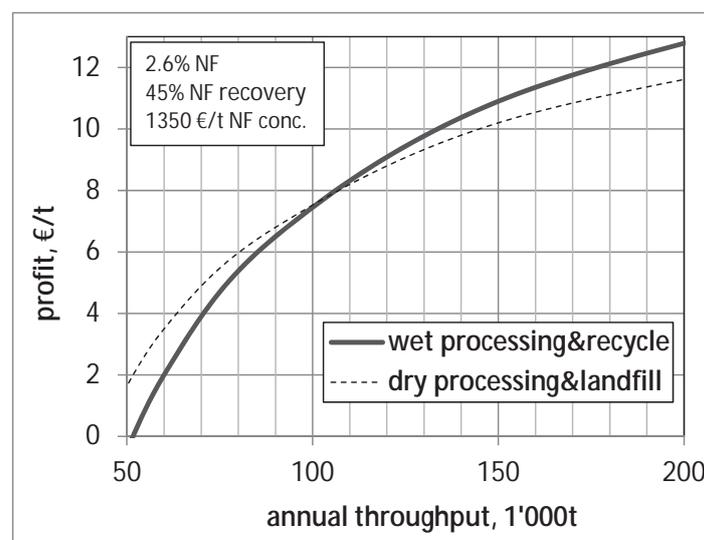


Fig. 7.2 The profit gained from BA processing depends considerably on the throughput of the plant.

### 7.3 Relevance of metal prices for the recovery of metals from BA

With a given BA, both metal recovery and LME price affect the profitability of the BA processing operation significantly. While plant operators have no influence on the LME metal prices, they can – to a certain extent - increase metal recovery in two ways. Increased recovery is of course best accomplished by making best use of the equipment at hand (by, for example, hiring highly qualified personnel). Provided that a plant is actually run at its optimum performance, further increase in recovery requires investment in more advanced hardware,

which also increases cost (e.g. retrofitting with high performance ECS or sensor sorters). The tradeoff between increased cost and additional revenue is being discussed below.

The data shown in Fig. 7.3 are extracted from Table 7.3 (which is based on data from Table 7.2). In the columns 1-3 of Table 7.3 the cost of various processing plants and the respective recoveries for NF and FE are shown at a fixed NF price (1.8 €/kg).

Low recoveries are the consequence of low-tech equipment, for example the installation of just one overhead magnet. This device may recover 50% of the FE at a cost of only 1 €/t (but no NF). If, instead, a low tech mobile processing unit was used (one drum magnet, one ECS, some handpicking), one may expect a FE recovery of 70% and a NF recovery of 20% at a cost of 11 €/t. For a plant as shown in the upper part of Table 7.2 the recovery of NF had been assumed 45% and that of FE 80% at a cost of 22.4 €/t. A high tech plant, that, for example, employs a sensor sorter as shown in Fig. 5.1 (advanced flowsheet), may recover 70% of the NF and 90% FE at a cost of 38 €/t. Going to the extreme, the recovery of 90% NF would require, in addition to the sensor sorter, a two-step comminution and several parallel trains of LIMS, HIMS, and ECS in series, raising the cost to 65 €/t of BA processed. With increased recovery the complexity of the plant and the respective cost per ton processed increases initially in a more or less linear fashion, but – as 100% recovery is approached – exponentially.

The revenue from the sale of the NF concentrate increases in a linear fashion with the amount of NF extracted, i.e. linearly with NF recovery. Consequently the profit, being revenue minus cost, initially increases until a maximum is reached and then rapidly declines. The maximum of the profit curve is the recovery that the plant should be designed for (i.e. around 50% in Fig 7.3 left).

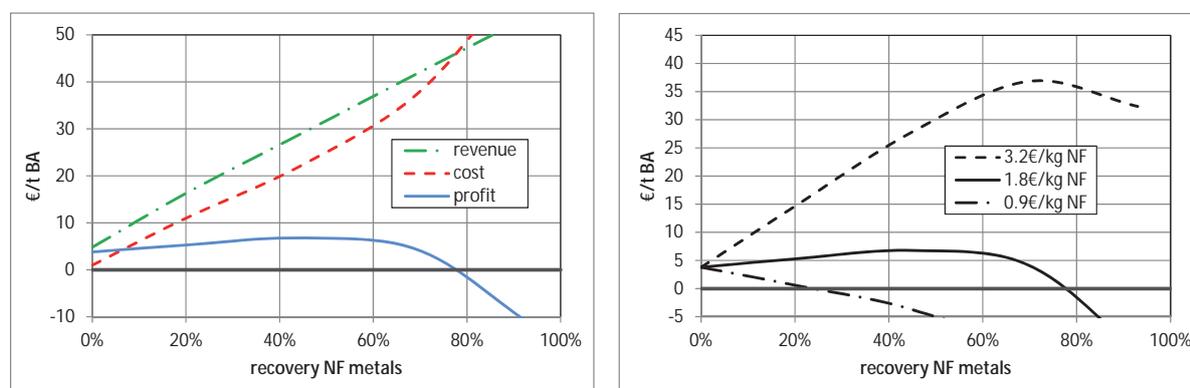


Fig. 7.3: Development of cost and revenue depending on the recovery of NF (left). The profit optimum is at approximately 50%. This optimum not only increases with increasing NF prices but also shifts towards higher recoveries (right).

Table 7.3: The data plotted in Fig. 7.3

1	2	3	4	5	6	7	8	9	10
cost per t processed	metal content NF: 26kg/t BA	metal content FE: 70kg/t BA	0.1 €/kg revenue FE	0.028€/kg saved disp. fee	1.8 €/kgNF revenue NF	total revenue	profit at 1.8€/kg NF	profit at 0.9€/kg NF	profit at 3.6€/kg NF
€/tBA	NF recovery	FE recovery	€/tBA	€/tBA	€/tBA	€/tBA	€/tBA	€/tBA	€/tBA
1	0%	50%	3.50	1.31	0.00	4.81	3.81	3.8	3.8
11	20%	70%	4.90	2.02	9.36	16.28	5.28	0.6	14.6
22.4	45%	80%	5.60	2.53	21.06	29.19	6.79	-3.7	27.8
38	70%	90%	6.30	3.03	32.76	42.09	4.09	-12.3	36.9
65	93%	98%	6.86	3.46	43.52	53.85	-11.15	-32.9	32.4

It is important to note that these considerations pertain to the design of a *new* plant. For an existing plant, that had been designed for a higher recovery, it would only make sense to run this plant at lower than design recoveries if the variable costs were significantly decreased (which is rarely the case once the equipment has been installed).

NF revenue not only depends on NF recovery but also on LME prices. Fig. 7.3 (right) shows what would happen if the assumed revenue of 1.8 €/kg NF was increased or decreased by a factor of 2. If the NF price was as low as 0.9 €/kg NF, only the recovery of the FE scrap by means of an overhead magnet would be profitable. If the LME prices doubled, however, not only would the profit increase at any given recovery but the optimum recovery would shift towards 70% instead of 50%. While plant operators have no control over LME prices, they must be acutely aware of the chances and risks for their operation associated with highly volatile metal prices.

As shown in chapter 2, countries that employ MWI also recycle separately collected wastes (e.g. metals), and this separate collection of metals is financially supported by the governments, either directly through a recycling fee or indirectly through legislation. Considering that less than half of the metals out of consumer wastes can be directly recycled (by way of separate collection), it is peculiar that their recovery from the residual waste, for example after incineration, does not receive any government support at all.

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