

# THE SUBMERGED ARC FURNACE (SAF) STATE OF ART IN METAL RECOVERY FROM NON-FERROUS SLAGS

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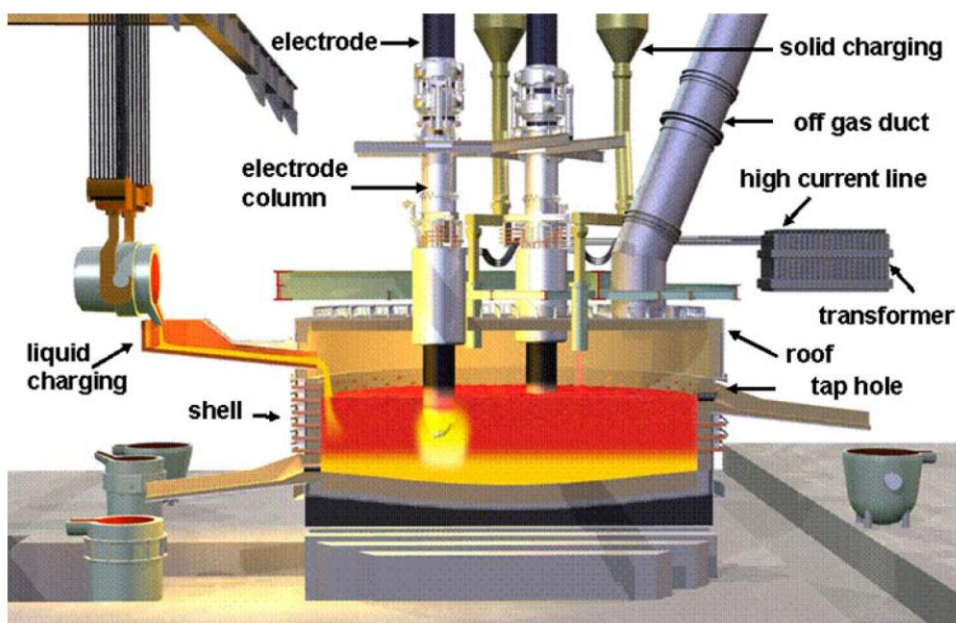
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## Introduction

In metallurgical production, the traditional aim is to increase recovery of metal to the maximum. It becomes apparent that the application of the submerged arc furnace (SAF) in non-ferrous metallurgy processes is increasing, in particular for the extraction of zinc, lead, copper and nickel, but also for the processing of residues and stripping of slag. SMS group has been developing this technology for 100 years, and has supplied a diverse market with about 700 furnaces and major furnace component<sup>2</sup>. During this time, numerous applications were constantly developed serving various user<sup>3</sup>. Especially slag cleaning furnaces have supplied more than 20 in the last 40 years. Slag cleaning furnaces are commonly connected to smelting units such as Teniente and Noranda converter, Outokumpu flash smelter, reverberatory furnaces. The main function of the furnace is the reduction of the matte and metal level in the slag. Depending on the upstream process, the slag is either liquid-charged *via* launders into the furnace or is cold-charged in solid form *via* conventional feeding systems.

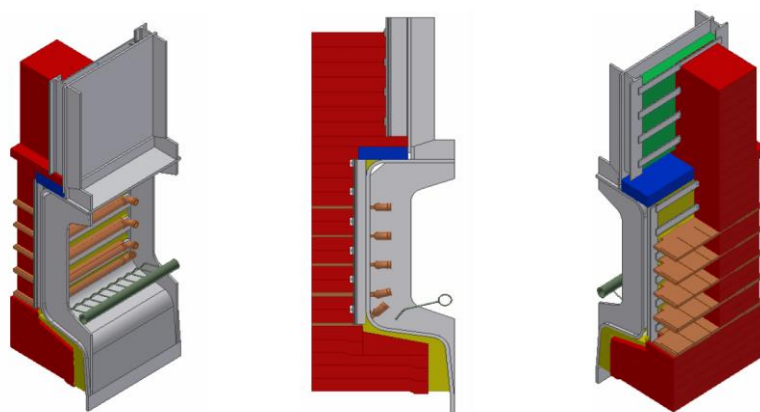
## Principle of SAF

The SAF works with electrical energy which is converted into heat, mainly using the electrical resistance of the burden or molten slag. The electrodes are immersed in the melt which provides the required energy exchange area between electrode and melt. The melt is the heat exchange medium which supplies the energy required for melting and chemical reactions of the charge<sup>1</sup>. The submerged arc furnace comprises the following major equipment parts as seen in Figure 1.



**Figure 1:** Main components of a submerged arc furnace<sup>1</sup>

A typical furnace with slag operation comprises a round or rectangular furnace shell with tap holes for slag, matte or metal. The furnace shell is refractory lined and – if additional shell cooling is required by the process – water-cooled. Various sidewall cooling systems are available depending on the specification of the process: rinse cooling, spray cooling, chamber cooling, channel cooling or Cu-strip cooling (see Figure 2). For safety reasons the cooling channels remain outside the furnace shell. In certain applications, such as PGM (Platinum Group Metals), pig iron and several ferroalloys and nonferrous processes, a sufficient heat removal rate will create a layer of frozen slag, the so-called “freeze line”, which protects the remaining sidewall lining. In this case a high thermal conductivity of the lining is of capital importance.

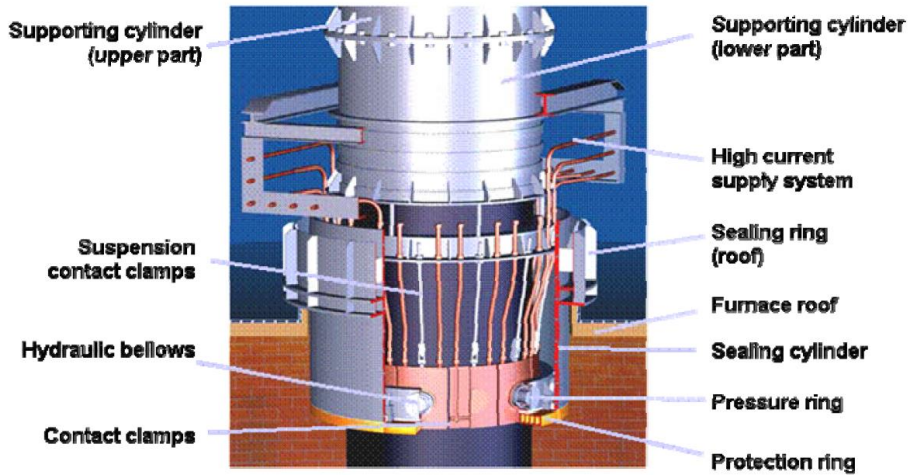


**Figure 2:** Side wall copper stripe cooling<sup>5</sup>

The shell bottom is cooled, if cooling is necessary, by forced air ventilation. In special applications, water cooling may be required. Furnace roofs, either brick type or water

cooled type, comprises all required glands, openings and sealings for the electrode columns, charging pipes and off-gas ducts.

The electrical energy is transferred into the furnace *via* self-baking carbon electrodes or pre-baked graphite electrodes. The electrode arrangements are depending on the process and the installed power. They are arranged either centrally (single electrode) or as triangle (three electrodes); in case of a rectangular furnace, they are arranged either as 3 or 6 electrodes along the centerline as 2 lines with 3 electrodes each. Therefore, one of the key components of the furnace is the electrode column (see Figure 3).



**Figure 3:** Electrode column<sup>4</sup>

The electrode is semi-automatically slipped into the bath, carried out under full electrical furnace load with no interruptions to furnace operation. The electrode column assemblies contain all facilities to hold, slip, backslip and regulate the penetration into the bath. All operations on the electrodes are executed hydraulically. The electrical power is normally transferred from the furnace transformers *via* high current lines, water-cooled flexibles, bus tubes at the electrodes and the contact clamps into the electrodes. Today the control and supervision is effected by a PLC (Programmable Logic Controller) and visualisation system. A back-up for manual operation is foreseen and is located in the control room<sup>4</sup>. The process gas created from the chemical reactions is provided at the gas stack connection at the furnace roof. Depending on the plant design, the process gas is combusted either inside of the furnace or outside the furnace by balanced addition of combustion and cooling air and sent to the filter system. If the process generates off-gas which contains a certain amount of CO in the process gas or other hazardous substances, the furnace is designed as a closed furnace type.

The above described furnace type is considered to be a typical design. Various other layout/design options meet the individual requirements of specific processes. Successful operation is always based on the right choice of furnace design and furnace dimensions. The choice of the raw material according to the customer's aspects has

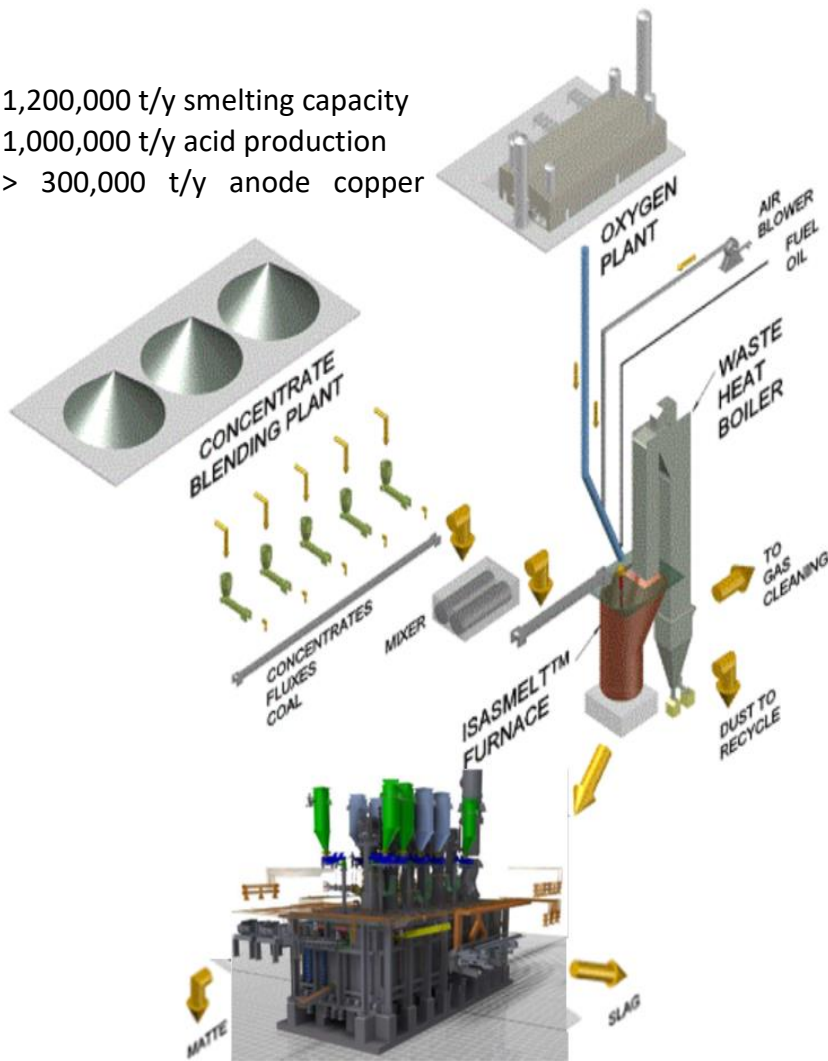
the biggest impact on the process. On the one hand, it affects the slag composition and on the other hand, the smelting pattern inside the furnace based on the physical properties and the amount of energy input.

## Exemplary results and Industrial applications

### Copper Slag Cleaning First Quantum at Kansanshi Copper Smelter

Copper Slag cleaning in (electrical) furnaces has been increasingly popular over the last decades. METIX, part of the SMS group, has recently constructed an electric furnace for the First quantum project at Kanshansi, Zambia. The plant was commissioned successfully in February 2015.

- 1,200,000 t/y smelting capacity
- 1,000,000 t/y acid production
- > 300,000 t/y anode copper



**Figure 4:** 3D illustration of Kansanshi Copper Smelter

First Quantum Minerals Limited, was already a major producer of SX-EW (Solvent extraction and electrowinning) copper at their operation in Kansanshi, Zambia when they chose to expand their operation by installing a copper smelter. An ISASMELT was

chosen for the smelting furnace application. The new smelter concept with an downstream Matte Settling Electric Furnace promises to provide a major benefit. The Kansanshi Smelter is producing 300,000 tonnes per annum of blister copper from a concentrate blend based on feed from the Kansanshi and Sentinel mines by treating up to 1.2 million dry t/y copper concentrates.

There is a trend towards to semi-continuous operating practice of the primary smelters (such as ISASMELT or Ausmelt) as well as of the slag cleaning furnaces. The rectangular SAF is more suitable for this task due to better geometrical conditions. It is expected that, for continuous operation, the recovery rate of a rectangular furnace can be (depending on the specific parameters) 0.1–0.4% higher in comparison to the conventional round type SAF's. This fact persuaded First Quantum at Kansanshi copper smelter in Zambia to install a six-in-line rectangular slag cleaning furnace of 12.5 MW downstream a continuously operating ISASMELT furnace<sup>3</sup>. The Matte Settling Electric Furnace (MSEF) is capable of processing copper matte, various converter slags, reverts from different sources and coke to produce a slag with a maximum copper content of 0.7%. The MSEF at Kansanshi is continuously charged *via* two launders with matte and slag coming from the ISASMELT. The launders enter the furnace shell in the gas area through the side wall and are positioned to convey the slag directly into the reaction area to prevent the precipitation of a major quantity of magnetite. Molten slag from the PSC is charged by ladle at regular intervals into the furnace by means of two dedicated cast iron launders in the eastern end wall from the converter aisle side. A mixture of reverts, coke and limestone is fed through the feed chutes into the furnace on top of the molten slag layer in the furnace. The coke and limestone act as modifiers to the slag to improve conductivity and viscosity to assist with the copper matte settling in the furnace, while reverts are fed back to ensure recovery of contained copper.

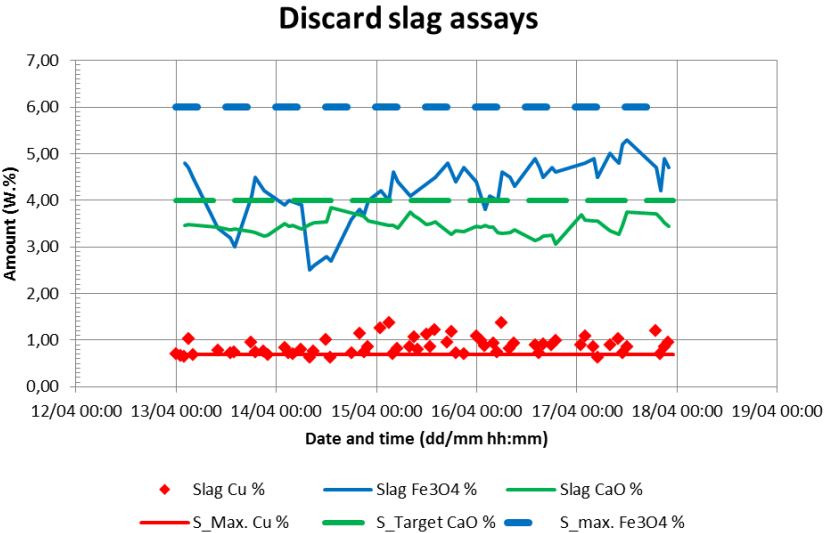
As mention above, the settling will be achieved by temperature increase and reduction of copper oxides and magnetite in the slag with carbon and maximising of the settling rate of the metallic or matte droplets by improving the viscosity of the slag. It is common agreement that reduction of oxides from a liquid slag with carbon:



is the first order reaction with respect to the oxide concentration, so the rate of magnetite reduction is represented by equation:

$$\frac{dC_{\text{Fe}_3\text{O}_4}}{dt} = k \frac{A_R \rho_S}{m_S} C_{\text{Fe}_3\text{O}_4} \quad (3)$$

where  $A_R$  represents reaction surface area and  $m_s/\rho_s$  ratio slag volume. Value of reaction constant  $k$  as the function of temperature has been determined experimentally by reduction of slag with coke. When reaching the fayalite point with approx. 29% of  $SiO_2$  the viscosity of the slag shows a maximum regarding the viscosity. A slag with 30 to 33%  $SiO_2$  will lead to poor slag cleaning results as it is very difficult for Cu and Cu matte droplets to settle. This is one of the reasons to keep the magnetite content in the range of 4 to 6%. Figure 5 shows representative discard slag results with the correlation between magnetite and total Cu content in MSEF slag.



**Figure 5:** Kansanshi MSEF discard slag assays CW16 2015

Tapping of the furnace is done through eight matte tap holes all situated on the same level and on both the north and south sides. The matte tap holes are equipped with a rail mounted mud gun and drill on each side of the furnace to assist with opening and closing of the tap holes. The tapped MSEF matte is ladled to Peirce-Smith converters and the MSEF slag will be granulated and discarded.

### Cobalt-Recovery from Copper-Smelter-Slags<sup>6</sup>

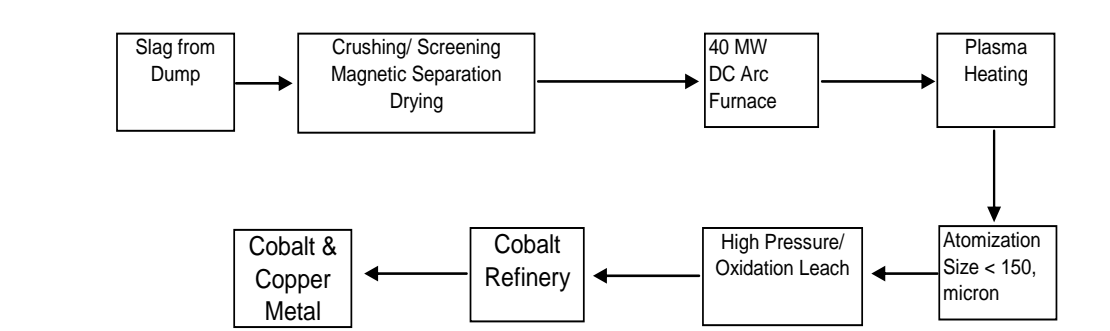
In January 2001, Chambishi Metals commissioned a 40 MW DC arc furnace using selective carbothermic reduction for processing reverberatory furnace slag stockpile of 20 million tons containing 0.34 to 4.5% cobalt, and an average of 1.1% copper. Cobalt occurs as fayalite whereas copper occurs as oxide and sulphide. An atomiser unit to atomise furnace product, molten Co/Cu/Fe alloy, and a pressure oxidation leach process to leach cobalt and copper and separate iron as goethite was also commissioned. Cobalt and copper containing solution is processed in the refinery for production of metals. Commercial application of the DC arc furnace on smelting of copper smelter slag was unique and therefore required extensive literature search to broaden the knowledge of the theory and the practice.

Mineralogical analysis of the slag carried out by MINTEK, RSA, as shown in Table 1, indicate that the cobalt is present as fayalite (Iron Silicate Matrix) whereas copper is present as oxide, sulphide and metal. Presence of significant level of sulphide and metallic copper are due to entrained losses.

**Table 1:** Mineralogical composition of the Rokana Reverberatory Slag

Group Total Co	Mineral Phases	% of Total Cu	% of Total Co
Slag	Fe, Ca, Al (Mg, K, Co, Cu) - Silicates	46.6	94.6
Spinel	Fe(Al, Cr, Ti, Ca, Co, Cu) - Oxides	1.1	5.2
Sulphide	Cu(Co, Fe) - Sulphide	39.3	0.2
Metal	Cu(Co, Fe) - Metal	13.0	< 0.1
	Total	100.0	100.0

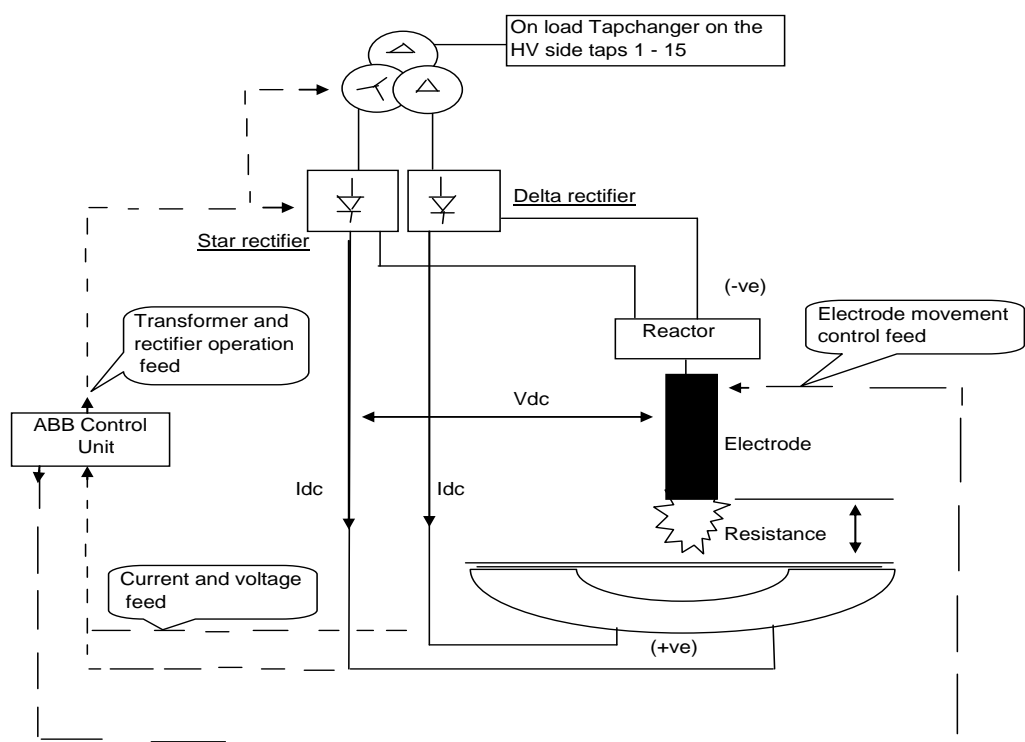
Following commissioning of the 40 MW DC Arc furnace in the year 2001, the Chambishi operation’s flow sheet is shown in Figure 6. The slag excavated from the dump, crushed and screened to +6, -16 mm particle size, beneficiated to high grade the furnace feed by 15–20%, dried to reduce moisture to < 1%, is fed to the furnace. Furnace product is an alloy of cobalt, copper and iron. Molten alloy is tapped and atomised to < 150 micron particle size which is pumped to the pressure/oxidation leach plant for leaching cobalt and separating iron as goethite. The leach solution containing cobalt, copper and other trace impurity elements is processed in cobalt refinery for cobalt and copper metal production.



**Figure 6:** Simplified Flow Chart of Chambishi Operation after 2001

Furnace was experiencing severe power fluctuations which caused erratic feed rates. This resulted in difficulties in controlling slag bath temperature to 1500–1550°C. Overfeeding was causing cooling of the bath whereas underfeeding causing high temperature, both changing the resistance. This frequent swing in the bath resistance triggered the electrode movement to maintain power set point, causing power fluctuations. Figure 7 gives the circuit control diagram originally built in PLC. The power fluctuations were experienced due to a sudden drop in resistance causing the power to begin to drop. The controller would try to compensate by increasing the current, thus maintaining the power at the set point, until it eventually reached a maximum

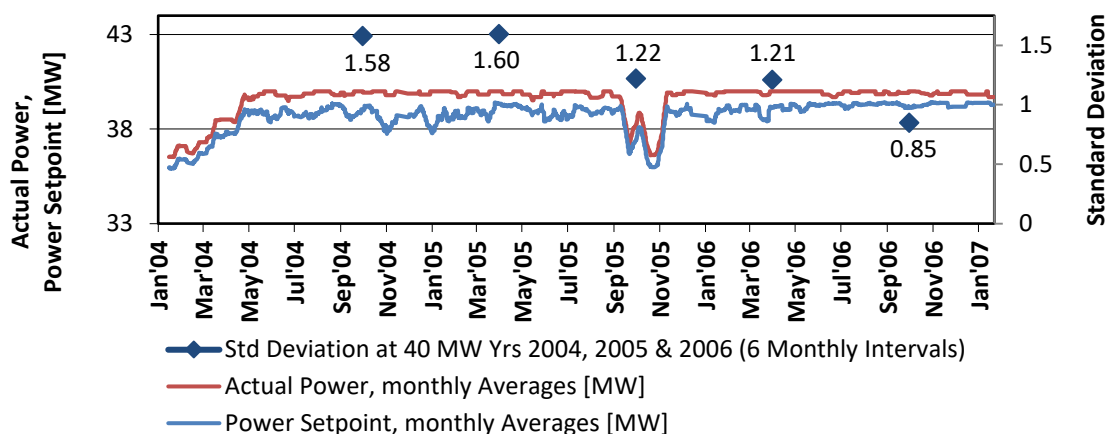
limit for the tap position. Once the current reached the maximum limit then the power would fall in line with the resistance. Under these circumstances the tap changer should tap down, thus increasing the allowable current. The ABB controller prevented the transformer from tapping down below tap 10. This was designed to limit the maximum current such that the current density remained under  $300 \text{ A/m}^2$ , specified by the supplier. The ABB was convinced to remove the tap changer control from their custom PLC. Chambishi then engineered this functionality into the Plant Scape control system.



**Figure 7:** Schematic electrical/control diagram for electrode movement for power control

The advantage of this change was that Chambishi had the flexibility to alter the philosophy whilst retaining the integrity of the safety features built into the ABB control system. The Plant Scape system requests a tap change from the ABB system, which executes it if it is safe to do so. The maximum limit for current was set at 70 kA ( $250 \text{ A/m}^2$ ). After a comprehensive change in tapping position was made to stabilise the power, the standard deviation improved to 0.84 from 1.6 in the year 2005, indicating a significant improvement as shown in Figure 8. The standard deviation improved from 1.6 to 0.847 by the end of the year 2006. The standard deviations are calculated for 6 monthly averages at 40 MW power only.





**Figure 8:** Furnace power set point vs. Actual power (Jan'04 – Jan'07)

### Valorisation of Pb containing Slags<sup>7</sup>

The lead and zinc contents of slags from lead production vary depending on the type of the production process and the process parameters. A slag from an oxidation step of crude lead production has a lead content of 50% or more, whereas a slag from a reduction step usually lies below 6% lead and 18% zinc. Therefore, one motivation for a subsequent treatment of the slag is to increase the metal yield of the production line and by doing so to increase the profitability of the process. If the metals contained in the otherwise discarded slag are regained and transferred into a product this can decrease the use of primary resources and thus increase the sustainability of lead production. This in turn can achieve or add to a positive company's perception of the public.

Facts about lead winning processes are published only fragmentary and intermittent. On the basis of the 2012 minerals yearbook for lead by the USGS<sup>9</sup> and the 2010 lead smelter survey<sup>8</sup>, it is possible to estimate the amount of slags that have been produced in 2008 after the reduction step of crude lead winning by smelters that treat only primary feed material or a mixture of primary and secondary feed materials. On the basis that 2008 the world lead production from primary sources is 3.9 million tons which represents a share of 75% in the primary and primary/ secondary smelter's lead production, and that 3% of the lead input of these smelters is lost in the slag, it can be calculated that the amount of slag is 5.3 million tons for 2008. If it is assumed that these slags contain in average 3% lead and 10% zinc, the lead amount in the slag is 160 000 tons, zinc amounts to 530 000 tons. Together, the metal value equals 982 million Euros. If all of these slags would have been treated towards a final slag with 0.1% lead and 1% zinc, more than 150 000 tons of lead and 370 000 tons of zinc could have been recovered, which represents a metal value of 751 million tons. In 2013, the lead production was 20% higher than in 2008, so that lead and zinc entrained in the slags will be considerably higher as well.

	conventional	direct wining			
crude lead winning	SM		ISA / Ausmelt	SKS OBBF	
oxidation		QSL	50 % 6 - 10 %	46 % 10 %	Kivcet (FS + EF)
reduction	BF 1.5 % 11 %	ISF 2.5 % 16 %	3 - 6 % 8 - 16 %	SKS OSBF 2 % 16 %	6 % 18 %
slag cleaning	FF	ISA / Ausmelt		FF	FF
reduction	0.1 % 3 %	1 % 6 - 8 %		0.5 - 1.5 % 2.5 - 4 %	0.1 % 2 %

X	process / furnace	SM:	sinter machine
X	not employed in all installations	BF:	blast furnace
		ISF:	Imperial smelting furnace
		OBBF:	Oxygen-Rich Bottom-Blowing Furnace
		OSBF:	Oxygen-Rich Side-Blowing Furnace
		EF:	electric furnace
		FS:	flash smelter
		FF:	fuming furnace

**Figure 9:** Comparison of relevant lead winning processes with lead and zinc contents of the produced slags

The slag fuming process and, to a minor extent, the ISA/Ausmelt-process are the standard processes for the treatment of lead slags from crude lead production. Slag fuming takes place in a rectangular water-cooled furnace and is usually carried out batch wise, with durations from 30 minutes up to three hours per batch. By injection of pulverised carbon and air an active gas-slag-interface is established that can be twenty times the size of the furnace surface. At temperatures between 1150°C and 1300°C lead and zinc are reduced and volatilised from the slag and subsequently oxidised with additional air over the melt, so that they can be recovered in the flue dust as oxides. The slag produced by fuming typically contains 0.5% to 1.5% lead and 2% to 4% zinc. The main disadvantages of the process are that lower zinc contents are not possible due to the risk of iron reduction and that it produces large amounts of off gas and waste heat that have to be treated. There are some publications about the possibility of using an electric arc furnace or plasma furnace for the treatment of slags, but its main use still is as holding or settling furnace. Nevertheless, electric arc furnaces offer multiple advantages for a slag treatment:

- high versatility in terms of feed material (e.g. solid / liquid), processing mode (e.g. EAF / SAF / SRF etc.) and parameters (e.g. temperature; continuous / batchwise)
- small off gas volume
- little space required
- high energy density and high space-time-efficiency

On the opposite, the disadvantages are low turbulences in the bath and a high demand of electrical energy which can require an expensive electrical infrastructure. In this work, it has been investigated if the electric arc furnace is a viable alternative for the treatment of lead slags. It can be added into an existing process line with only little demand for space and off gas treatment. It certainly makes most sense to process the liquid slag inline of the main process, but there still is the option to feed solidified slag and to melt it before treatment, too. In any case the electric arc furnace is able to adjust the slag temperature as needed. It is possible to generate stronger reducing conditions than in the slag fuming process, because if iron is reduced to metallic state it can be molten and consumed as reducing agent. This should enable the electric arc furnace to produce slags which are low both in lead and zinc.

### Experimental

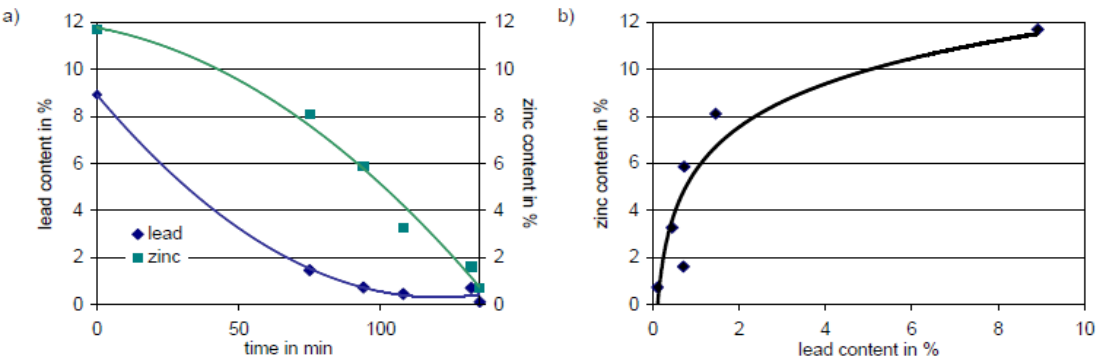
On the basis of the results of the preparatory work several test series in a 500 kW pilot-scale DC electric arc furnace have been conducted. In total more than six tons of six different industrial lead slags have been treated in 25 tests. The aim was to reduce the lead content below 0.1% and the zinc content below 1% and thereby to confirm and if possible extend the findings of the previous work, so that a scale-up into commercial scale is possible. Table 2 gives the initial chemical composition of the slags treated at pilot-scale at IME. Additionally, the average trial results are given in terms of final Pb and Zn concentration in the slags. The methods of coke additions to the melt are furthermore mentioned.

**Table 2:** Composition of treated slags and average final experimental results from 25 trials

slag type	Pb	Zn	Fe	S	SiO <sub>2</sub>	CaO	MgO	Al <sub>2</sub> O <sub>3</sub>
	[%]	[%]	[%]	[%]	[%]	[%]	[%]	[%]
I	8.9	11.7	28.2	0.1	25.1	9.8	1.3	2.3
II	1.1	5.1	25.5	2.6	24.6	16.4	4.4	7.9
III	55.3	5.9	10.3	0.1	7.1	3.4	0.4	0.7
IV	54.4	7.7	9.5	0.4	7.4	3.1	--	1
V	6.8	12.4	23.4	0.3	23.5	9.9	2.2	2.4
VI	5.8	11.4	24.5	0.4	23	10.8	1.5	2.8
slag type	method of coke feeding*	number of test runs	final slag					
			Pb [%]		Zn [%]			
			average	best	average	best		
I	h. el.	6	0.80	0.08	3.50	0.10		
II	h. el.	7	0.12	0.02	0.24	0.10		
III	h. el.	3	16.6	0.29	0.09	0.05		
IV	pre.	2	5.85	3.30	1.44	1.36		
V	h. el.	1	0.39	0.39	2.98	2.98		
	p.i.	4	0.78	0.38	6.09	3.89		
VI	p.i.	2	0.49	0.32	2.94	2.49		

\* "h.el."-hollow electrode, "pre"-premixed (slag & coke), "p.i."-pneumatic coke injection

Despite of the partly insufficient results (in terms of the final lead and zinc contents) important correlations can be derived from the test runs. At first, all test runs show a comparable interdependence of the zinc content upon the lead content throughout the duration of the test runs. Figure 10 shows exemplary a test run with slag type I. The lead content in the slag decreases faster than the zinc content (left). This could be expected, but more interesting is that the decline of the zinc content shows a logarithmic dependency on the lead content of the slag (right).



**Figure 10:** Slag type I: lead and zinc content vs. duration of a test run (left) and zinc content vs. lead content (right)

In consideration of the empirical variation in the samples, this dependency can be found in all test runs. For each slag type the logarithmic correlation between lead and zinc is unique, so that by averaging the logarithmic functions of the slag types one mean function for each slag type can be obtained. Generally speaking all of these trends can be expressed by an equation of the form

$$Zn_t = f(Pb_t) = A \cdot \ln(Pb_t) + B \quad (4)$$

$Zn_t$ : Zn content in % at time  $t$ ;  $Pb_t$ : Pb content in % at time  $t$ ; A: Factor; B: Constant

Altogether by calculating the factors A and B the time-dependent Zn concentration in relation to the Pb content may be expressed as:

$$Zn_t = Zn_0 - (-0.01953 \cdot Pb_0 + 4.455 - 0.0063 \cdot Pb_0^2 + 0.5552 \cdot Pb_0 - 3.436) \cdot \ln(Pb_t/Pb_0) \quad (5)$$

### Reduction Smelting of Redmud<sup>10</sup>

Red mud is the so-called bauxite residue after the solid-liquid separation from alkaline high pressure leaching of aluminium bearing bauxite ore. Depending on the composition of the primary mineral deposit, 4-7 tons of bauxite are necessary to gain 2 tons of alumina in order to produce 1 ton of aluminium. The leaching process is very selective on the extraction of aluminium and gallium and the undissolved minerals remain in the residue. Due to its high hematite content, the bauxite residue has a reddish colour and is therefore called red mud. With each ton of primary produced aluminium, 1-2.5 tons of red mud are generated. The composition of red mud varies strongly dependent on the composition of the original bauxite and the employed

process parameters (digestion temperature, caustic concentration, digestion time, etc.). Table 3 shows the average ranges in composition of commonly produced red muds. It can be seen that the predominant phases are iron compounds like hematite or goethite which can be easily recovered by carbothermic reduction.

**Table 3:** Composition of different red muds and composition of the raw material used in the experiments

Component in wt%	Average red mud	Red Mud Lünen (Germany)	After re-leaching	After re-leaching with CaO (CaO/SiO <sub>2</sub> ~ 0.5)
Fe <sub>2</sub> O <sub>3</sub>	30-50	29.5	35.5	34.7
Al <sub>2</sub> O <sub>3</sub>	10-20	27	18.3	17.7
SiO <sub>2</sub>	5-20	13.1	14.9	15
TiO <sub>2</sub>	3-15	8	9.3	9
Na <sub>2</sub> O	3-7	7	9.3	9.1
CaO	1-8	3.8	4.7	7.7
Cr <sub>2</sub> O <sub>3</sub>	-	0.35	0.41	0.4
P <sub>2</sub> O <sub>5</sub>	-	0.22	0.25	0.27
SO <sub>3</sub>	-	0.47	0.5	0.58

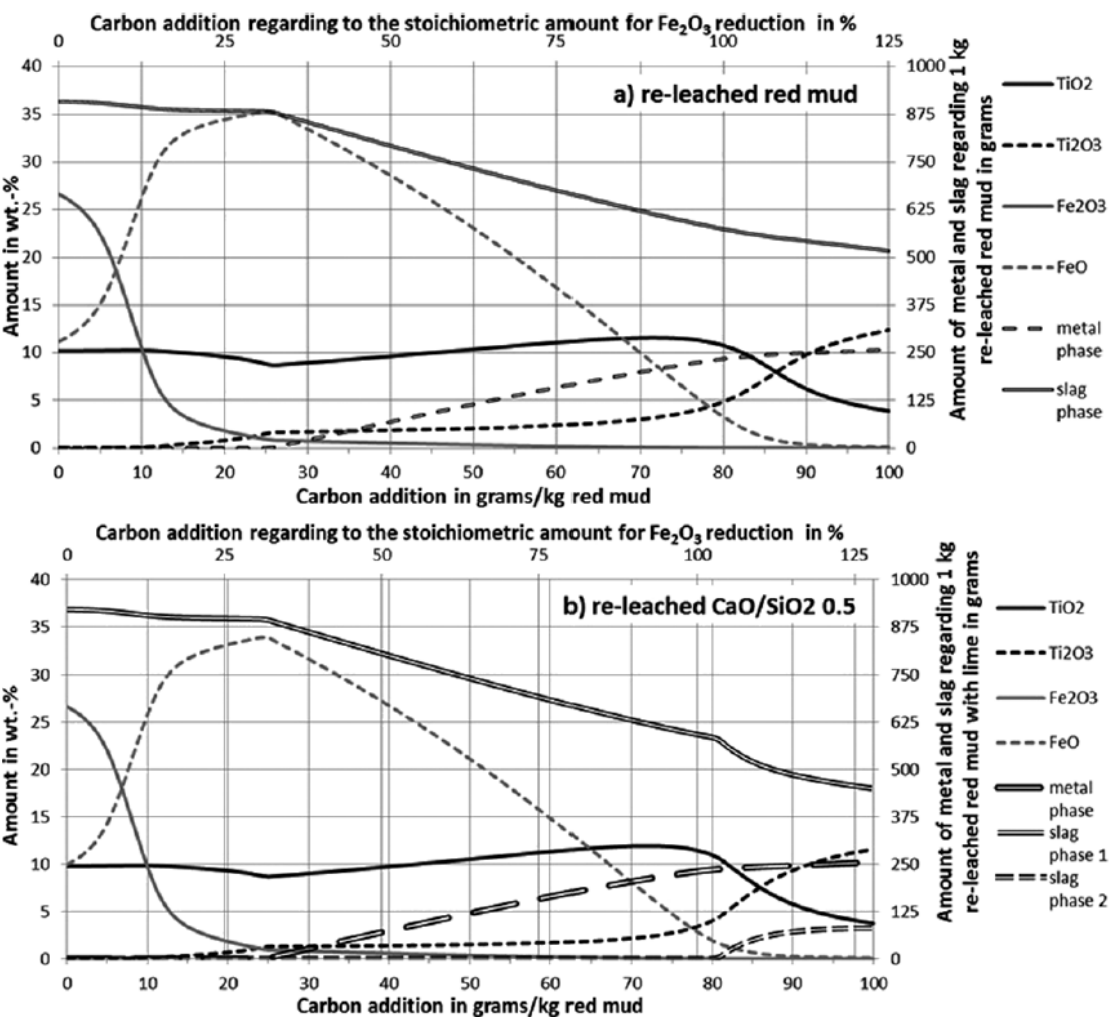
The examined red mud comes from the landfill of the former “Vereinigte Aluminiumwerke” near Lünen, Germany. The original composition is shown in column 1 and indicates high amounts of remaining alumina. In former times, high throughputs and cheap but poorly digestible bauxites as raw material refused a high alumina recovery. Therefore, X-ray diffraction patterns show, the predominant phases are hematite and still aluminium hydroxides (gibbsite and boehmite). The exact aluminium content is measured by x-ray fluorescence and the aluminium values in Table 3 are converted into alumina as quite common in literature. Therefore, the sum of all components does not reach 100%. This original red mud from landfill is one material for the following test trials.

Moreover, the entire research project includes a recovery of the main parts of the remaining alumina content. Thereby the red mud from the landfill is leached a second time employing the Bayer-process with optimised parameters. From this second leaching originates a red mud with the composition called after “re-leaching” (see Table 3) which is also examined for iron recovery.

The third examined raw material for carbothermic iron recovery experiments is presented in the last column of Table 3. In this case, small amounts of lime are added during the second leaching of the initial red mud in order to increase the recovery of alumina in the leaching step. Experiments confirmed an increased alumina yield of about 10% since the molar ratio CaO/SiO<sub>2</sub> increased from 0.31 in the initial red mud to 0.5 by the addition of lime. Thus, the remaining leaching residue is increased in its lime content, while the other constituents are slightly diluted.

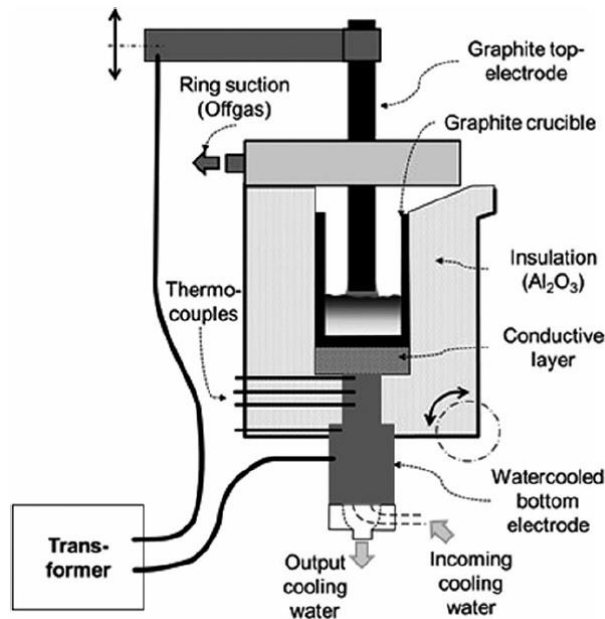
Theoretical considerations

It can be seen from Figure 11 that firstly the hematite is reduced to FeO. At about 15 grams carbon addition the co-reduction of titania to  $Ti_2O_3$  starts but stagnates on a low level. Only 15-35% of the titania is reduced to  $Ti_2O_3$ . By adding 26 g carbon per kg re-leached red mud, the hematite is almost completely (remaining  $Fe_2O_3$  content <0.9 wt%) converted into FeO. If the addition of the reducing agent is further increased, a phase of metallic iron will be produced and the FeO-content of the slag declines. At about 80 g carbon addition to the FeO activity descends rapidly and the titania activity increases dramatically which leads to a significant titania reduction. At 87.5 grams carbon addition the iron oxide content in slag is below 1 wt% so the reduction is completed. The metal phase firstly forms after the addition of 26 g carbon and at about 80 grams addition, carbon is even dissolved in the metal phase. At about 90 grams carbon addition the silica reduction to metallic silicon which is collected in the metal phase starts and at about 96 grams carbon, the reduction to titanium slightly begins (threshold 0.1 wt%).



**Figure 11:** Component distribution of oxides and sub-oxides during the reduction of a) re-leached red mud and b) re-leached red mud with lime addition calculated at 1650°C

The slag must not be contaminated by the used refractory and the aim of all experiments was a complete reduction of iron-oxide. Therefore, the raw material was smelted in a graphite crucible with the dimensions of 150/120 mm outer/inner diameter and 200 mm depth. To minimise the graphite consumption of the crucible due to the carbon absorption by the slag and to cover the melt from reoxidising at the surface, approximately 200 g (depending on the hematite content of the raw material) of lignite coke with a grain size below 1 mm have been fed together with 3 200 g pre-dried and lumpy raw material of a particle size 10-30 mm. All experiments were conducted in a tiltable AC electric arc furnace, which is presented in Figure 12. Although the set up with one top electrode made of graphite in combination with a water-sprayed copper bottom electrode is typical for DC mode, the furnace was driven without a rectifier. In order to establish a good electrical contact, a thin layer of graphite powder was put between the bottom electrode and the graphite crucible. The electrical power during the smelting process was 12-15 kW. A process temperature of 1600°C was aspired and controlled discontinuously by a pyrometer. Actually, the temperatures varied from 1600°C to 1700°C. The feed rate of the input material was continuously 2 500 g/h. After 90 min, the whole material has been fed and the melt was hold for additional 10 min to complete the hematite reduction. Afterwards the entire melt was tapped into a steel mould, in which metal and slag phase were separated due to the settling behaviour of the higher density metal droplets.



**Figure 12:** Sketch of the laboratory electric arc furnace

After the carbothermic reduction of red mud, the hazardous residue from the Bayer-process, a salable pig iron with about 4 wt% carbon and less than 0.5 wt% silicon and 0.2 wt% titanium can be obtained. The slag is by the addition of lime adjustable in its viscosity and the sodium content can be varied by the length of the reduction time. An important role during the reduction process plays the slag viscosity, since many

reduction products like CO, Na and SiO are gaseous species and have to pass through the melt. At high viscosities, the ascend of gas bubbles is hindered and leads to a foaming slag. The addition of lime during the prior leaching step affects the process handling in the smelting step positively by decreasing the slag viscosity which was obvious during the experiments. Unfortunately, the calculation of the viscosity is not that easy because the high content of titania in trivalent and tetravalent state is not exactly defined in any known model for slag viscosities. But FactSage® states the viscosities of all slags in the range of 0.27-0.36 Pa\*s with positive effect on lowering the viscosity by the addition of lime even though lime increases the liquidus temperature by 100°C to 1500°C.

## Summarising Remarks

The first SAF was commissioned 100 years ago in Germany. Since then a tremendous development of this smelting tool was recognised all over the world and submerged arc furnaces (SAF) are now operating in at least 20 different main industrial fields. In metallurgical production, the traditional aim of the SAF is to increase recovery of metals from slags to the maximum. SMS group has supplied numerous furnaces for this application. Especially in the field of rectangular furnace technology, the SAF could enhance its market position for slag cleaning.

The last order in rectangular furnace for the First quantum project at Kanshansi, Zambia demonstrates convincing results and confirms the intelligent solution (such as side wall cooling system, furnace integrity). The stirring reactor as a new unit in the line in copper production enables new dimensions in the industrial improvement of the iron silicate product, commonly only named slag. With the copper content in the by-product slag being able to be reduced by 30-50%, depending on the starting raw materials, the raw material copper is better exploited. Furthermore, additional by-products, which are unwanted for reuse of the iron silicate product, are to some extent even more significantly reduced than copper.

Furthermore, over the last decades the SAF has taken an enormous position in non-ferrous metallurgy research. Numerous dissertations have been published and underline the versatility on this type of furnaces in many different industrial and scientific applications. Some examples of recent scientific results have been demonstrated. The applications for SAFs range from the reuse of previously landfilled Redmud, Cobalt-recovery operations and the reduction and removal of Pb and Zn from slags to future applications for example in the pyrometallurgical treatment of deep-sea minerals such as polymetallic Mn-nodules.

The close cooperation of University researchers and experienced industry engineers is fundamental to the advancement and optimisation of industrial processes. The cooperation between SMS and IME shows that the entire size spectrum from lab- and technical-scale up to pilot- and industry-size research may be realised and leads to



important process improvements. Generally, it was shown that the application of the submerged arc furnace technology to the valorisation of slags enables the production of low-level heavy metal contamination of the slags while simultaneously allowing the recovery of valuable metals from the slags.

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