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New Technological Solutions to Reduce the Copper Content in the Slags of the Oxygen-Flare Smelting of Sulfide Copper Concentrates

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Abstract: *Uzbekistan is one of the countries with a highly developed metallurgical industry. The development of this branch of the industry was facilitated by rich mineral resources, modern enterprises for their processing and highly qualified engineering and technical personnel. The severance of economic ties caused by the collapse of the former union, the global financial and economic crisis, undoubtedly, posed big problems for the metallurgical industry of the republic. The collapse of the union caused the need to localize the production and release of metal based on local raw materials. The global crisis has caused a decrease in demand for our products and, associated with this, a decrease in their cost in the world market. But, in spite of these difficulties, the main metallurgical enterprises of the republic are operating normally and, practically, are at the level of the pre-crisis state. What are the main directions of development of metallurgy in Uzbekistan in the post-crisis period? Here is an analysis of several metals that form the basis of the republic's metallurgy.*

Keywords— *metallurgy, slag, copper, recycling, decoppering process, factors, Oxygen-Flare smelting.*

1. INTRODUCTION

Copper is one of the suppliers of foreign currency to the republic. At present, copper is produced only by AMMC in the amount of about 100 thousand tons per year. Melting units are designed for this capacity, although the capacity of the electrolytic department is twice as large [1-5]. In the post-crisis period, metal production may be slightly increased. The construction and commissioning of a new liquid bath smelting process will improve the process.

Several hundreds of copper occurrences are known on the territory of the republic. To date, more than 20 deposits and promising ore occurrences of copper have been identified in Uzbekistan, four deposits have been explored (Kalmakir, Sarycheku, Dalnee, Kyzata). The Kalmakir and Sarycheku copper deposits are being developed by AMMC, Dalnee and Kyzata are reserved. The rest of the deposits are not involved in industrial processing [6-10].

Non-traditional resources can also be used as a raw material base:

1) dumps of mining production;
2) tailings dumps;
3) wastes from metallurgical production.

The amount of this waste is estimated in hundreds of millions of tons and can serve as an additional source of obtaining copper, gold, silver, selenium, tellurium, rhenium, indium and other metals [7-8].

2. THE USE OF ENERGY-SAVING TECHNOLOGIES IN THE PRODUCTION OF COPPER

At the Almalyk copper-smelting plant, an oxygen-flare smelting complex (OFS) for the processing of sulfide copper concentrates has been built and is being successfully operated. The creation of the industrial complex of OFS was preceded by a large amount of research work, which is currently ongoing for the further improvement of equipment and technology [9-12].

When the blast is enriched with oxygen, the process temperature usually rises. The oxidation process in this case is more intense. For many sulfide oxidation reactions, the activation energy is 83.6 - 167.6 MJ / mol. As the temperature rises from 500 to 1000 K, the reaction rate increases 20,000 times [13]. Oxidation of sulfides can be expressed by the overall reaction:

$$MeS + 1{,}5O_2 = MeO + SO_2$$

Flare processes are becoming more widespread in nonferrous metallurgy in many countries. The essence of flare processes lies in the fact that the processed concentrate as part of the charge is introduced into the combustion chamber of the furnace by the formed blast stream: this leads to the formation of a dust-gas torch, in which the concentrate is oxidized, the charge melts and slag-forming oxides are partially fluxed [14-17].

In flare melting, the surface of particles interacts with gases by one or two orders of magnitude more than in reflective melting. This is the decisive factor in the efficiency of flare processes in comparison with traditional processes with a compact bed of charge [18].

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With OFS, the same reactions occur as in traditional metallurgical processes of roasting, smelting and converting. A distinctive feature of the oxygen process is that it does not require external heat supply, therefore the process is autogenous, without fuel consumption. This is especially important, since the reserves of hydrocarbon fuels are decreasing everywhere in the world and their cost is constantly increasing [19-23].

The oxygen-flare process is characterized by operation on relatively rich mattes (up to 45% and more), as well as a high oxidation potential of the medium. Under these conditions, the slags inevitably contain a fairly high copper content, reaching 0.70-0.90% and more. This grade is almost twice the copper grade in the original ore and must be disposed of [24-27].

The issue of slag processing and reduction of the copper content in them is very difficult and has not yet found its satisfactory solution [28].

The process of de-curing the slag inside the OFS by blowing in pyrite did not justify itself and was canceled at one time. The practice of depletion of OFS slags by the flotation method can hardly be considered economically justified. This is due to the difficulties in grinding very hard slags, as well as the relatively low rates of metal recovery. This makes it necessary to search for alternative ways to reduce the copper content in the OFS dump slags [29-33].

3. IMPROVEMENT OF THE SMELTING PROCESS IN AN OXYGEN-FLARE FURNACE

By At the Department of Metallurgy of the Tashkent State Technical University, together with the employees of the AMMC, an analysis of technological solutions to reduce the copper content in OFS slags on the existing equipment was carried out with minor changes or without deviations from the technological regulations as a whole [34].

As a starting point for our research, we chose the established fact of uneven copper concentration along the height of the slag bath. The OFS furnace practically operates in a stationary mode, when the concentration of copper in different layers of the slag does not change over time. We took samples from different levels of the slag bath. Since the layer-by-layer analysis of the copper content in the slag bath is technically difficult, we took samples from the slag jet during tapping. Moreover, we took 3 samples from each slag outlet: at the beginning, middle and end of the outlet [35-38]. To a certain extent, these samples simulated the copper content along the height of the tapped part of the slag: at the lowest point (the richest in copper), in the middle and in its upper part (the poorest in copper). In total, 12 samples were taken (4 times 3 samples) at different releases. The results of chemical analysis of samples are shown in Table 1.

The data in Table 1 clearly indicate a decrease in copper content as the height of the slag bath increases.

According to our observations, the slag tapping before filling one bowl was carried out in 25-30 minutes. The total height of the slag bath is 60-70 cm.

Table 1. Results of analysis of OFS slag for copper content

Slag outlet number	Copper content when tapping slag, %		
	at the beginning of release	in the middle of release	at the end of release
1	0,75	0,7	0,6
2	1,61	1,37	1,14
3	1,54	1,46	0,97
4	0,74	0,36	0,34

Let us determine the dynamics of matte and slag production. The calculation is based on the throughput capacity of 1500 tons of charge per day. According to our observations, with such a furnace productivity, an average of about 590 tons of matte and 685 tons of slag is formed per day. According to factory practice, the capacity of the matte ladle is 20 tons, and the slag bowls are 28-30 tons. The number of matte releases per day is 590: 20 ≈ 30 times or every 28:30 = 0.8 hours or 48 minutes. Slag tapping number 685: 28 (30) ≈ 24 times or every 24: 24 = 1 hour (or 60 minutes). In the course of our observations, depending on the composition of the charge, its consumption, density and temperature of the slag, the number of slag outlets varied within 24-33 times a day [39-40].

The dynamics of matte production depends on the operating mode of the converters, which determines significant fluctuations in the level of the matte bath [41].

According to our calculations and observations, 27.5 tons of slag or 27.5: 3.5: 120 = 0.0655 m or 6.65 cm at the level of the bath are released at one time. When calculating the density of the slag is taken as 3.5 t / m^3, and the area of the hearth of the furnace is 120 m^2.

To reduce the copper content in the slag, we propose to raise the level of the slag window to the height of one brick, i.e. 6.5 cm. In this case, the starting point of slag discharge will be the end point of the existing option [42-43].

Since the height of the molten bath does not fall below 90 cm, a change in the height of the slag windows will not affect the dynamics of slag production.

Such a small change in the design of the furnace will lead to the fact that the slag will be tapped not from the copper-rich part adjacent to the matte, but from its poorest in specific conditions, its height. Moreover, during the release, the copper content in the slag will further decrease. So, if the technological process is carried out under the conditions of experiment No. 4, then at the initial moment the copper content will be 0.34%, with its further decrease to 0.20 - 0.30%. At the same time, no additional technology for its processing is required. Even when operating in other technological modes (experiments 1-3), in any case, it is possible to achieve a reduction in the copper content in the final slag by 20 - 35% [44].

Another advantage of changing the height of the change in the height of the slag window is a decrease in the hydrostatic pressure of the column of molten slag, determined by the formula:

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$$P = \rho g h$$

Slag pressure in the existing design:

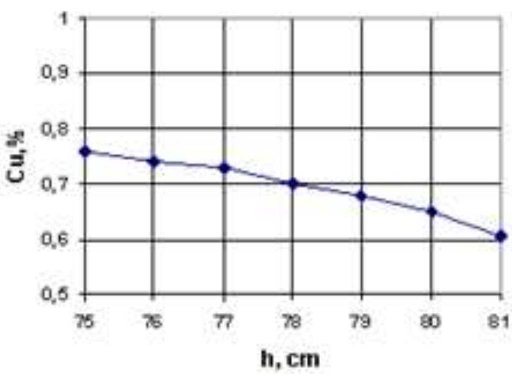
$$P = 3{,}5 \cdot 9{,}8 \cdot (1{,}20 - 0{,}75) = 15{,}435 \text{ n/m}^2$$

Here, the total height of the melt is taken equal to 1.2 m, and the slag column - 0.75 m.

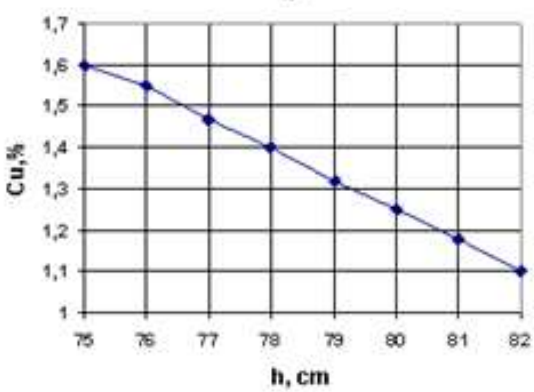
The slag pressure in the case of raising the height of the slag window will be:

$$P = 3{,}5 \cdot 9{,}8 \cdot (1{,}20 - 0{,}815) = 13{,}206 \text{ n/m}^2.$$

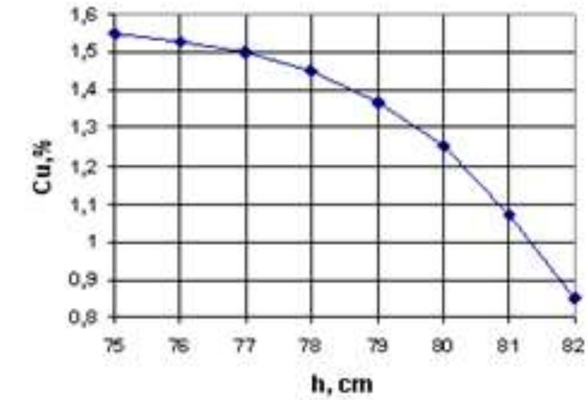
A decrease in hydrostatic pressure will lead to a weakening of the jet energy, and this, in turn, will reduce the amount of mechanical entrainment of matte drops with slag. According to the calculation, the height of the bath when draining one bowl decreases by 6.6 cm. The change in the copper content along the height of the slag bath is shown in Fig. 1. Figure 1 shows that in all experiments there is a decrease in the copper content along the height of the slag column [45].



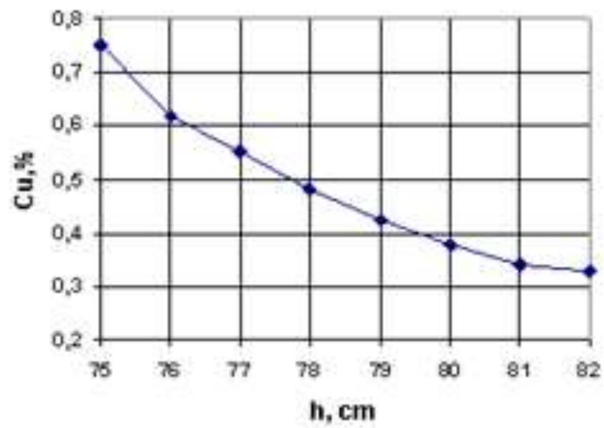
a)



b)



c)



d)

Fig. 1. Reduction of copper content along the height of the slag column: a) experiment 1; b) experiment 2; c) experiment 3; d) experiment 4.

We have also studied some more possibilities of reducing copper losses with waste slag. At the same time, we took the very essence of oxygen-flame smelting as a basis. So, matte and slag formation during OFS occurs in a torch and an oxide-sulfide emulsion enters the surface of the bath. The deposition of melt particles occurs over the entire length of the bath due to the considerable aerodynamic length of the torch. In this case, the process of complete settling and phase separation is difficult [46].

As a result, we studied the issue of the possibility of a partial change in the melting schedule in order to create conditions for a more complete settling of the melt and separation of phases by density. For this purpose, we suggest stopping one burner for 3-5 minutes before each slag discharge.

To substantiate our choice, consider several more possible options for settling the melt in the furnace:

a) complete shutdown of the burners for the specified time;

b) stopping the supply of the charge without switching off oxygen for the specified time;

c) supply of pyrite instead of a charge for a specified time.

In case (a), this option is possible, however, the furnace will lose some amount of heat, which will lead to the cooling of the slag and an increase in its viscosity. At the same time, the concentration of SO_2 in gases will sharply drop, which will affect the operation of the SKTs. Therefore, this option is unlikely to be effective.

In case (b), the slag will undergo peroxidation, which will lead to a change in its physicochemical properties, the formation of an excess amount of magnetite and heterogenization of the melt. For these reasons, this option is unlikely to be acceptable.

In case (c), the temperature of the melt and the concentration of SO_2 in the gases will be maintained during the combustion of pyrite. However, when iron is oxidized, the chemical composition and physical properties of the slag will drastically change, which ultimately may affect the violation of the technological regulations of the OFS.

Of all the options considered, in our opinion, the most acceptable may be the shutdown of one burner. In this case, the furnace charge performance and the volume of sulfuric acid produced will decrease slightly.

The capacity of one burner is 750 t / day. With 24 burner shutdowns for 5 minutes, the burner will be idle for one day

$24 \cdot 5 = 120$ minutes (or 2 hours). During this time, the charge will not be melted.

$750 \cdot 2: 24 = 62.5$ t

The real productivity of the furnace will be equal to

$1500 - 62.5 = 1437.5$ t / day

When processing such a quantity of charge per day, matte will be obtained:

$1437.5 \cdot 39.33: 100 = 565.45$ t

Slag will be obtained:

$1437.5 \cdot 45.69: 100 = 656.88$ t

With an average content of 0.34% Cu in the slag, the losses will be:

$656.88 \cdot 0.34: 100 = 2.233$ tons of copper.

The amount of copper in the charge:

$1437.5 \cdot 16.22: 100 = 233.16$ t

will go into matte:

$233.16 - 2.23 = 230.93$ t.

When 1,500 tons of charge is melted per day, the following will be lost:

$236.01 - 230.93 = 5.08$ tons of copper,

what is:

$5.08: 236.01 \cdot 100\% = 2.15\%$.

In the basic version, the copper content in the slag is 0.71%. When 1,500 tons of charge is melted per day, the following is lost with slags:

$685 \cdot 0.71: 100 = 4.86$ tons of copper.

According to the proposed technology, copper losses will amount to 2.233 tons, i.e. will be half as much.

4. Conclusion

Thus, with the introduction of the new technology, the real losses of copper with slag will decrease, and an additional 2.63 tons will be obtained. This will more than half compensate for copper losses from a decrease in productivity from a temporary shutdown of one burner.

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