

New Technology for Decreasing Copper Content in Dump Slags: A Review

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Abstract: *The metallurgy of copper and other heavy non-ferrous metals is the leading link in the domestic non-ferrous metallurgy. The share of heavy non-ferrous metals in Uzbekistan accounts for a significant part of the gross output of the industry. The value of copper is increasing from year to year, especially in connection with the rapid development of energy, electronics, mechanical engineering, aviation, space and nuclear technology. This paper will consider the technology for reducing the loss of copper with slags during smelting processes. Numerous parameters affect the copper content in slags, of which the following should be considered the most important: preparation of the charge, parameters of the technological mode of smelting, physicochemical properties of melts, design of units, organization of work, etc. It is very difficult to give an unambiguous assessment of the influence of all parameters acting separately or in aggregate, and is hardly possible at all. In this regard, we will try to assess individual process parameters from the point of view of their influence on the depletion indicators and the possibility of changing their values in real metallurgical smelting.*

Keywords— *metallurgy, slag, copper, recycling, decoppering, depletion process, factors, reducing-sulfiding complex.*

1. INTRODUCTION

The development of copper metallurgy in recent years is characterized by an increase in the complexity of the use of raw materials, the increasing use of oxygen in pyrometallurgical and hydrometallurgical processes, the creation of mechanized and automated continuous production [1].

The copper industry in developed countries continues to develop on the basis of raw materials from third underdeveloped countries. The USA, Japan, Germany, France, Belgium annually import hundreds of thousands of tons of concentrates from Chile, Zambia, Zaire, etc [3].

Production capacities in developed and developing countries are loaded on average by 93% (in the USA by 85-88%, in Canada, Chile, Peru, Zaire, Zambia - by 97%) [4].

The main amount of copper abroad and in the CIS, including Uzbekistan, is obtained according to the standard pyrometallurgical scheme: smelting - converting - refining; the share of the hydrometallurgical method accounts for 12-16% [5].

In recent years, in a number of countries abroad and the CIS, attention has increased to hydrometallurgical methods for extracting copper from lost and off-balance raw materials (heap, underground leaching), rich oxidized ores (heap, percolation and agitation leaching) [6]. Mixed ores are processed according to the scheme of leaching - cementation - flotation followed by pyrometallurgical processing of the resulting concentrates. Research is being carried out for hydrometallurgical processing of sulfide copper-containing materials using the autoclave method, salt leaching, sulfatization, but most of the methods have not gone beyond

semi-industrial and industrial tests. The autoclave method is successfully used to obtain copper powder from ammonia solutions from leaching of secondary materials [7].

Sulfide copper-nickel ores are subjected to flotation concentration to obtain high-quality concentrates of the same name [8].

The development of the copper industry in recent years has been characterized by the modernization and expansion of existing enterprises based on new technology [9]. There has been a tendency towards the organization of large associations and plants that include a full production cycle - from the extraction of ore to the production of finished metal and its further processing (production of copper wire rod, rolled metal, etc.) [10].

Significant successes have been achieved in increasing the complexity of the use of raw materials by expanding the range of products, organizing dust collection, more complete use of sulfur-containing gases, as well as the use of secondary energy resources. The achieved achievements are largely associated with the widespread introduction of oxygen in the autogenous processes of the copper industry [11]. So, in recent years, the production of copper using oxygen has increased significantly.

A distinctive feature of copper smelters in the CIS countries and Uzbekistan (in comparison with foreign ones) is the use of a larger volume of poor (15-34%) complex concentrates [12]. Up to 50% of copper is smelted from mattes obtained in reverberatory and electric furnaces, 50% in furnaces of autogenous processes. Shaft furnaces are used only in old factories for processing rich ores (semi-pyrite) and sulphurous ores (improved pyrite), as well as for substandard secondary raw materials [13].

In the field of converting mattes and refining copper, the performance indicators of factories in the CIS and Uzbekistan are at the level of leading foreign firms. The production and continuous casting of oxygen-free copper, the production of copper wire rod, etc. have been mastered.

Currently, in the production of copper, more than 12 components are extracted from raw materials and more than 20 products are manufactured [14].

At the same time, it should be noted that the introduction of new technological processes and schemes is carried out relatively slowly [15]. The reason for this is: 1. Significant lag of the level of hardware design of circuits from the level of technological developments; 2. Unfavorable economics of these extremely necessary processes due to the still high cost of equipment, new solvents, sorbents, extractants due to the lack of industrial production of these reagents; 3. The issue of processing semi-products of autogenous processes has not been resolved [16]. For example, slags from autogenous processes of the copper industry contain 0.8-1.2% copper, with a standard for dump slags of 0.35%. It should be noted that practically all slags of autogenous smelting are stored, awaiting the development of an economically acceptable technology for their processing. Work in progress is growing, working capital is frozen, increasing the cost of production [17].

2. IMPROVEMENT OF THE TECHNOLOGY OF REVERBERATORY SMELTING OF SULPHIDE COPPER CONCENTRATES

The global economic crisis, which began in the second half of 2008 and continues at the present time, made it necessary to improve technological processes aimed at increasing the complexity of raw materials, involving man-made waste and intermediate products in processing, reducing the cost of final products, switching to low-waste technology, etc [18].

At the Almalyk Mining and Metallurgical Combine, two smelting complexes are successfully operating: reverberatory redistribution and oxygen-flare smelting (OFS). The OFS complex makes great demands on the quality of charge materials. Reducing the content of copper in the original ore, obtaining more and more concentrates pose great problems in the operation of OFS, and especially in ensuring its main advantage – autogenicity [19].

Reverberatory smelting is the most common method for processing copper flotation concentrates. In global copper production, more than half of the primary blister copper is smelted using reverberatory smelting. This process has great advantages: the ability to process a wide range of raw materials, high copper recovery, low dust entrainment, good slag settling conditions, simple design of the unit and easy temperature control in the furnace. In this process, converter slags are traditionally depleted [20].

But for reverberatory smelting, serious disadvantages are also characteristic: relatively low specific productivity, high fuel consumption, high copper content in waste slag, large output of exhaust gases, which does not allow the utilization

of sulfur, rare and scattered elements. These disadvantages of reverberatory smelting determined the development in a number of countries of processes for suspended smelting of copper concentrates on hot air and oxygen blast, etc [21].

However, the introduction of new methods of processing copper concentrates does not mean that in the near future, reverberatory smelting will be everywhere replaced by these processes, since this is associated with the difficulties of reconstruction of existing enterprises and significant foreign exchange capital costs [22]. In addition, in this case, each enterprise faces a set of questions regarding the processing of converter slags and various other copper-containing materials, which are currently being processed in reverberatory furnaces. In this regard, it can be assumed that in the near future the processing of concentrates by reverberatory smelting will continue, and its share in the production of primary copper, blister copper will remain quite high [23].

At present, there is a rather intensive process of improving reverberatory melting. These works are aimed at reducing the impact of its shortcomings on the technical, economic and technological indicators of reverberatory melting [24].

One of the big problems of full-cycle copper smelters is the processing of converter slag. Converter slags are recycled material, contain 2.5-3.5% copper, a significant amount of precious metals and must be recycled [25].

World experience shows that converter slag is mainly processed in reverberatory furnaces. Moreover, as much converter slag is loaded into the reverberatory furnace, as much as it can process. At AMMC, it is practiced to process the part of the converter slag not loaded into the furnace by the method of flotation [26]. However, this processing method is unlikely to be promising, since there are big problems with grinding very hard slag and low copper recovery into concentrate. In addition, there are quite big problems with the extraction of copper into matte when smelting concentrate obtained during the enrichment of slags [27].

Based on this, it can be concluded that in the near future the trend of processing converter slag in a reverberatory furnace will continue [28].

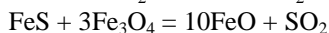
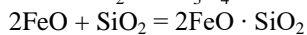
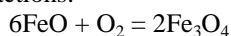
However, this process has a negative impact on the operation of the reverberant oven. At AMMC, large-scale industrial studies were carried out to determine the effect of converter slag on the operation of a reverberatory furnace [29]. Comparative melts were carried out in a reverberatory furnace without pouring converter slag and with pouring. The results of very reliable tests showed that when operating without pouring liquid converter slag, the specific melt of the furnace increases by 2.33%, the copper content in the matte increased from 18.3 to 21.1%, the output of the waste slag and the copper content in it significantly decreases, as a result of which the absolute loss of copper with slag decreased by 7.5 times, and extraction into matte increased by 2.3%. The recovery of copper from the liquid converter slag with a 2.3% content was 75.5% in reverberatory

melting. These data convincingly show the negative effect of the liquid converter slag on the technical and economic indicators of reverberatory smelting [30].

An analysis of the results of industrial tests showed that one of the main reasons for the deterioration of the technical, economic and technological indicators of reverberatory smelting when pouring converter slag is the presence of magnetite in high concentrations in it [31].

During the industrial tests, converter slags contained 22-24% SiO₂ and 23-30% Fe₃O₄.

The formation of magnetite in the process of converting is considered in sufficient detail in the works of R. Ruddle [32] and H.K. Avetisyan [33]. These works indicate that the production and reduction of magnetite in the converter depends mainly on the possible rate of the following reactions:



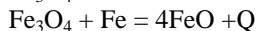
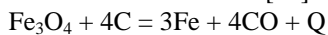
Magnetite, being formed in large quantities, degrades the process technology and causes build-up in reverberatory furnaces. N.M. Dergachev and N.I. Utkin [34] point to the fact that a high content of magnetite in converter slags leads to the formation of an intermediate layer in reverberatory furnaces and high losses of copper with slags.

In the countries of the far and near abroad, attempts were made to restore magnetite in the bath of reverberatory furnaces with various reducing agents, but no positive results were obtained [35].

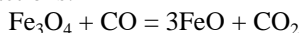
In AMMC, the degree of reduction of magnetite of liquid converter slag is 11.53%, which indicates a low development of the process of reduction of magnetite of converter slag in the furnace bath [36].

At the Department of Metallurgy, Tashkent State Technical University, a technology has been developed for loading pre-reduced converter slag into a reverberatory furnace [37].

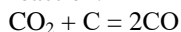
The preliminary reduction is carried out in ladles, in which the liquid converter slag is transported to the reverberatory furnace. For this purpose, zinc production clinker is loaded into the ladle before pouring the liquid converter slag in an amount of 5-10% by weight of the slag. The converter slag jet creates almost ideal conditions for mixing the converter slag and clinker. As a result of this, reactions occur in the ladle [38]:



The evolved CO also takes part in the reduction reactions:



The evolved CO₂, in turn, reacts with CO according to the reaction:



Metallic iron and carbon are found in sufficient quantities in clinker [39].

All reactions (except for the carbon gasification reaction) are exothermic and serve as additional heating of the melt in the ladle [40].

In a series of laboratory experiments, we have achieved the degree of magnetite reduction from 40 to 85%. This indicator is influenced by the composition of the slag, the consumption of the reducing agent, the transportation time, etc [41].

The pre-reduced converter slag has very little negative impact on the reverberatory furnace operation. In this case, the amount of converter slag fed into the reverberatory furnace can be significantly increased up to a complete conversion. In this case, there is no need to melt them in the fourth converter, as well as flotation concentration of converter slag [42].

3. NEW TECHNOLOGY FOR REDUCING COPPER CONTENT IN WASTE SLAG

By processing copper concentrates to obtain blister copper, in our republic and abroad, about 85% of copper is obtained by the pyrometallurgical method, and 10-15% by the hydrometallurgical method [43]. The classic pyrometallurgical scheme for the production of blister copper at most plants in the world, including the Almalyk Mining and Metallurgical Combine, includes melting the charge for matte in a reverberatory electric furnace or in furnaces of an autogenous process, oxygen-flare smelting, in Vanyukov furnaces, by means of suspended smelting, flare-bubbling melting, as well as converting mattes. With such a technological scheme, the products of production are blister copper, matte, reverberatory and electric smelting slags, autogenous smelting slags, converter slags [44].

At present, more than 13 million tons of waste slag from the copper smelter have accumulated on the dumps of Almalyk MMC as a result of many years of processing copper ores. Annually, about 400 thousand tons of waste slag with iron content up to 35-40%, copper up to 0.7%, gold 0.2-0.4 g / t are stored here, which can be processed. Dozens of hectares of land, including fertile, are occupied by dumps. Large funds are spent annually on the maintenance of dump farms. A very large volume of slags with valuable components formed during the processing of copper ores determines the urgency of the problem of their rational use [45].

It should be noted that due to the continuous decrease in the base metal in the ore, a concentrate with a low copper content is obtained, respectively, and the matte is poor, increasing the amount of converter and dump slags per unit of output. Ultimately, there is an increase in irrecoverable losses with waste slag. At the same time, their rapid accumulation occurs, and even as a recycled product, converter slag is processed only partially from the volume obtained, although the copper content in it reaches an average of 3%. The remaining amount is stored in special storage facilities.

For several decades, the attention of scientists and specialists has been directed to finding methods for

processing slags from copper smelting production. Hundreds of technologies and recommendations have been developed, which are usually local in nature and applicable to solving specific problems. This is due to the variety of composition, properties and mechanism of slag formation, which requires individual technology for its processing. There is currently no unified technology applicable to solving all the problems of processing copper-containing slags.

All known methods of processing slags from copper smelting production are classified in the following areas: hydrometallurgical, flotation, pyrometallurgical and combined. Each direction has its own advantages and disadvantages, which depends on the composition of the feedstock and fluxes, as well as on the specific conditions of the enterprise [46].

The hydrometallurgy of slag processing is based on the transfer of copper into a solution, followed by its isolation by one of the known methods. Dissolution is carried out by treating the slag after crushing and grinding with a solvent with the possibility of chemical treatment for additional conversion of copper into soluble forms, or without it. Conversion of copper into soluble forms can be carried out by sulfatization, chlorination, biotechnological and other methods.

The advantage of slag hydrometallurgy is the ability to achieve high copper recovery into a highly concentrated sludge in the absence of process gases, thereby eliminating air pollution. However, this method also has significant drawbacks: a low leaching rate, the formation of a large volume of solutions per unit of production, the difficulty of extracting precious metals and removing sulfur in elemental form.

In this regard, accordingly, it is necessary to suspend the accumulation of slags, receiving dump slags with a standard copper content in them.

This task can be achieved by reducing the excess content of ferric iron (magnetite) of the converter slag in the process of converting copper mattes. The high content of ferric iron in iron silicate slags increases the copper content in them in both dissolved and mechanical form. Based on the studies carried out on a comparative assessment of the effectiveness of various materials for restoration, we have selected a relatively cheap reducer produced in our Republic of Uzbekistan. This reducing agent is a composition based on clinker - a zinc production by-product obtained from the Waelz zinc cakes, which has also accumulated hundreds of thousands of tons in dumps and continues to accumulate. The composition based on waste from non-ferrous metallurgy - clinker from Waelz kilns from various zinc plants has a very variable composition. The concentration of the main components fluctuates within the following limits, wt. %: C - 15-28; Fe_{met} - 10-25; Cu - 0.5-2.5; Au - 1.0-10.0 g / t; Ag - 100.0-700.0 g / t; Pb - 1.0-2.5; Zn 1.0-2.0; SiO₂ - 10.0-25.0; S - 1.0-3.5. As can be seen from the above composition, in addition to non-ferrous metals, clinker contains more than 40% of reducing agents in the form of

unburned carbon and metallic iron, the bulk of which is in the composition of cast iron .

Reduction of excess magnetite of converter slag with a solid reductant (zinc production clinker) to a residual content of 2-7%, followed by pouring into reverberatory and electric furnaces should reduce the copper content in waste slag to the standard, since during operation of reverberatory furnaces without pouring converter slag, waste slag contain copper 0.1-0.35%, magnetite 1-5%. It follows that when pre-reduced converter slags with a residual magnetite content of 2-7% are poured into a reverberant furnace, part of the latter is reduced by iron sulfides of the charge to 1-5% during reverberatory melting, and the slags will contain 0.15-0.4% copper. Based on the above, laboratory studies of smelting sulphide concentrates, fluxes and pre-reduced converter slags were carried out under conditions close to reverberatory smelting, and slags with a copper content of 0.19 -0.4% were obtained. Such slags can be transferred to the construction industry, and not accumulated in dumps, thereby improving the ecological situation of the slag dumps of the AMMC.

4. THE EFFECT OF TEMPERATURE AND OXYGEN ON THE COPPER LOSS WITH SLAG

Losses of copper with slags in the AMMC are as follows. In slags with a degree of acidity of 0.95 - 1.3 (29-35% - SiO₂) copper losses are greatest: total 0.47-0.54%; mechanical 0.22-0.28%, dissolved 0.21-0.29%. General (0.28-0.44%) and mechanical losses (0.23-0.36%) are also quite high in slags with an acidity of 1.9-2.5 (46-52% SiO₂). The smallest total (0.15-0.38%) and mechanical losses (0.05-0.15%) are observed in slags with a degree of acidity from 1.3 to 1.9 (36-45% SiO₂). The ratio of mechanical and dissolved copper losses for the first slags is about 1, for the second - about 3, and for the third - about 0.7.

From the graphs it follows that the total minimum losses are observed when the content of the slags is 42-45% SiO₂, the dissolved losses decrease as the content of this component increases. This effect of silica on losses is associated with a significant increase in the interfacial tension with an increase in the SiO₂ content. However, when the content of silica is over 45-46% (at a constant temperature of the slag), the viscosity increases sharply, the effect of which on the deposition of particles becomes greater than the effect of interfacial tension.

An increase in the content of calcium oxide to 12-13% reduces all forms of copper loss. This component has a greater effect on the reduction of dissolved losses, less on a decrease in mechanical losses. This is due to the fact that interfacial tension and viscosity decrease with increasing calcium oxide content.

Iron oxide, with an increase in its content from 26 to 52%, increases all forms of copper loss, which is explained by an increase in the solubility of copper in ferrous slags.

The data presented show that the minimum copper losses are achieved when slags of the following composition are obtained:

42-45 % SiO₂, 12-13 % CaO, 22-25 % FeO

It is to work with such a slag composition that the creation of compositions of reducing-sulfiding complexes will be directed.

To regulate the composition of the slag, limestone, tailings of the lead and copper concentrating plants of AMMC will be used. In these materials, along with the components regulating the composition of the slag, there is a noticeable amount of non-ferrous metals, the additional extraction of which will have a favorable effect on the technical and economic indications of the entire depletion process.

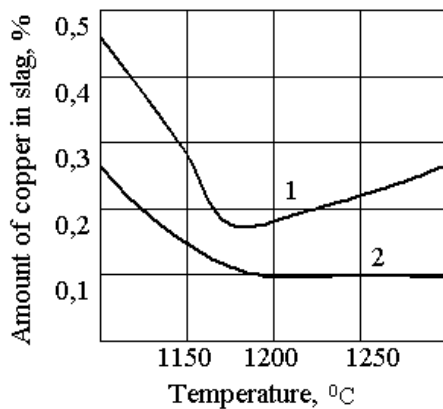


Fig.1. Influence of temperature on the loss of copper with slags

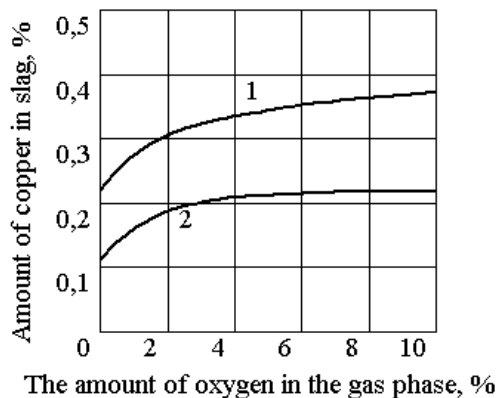


Fig.2. Influence of oxygen in the gas phase on the loss of copper with slags

The effect of temperature on copper loss depends on the composition of the slag (Figure 1). An increase in temperature determines a decrease in the content of copper in the acidic slag. As the temperature rises, the copper content in the main slag first decreases due to a decrease in mechanical losses, and then increases due to an increase in the solubility of copper. The effect of oxygen in the gas phase on the loss of copper with slags is shown in Fig.2. An increase in the oxygen concentration in the gas phase leads

to an increase in the copper content in the slags. This can be explained by the formation of magnetite in the process of decopperization. Magnetite increases the oxidation potential of the slag, which leads to an increase in the solubility of copper in it.

Thus, for basic slags, there are optimal temperature conditions at which copper losses are minimal. For example, for the main slags of the AMMC, the optimal temperatures are 1180-1200 ° C. The effect of magnetite on the loss of copper with slags is shown in Figure 3.

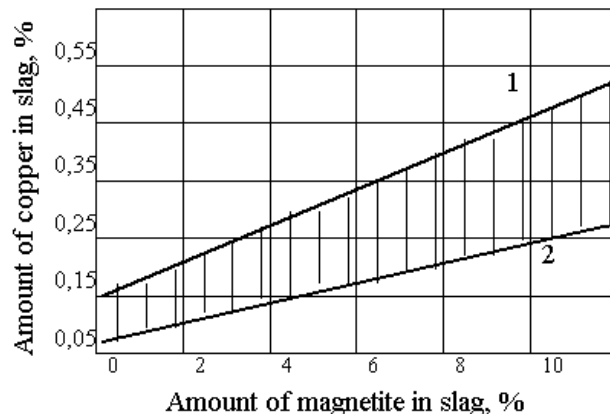


Fig.3. The effect of magnetite on the loss of copper with slags: 1-total losses; II-dissolved:
 1 - 31 % SiO₂; 45,6 % FeO; 2,6 % CaO; 7,7 % Al₂O₃
 2 - 41,4 % SiO₂; 23,5 % FeO; 12,3 % CaO; 10,2 % Al₂O₃

An increase in magnetite content determines an increase in mechanical and dissolved copper losses with slag. So, with an increase in its concentration from 0 to 11%, mechanical losses increase from 0.05 to 0.23%, dissolved from 0.13 to 0.23%. For each percent of magnetite, the first losses increase by 0.016%, the second by 0.014%. The increase in mechanical losses of copper is associated with a deterioration in the conditions for coarsening the size of suspended particles, and an increase in dissolved losses is associated with an increase in the oxidizing potential of the slag with an increase in the concentration of magnetite.

The theoretical and practical provisions considered in the previous sections allow a reasonable approach to the choice of technological modes of smelting in order to minimize the loss of copper with slag.

The choice of the optimal slag composition is one of the most important reserves for increasing metal recovery. The slag must be sufficiently low-melting, liquid-flowing and have a low density. The solubility of non-ferrous metals in the slag should be minimal, and the interfacial tension at the matte-slag interface should be as high as possible. Moreover, the output of slag should be minimized.

Slags with a high concentration of iron oxides dissolve more non-ferrous metals. The interfacial tension on the matte border with such melts is low. All this leads to increased losses of copper, which are in such a melt, both in

a dissolved state and in the form of a finely dispersed suspension.

The effect of iron oxides is much less harmful under reducing conditions, when the Fe^{3+} content is reduced to a minimum. In order to reduce the concentration of FeO, fluxes are introduced into the melt: quartz and limestone. An increase in the concentration of SiO_2 in the melt significantly reduces the solubility of non-ferrous metals in the slag and increases the interfacial tension of the system.

A prerequisite for depletion of the slag is its reduction in order to reduce Fe^{3+} in it and, at the same time, dissolved metals.

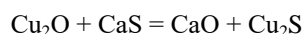
It is most rational to recover slags by blowing them with a reducing agent (gaseous, liquid or coal dust). Obviously, it is most convenient to use natural gas for recovery. However, the utilization rate of unconverted gas is very low. The reduction can be carried out in a fixed bed, for which a solid reductant is loaded onto the surface of the melt. Probably in this case, due to the low rate of mass transfer processes, the recovery will proceed slowly. Consequently, preference should be given to processes combining reduction with stirring of the melt.

Due to the oxygen dissolved in the melt, as well as the high oxidation potential of the gas phase, up to 10% of the copper in the slag is in the oxidized state. If you do not take special measures to sulfidize them, they will be almost completely lost with the slag. It is proposed to use zinc tailings from the AMMC lead concentration plant and phosphogypsum from the Almalyk chemical plant as sulfidizers.

The selection of the optimal composition of the slag, its reduction and sulfiding, although necessary, but insufficient operations for deep depletion. The fact is that the metal and sulfide particles formed during these operations are very small in size and for their sufficiently complete sedimentation, long exposures (10-15 hours) are required, which is not technologically and economically feasible.

The process of separating fine suspended particles from slag can be accelerated in various ways. In particular, bubbling of the bath with gaseous products of reduction reactions, mixing of slag with matte can be used. Both of these possibilities, to one degree or another, take place during the treatment of copper slags with reductive sulfiding complexes.

The technology for depletion of copper slag can be successfully implemented if it does not pollute the air basin with sulfur-containing gases outside the existing standards. To prevent the release of sulfur dioxide into the environment, limestone is introduced into the charge, which binds sulfur into a very strong compound CaS - calcium sulfide. Calcium sulfide, in turn, participates in the sulfidation of oxidized copper compounds according to the reaction:



Thus, at the same time, two problems are solved: sulfiding oxidized copper compounds and preventing the emission of harmful gases into the air basin, which is technologically very advantageous.

When solving the complex problem of processing slags from copper production, it is very important to establish technological and economically viable boundaries for the depth of copper extraction. The ideal option is the complete disposal of copper and all associated valuable elements. However, in production this is an almost impossible task. The fact is that up to 10% of copper is embedded in the lattice of other compounds (for example, wustite) and it is impossible to extract it without completely destroying the structure of the solid. Part of the copper is in the form of very thin inclusions and, in a technologically acceptable time (1.5 - 2 hours), does not have time to settle into the bottom phase. This is about 20-40% of the total copper losses with slag. Part of the copper is in the slag in an oxidized state and does not have time to sulfidize and go into the matte phase. This part of the copper is irretrievably lost with slags. Thus, there are always objective reasons for the presence of a certain amount of copper in the slags and losses are inevitable here.

A study of the literature and analysis of the operation of operating copper smelters allows us to conclude that the economically and technologically acceptable residual copper content in slags is 0.30-0.35%. Deeper de-crimping causes great technological difficulties and is associated with significant economic costs. Experience shows that with such a residual copper content, it is possible to organize waste-free production by transferring decopperized slags to the construction industry.

5. CONCLUSION

The reduction process is so intense that during the first 3-4 minutes, in the presence of a reducing agent in the melt bath, there is a sharp decrease in ferric iron to 50-60% and a decrease in the copper content to 40% in the converter slag, which is the goal of the first stage of industrial tests.

But when the reducing agent is assimilated in a bath furnace, ferrous iron is again intensively oxidized to ferric iron in a very short period of time. In the first stage of industrial testing, our goal was to reduce the copper content in the converter slag as much as possible. At the second stage of industrial testing, we set the goal of minimizing the content of ferric iron in the converter slag fed to the reverberatory furnace, in order to reduce the negative effect of the latter on the reverberatory smelting process. Accordingly, this will lead to a decrease in the copper content in the waste slag. We are trying to solve this goal by industrial tests in smelting units - in converters, a reverberatory furnace under the conditions of the AMMC. When the clinker is loaded with 1-5% of the converter slag weight in the converter, the purge is 10-90 seconds, then the converter slag is discharged with partially unreacted clinker floating above the slag melt in the converter. Since the reactions of reducing ferric iron to ferrous are very intensive,

the process of reducing ferric iron will continue when the slag is drained from the converter into the ladle, also in the ladle itself during the drain period, since the slag stream will mix the reducing agent and the converter slag, thereby reducing the likelihood of oxidation converter slag.

Reducing the ferric content will accordingly reduce the copper content in the reverberatory furnace slag, thereby increasing the blister copper yield.

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