MINISTRY OF EDUCATION AND SCIENCE OF THE REPUBLIC OF KAZAKHSTAN

K.I. Satpayev Kazakh National Research Technical University

Mining and Metallurgical Institute named after O.A. Baikonurov

Department of Metallurgy and Mineral Processing

Galipauly Shyngys

MASTER'S DISSERTATION

Research and development of ways to improve the efficiency of processing of titanomagnetites

7M07204 – Metallurgy and mineral processing

Almaty 2022

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For an academic master's degree

Thesis title:	Research and development of ways to improve the efficiency of processing of titanomagnetites
Direction of training:	7M07204 – Metallurgy and mineral processing

Scientific adviser:
Doctor Ph.D.
assistant professor
_________ G. M. Koishina
«06» _____06_____ 2022

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THE TASK
for the Master's dissertation

Master 's student: *Galipauly Shyngys*

Subject: *Research and development of ways to improve the efficiency of processing titanomagnetite*

Approved by *order of the Rector of the University* No. 2026-m of 03.11.2020

Deadline for completion of work is *«8» June 2022*

Initial data for the master's thesis: *Selection of charge components and chemical compositions of selected components*

The list of questions to be developed in the master's thesis:

a) preparation of laboratory samples from multicomponent oxide materials;

b) calculation of stoichiometric consumption of solid carbon for the reduction of metal oxides;

c) preparation of special samples of ore-coal mixture required for the process of direct reduction of metals and obtaining metallized materials from them;

d) development of technology of laboratory reduction and melting of multicomponent concentrates, preparation and testing of laboratory melting units. Analysis of the obtained research results.

SCHEDULE
of preparation of the master's thesis

The name of the sections, the list of issues being developed	**Deadlines for submission to the scientific adviser**	**Notes**
Introduction	06.09.2021	
Analytical review of the literature	14.12.2021	
Research methodology	21.02.2022	
The experimental part	07.04.2022	
Conclusion	15.05.2022	

Signatures
of consultants and the norm controller for the completed master 's thesis with an indication of the relevant sections of the dissertation

Section name	Consultants (academic degree, title)	Date of signing	signature
Introduction	Doctor Ph.D. assistant professor G. M. Koishina	06.09.2021	Khoishina
Analytical review of the literature		14.12.2021	Khoishina
Research methodology		21.02.2022	Khoishina
The experimental part		07.04.2022	Khoishina
Conclusion		15.05.2022	Khoishina
Normcontroller	senior Engineer of the Department of MaMP A. N. Taimassova	06.06.2022.	

Scientific adviser Khoishina G. M. Koishina

The master accepted the task ___________Sh. Galipauly

Date «06» 09 2021

АҢДАТПА

Диссертациялық жұмыста 54 парақ, 14 сурет, 23 кесте, 65 атаудан тұратын библиография бар.

Түйін сөздер: титаномагнетит кендері, концентраттар, агломераттар, жентек, шихта, редукция.

Зерттеу жұмыстарын жүргізу үшін құрамында титан, ванадий сияқты өндірістік маңызы бар легірлеуші элементтер кездесетін титан-магнетитті және ильменит концентраттары алынды.

Диссертациялық жұмыс қиын балқитын металдардың кешенді концентраттарын редукциялап-балқыту технологиясын жетілдіруге бағытталған жұмыстардың нәтижелерін қамтиды, титан карбидтерінің пайда болуымен байланысты қауіптерді азайту және жалпы металлургияда үлкен қызығушыққа ие экологиялық-энергетикалық-экономикалық мәселелерді шешуге арналған.

АННОТАЦИЯ

Диссертационная работа содержит 54 листов, 14 рисунок, 23 таблицы, и библиографию, состоящая из 65 названия.

Ключевые слова: титаномагнетитовые руды, концентраты, агломераты, окатыш, шихта, чугун и шлак.

Для проведения исследований были получены титано-магнетитовые и ильменитовые концентраты, в составе которых содержатся легирующие элементы производственного значения, такие как титан, ванадий.

Диссертационная работа содержит результаты работ, направленных на совершенствование технологии восстановительной плавки комплексных концентратов тугоплавких металлов, призвана минимизировать риски, связанные с образованием карбидов титана, и решить эколого-энергетические и экономические проблемы, имеющие большой интерес в металлургии.

ABSTRACT

The dissertation work contains 54 sheets, 14 figures, 23 tables, and a bibliography consisting of 65 titles.

Keywords: titanomagnetite ores, concentrates, agglomerates, pellets, charge, cast iron and slag.

For research, titanium-magnetite and ilmenite concentrates were obtained, which contain alloying elements of industrial significance, such as titanium, vanadium.

The dissertation work contains the results of work aimed at improving the technology of reduction smelting of complex concentrates of refractory metals, designed to minimize the risks associated with the formation of titanium carbides and solve environmental, energy and economic problems of great interest in metallurgy.

Content

Introduction

The global trend in the development of metallurgy is such that there is a rapid depletion of iron ore reserves, so there is an urgent need to switch in the near future to the use of complex ores of complex composition. Such ores include titanomagnetites, which are widely distributed and have huge reserves in the bowels of the earth.

Melting on a high-titanium charge is one of the most difficult in the practice of blast furnace production, because along with compact masses of cast iron and slag, sedentary conglomerates of coke fines, metal, slag and carbonitrides are formed in the furnace. In this regard, the filtration of melts worsens, metal losses increase, the gas pressure drop in the lower part of the furnace increases and its productivity decreases. Under blast furnace smelting conditions at "JSC EVRAZ Nizhnetagilsky Metallurgical Combine" (JSC "EVRAZ NTMK"), the content of titanium oxide in the charge is 48-50 kg per 1 ton of cast iron, which is almost twice the maximum allowable amount of titanium in the charge at metallurgical plants in other countries, for example, Japan. In the future, the titanium content in the charge will increase.

Relevance. On the territory of Kazakhstan there are ilmenite and studied, but not processed titanomagnetite ores. On the territory of Russia (in the Ural region) a large deposit of titanomagnetite has been studied and processed. Titanium magnetite concentrate containing 56.0-62.0% Fe; 4.0-5.0% TiO_2, 0.50-0.6% V_2O_5 and 0.12-0.15% Mn are produced by the Kachkanar GOK and a large metallurgical plant of JSC EVRAZ NTMK. The most important issue is the development of new progressive technologies that ensure the direct extraction of natural alloyed steels from them as a result of the processing of complex in chemical and mineralogical composition, hematite, magnetite and complex iron ore raw materials. To do this, it is necessary to create the scientific foundations of technologies that make it possible to leave valuable elements contained in raw materials, in the right amount, in the composition of steels obtained from them.

Among the complex iron ore raw materials, the most important are titanium magnetite, ilmenite ores, which contain such precious metals as V, Ti, Mn and Cr. Titanium oxide (TiO_2), which is contained in the raw material, and is widely distributed in the iron crust. The chemical strength of titanium oxide is several times higher than that of iron oxide. Therefore, titanium oxide reduction will be very difficult. In this connection, scientific research of the process of consistent reduction of iron and alloying metals, development of technology of processing of titanium magnetite and ilmenite concentrates is topical.

Purpose of the work. The purpose of this dissertation is to study the processing of titanomagnetite ores or concentrates that have a significant impact on the recovery processes, taking into account the development of production and the trend of changes in raw material conditions.

Main tasks:

– preparation of laboratory samples from multicomponent oxide materials;

– calculation of stoichiometric consumption of solid carbon for the reduction of metal oxides;

– preparation of special samples of ore-coal mixture required for the process of direct reduction of metals and obtaining metallized materials from them;

– development of technology of laboratory reduction and melting of multicomponent concentrates, preparation and testing of laboratory melting units. Analysis of the obtained research results.

The object of study is titanomagnetite ores, concentrates.

The subject of research is the technology of processing titanium-magnetite ores.

Scientific novelty of the research:

– the regularities of the sequential reduction of iron, vanadium, manganese, chromium and titanium using solid carbon have been revealed, depending on the chemical strength and temperature of the corresponding metal oxides;

– determined the kinetic parameters of the metal reduction process obtained by sequential-phase change with solid carbon from each high oxide to metal;

– it has been established that the composition of gases formed during reduction (CO, CO_2) varies depending on the phase transformation of oxides, the composition of which is important for controlling the phase composition of the charge and molten metal;

– based on the results of scientific research, a technology has been developed for obtaining alloyed steel directly from complex titanomagnetite and (or) ilmenite concentrates that are difficult to process.

Practical significance. The proposed method makes it possible to solve a wide range of technological problems arising during the processing of titanium-magnetite raw materials, which makes it possible to eliminate problems associated with carbide formation.

1 The state of the problem of processing titanomagnetites

1.1. Features of blast furnace melting of titanomagnetites

Titanium magnetites are quite common type of iron ore raw materials, for example, their share in the industrial reserves of iron ores of the Urals is 76.8% [1].

Titanomagnetites are complex ores containing mainly the following minerals:

- ilmenite $FeTiO_3$ (36.8% Fe, 31.6% Ti; 36.6% O) ~ 30 %;
- magnetite Fe_3O_4 or $FeO \cdot Fe_2O_3$ (74.4% Fe; 27.6%O) ~ 50 %;
- hematite Fe_2O_3 (70% Fe; 30% O) ~ 15%.

Ilmenite is rarely pure and is more commonly found in association with magnetite. This kind of structure is a consequence of decay of solid solution ilmenite in magnetite during solidification. Under different crystallization conditions, the ore can be coarse-grained, allowing for the separation of titanium from iron by magnetic separation, but sometimes the germination is so thin that the process becomes impossible [6].

Titanomagnetites typically contain 4 to 18% TiO_2 and 0.1 to 0.8 % V_2O_5. The reason for the presence of vanadium in these ores is due to the fact that the primary form of vanadium in the Earth's crust is its trioxide - V_2O_3, which is part of most major eruptive rocks, concentrating in very small but certain quantities [2], $(Me^{1+,2+})_2(Me^{2+,3+})_5[(OH,F)|(Si,Al)_4O_{11}]_2$ - (0.03 - 0.06%), pyroxenah - (Me^{2+},Me^{3+}) $[Si_2O_6]$- (0.03.03 - 0.06%), $K(Mg,Fe^{2+},Mn)_3[(OH,F)_2|(Al,Fe^{3+}) \cdot Si_3O_{10}]$ – (0,07 – 0,127 % V_2O_3). The same primary vanadium, which is in the form of V_2O_3 in magmas, is partially released to ore sites upon differentiation, replacing isomorphically Fe_2O_3 in magnetite or entering the iron lattice due to the very close similarity in the electronic shell structure and ion radii (for Fe - 0.67 Å, for V - 0.65 Å) [2, p.23].

The features of blast furnace melting of titanomagnetites are generalized in the works and can be summarized as follows.

The increased content of TiO_2 in blast furnace charge (about 50 kg/t of cast iron) causes technological difficulties in the smelting of vanadium cast iron. In the temperature conditions of the blast furnace, gamma oxides of titanium (TiO_2, Ti_3O_5, Ti_2O_3, TiO), are formed, which contribute to the formation of refractory carbonitride compounds. The latter accumulate on the interfacial boundaries and surface of the coke, making it difficult to coagulate the kings of the metal [19, p.5]. The consequence of this is an increased loss of cast iron with slag, a deterioration of the drainage capacity of the horn [20, p.70-75], which creates conditions for the formation of non-active «soft» masses in the mountains. It is no coincidence that in the world practice of blast-furnace production, the allowable quantity of TiO_2 in charge is limited to 10 kg/t of cast iron [21, p. 20].

The increased content of titanium dioxide in the initial ore, as well as production failures (in the form of delayed releases, quiet strokes, parking lots and sediments) enhance and accelerate the processes of furnace growth by titanium carbonitride, Blast furnace operation becomes extremely unstable and is characterized by frequent cluttering of the mountain by non-polar masses. Therefore, titanium magnetite processing efficiency reserve is largely related to the improvement of the slag mode [3].

Titanium dioxide (TiO_2) in titanium-containing iron-ore materials is successively restored in the blast furnace by the following reactions:

$$TiO_2 + C = TiO + CO, \tag{1.1}$$

$$TiO + C = Ti + CO. \tag{1.2}$$

Thermodynamic analysis of the above reactions shows that recovery of titanium oxides at a noticeable rate begins at 1300 ºC.

The resulting metallic titanium interacts with carbon as well as nitrogen to produce carbides (TiC), carbonitride (TiCN) and titanium nitride (TiN):

$$Ti + C = TiC, \tag{1.3}$$

$$Ti + C + N = TiCN, \tag{1.4}$$

$$Ti + N = TiN. \tag{1.5}$$

TiC is formed with a large release of heat (239.694 kJ/mol), the conditions for its formation when carbon is exposed to TiO_2 are very favorable. The resulting compounds have exceptionally high melting points – more than 3000 ° C (for example, TiC – 3420 ° C). Since the temperature of cast iron and slag during blast furnace melting rarely exceeds 1450 – 1500 ° C, these carbides and carbonitrides of titanium are in melts in the form of solid phases.

During the normal course of the furnace, 7-10% of titanium is restored and passes into the metallic phase, but, as the metal is carburized and silicon is reduced to cast iron, the solubility of titanium in it decreases and excess titanium is released, concentrating (together with the newly formed) on the metal-slag, slag-coke contact surfaces, increasing the adhesion of the metal to slag and slag to coke. This is the reason for the poor filterability of the slag through the coke nozzle, which is expressed, in particular, in the appearance of slag on the tuyeres, especially before releases and when removing the blast, as well as the cause of increased losses of metal with slag, mainly in the form of so-called grenal, which is a crown of metal in a slag shell enriched with carbides and oxycarbonitrides of titanium.

As the temperature and time of the melts in the mountain furnace increase, the formation of TiC and TiCN progressively increases, which is accompanied by their accumulation in the blast furnace mountain. The density of titanium carbide and carbonitride formations is lower than that of cast iron (titanium carbide density 4.93 g/cm^3 and cast iron density 6.9 g/cm^3), but higher than slag density (slag density 2.8 g/cm^3). Said formations, which are not soluble in cast iron and slag, form a third phase - a grease, which from the main mountain trough enters the slag (i.e. is lost with slag) and cast iron carriers, depositing on the lining of the latter. This leads to an increase in the specific iron consumption by 1t of commercial cast iron [24].

The presence of titanium carbide and carbonitride solids in the slag makes the slag a heterogeneous system, accompanied by an increase in the apparent viscosity of the slag. When the furnace furnace horn is heated and the heating temperature of the cast iron and slag melts increases, the recovery of titanium oxides and the formation of titanium carbides and carbonitrids are intensified, causing a progressive increase in the viscosity of the heterogeneous slag melt.

The appearance in the melts of cast iron and slag of solid phases of carbides and titanium carbonitride has another negative side. In this case, solid shells from TiC and TiCN prevent the iron droplets from becoming larger, resulting in an increase in the loss of cast iron from the main distribution chute when the cast iron is released from the furnace.

The increase in metal and slag losses due to increased slag viscosity and small iron droplet size is related to the low deposition rate of iron droplets in slag, which is described by Stokes equation:

$$W = (2/9)\{[g \cdot r^2 \cdot (\rho - \rho_{sl})]/\eta\}, \qquad (1.6)$$

where W– the deposition rate of iron droplets in the slag;

g – the acceleration of gravity;
r – the radius of the drop of cast iron;
ρ – density of cast iron;
ρ_{sl} – slag density;
η – viscosity of the slag.

Measures to reduce titanium carbide formation during blast furnace smelting of titanomagnetites included the following.

The first successful implementation of vanadium pig iron smelting from titanium magnetite (in 30-40 years) was made possible by the use of such a technique as increasing the number of slags [23, p.488]. In other words, this was due to an artificial increase in the specific output of slag due to the use of low-grade ores in the charge. As a result, the concentration of titanium compounds in the slag and, consequently, their impact on the physical properties of the fire-liquid slag decreased. Naturally, this approach worsened the technical and economic performance of production in general.

As the blast furnace melting of titanomagnetites was developed, the optimal thermal mode was established, which corresponded to the silicon content in the cast iron range 0.2 - 0.4%. Successful operation of blast furnace with melting of titanomagnetites is possible only with high stability of thermal state and physico-chemical processes flowing in it. In this case, the heat level of the kiln should be lower the higher the concentration of titanium in the charge [18, p.5]. This mode of operation has a inhibiting effect on carbide formation. However, a prolonged campaign leads to a gradual increase in garnish, a decrease in drainage ability and mountaineering.

Until recently, the NTMC used to intermittently switch from vanadium pig iron to conventional pig iron, which allowed to eliminate the above negative aspects. This was allowed with six blast furnaces and more than enough iron ore base [27]. In the future, there will be three furnaces in the blast furnace shop, and the possibilities for the production of iron ore sinter have already decreased by 2.5 times.

The use of slag-liquefying additives, such as fluorspar and alkaline compounds, in experimental smelting showed high efficiency. However, their high cost, difficulties in supply, preliminary preparation and introduction into the charge, as well as the negative impact on the lining of the blast furnace, did not allow the introduction of this technology into permanent practice.

A fairly effective means of combating titanium carbide formation (more precisely, the destruction of carbide compounds) is the introduction of solid oxidants into the high-temperature zone of the blast furnace. Such oxidizing agents include hard-to-recover materials, for example, welding slag. Welding slag was introduced and is being introduced into the blast furnace charge [27, p.152]. Due to the low-porous macrostructure, as well as the high content of hard-to-recover fayalite in it, a significant amount of iron oxide entered the furnace, but not so much affected titanium carbides by reaction:

$$TiC + 3FeO = TiO_2 + CO + 3Fe. \qquad (1.7)$$

After the closure of the crimping workshops at the combine, the use of such materials is possible only by purchasing on the side, which is very problematic.

The features of the blast furnace melting of titanomagnetites are associated with the need to achieve the most complete extraction of vanadium into cast iron and prevent (or minimize) titanium carbide formation. The degree of vanadium transition to cast iron depends on numerous factors such as the basicity of the slag, its quantity, the temperature of the cast iron, etc. Considering that the lower vanadium oxides are difficult to recover and require significant heat costs, an increase in the completeness of recovery and the degree of extraction of vanadium into cast iron can be achieved by increasing the temperature level of the process. However, in real conditions, an increase in the thermal state of the furnace leads to the intensive formation of refractory titanium carbonitrides, which dramatically increase the apparent viscosity of the slag [12, p.84,29]. Therefore, blast furnace melting of titanium-magnetite raw materials, on the other hand, is forced to be carried out at moderate temperatures, i.e. to a threshold at which the appearance of viscous dense masses of coke, slag and cast iron is not caused in the furnace. At the same time, the more titanium in the charge, the lower the heating level of the furnace and the higher the requirements for its operation [29, p.315,30]. But, despite the fairly stable operation of the furnaces at low furnace heating, the specifics of melting titanium–magnetite raw materials are still manifested during the processing of melting products and in increased losses of metal with slag (5-7%).

Improving the slag mode of melting only by optimizing its basicity is not entirely effective. An increase in the basicity of slag contributes to both a more complete recovery of vanadium and the binding of titanium oxides, but increases the yield of slag, which leads to, in addition to a deterioration in melting performance, an increase in vanadium losses.

Studies of the effect of boron-containing additives on the agglomeration and blast furnace process.

Reducing the true viscosity can be achieved by using slag-liquefying additives. At the same time, the type of additive and the method of its introduction are of no small importance [29, p.315, 30, p.9-10]. One of such additives is boric anhydride – B_2O_3, which is an effective means for liquefying slags and reducing their melting temperature.

The development of technology for the use of boron-containing additives in the melting of titanomagnetites began with the use of borate ore (flux) Inderskoye field (Republic of Kazakhstan) [55, 56]. The ore had a wide range of fineness (0 – 400 mm), high moisture content (up to 20%) and sulfur (more than 8%) and an unstable mass fraction of boron oxide (16 – 30%). The approximate chemical composition of borate ore is given in Table 1.1. It is clear that the use of such material directly in the blast furnace process is unacceptable and its preparation is necessary, for example, together with the agglomeration of iron ore raw materials.

Table 1.1 - Average chemical composition of borate ore (%)

B_2O_3	CaO	SiO_2	S	MgO	Al_2O_3	w.r.
16.1	26.1	3.5	11.3	5.0	0.37	14.6

Initially, laboratory studies were carried out to determine the effect of the addition of on-board flux (containing 37% B_2O_3) to the agglomeration charge of the Kachkanar GOK on the agglomeration process indicators. It has been established (Table 1.2) that when borate flux is introduced (with a B_2O_3 content of more than 25%) in an amount of 3 to 5%, a fluxed agglomerate containing 1.1 – 1.5% B_2O_3 is obtained. With an increase in the proportion of borate flux in the charge, the specific productivity of the sinter plant increases significantly (up to 70%), the strength of the agglomerate (up to 25% (rel.)), but the mass fraction of iron in the agglomerate decreases by 0.3 – 0.5%. The sulfur content in the agglomerate does not increase due to an increase in the degree of its removal. During sintering, the volatilization of boron oxide in the amount of 10 – 15% was determined.

Table 1.2 - Results of laboratory sintering of Kachkanar sinter with the addition of Inder borate flux (IBF)

Indicators	Sintering series		
	basic	3 % IBF	5 % IBF
Composition of the charge, %			
Kach. GOK concentrate	89.3	87.8	86.7
Limestone	10.7	9.2	8.3

Continuation of table 1.2

Indicators	Sintering series		
	basic	3 % IBF	5 % IBF
Borate flux	0	3	5
C	3.6	3.6	3.6
Return	40	40	40
Charge humidity, %	7.1	7.0	7.2
The content of S in the charge, %	0.077	0.09	0.13
Specific productivity, t/m2·h	0.782	1.296	1.332
Suitable output (+5 mm), %	66.6	81.1	83.8
Composition of agglomerate, %			
Fe	56.9	56.5	56.3
FeO	13.9	14.1	16.6
S	0.03	0.02	0.02
CaO	7.22	6.88	6.9
SiO_2	5.47	5.07	5.05
V_2O_5	0.44	0.5	0.54
TiO_2	1.74	1.75	1.82
B_2O_3	-	1.14	1.58
CaO/SiO_2	1.32	1.36	1.37
Removal rate S, %	60.54	83.8	91.17
Strength	55.0	68.5	69.1
Abradability	9.5	6.2	6.1

However, the implementation of industrial approbation of this technology directly on the Kachkanar GOK was difficult in terms of organizational-technical reasons, and the study of the high temperature properties of the agglomerate and the obtained melts showed the irrationality and unreasonableness of such a high consumption of boron compounds.

In this connection, the possibility of producing boron agglomerate on a regular Lebyazhinsky agglomerate load for subsequent use as an additive in titanium magnetite smelting was raised. The principal possibility of producing boron agglomerate was identified by laboratory sintering. As a result of these studies it has been determined (Table. 1.3) that when added 3 - 5% of borato flux (with a content of B_2O_3 16%) to the usual planned charge of Lebyazhinsky agglomerate results from the lumbar agglomerate containing ~ 0.2% B_2O_3, and about 25% of the boron evaporates with process gases. In the agglomerate the mass fraction of iron decreases by ~ 0.4%, the unit specific performance does not change, the strength characteristics of the agglomerate slightly improve, the solid fuel consumption decreases by 3 - 4 kg/t agglomerate.

In the course of the industrial experiment, just under 11.000 tonnes of agglomerate were produced with an average B_2O_3 content of 0.4%. Borato ore was fed to the charge as a mixture with limestone in a ratio of 1:2 and after grinding to a fraction of 0 - 3 mm. The share of borato ore in the flux mixture was 23%, and the mixture contained 2.75% B_2O_3 and 47.5% CaO.

Table 1.3 - Results of laboratory sintering of Lebyazhinsky sinter with the addition of Inder borate flux (IBF)

Indicators	Sintering series		
	basic	3 % IBF	5 % IBF
Composition of the charge, %			
Borate flux	0	3	5
C	3.0	2.8	2.5
The content of S in the charge, %	0.917	1.295	1.63
Specific productivity, t/m^2·h	1.297	1.267	1.316
Suitable output (+5 mm), %	82.2	86.8	85.6
Composition of agglomerate, %			
Fe	54.4	53.9	53.6
FeO	15.9	16.8	13.8
S	0.09	0.14	0.15
CaO	11.0	11.41	11.64
SiO_2	9.62	9.5	9.2
B_2O_3	0.01	0.16	0.25
CaO/SiO_2	1.32	1.36	1.37
Removal rate S, %	91.6	91.1	92.1
Strength	64.1	65.9	65.7

The production of the experimental agglomerate (Table. 1.4) was characterized by a significant increase in the current downtime (due to the high humidity of the fluxes), a reduction in the solid fuel consumption by 3.3 kg/t of the agglomerate. The balance sheet shows the volatilization of boron to 30% of its arrival. The specific performance of agglomerates remained virtually unchanged and the agglomerate met the requirements (for a conventional agglomerate).

Table 1.4 – Sintering indicators of the first experimental batch of boron-containing agglomerate

Agglomerate quantity, t	10786.7
Composition of the charge, %	
MOF concentrate	33.0
Aglorood mixture	49.6

Continuation of table 1.4

Dust	2.2
Limestone	11.7
Borate flux	3.5
Solid fuel consumption, kg/t	60.1
Speed of agolo machines, m/min	3.4
Specific productivity, $t/m^2 \cdot h$	1.389
Current downtime, %	22.5
Agglomerate composition, %	
Fe	53.04
FeO	12.9
S	0.089
CaO	11.03
SiO_2	10.59
B_2O_3	0.39
CaO/SiO_2	1.04
Detail content 5-0 mm, %	8.5
Strength	60.6
Abradability	6.8

The resulting agglomerate was used as an additive (~ 2%) to the main charge of the NTMK blast furnace No. 1 smelting vanadium cast iron. This made it possible to achieve a reduction in coke consumption by 6 kg / ton of cast iron, increase the vanadium recovery rate by 5%, and reduce metal losses with slag by 4 times. There was also an improvement in the furnace stroke and the quality of cast iron in terms of sulfur content [39].

These results were a prerequisite for repeated industrial tests, during which about 11.000 tons of boron–containing Lebyazhinsky agglomerate were also produced (with an average B_2O_3 content of 0.44%), and the resulting agglomerate was used as an additive already on all blast furnaces smelting vanadium cast iron. Comparison of technological indicators (Table 1.5) the production of boron-containing and conventional agglomerate showed that there were no significant changes in the process parameters in this case: specific productivity, current downtime and strength remained at the same level. The content of fines in the agglomerate has noticeably decreased, but at the same time the sulfur content has increased in it. A decrease in solid fuel consumption (by 3.6 kg/ton of agglomerate) and boron loss in the gas phase in the amount of 25-30% was confirmed.

Table 1.5 - Comparison of agglomerate production

Indicators	Agglomerate	
	borate	ordinary
Agglomerate quantity, t	10 972.8	226 132

Continuation of table 1.5

Indicators	Agglomerate	
	borate	ordinary
Composition of the charge, %		
MOF concentrate	39.2	37.0
Aglorood mixture	43.0	46.7
Dust	1.7	2.3
Limestone	8.1	14
Borate flux	8.0	-
Solid fuel consumption, kg/t	67.0	70.6
Speed of agolo machines, m/min	3.2	3.1
Specific productivity, t/m2·h	1.315	1.306
Agglomerate composition, %		
Fe_{total}	53.82	53.87
FeO	13.5	12.95
S	0.096	0.071
CaO	9.92	11.72
SiO_2	8.62	9.36
B_2O_3	0.39	-
CaO/SiO_2	1.151	1.252
Detail content 5-0 mm, %	5.5	8.3
Strength	57.6	58.4
Abradability	6.2	6.2

The involvement of vanadium furnaces in the blast furnace charge up to 5% of boron-containing Lebyazhinsky agglomerate significantly improved the melting performance (Table 1.6): coke consumption decreased by 2.4 kg / ton of cast iron, metal losses with slag decreased by 2 times, the vanadium extraction coefficient increased by 0.6%. The sulfur distribution coefficient and the degree of carbon monoxide use were also improved, respectively, by 1.0% and 0.3%. An increase in the mass fraction of titanium compounds in slags was noted, which is a consequence of the inhibition of coal formation in the blast furnace.

The conducted studies have shown the effectiveness of the use of boron-containing additives in the blast furnace melting of titanomagnetites. At the same time, the sufficiency of boron oxide content in blast furnace slags providing a B_2O_3/TiO_2 ratio of 0.01 was determined. However, the geopolitical changes that have taken place, the decrease in the reserves of raw materials of the Indeborskoye deposit, the negative properties of borate flux – high sulfur and moisture content, as well as significant transportation costs did not allow the developed technology to be put into permanent practice. But this prompted the search for a replacement for natural boron-containing material.

In this regard, studies were conducted on the possibility of attracting waste from the production of boron-containing glass into the agglomeration charge of the

Kachkanar GOK on the indicators of the agglomeration process. It is known that the introduction of mineral fiber and even glass in small quantities into the charge for the production of dipped iron ore raw materials increases the strength and accelerates the process.

Table 1.6 - Technical and economic indicators of the work of the blast-furnace vanadium conversion without the involvement of (basic) and with the involvement of (experimental) boron-containing Lebyazhinsky agglomerate

Indicator name	Periods	
	Basic	experienced
Cast iron production, t/d	8 985	8 993
Coke consumption (dry), kg/t of cast iron	450.7	448.3
Consumption of ore raw materials, kg/t of cast iron	1676.3	1668.0
Consumption of iron, kg/t of cast iron	969.9	967.1
Consumption of natural gas, m^3/t of cast iron	91.4	90
Mass fraction of Fe in the charge, %	57.86	57.98
Extraction coefficient V, %	80.6	81.2
L_S	30	31
The degree of use of CO, %	45.2	45.5

The research used waste from the production of "Glass Fiber" (Polotsk), the following chemical composition (%): 10.5 B_2O_3; 16.5 CaO; 49.2 SiO_2; 4.6 MgO; 13.5 Al_2O_3; w.r. and S – traces. The annual generation of aluminum-borosilicate fiber waste in the industry significantly exceeds the needs of the blast-furnace vanadium conversion in Russia (NTMK and Chusovskaya Metallurgical Plant). At the same time, the criterion of optimality of the concentration of boron in the charge of blast furnace vanadium smelting found when working with boron-containing ore is similar for the case of using aluminum borosilicate fiber waste [45-46].

The iron ore part of the charge consisted of 100% Kachkanar concentrate. The content of coke fines and return was constant and was equal to 4 and 35% of the total mass of the dry charge, respectively. The amount of glass added, pre-crushed in a ball mill, was 0.5 and 1.0%, which provides the required ratio of boron oxide to titanium dioxide in the slag of the blast furnace during the smelting of vanadium cast iron.

The data on the sintering results (Table 1.7) show that the introduction of boron-containing glass additives in such small quantities increases the yield and productivity of the sintering plant, and the first portions of the added glass have the greatest effect. The strength of the agglomerate increases according to GOST 15137 in terms of X_1 by 4.7 and 2.6% in the presence of 0.5 and 1% fiberglass in the charge, respectively, and the abrasion index decreases as the additive consumption increases by 2.7%.

Table 1.7 - Sintering results

Indicators	Consumption of fiberglass in the charge, %		
	0	0.5	1.0
Initial vacuum, kPa	8.01	8.20	8.48
Final vacuum, kPa	6.20	5.10	4.60
Suitable output (+ 5 mm),%	66.5	78.3	77.2
Strength according to GOST15137, %:			
X_1	64.2	68.9	66.8
X_2	7.4	5.7	4.6

To determine the cause of such a positive effect on the agglomeration process, the glass fiber suspension was subjected to a thermal study on a derivatograph (Fig. 1.1).

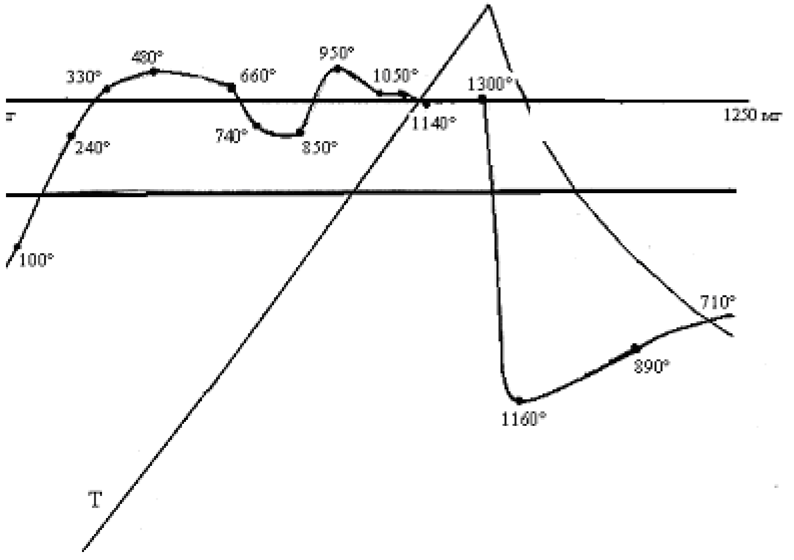

Figure 1.1 - Thermoderivatogram of boron-containing glass fiber

Analysis of thermoderivatograms shows that the transition of glass fiber from a solid to a ductile-viscous state occurs already at a temperature of about 800 °C, and at a temperature above 1000 °C, a sufficiently liquid-mobile melt is formed.

Therefore, the mechanism of action of glass fiber in the process of agglomeration is to intensify the formation of sinter melt and reduce its viscosity. The development of the melt contributes to the solubility of the charge minerals, a deeper passage of physico-chemical sintering processes and the formation of a uniform macro- and microstructure of the agglomerate, which ultimately determines its strength. It is also known that with the greater development of the melt, the processes of isomorphic substitutions take place more intensively in magnetite, leading to an increase in Mg, Al and Ti in its composition [52-54].

Analysis of softening curves shows that the temperature of the beginning of softening with boron-containing glass additives not only does not decrease, but also increases markedly. This is explained, as mentioned above, by the more uniform structure of the agglomerate. The porosity and reducibility of the agglomerate with the addition of 0.5% fiber is slightly lower than the base agglomerate, which is due to a more fused structure. Of interest is a noticeable increase in porosity and recoverability with a consumption of fiberglass in the charge of 1%. It is characteristic that similar data were obtained with the addition of the same amount of window glass.

In the future, porosity according to GOST 26293, reducibility and soften ability of the obtained agglomerates were studied according to the MISIS methods. The results of these studies are shown in Table 1.8.

Table 1.8 - Characteristics of agglomerates

Material	Content in the agglomerate, %			Softening start temperature, °C	Porosity, %	Recoverability, %
	B_2O_3	FeO	S			
Agglomerate without additives	-	16.8	0.03	970	33.02	72.1
Agglomerate with 0.5% glass fiber additive	0.06	13.8	0.02	1025	29.2	71.0
Agglomerate with 1.0 % glass fiber additive	0.1	11.7	0.02	990	35.72	79.9

The mass fraction of S and FeO decreases in the agglomerate, since an increase in the development of the melt facilitates the transport of oxygen for the formation of hematite and sulfur dioxide. When calculating the boron balance, it was found that when boron-containing glass is introduced into the sinter, boron oxide, in this case, does not evaporate and completely passes into the agglomerate.

It should be noted that as the content of borosilicate fiber in the charge increases further, the agglomeration process indicators decrease. This is due to the deterioration of the lumpiness of the charge, since the glass is practically not wetted with water, and an increase in the gas-dynamic resistance of the layer due to an excessive amount of melt. Therefore, there is a critical consumption of fiberglass in the charge, above which the technical and economic indicators of the agglomerate become lower than when sintering without additives. However, in these studies, the task was not to determine the optimal consumption of boron-containing fiber in the charge, and the use of these amounts of additives to ensure the required ratio of

B_2O_3/TiO_2 is very effective, and the decrease in iron content in the agglomerate is insignificant and can be compensated by a decrease in basicity.

As a result of these stages of research, the following has been established. The use of boron-containing additives in agglomeration and blast furnace melting of titanomagnetites can significantly increase the efficiency of end-to-end processing while improving the quality of products, reduce total energy consumption by 200 kJ/ton of cast iron. It is most rational to use aluminum borosilicate fiber waste in an amount of up to 1% as a boron-containing additive to the titanomagnetite agglomeration charge, which makes it possible to simultaneously dispose of these wastes and reduce emissions of sulfur dioxide and boron compounds into the atmosphere. The effect of introducing boron-containing agglomerate titanomagnetites into the blast furnace charge is investigated. Industrial testing of this technology has shown that as a result of reducing the viscosity of slag, cast iron losses are reduced. This event should be considered promising with an increase in the concentration of titanium in ore raw materials.

Technological difficulties of blast furnace melting of titanomagnetites are associated with the formation of refractory titanium compounds, which worsen the drainage ability of the furnace. Despite the systematic improvement of smelting technology and improvement of the quality of iron ore raw materials, the intensity of the slag regime does not weaken and even increases. This is due, in particular, to a deeper enrichment of titanomagnetites, which, along with an increase in the iron content, leads to an increase in the concentration of titanium dioxide and alumina in iron ore raw materials and, even more, in slags. The presence of these components in slag melts significantly increases their heterogeneity and, consequently, their apparent viscosity, which, in turn, increases metal losses at the outlets and, in general, reduces the technical and economic indicators of production. Therefore, the reserve of efficiency of processing of titanomagnetites is largely associated with the improvement of the slag regime. The latter can be achieved by using slag-liquefying additives and suppressing the formation of titanium carbonitrides.

An interesting idea of extracting iron from polymetallic ores, including titanium, is used in the Australian patent [50]. According to it, titanium-containing polymetallic ores are reduced to a temperature of 1000-1400 ° C, under conditions of preserving their original shape and non-melting, forming the necessary metal layer, up to coagulation of reduced iron particles. In this case, free oxides of rocks containing titanium oxides coagulate in the molten state and form an inner core. The separation of metal from iron and titanium is achieved by grinding and magnetic separation.

Conclusion on the first chapter, purpose and objectives of the study

It should be noted that during the two-stage processing of ilmenite and titanomagnetite concentrates, there are many ways to pre-extract iron. In addition, it should be noted that many methods are not used in industry, while others are used on a very limited scale. One of the main reasons for this is the use of processes and installations for the extraction of iron, titanium and other components from a special ("multicomponent") raw materials. On the one hand, due to the reduction of such raw materials, on the other hand, due to the growing demand for metals, there is an increasing need for the introduction of polymetallic ores into the industry and the development of methods for their processing.

Disadvantages:

- The mechanism of reduction of difficult-to-reduce metals is not fully considered and studied in the technical literature;

-The technology of obtaining alloy steel directly from hard-to-process titanium-magnetite and ilmenite ores is not approved.

To solve these problems, the following tasks were set in the dissertation work:

- preparation of laboratory samples from multicomponent oxide materials;
- calculation of stoichiometric consumption of solid carbon for the reduction of metal oxides;
- preparation of special samples of ore-coal mixture required for the process of direct reduction of metals and obtaining metallized materials from them;
- development of technology of laboratory reduction and melting of multicomponent concentrates, preparation and testing of laboratory melting units. Analysis of the obtained research results.

2 Methodology of practical research

2.1 Description of the components used

Complex ilmenite (Satpayev field) and titanium-magnetite concentrates, as well as agglomeration process wastes (okalina) were used for the organization and implementation of experimental research. In addition to iron oxides, the complex ore contains alloys of manganese, chromium, vanadium, titanium metals, which have a higher chemical bond strength than iron oxides. Therefore, the method of reduction processes is directly related to the choice of reduction reagents and the mechanism of the reaction.

In the modern educational and scientific literature, the mechanism of the reduction process is explained in terms of the adsorption-autocatalytic mechanism based on the adsorption of CO, H_2 by gas reducing agents [56,57]. According to this mechanism, the role of solid carbon is not taken into account, as only the contact-diffusion interaction is observed. However, as we know, the above-mentioned difficult-to-reduce oxides are not significantly reduced due to the lack of reduction potentials with CO and H_2 gases. This indicates the impossibility of using the adsorption-autocatalytic mechanism in the reduction of complex ores.

The possibility of the interaction of carbon with metal oxides has been identified in practice and a new "dissociation-adsorption mechanism" of metal reduction has been proposed. The study of the mechanism and kinetics of difficult-to-reduce metals is a very complex and topical issue [59].

A number of experimental studies conducted for the direct reduction of metals used the method of interaction of dissolved carbon in the oxide system. And taking into account the high reduction potential of free and dissolved carbon, the method of experimental research is based on the organization of direct reduction of iron and associated difficult reduction metals in complex titanium-magnetite raw materials.

The chemical composition of the charge components used in experimental studies is given in Table 2.1.

As can be seen, the main oxide in Satpayev concentrate is TiO_2. Iron concentration 32.53% consists of two oxides Fe_2O_3= 26.67% and FeO = 16.5%. Oxides of iron, manganese, vanadium, titanium and chromium are reduced by solid carbon and converted into metal, which contributes to improving the quality of the molten metal.

Along with alloys of alloying metals - MnO, Cr_2O_3, V_2O_5, high chemical strength of titanium oxide TiO_2 remains a big problem in these complex raw material resources. This is due to the fact that the alloys of alloying metals have a very high chemical strength TiO_2. If the lower iron oxide FeO is well reduced by CO gas and solid carbon at a temperature of 1000-1200 K, then the oxides of MnO, Cr_2O_3, V_2O_5, TiO_2 can not be reduced very effectively by CO gas, and by solid carbon are reduced at very high temperatures.

According to the chemical strength of oxides, the reduction of individual oxides with solid carbon takes place at different temperatures. Among these oxides,

TiO_2 has a special place in the organization of reduction processes. Titanium reduction begins only at a temperature of 1250 ºC. Such a difference between the temperature level of these metals and the kinetics of direct reduction causes certain problems in the processing of titanium-magnetite concentrates. However, to some extent, the solution can be achieved by increasing its activity by adjusting the state of solid carbon at the entrance to the described system and a defined approach to the direct reduction of titanium, taking into account the solubility of titanium in metal.

Preparation of dispersed ore is not only theoretically based, but also based on the methods of experimental research on the organization of reduction and melting processes.

2.2 Theoretical bases of charge preparation

Satpayev ilmenite concentrate and Arcelor Mittal Temirtau okalina were used as components of the initial charge. They consist mainly of Fe, Mn, Cr, V and Ti oxides.

The amount of oxygen gaseous from reduced metal oxides was determined by the principle of sequential phase transformation of the chemical composition of concentrates [59].

The concentrates are pre-ground to a fraction of ~ 0.074 mm and their chemical composition determines the amount and phase distribution of oxygen gasified in the charge. At the same time, attention is paid primarily to the gasification of oxygen from higher oxides, such as Fe_2O_3 - to FeO, MnO_2 - to MnO, V_2O_5 - to V_2O_3, TiO_2 - to Ti_2O_3. Oxidation of oxygen to Cr_2O_3- Cr, V_2O_5- V and TiO_2 - Ti takes place at the final stage of the process.

As a reduction reagent, charcoal or granulated cast iron (depending on the amount of dissolved carbon in the cast iron) is added in a stoichiometric ratio to reduce them from oxides of Fe, Mn, Cr, V and Ti.

2.3 Used laboratory facilities and equipment

To obtain the components of the crushed charge. A laboratory electric vibrator was used to obtain the crushed materials (Figure 2.1). The charge components were crushed individually to a fraction of ~ 0.074 mm.

Extraction of charge components. According to the chemical composition of the charge components, the amount of oxygenated gas during reduction and the consumption of reduction reagents per unit of oxide components - charcoal and granulated cast iron.

At the same time, during direct reduction, attention is paid to the fact that the oxides of oxygen, such as Fe_2O_3 – Fe_{met}, MnO_2- MnO, V_2O_5 - V_2O_3, TiO_2- Ti_2O_3, are gassed first. Oxidation of oxygen from Cr_2O_3 - Cr, V_2O_3 - V and TiO_2 - Ti oxides takes place at the final stage of the process. From the balance of gaseous oxygen oxide charge and charcoal and 1-2.0 mm fr. The mass ratio of the reduction reagent in the form of granular cast iron was determined. The charge mixture is prepared according to the given ratios.

Production of metallized products. The mixture of charge materials is introduced into a hermetic test chamber made of quartz tube with a diameter of 30 mm. The cell is installed in the SUOL-044 12-M2 tubular electric furnace shown in Figure 2.3. As the system heats up, a solid phase reduction product - gas - is released from a temperature of 700 ° C. The release and composition of the gas mixture containing CO_2 and CO is monitored by gas meters and gas analyzers.

Figure 2.1 – Laboratory grinder

Figure 2.2 – Disc granulator

Figure 2.3 – SUOL-044 12-M2 tubular electric furnace

Figure 2.4 – Gas analyzer KGA 1-1

Figure 2.5 – High temperature tubular furnace

A new three-dimensional research method - 3D was used for a more reliable quantitative and qualitative assessment of the degree of metal contamination by non-metallic inclusions.

Figure 2.6 - Device for the detection of non-metallic impurities by electrolytic extraction

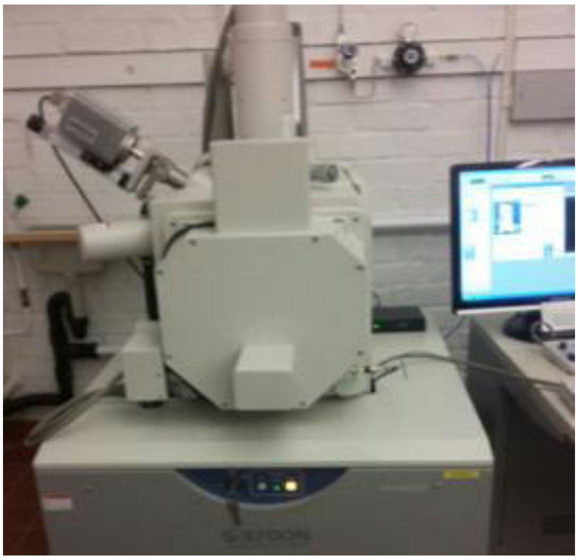

Figure 2.7 - SEM S-3700N

We investigate the compounds (metal and non-metallic) isolated on the surface of membrane film filters with the help of S-3700N brand SAM, connected to the computer.

Conclusion on the second chapter

- In order to achieve the objectives set for the production of complex charge, the fine concentrate Satpayev ilmenite, rolling scale Arcellor Mittal Temirtau, as well as charcoal as reducing agents are used;

- The mixture of carbon raw material is produced by mixing the calculated mass quantity of the specified components of the charge;
- Chemical and mineralogical samples of products obtained during the metallization and smelting of coal-ore loads shall be carried out in accredited laboratories and on certified devices;
- Theoretical research and determination of stoichiometric amounts of carbon spent on iron oxides, manganese, chromium, titanium, vanadium is based on the theory of the Doctor of Technical Sciences, Academician S. M. Tleugabulov on reduction of metal oxides in the solid phase;
- It is proposed to carry out the process of metallization in the reduction atmosphere for the production of metallized material from titanium-magnetite and ilmenite ores of complex processing;
- Metallized product sample reduction - the melting process is carried out in the High temperature tubular furnace.

3 Full charge calculation, material and thermal balance reduction-melting process

Laboratory studies on involvement in the processing of titanium magnetite ores were conducted. The table shows the chemical composition of the studied concentrates:

Table 2.1 - The chemical composition of the charge components

Names of materials	Chemical composition ,%												
	Fe	FeO	Mn	SiO_2	CaO	MgO	V_2O_5	TiO_2	Cr_2O_3	Al_2O_3	P	S	C	w.r.
Satpayev ilmenite concentrate	32.53	16.54	0.7*	5.80	0.50	0.33	0.22	49.65	0.27	1.25	0.005**	-	-	7.73
Arcelor-Mittal rolling scale	67.65	28.25	0.23	-	-	-	-	-	0.51	-	-	0.017	-	-
Charcoal***	-	-	-	0.48	-	-	-	-	-	0.12	-	0.06	98.0	-
Notes: *- MnO; **-P_2O_5; ***- other ash A^c=1,34.														

Determine the concentrations of high metal oxides- Fe:

In the process of reducing firing, metallization and melting of the charge, the oxygen of the following oxides is gasified:

$Fe_2O_3 \rightarrow Fe_3O_4 \rightarrow FeO$;

$MnO_2 \rightarrow Mn_2O_3 \rightarrow Mn_3O_4 \rightarrow MnO$;

$V_2O_5 \rightarrow V_2O_3 \rightarrow VO$;

$TiO_2 \rightarrow Ti_2O_3$;

$Cr_2O_3 \rightarrow Cr_3O_4 \rightarrow CrO$.

Oxygen purified oxides are gasified with a solid CO_2 and CO. The CO_2 and CO ratio in the existing gas will be related to the oxides recovery stage and the temperature of the process. In the context of renewable energy sources in relation to each metal, it is possible to predict their restoration or restoration. Pressure, step strength of the η_{Fe}=0.999; η_{Mn}=0.80%; η_V=0.90. In addition to these metals, silicon is partially reduced depending on the excess carbon and temperature. Therefore, the amount of carbonated oxygen of silicon oxide may not be accounted for or taken into account at an expected metal concentration of 0.18-0.3%.

Determination of gasifiable oxygen of charge

1 Recovery of iron. First of all, we will estimate the ratio of higher and lower oxides according to the average composition of the ore part of the charge. Data on total iron concentration Fe=50.4% and iron oxide FeO=23.28% allow a clear estimate of the amount of oxygen bound in FeO and Fe_2O_3.

Fe_1= Fe-0.777FeO=50.4-0.777·23,28=32.31%

Further on the methodology we determine the amount of gasified oxygen by the stages of recovery.

$Fe_2O_3 \rightarrow Fe_3O_4$; $\Delta Ơ_1 = 10^{-2} \cdot 0.4285 \cdot Fe_1 \cdot 0.111 = 0.0476 \cdot 10^{-2} \cdot 32.31 = 0.0154$ kg/kg of charge

$Fe_3O_4 \rightarrow FeO$; $\Delta Ơ_2 = 10^{-2} \cdot 0.4285 \cdot Fe_1 \cdot 0.222 = 0.0951 \cdot 10^{-2} \cdot 32.31 = 0.0307$ kg/kg of charge

The incomplete iron remains as FeO and can be counted at the last step multiplied by the corresponding value of the assimilation degree, i.e.

$FeO \rightarrow Fe$; $\Delta Ơ_3 = 10^{-2} \cdot \eta_{Fe}(0.4285 \cdot 0.667\ Fe_1 + 0.222\ FeO) = 10^{-2} \cdot 0.999 \cdot \cdot (0.2858 \cdot 32.31 + 0.222 \cdot 23.28) = 10^{-2} \cdot 0.999\ (9.23 + 5.17) = 0.144$ kg/kg of charge

Total oxygen gasified during iron recovery:

$\Delta Ơ_{Fe} = \Delta Ơ_1 + \Delta Ơ_2 + \Delta Ơ_3 = 0.0154 + 0.0307 + 0.144 = 0.1901$ kg/kg of charge

2 Recovery of manganese. In the initial charge of all manganese Mn=0,48%

$\Delta Ơ_{Mn} = 10^{-2} \cdot Mn[0.1455(1 - \Omega_{Mn}) + 0.2908 \cdot \eta_{Mn})] = 10^{-2} \cdot 0.48(0.1455 \cdot 0.548 + 0.23264 = 0.0015$ kg/kg of charge

The amount of gasified oxygen during the reduction of manganese is distributed by steps in the following order.

$4MnO_2 \rightarrow 2Mn_2O_3 + O_2$

$\Delta O_{1\ Mn} = 0.632 \cdot 10^{-2} \cdot \Omega_{Mn} \cdot Mn = 0.632 \cdot 10^{-2} \cdot 0.217 = 0.0014$ kg/kg of charge;

$6Mn_2O_3 \rightarrow 4Mn_3O_4 + O_2$

$\Delta O_{2\ Mn} = 0.111 \cdot \Delta O'_{Mn} = 0.00016$ kg/kg of charge;

$2Mn_3O_4 \rightarrow 6MnO + O_2$

$\Delta O_{3\ Mn} = 0.222 \cdot \Delta O'_{Mn} = 0.00033$ kg/kg of charge;

$2MnO \rightarrow 2Mn + O_2$

$\Delta O_{4\ Mn} = 0.667 \cdot \Delta O'_{Mn} = 0.0010$ kg/kg of charge.

Total oxygen gasified during manganese reduction:

$\Delta O'_{Mn} = \Delta O_{1\ Mn} + \Delta O_{2\ Mn} + \Delta O_{3\ Mn} + \Delta O_{4\ Mn} = 0.0014 + 0.00016 + 0.00033 + 0.0010 = 0.00289$ kg/kg of charge.

3 Recovery of vanadium. All vanadium in the initial charge is in the form of V_2O_5. The degree of oxidation of vanadium according to equation $\Omega_V = 0$. Then

$\Delta O'_V = 10^{-2} \cdot V[0.4705 \cdot \eta_V - 0.1568(1 - \Omega_V)] = 10^{-2} \cdot 0.061\ (0.174 - 0.1568) = = 0.0018$ kg/kg of charge.

The amount of gasified oxygen during vanadium recovery will be distributed in the following order.

$V_2O_5 \rightarrow V_2O_3$ $\Delta O_{1V} = 10^{-2} \cdot 0.3137 \cdot \Delta O'_V = 0.00056$ kg/kg of charge;

$V_2O_3 \rightarrow VO$ $\Delta O_{2V} = 10^{-2} \cdot 0.1568 \cdot \Delta O'_V = 0.00028$ kg/kg of charge;

$VO \rightarrow V$ $\Delta O_{3V} = 10^{-2} \cdot 0.3137 \cdot \Delta O'_V \cdot \eta_V = 0.00208$ kg/kg of charge.

Total oxygen gasified during vanadium reduction:

$\Delta O'_V = \Delta O_{1\ V} + \Delta O_{2\ V} + \Delta O_{3\ V} = 0.00056 + 0.0028 + 0.00208 = 0.00293$ kg/kg of charge.

4 Recovery of chromium. In the initial charge of Cr_2O_3 chromium oxide = 2.7%, of which in the form of Cr is:

$Cr = 0.684 \cdot Cr_2O_3 = 0.684 \cdot 2.7 = 1.84$ %.

Next, we determine the amount of gasified oxygen by the reduction steps:

$Cr_2O_3 \rightarrow Cr_3O_4$: $\Delta O_{1Cr} = 10^{-2} \cdot 0.4615 \cdot 0.111 \cdot \Delta O'_{Cr} = 10^{-2} \cdot 0.4615 \cdot 0.111 \cdot 1.84 = 0.00094$ kg/kg of charge;

$Cr_3O_4 \rightarrow CrO$: $\Delta O_{2\,Cr} = 10^{-2} \cdot 0.4615 \cdot 0.222 \cdot \Delta O'_{Cr} = 10^{-2} \cdot 0.4615 \cdot 0.222 \cdot 1.84 = 0.0019$ kg/kg of charge;

$CrO \rightarrow Cr$: $\Delta O_{3\,Cr} = 10^{-2} \cdot 0.4615 \cdot 0.667 \cdot \Delta O'_{Cr} = 10^{-2} \cdot 0.4615 \cdot 0.667 \cdot 1.84 = 0.0057$ kg/kg of charge.

Total oxygen gasified during vanadium reduction:

$\Delta O'_{Cr} = \Delta O_{1\,Cr} + \Delta O_{2\,Cr} + \Delta O_{3\,Cr} = 0.00094 + 0.0019 + 0.0057 = 0.00854$ kg/kg of charge.

5 Recovery of silicon and titanium. In ore materials, titanium and silicon are found as TiO_2 and SiO_2 respectively, the oxygen content of titanium oxides and silicon is calculated at 0.10-0.3% of its expected concentration in metal:

$\Delta O'_{Si} = [Si] \cdot 10^{-2} \cdot e_M \cdot 1.1428 = 0.9 \cdot 10^{-2} \cdot 1.428 = 0.0077$ kg/kg of charge.

$\Delta O'_{Ti} = [Ti] \cdot 10^{-2} \cdot e_M \cdot 0.667 = 0.4 \cdot 10^{-2} \cdot 49.69 \cdot 0.3 = 0.0059$ kg/kg of charge.

Total amount of gasified oxygen:

$\Delta O'_{ch} = 0.1901 + 0.00289 + 0.00293 + 0.0077 + 0.00854 + 0.0059 = 0.218$ kg/kg of charge.

Determination of carbon consumption as a reducing agent

Since the formation of CO_2 and CO depends on the degree of reduction of oxides and temperature, first of all it is necessary to set the reduction mode, select the process temperature. The process of reducing iron is completely completed at 1100 ° C, manganese and vanadium – at 1300-1400 ° C. However, the higher oxides of manganese and vanadium are quite intensively reduced at 1000-1100 ° C, i.e. in the temperature range of iron reduction. Therefore, to determine the flow rate, we will use the table where the ratios of CO_2 and CO for iron, manganese, chromium, titanium and vanadium oxides are presented.

Carbon consumption for iron recovery

We determine the oxygen distribution coefficients at 1100 ° C according to the equations.

$Fe_2O_3 \rightarrow Fe_3O_4$; $\gamma_{01} = 0.974$; $\gamma_{02} = 0.974$;
$Fe_3O_4 \rightarrow FeO$; $\gamma_{01} = 0.70$; $\gamma_{02} = 0.974$;
$FeO \rightarrow Fe$; $\gamma_{01} = 0.70$; $\gamma_{02} = 0.974$.

By the equation for iron:

$g'_{C,Fe} = 0.75 \cdot \Sigma \Delta O'_i (1 - 0{,}5 \cdot \gamma_{01,2}) = 0.75\,(0.0079 + 0.019 + 0.1152) = 0.1421 \cdot 0.75 = 0.106$ kg/kg of charge.

For recovery chromium:

$g'_{C,Cr}$=0.75(0.00094·0.51+0.64·0.0019+0.0057) =0.104+0.024 =0.0734 kg/kg of charge.

For recovery vanadium:

$g'_{C,\,V} = 0{,}75\cdot\Sigma[\Delta O_{iV}\cdot(1-0{,}5\cdot\gamma_{0V})\cdot i]=0.0002745$ kg/kg of charge.

For recovery manganese:

$g'_{C,\,Mn}$ =0.75[0.0014 ·0.5 + 0.00016(1-0.5·0.888) +0.00033(1-0.5·0.57) + 0.0010· 0.5= kg/kg of charge.

For recovery titanium:

$g'_{C,\,Ti}$ =0.75(0.0059 ·0.655) =0.021 kg/kg of charge.

For recovery silicon:

$g'_{C,\,Si} = 0.75\cdot\Delta O_{Si}=0.75\cdot 0.0077=0.0057$ kg/kg of charge.

Total carbon reagent consumption,

$g'_{C}= g'_{C\,Fe} + g'_{C,Mn} + g'_{C,Ti}\; g'_{C,V} + g'_{C,Si} + g'_{C,Cr} = 0.228$ kg/kg of charge.

This amount of carbon is supplied in the form of a coke trim, in which the carbon content is 85% and the iron in the form of oxides is 1.24%. To recover the own iron is used part of the carbon. In this way the effective carbon concentration in the coke breeze will be reduced to the C_{ef} level, the value of which is determined by the formula,

$C_{ef} = C-0.75\cdot Fe[0.1426(1-\Omega'_{Fe})+0.286(0.5\cdot\gamma_{01}+\gamma_{02})] = 84.97\%$.

With the carbon content adjustment, we determine the consumption of coke residue –reagent,

$g'= g'_{c}/10^{-2}C_{ef}=0.228/0.8497=0.26$ kg/kg of charge.

Determination of output metal

The main components of the metal are Fe, Mn, V, Si. However, before the melt is obtained, the composition of the metallized product is formed. The MP output is easily determined by the mass loss and the formula

$$L_{MP}=1-0.2255-0.0351=0.7394 \text{ kg/kg of charge.}$$

If the final product is MP, then further calculation of the material and thermal balances can be carried out by 1 kg of MP.

In this case, the task is to obtain a metal melt. Therefore, the calculation is carried out further.

$l''_{sl}=10^{-2}\cdot(2.35-0.014+2.6+0.32+0.087+0.123+0.3+0.007+0.019+0.06)= -1.18\cdot 10^{-2}$ kg/kg of charge.

This amount of slag is in MP. Then output of the metal is:

$$l''_{\text{м}}=0.7394+0.0118=0.7512 \text{ kg/kg of charge.}$$

The amount of metal and slag increases due to the ash of coke fines-reagent:

$\Delta\, l''_{\text{м}}=10^{-2}\cdot Fe_{\text{кл}}\cdot g_{\text{кл}}\cdot\eta_{Fe}=10^{-2}\cdot 0.99\cdot 0.106\cdot 0.26=0.00027$ kg/kg of charge.

$\Delta\, l''_{\text{шл}}=10^{-3}\cdot A\cdot g_{\text{кл}}-1.4285\cdot\ell'_{\text{м}}= 10^{-3}\cdot 12.97\cdot 0.26-1.4285\cdot 0.00027=0.032$ kg/kg of charge.

Corrected metal output:

$$l'_{\text{м}} = l''_{\text{шл}} + \Delta\, l''_{\text{шл}}=0.7512+0.00027=0.751 \text{ kg/kg of charge.}$$

Slag output:

$$l'_{sl} = l''_{sl} + \Delta l''_{sl} = -0.0118+0.032=0.0202$$ kg/kg of charge.

Consumption indicators are recalculated per unit of metal.

Consumption of the ore part of the charge:

$$P=1/ l'_{м} =1/0.751=1.33$$ kg/kg of metal.

Consumption of coke fines –reagent:

$$K_p=0.26/0.751=0.34$$ kg/kg of metal.

Slag output:

$$\Delta l_{sl}=l'_{sl}/l'_{м}=0.0202/0.751=0.026$$

Volumetric gas output:

$$V_{cp}=1.866\cdot 10^{-2}\cdot C\cdot K_p=1.866\cdot 10^{-2}\cdot 85\cdot 0.34=0.53\ m^3/kg$$ of metal.

To determine the mass amount of gas, it is necessary to calculate the component output. To calculate, use the formulas:

$\upsilon_{CO2}=0,7\Sigma g_{0,i}\cdot \gamma_{01\ i} = 0.7\cdot 1.33(0.0154\cdot 0.974 + 0.0307\cdot 0.7096+ 0.144\cdot 0.40 + +0.00019\cdot 0.974 +0.000095\cdot 0.4 +0.00017\cdot 1+0.045\cdot 0.974+0.039\cdot 0.7096 = 0.19$ m^3/kg of metal.

$\upsilon_{CO}=1,4\cdot P(\Delta O_{,i}\cdot \gamma_{01,i}) = 1.4\cdot 1.33(0.0154\cdot 0.026 + 0.0307\cdot 0.2904 + 0.144\cdot 0.6+ 0.00019\cdot 1+0.000095\cdot 0.112+0.00017\cdot 0.43+0.04176\cdot 1+0.0057\cdot 1 + 0.0453\cdot 0.2904+ +0.039\cdot 0.6 =0.33\ m^3$/kg of metal.

The sum of CO_2 and CO corresponds to V_{cp}

$$\upsilon_{CO2} + \upsilon_{CO} =0.33+0.19=0.52\ m^3/kg$$ of metal.

Volatiles turn into gas RO = CO_2. In terms of volume will be

$$\upsilon_{L\ CO2}=P\cdot 22,4/44\cdot g'_1=22.4/44\cdot 0.035\cdot 1.33=0.023\ m^3/kg$$ of metal.

Total amount of gas

$$V'_{r}=0.52+0.023=0.543\ m^3/kg.$$

Gas composition, %:

$$CO_2=(0.19+0.023)\cdot 100/ V'_g$$

$$CO_2=(0.19+0.023)\cdot 100/0.543=39.2$$

$$CO =0.33\cdot 100/0.543=60.7$$

Specific mass of gas:

$$\gamma'_g=10^{-2}/22.4(44\cdot CO_2+28\cdot CO) = 10^{-2}/22.4(44\cdot 39.2+28\cdot 60.7)=1.528$$

Mass gas output:

$$G_g= V'_g\cdot \gamma'_g= 0.543\cdot 1.528 = 0.82$$ kg/kg of metal.

Heating of the system, reduction of metals with solid carbon, melting and overheating of the charge are accompanied by heat absorption. To provide the system with thermal energy, it is necessary to burn fuel. Such fuel is mainly coke, the consumption of which is determined sequentially according to the articles of the thermal balance. After determining the fuel consumption and flux for fluxing the charge and fuel ash, it will be possible to summarize the material and thermal balance of the process.

Material and thermal balance of melting

At the end of the metallization process we have:

a) metallized product

b) a gaseous product of reducing reactions.

The heat consumption for heating the charge is equal to the sum of the heat content of the gas at the outlet and the metallized product before melting.

The grate gas at the outlet of the furnace has a temperature of 250 ° C.

The heat capacity of CO_2 and CO gases at this temperature according to reference data: with c_{CO2}=1,825; c_{CO} =1.311 kJ/m^3·deg

Heat content of the gas

$q_g = 10^{-2} \cdot V'_g \cdot t_{kg} \cdot (c_{CO2} \cdot CO_2 + c_{CO} \cdot CO) = 10^{-2} \cdot 0.543 \cdot 250 \cdot (1.825 \cdot 39.2 + 1.311 \cdot 60.7) = 152$ kJ/kg

The heat content of the metallized product at t_{ml} = 1500 ºC; c_{ml}=0.965;

$q_{Mp} = l'_{ml}/l'_m \cdot c_{ml} \cdot t_{ml} = 0.7394/0.751 \cdot 0.965 \cdot 1500 = 1425$ kJ/kg of metal.

Heat consumption for heating the system:

$Q_h = q_g + q_{mp} = 152 + 1425 = 1577$ kJ/kg of metal.

Heat consumption for melting and overheating of the melt:

$Q_{ml} = [(377 + c_{sl} \cdot \Delta t)\, l_{sl} + 326 + c_m \cdot \Delta t] = (377 + 1.34 \cdot 150) \cdot 0.026 + 326 + 0.76 \cdot 150 = 455$ kJ

Heat consumption for endothermic reactions:

$Q_{en} = 1.33[0.0154(0.974 \cdot 1436 + 0.026 \cdot 6882) + 0.0307(0.7096 \cdot 6775 + 0.2904 \cdot 1215 + 0.144 \cdot (0.4 \cdot 4550 + 0.6 \cdot 9925) - 0.00019 \cdot 1 \cdot 7750 - 0.000095(0.88 \cdot 3460 - 0.112 \cdot 1940 + 0.00017(0.57 \cdot 1960 + 0.43 \cdot 7360) + 0.045(0 + 17300) + 0.039(0.7096 \cdot 2120 - 0.2904 \cdot 3270)] = 2197$ kJ/kg of metal.

Total heat consumption:

$Q_{\Sigma} = Q_h + Q_{sl} + Q_{en} = 1577 + 455 + 2197 = 4229$ kJ/kg of metal.

The effective calorific value of the fuel, taking into account the smelting of slag from ash and the decomposition of limestone:

$Q_{ef} = Q_{к} - 10^{-2} \cdot A[c_{ash} \cdot t + 10^{-2} \cdot B \cdot SiO_2(c_{lime} \cdot t + q)] = 9800 - 0.13(2144 + 118) = 9505$ kJ/kg

The following amount of blast is consumed for burning 1 kg of coke

$\upsilon_b = 10^{-2} \cdot c_k / 0.536(2r_0 + \varphi) = 0.85/0.536(2 \cdot 0.21 + 0.025) = 3.56$ m^3/kg,

where r_0, φ are the volume fractions of oxygen and moisture in the blast.

Blowing heat saves heat in the furnace.

Blowing heat:

$$I_b = c_b \cdot t_b = 1.46 \cdot 800 = 1168 \text{кJ/m}^3 \text{ blow.}$$

Considering the heat content of the blast determine the consumption of coke-fuel:

$$К = \frac{Q\Sigma}{Qef \cdot \eta + \upsilon \cdot i} = \frac{4229}{0.86 \cdot 9505 + 3.56 \cdot 1168} = 0.35$$

Coke in the form of fuel 0.463 kg and in the form of a reagent 0.2604 kg produces ash in a quantity in which the content of SiO_2=52% and CaO =2.23%.

Determine the consumption of limestone for coke ash and slag-forming ore part of the charge:

$$L = \frac{P(B \cdot SiO2 - CaO) + g(B \cdot SiO2 - CaO)}{CaO - B \cdot SiO2} = 3.325 + 78.82 = 1.57 \text{ kg/kg of metal.}$$

Coke-fuel ash and the residue from the decomposition of limestone are additionally part of the slag:

$$\Delta l_{sl}=10^{-2}\cdot A\cdot 0.35+0.58\cdot 1.57=0.9556 \text{ kg/kg of metal.}$$

Then the total output of slag will be:

$$L_{sl}=0.026+0.9556=0.9816 \text{ kg/kg of metal.}$$

Burning this amount of coke produces the following amount of gas:

$$\upsilon_g=K\cdot \upsilon_l(l+r+\varphi) = 0.35\cdot 3.56(1+0.21+0.025) = 1.53 \text{ m}^3\text{/kg of metal;}$$

with the components:

$$\upsilon_{co}= K\cdot \upsilon_d(2r_0+\varphi) = 0.35\cdot 3.56(2\cdot 0.21+0.025) = 0.55 \text{ m}^3\text{/kg;}$$

$$\upsilon_{n2}= K\cdot \upsilon_d \cdot\varphi=0.35\cdot 3.56\cdot 0.025=0.03 \text{ m}^3\text{/kg;}$$

$$\upsilon_n= K\cdot \upsilon_l(l-r_0-\varphi) = 0.35\cdot 3.56\cdot(1-0.21-0.025)=0.953 \text{ m}^3\text{/kg.}$$

Limestone decomposition additionally produces CO_2:

$$\upsilon_{co2}=22.4/44\cdot 0.42\cdot L=22.4/44\cdot 0.42\cdot 1.57=0.335 \text{ m}^3\text{/kg of metal.}$$

Total amount of gas:

$$\upsilon_{gt}= \upsilon_g+ \upsilon_{co2}=1.53+0.335=1.865 \text{ m}^3\text{/kg of metal.}$$

Gas composition:

$CO=0.55\cdot 100/1.865=29.4$

$H_2=0.03\cdot 100/1.865=1.60$

$N_2=0.95\cdot 100/1.865=51$

$\underline{CO_2=0.335\cdot 100/1.865=18}$

100

Specific gas mass

$$\gamma_{gt}=\frac{10^{-2}}{22.4}(28(CO+ N_2)+2H_2+44CO_2)= \frac{10^{-2}}{22.4}(28\cdot 80.4+3.2+792)=1.34 \text{ m}^3\text{/kg}$$

The mass of the combustion gas:

$$G_{cg}=1.865\cdot 1.34=2.49 \text{ kg/kg of metal.}$$

Blast mass:

$$G_b=3.56\cdot 0.35\cdot 1.29=1.607 \text{ /kg of metal.}$$

Table 2.1 - Total material balance

Expense item	Quantity,kg/kg of metal	Article of arrival	Quantity,kg/kg of metal
Ore part of the charge	1.33	Metal	1.000
Coke reagent	0.34	Slag	0.9816
Coke-fuel	0.35	Gas-prod. combustion	2.490
Blowing	1.607	Gaz-prod.reactions	0.820
Limestone	1.570	Dust	0.406
Total	5.197	Total	5.697

Thermal balance

Heat consumption for melting and overheating of additional slag.

$$\Delta Q_{ml}=0.2453(q_{ml}+c\cdot 1600)=471 \text{ kJ/kg}$$

Heat consumption for limestone decomposition

$$Q_{dec}=L\cdot 10^{-2}\cdot CaO\cdot q_{dec}=1.57\cdot 10^{-2}\cdot 54\cdot 2046=1735 \text{ kJ/kg of metal.}$$

Heat coke burning $Q_h=q_h\cdot K=0.35\cdot 9800=3430$ kJ/kg.

Blowing heat $Q_b=0.35\cdot 3.56\cdot 2268=2825$ kJ/kg.

Table 2.2 - Total thermal balance

Expense item	**Quantity, kJ/kg**	**Article of arrival**	**Quantity, kJ/kg**
Charge heating	1577	The heat of combustion of fuel	3430
Melting	455	Heat content of the blast	2825
Endotherm	2197		
Decomp. of limestone	1735		
Melting of add. slag	471		
Total	6435	Total	6255

Conclusion on the third chapter

-Under the conditions of metal reduction with solid carbon, an accurate calculation of carbon was made to avoid the formation of carbide;

- Total material and thermal balance calculated.

4 Experimental research

4.1 Carrying out experiments on the reduction of pure titanium oxide with solid carbon

As shown in the section of theoretical research, the problem of reduction smelting of ilmenite and titanium-magnetite concentrates is associated with high concentrations of TiO_2. The bulk of these concentrates are iron and titanium oxides. Therefore, the main process of direct reduction of iron and titanium is of interest. In connection with this task, preliminary experiments were performed using chemically pure oxides.

We mixed 50 g of technically pure TiO_2 coals with a stoichiometric amount of charcoal. The mixture with a fraction of less than 1.0 mm is injected into the reaction cell prepared from a quartz tube and installed in the reaction zone of a tubular electric furnace SUOL-44 12m.

Gas separation was expected only in the possibility of the following reactions:

$$TiO_2+C=Ti+2CO \qquad (4.1)$$

$$TiO_2+3C=TiC+2CO \qquad (4.2)$$

Table 4.1 – Results of the experiment on the reduction of pure titanium oxide with solid carbon

Time, min	T, °C	V_{gas}, dm^3	Gas composition, %		$\Delta O_{(CO)}$, g	$\Delta O_{(CO_2)}$, g	, ΔO i, g	ΔO , g	R	$W_B \cdot 10^{-3}$, min^{-1}
			CO_2	CO						
0	600	The beginning of gas release								
22	750	0.4	23	77	0.218	0.131	0.349	0.349	0.025	0.001
15	870	0.9	44	56	0.359	0.565	0.924	1.273	0.093	0.004
13	950	0.85	22	78	0.473	0.267	0.740	2.013	0.148	0.004
13	1010	1.0	12	88	0.628	0.171	0.799	2.812	0.207	0.004
10	1050	1.0	9	91	0.649	0.128	0.777	3.589	0.264	0.005
10	1090	1.6	7	93	1.062	0.160	1.222	4.811	0.354	0.009
10	1120	1.5	3	97	1.039	0.064	1.103	5.914	0.435	0.008
8	1140	1.4	7	93	0.929	0.139	1.068	6.982	0.514	0.009
4	1150	1.3	5	95	0.882	0.092	0.974	7.956	0.586	0.018
7	1170	1.55	5	95	1.051	0.110	1.161	9.117	0.672	0.012
6	1160	1.2	3	97	0.831	0.051	0.882	9.999	0.737	0.010
7	1150	1.2	2	98	0.839	0.034	0.873	10.872	0.801	0.009
7	1150	1.2	5.5	94.5	0.809	0.094	0.903	11.775	0.867	0.009
7	1150	1.0	1.5	98.5	0.703	0.021	0.724	12.499	0.92	0.007
7	1150	0.75	4	96	0.514	0.042	0.556	13.055	0.961	0.006
19	1150	0.45	3	97	0.311	0.019	0.430	13.485	0.99	0.001

The system temperature was increased to 1000°C at a speed of 10-12 °C/min. The laboratory furnace, where an isolated cell is installed, is connected to a gas meter and a gas analyzer. No gas release was observed when the temperature was increased.

When the temperature reaches 1000°C, we keep the system at this temperature for 20 minutes. However, no gas release was observed either during heating or when the temperature was maintained at 1000°C, which indicates the absence of a clear picture of titanium reduction reactions.

4.2 Carrying out experiments on the reduction of iron oxides and pure TiO_2 titanium oxides with solid carbon

Experiments were carried out using iron oxides in the form of rolled okalina and TiO_2 in the mass ratio of 70 g ÷ 30 g, respectively. I mixed the mixture of oxide components with the stoichiometric consumption of charcoal. I introduced an oxide-carbon mixture with a fraction of less than 1.0 mm into the test cell and placed it in the reaction zone of the furnace SUOL-044 12M2. According to the first experiment, during the rapid heating of the system, the release of gas was observed when the temperature reached 600 ° C. The gas analysis showed that the emitted gas was a product of the process of direct reduction of the metal - CO and CO_2 gases. As the temperature increases, the rate of gas evolution increases, which reaches a maximum at a temperature of 1000-1050 ° C. Then the temperature is set at 1150 ° C for 30 minutes. I held on all the time. Over time, the rate of gas evolution decreased and dropped to zero by the end of the metallization process, indicating the completion of iron reduction. As for titanium, at this temperature only a small part of it is reduced.

Samples of metallized products were completely cooled, removed from the cell, and melted in a crucible in the High temperature tubular furnace. The analysis of molten metal samples is shown in table 4.2.

Table 4.2 - Chemical composition of molten metal

Name	Chemical composition, %						
	[S]	[P]	[C]	[Ti]	[Cr]	[Mn]	[Si]
Test №1	0.014	0.095	0.31	0.29	0.24	0.27	0.3
Test №2	0.015	0.08	0.3	0.3	0.24	0.27	0.51
Test №3	0.015	0.08	0.3	0.29	0.23	0.26	0.6

It is proved that the degree of reduction of metals is determined by the amount of gaseous oxygen, due to the fact that the process of reduction of metals takes place through the conversion of oxygen from metal oxides to carbon with gas.

Table 4.3 – Results of the experiment on the reduction of iron oxides and pure TiO_2 titanium oxides with solid carbon

Time, min	T, °C	V_{gas}, dm^3	Gas composition, %		$\Delta O_{(CO)}$, g	$\Delta O_{(CO_2)}$, g	ΔO i, g	ΔO , g	R	$W_B \cdot 10^{-3}$, min^{-1}
			CO_2	CO						
0	600	The beginning of gas release								
13	787	1.9	40	60	0.814	1.085	1.899	1.899	0.104	0.008
14	890	1.1	37	63	0.494	0.581	1.075	2.974	0.162	0.004
8	940	1	41	59	0.421	0.585	1.006	3.980	0.217	0.006
5	970	1.2	35	65	0.557	0.599	1.156	5.136	0.28	0.012
7	1000	1.6	27	73	0.834	0.617	1.451	6.587	0.359	0.011
6	1025	1.75	26.5	73.5	0.918	0.662	1.580	8.167	0.445	0.014
5	1045	2.1	22	78	1.169	0.659	1.828	9.995	0.545	0.02
4	1060	1.7	21	79	0.959	0.509	1.468	11.463	0.625	0.02
3	1070	1.75	24	76	0.949	0.599	1.548	13.011	0.709	0.028
5	1085	1.8	20.5	79.5	1.022	0.527	1.549	14.56	0.793	0.018
3	1095	1.6	19	81	0.925	0.434	1.359	15.919	0.867	0.024
3	1105	1.4	17	83	0.829	0.339	1.168	17.087	0.931	0.021
4	1115	1.4	16	84	0.839	0.319	1.158	18.245	0.99	0.014

The amount of oxygen in the prepared carbon charge in the form of oxides with Fe, Cr, V, Ti and Mn is 99% of total oxygen. In the temperature range of 1150-1200ºC the content of carbonated oxygen is 85-87%, ie iron oxides in the charge are completely reduced, and oxides Cr, Mn, V are reduced from high oxides to low oxides.

The maximum value of CO_2 in the exhaust gas is observed for solid carbon at a temperature of 850-950 ° C, at temperatures above 950 ° C the normal content of this gas gradually decreases, as explained by the completion of the reduction of high Fe oxides.

The exact extremum of the total reduction rate depends on the phase and temperature, but in the temperature range of 900-1050 ° C the reduction rate doubles and reaches its minimum value at 1165 ° C. This indicates a decrease in the rate of reduction due to the completion of the reduction of Fe.

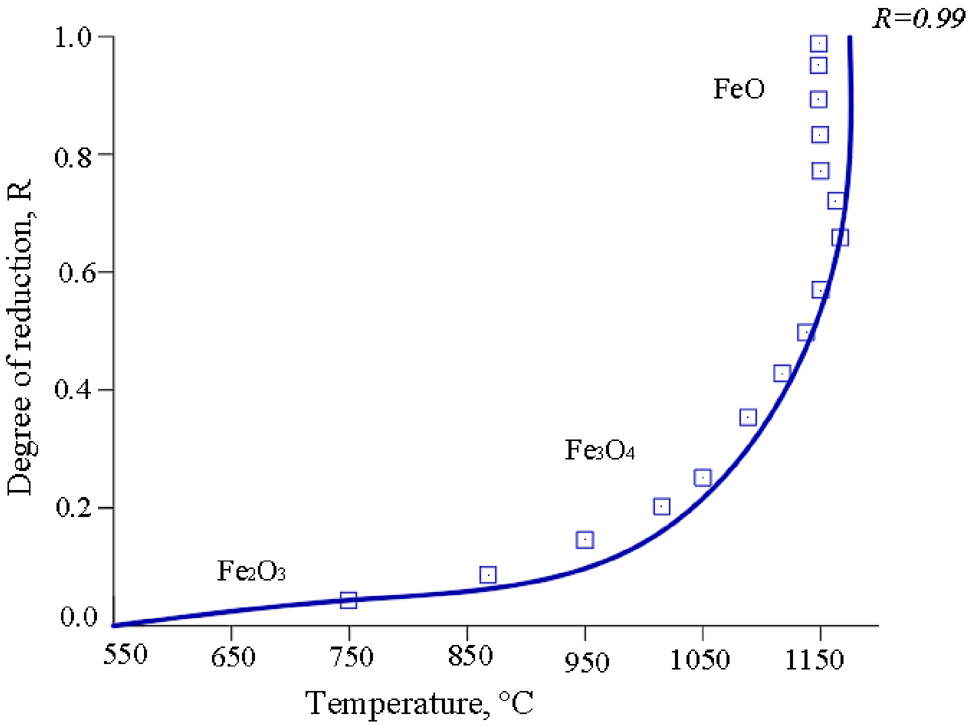

Figure 4.1- Temperature dependence of Fe reduction

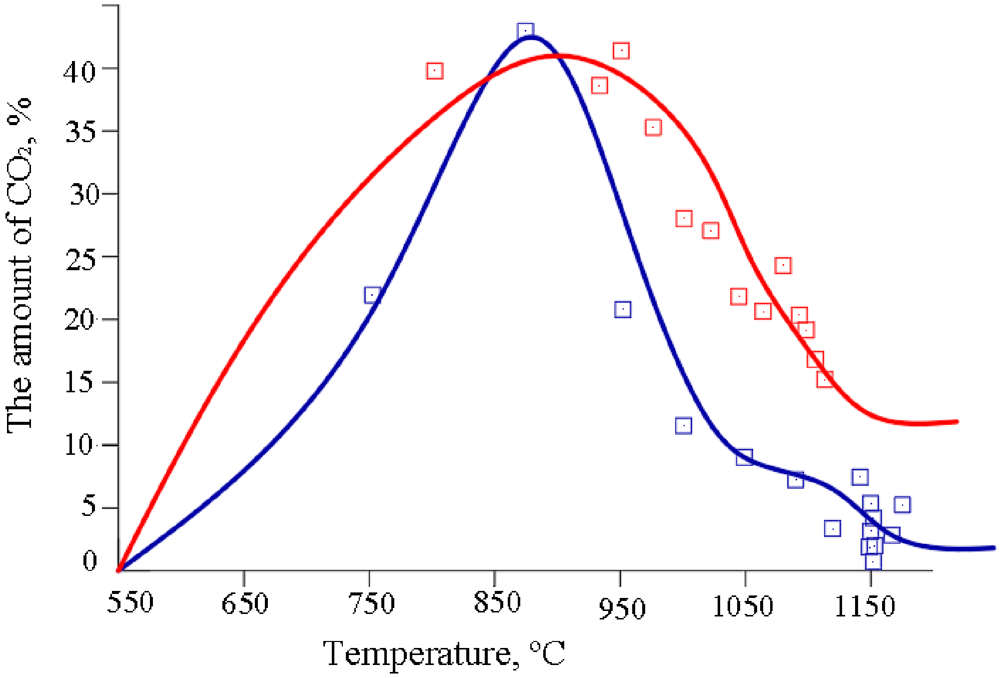

Figure 4.2 - Temperature dependence of CO_2 emissions

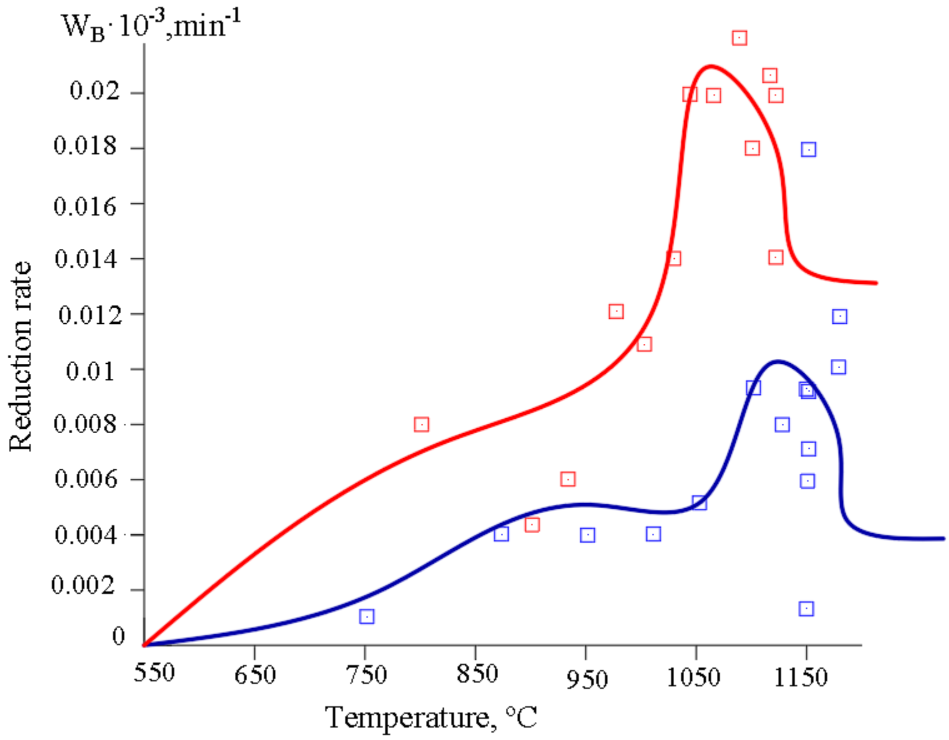

Figure 4.3 - Temperature dependence of the reduction rate

4.3 The effect of temperature and duration of heat treatment on the degree of reduction of iron oxide

In order to mathematically process the results obtained during the experiment, a special program was developed to calculate the key indicators needed to build forecasting models. Thus, the results of experimental data for the mathematical processing of a complete array are given in table 4.4, where, as an example, the results of a selected experimental array are given.

As a result of mathematical processing of the experimental array, a regression equation was obtained, which equivalently describes the dependence of the iron oxide reduction process on the temperature:

$$[R] = -1.263 + 0.02 \cdot [T],\ r = 0.81 \qquad (4.3)$$

where [R] is the degree of reduction,

[T] – temperature, K.

Average value of the multiple correlation coefficient ($r = 0.81$)

Table 4.4 - Results of experimental data for mathematical processing of the selected array

Temperature,°C	R	W	CO/CO_2
X	Y_1	Y_2	Y_3
600	0	0	0
750	0.025	0.001	3.3
870	0.093	0.004	1.3
950	0.148	0.004	3.5
1010	0.207	0.004	7.3
1050	0.264	0.005	10.1
1090	0.354	0.009	13.3
1120	0.435	0.008	32.3
1140	0.514	0.009	13.3
1150	0.586	0.018	19.0
1170	0.672	0.012	19.0
1160	0.737	0.01	32.3
1150	0.801	0.009	49.0
1150	0.867	0.009	18.9
1150	0.920	0.007	49.3
1150	0.961	0.006	24.0
1150	0.990	0.001	32.3

Table 4.5 - Regression statistics

Multiple R	0.813754
R-square	0.662196
Normalized R-square	0.639675
Standard mistake	0.20574
Observations	17

Table 4.6 - Analysis of variance

	df	*SS*	*MS*	*F*	*Significance of F*
Regression	1	1.244662	1.244662	29.4044	7.06E-05
Remains	15	0.634936	0.042329		
Total	16	1.879598			

Table 4.7 - Conclusion of results

	Coefficients	Standard mistake	*t-statistics*	*P-Value*	*Lower 95%*	*Upper 95%*	*Lower 95.0%*	*Upper 95.0%*
Y-intersection	-1.2627	0.329667	-3.83022	0.001639	-1.96536	-0.56003	-1.96536	-0.56003
Variable X 1	0.001687	0.000311	5.422583	7.06E-05	0.001024	0.00235	0.001024	0.00235

4.4 X-ray fluorescence analysis of a steel sample

X-ray fluorescence analysis of a steel sample (XRFA) is one of the modern spectroscopic methods of studying a substance to obtain its elemental composition, i.e. elemental analysis. You can use it to find various elements. The XRFA method is based on the collection and analysis of the spectrum that occurs during X-ray irradiation of the studied material. When interacting with high-energy photons, the atoms of a substance go into a state of shock, which is expressed in the transition of electrons from a lower Orbital to a higher energy level even before atomic ionization. The atom is in an excited state for a short time, about one microsecond, after which it returns to a calm state (its main state). At this point, the electrons of the outer shell fill the resulting gaps, and the excess energy is either released as a photon, or the energy is transferred to another electron of the outer shell. Here, each atom emits an energy photon that has a strictly defined value, for example, iron emits $K\alpha = 6.4$ kV photons when irradiated with X-rays. Further, the structure of matter is determined by the amount of energy and quantum, respectively. X-ray tubes or isotopes of certain elements can be used as radiation sources. Since each state has its own requirements for the input and output of radiation isotopes, it has recently been trying to use an X-ray tube in the production of X-ray fluorescent equipment.

After entering the detector, the photon is converted into a voltage pulse, which is calculated by computing electronics and the value is transmitted to the computer. Based on the obtained spectrum peaks, it is possible to qualitatively identify the elements present in the sample. The X-ray fluorescence method is widely used in industry and scientific laboratories. Due to its simplicity, the possibility of rapid analysis, accuracy, and the absence of complex sample preparation, the areas of its application are expanding.

The composition of the charge was formed at the following mass ratios of Satpayev concentrate and rolling scale: 0.50-0.50; 0.40-0.60; 0.30-0.70. in addition, a small proportion of burnt lime ($CaO = 93\%$) was introduced into the charge to ensure the liquid mobility of the slag.

To conduct experimental studies, the following components were used, the chemical compositions of which are presented in table 4.8.

Table 4.8 – Chemical composition of the components used for experimental studies

Name of materials	Chemical composition, %													
	Fe	FeO	MnO	SiO_2	CaO	MgO	V_2O_5	TiO_2	Cr_2O_3	Al_2O_3	P	S	C	w.r.
Satpayev concentrate	32.53	16.54	0.70	5.80	0.50	0.33	0.22	49.65	0.22	1.25	-	0.017	-	7.73
Rolling scale	67.65	28.25	0.30	-	-	-	-	-	-	-	-	0.017	-	-
Charcoal	-	-	-	-	-	-	-	-	-	-	-	0.37	98.0	-

The weighted average chemical composition of the mixture of Satpayev concentrate and rolled scale in the specified mass ratios are presented in table 4.9.

Table 4.9 – Weighted average chemical composition of the charge at the specified mass ratios of Satpayev concentrate and rolling scale

Mass ratio	Chemical composition, %												
	Fe	FeO	MnO	SiO_2	CaO	MgO	V_2O_5	TiO_2	Cr_2O_3	Al_2O_3	P	S	w.r.
0.50-0.50	50.08	22.40	0.50	2.9	0.25	0.165	0.11	24.82	0.14	0.625	-	0.017	3.86
0.40-0.60	53.60	23.56	0.46	2.32	0.20	0.13	0.088	19.86	0.088	0.50	-	-	3.09
0.30-0.70	57.11	24.74	0.42	1.74	0.15	0.10	0.066	14.89	0.066	0.37	-	0.016	2.32

The complete reduction of iron from the presented composition of the charge was realized by the introduction of free carbon in the form of charcoal at a heating temperature of up to 1100 ° C.

Calculated data

Determination of the amount of gasified oxygen of the charge and the consumption of carbon as a reducing agent at a temperature of 1100 ° C. According to preliminary studies, it is known that vanadium oxide (V_2O_5) is reduced by 85%, and chromium oxide (Cr_2O_3) 80%,

$$g'_{C,\,Fe} = 0.75\cdot[0.333\cdot Fe - 0.076\cdot FeO + 0.170\cdot MnO + 0.253\cdot Cr_2O_3 + 0.374\cdot V_2O_5 + \\ + 0.03\cdot TiO_2]/95$$

Test №1

$$g'_{C,\,Fe} = 0.75\cdot[0.333\cdot 50.08 - 0.076\cdot 22.40 + 0.170\cdot 0.5 + 0.253\cdot 0.14 + 0.374\cdot 0.11 + \\ + 0.03\cdot 24.82]/95 = 0.12 \text{ g/g charge}$$

Test №2

$$g'_{C,\,Fe} = 0.75\cdot[0.333\cdot 53.6 - 0.076\cdot 23.56 + 0.170\cdot 0.46 + 0.253\cdot 0.088 + 0.374\cdot 0.088 + \\ + 0.03\cdot 19.86]/95 = 0.135 \text{ g/g charge}$$

Test №3

$$g'_{C,\,Fe} = 0.75\cdot[0.333\cdot 57.11 - 0.076\cdot 24.74 + 0.170\cdot 0.42 + 0.253\cdot 0.066 + 0.374\cdot 0.066 + \\ + 0.03\cdot 14.89]/95 = 0.142 \text{ g/g charge}$$

Results of experimental studies (first stage of metallization with free carbon)

Test №1

Tube - 211.3 g.; Total weight - 271.3 g.

The composition of the charge: 50% mill scale + 50% Satpayev concentrate + 5.6% CaO + 12% C (charcoal).

Charge weight - 60 g. The weight of the metallized product is 40.5 g.

Table 4.10 – Results of test №1

Time	Temperature, ° C	CO_2 content	V, dm^3
15.50	600	Start of gas release	
16.28	660-811	23	150
17.06	900	26	1200
17.15	950	15	1800
17.29	1000	2	3100
17.44	1050	4	2575
18.03	1100	0	2050
18.14	1100	0	500

Test №2

Tube - 196.1 g.; Total weight -256.1 g.

The composition of the charge: 40% mill scale + 60% Satpaevsky concentrate + 5.7% CaO + 13.4% C (charcoal).

Charge weight - 60 g. The weight of the metallized product is 40.0 g

Table 4.11– Results of test №2

Time	Temperature, ° C	CO_2 content	V, dm^3
15.00	600	Start of gas release	
15.14	700-800	26	300
15.50	870	55	1100
15.55	900	54	2500
16.08	950	27	3630
16.29	1000	22	2200
16.54	1050	12	1500
17.25	1100	0	254

Test №3

Tube - 211.3 g.; Total weight - 271.3 g.

Charge composition: 70% mill scale + 30% Satpayevskiy concentrate + 5% CaO + 14% C (charcoal)

Charge weight - 60 g. The weight of the metallized product is 36.6 g.

Table 4.12– Results of test №1

Time	Temperature, ° C	CO_2 content	V, dm^3
16.24	600	Start of gas release	
16.30	800	16	650
16.46	900	55	850
16.59	950	18	1565
17.12	1000	18	3800
17.31	1050	10	4400
18.54	1100	0	1300
18.20	1100	0	200

The reduction of titanium was organized from a metallized charge at a higher temperature of 1150-1500 ºC with the additional introduction of dissolved carbon in the form of cast iron granules.

In this case, it is possible to plan and achieve the degree of reduction of titanium metal at the level of 0.2-0.3. In accordance with reaction, the consumption of dissolved carbon for the restoration of a unit mass of titanium will be (g/g or kg/kg).

Test №1

Figure 4.4 - Smelted metal samples (second stage of metallization with dissolved carbon)

Thermodynamic studies

The effect of surface atoms with carbon due to 5-6% O_2.

Mechanical properties of the obtained samples

at T = 20°C.
Test 1
5 mm-S_V =1500 MPa
20 mm-S_V = 980 MPa
H_B 10^{-1} = 217 MPa
S_v - the limit of short-term strength
H_B - Brinell hardness

According to the results of experimental data, the obtained sample is similar in characteristics to steel 18XGT and 30XGT.

Sample 1 (not the shiny side)

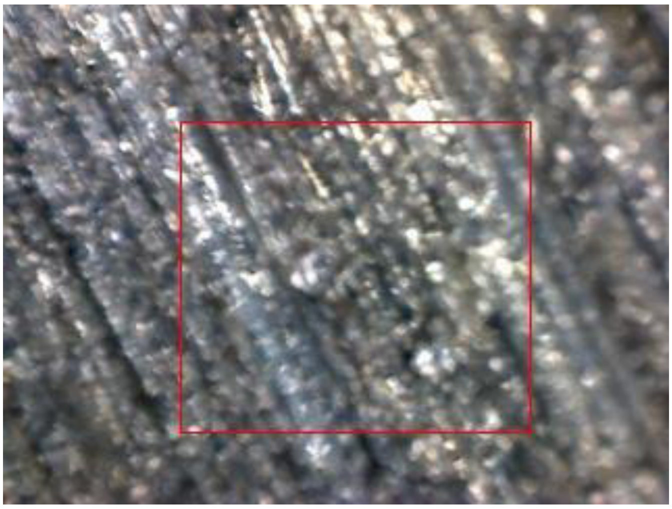

Figure 4.5 - The shooting area. Magnification X400.

The intensity graph is built in a logarithmic scale along the intensity axis to fit all the intensity values.

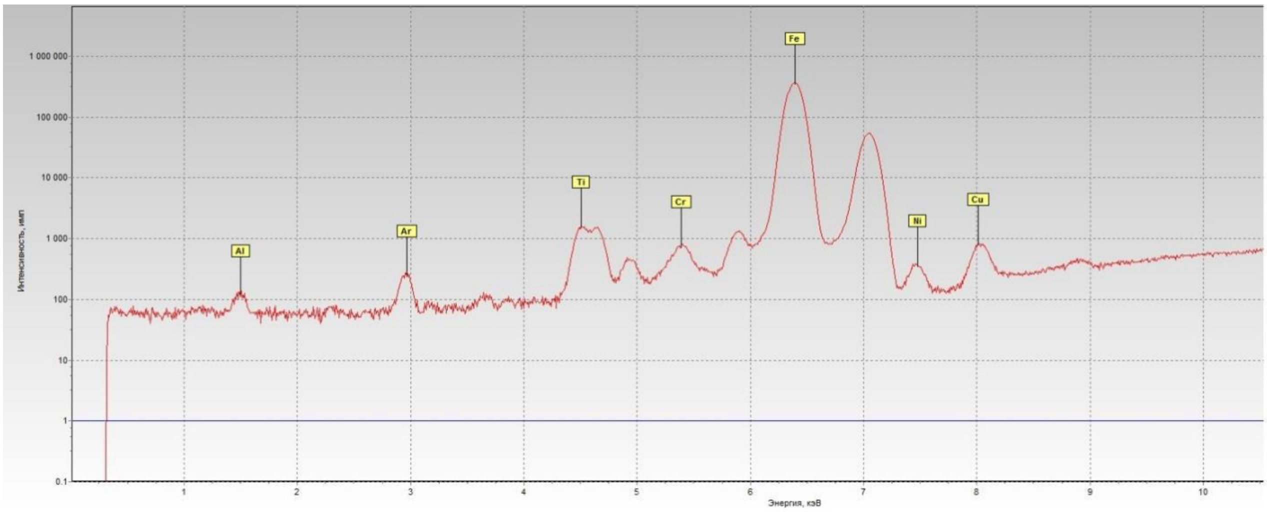

Figure 4.6 – Energy spectrum of the metal sample after solid phase reduction

The Argon line is because of the carrier gas. It is not taken into account. The unsigned peaks are the lines from the betta lines of the elements.

Qualitative analysis: Ti, Cr, Fe, Ni, Cu.

Table 4.13 Quantitative analysis of this sample taking into account manganese

Elements:	Ti	Cr	Fe	Ni	Cu	Mn
Concentrations:	0.6	0.3	98.3	0.1	0.4	0.3

CONCLUSION

An overview of the technologies for processing titanium-magnetite concentrates was presented, their achievements and prospects were demonstrated. Taking into account such difficulties of well-known technologies, the dissertation work has developed a methodology for processing titanium-magnetite concentrate with a new technology.

A method of direct reduction of iron and chemically high-strength metal oxides in titanium-magnetite concentrate with solid carbon has been developed. The main feature of the methodology is mixing carbon with titanium-magnetite oxide concentrate in stoichimetric quantities and grinding the mixture to a fraction of 1.0 mm. In this case, the course of the reduction process has shifted from a topochemical (diffusion) mode to a kinetic one.

It was found that the kinetic indicators of the process of chain-Phase reduction of iron and alloying metals with solid carbon are more effective in comparison with the indicators of the process of reduction of lump raw materials with gas.

The process of direct reduction of iron and alloying metals with solid carbon was carried out at 1100-1150 °C and a high-quality metallized product was obtained.

Natural alloy steel was obtained by reducing and melting metallized products in a High temperature tubular furnace at temperature of 1600-1650 °C.

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ТИТАНДЫ БОЛАТТАҒЫ ҚОСЫНДЫЛАРДЫ 3-D ӘДІСІМЕН ЗЕРТТЕУ

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Аңдатпа. Әртүрлі болаттан жасалған соңғы өнімдердің сапасы мен қасиеттері олардың құрамына, құрылымына және болаттардағы металл емес қосындылардың (МЕҚ) сипаттамаларына байланысты. Дегенмен, кейбір қосындылар болаттың соңғы қасиеттеріне өте зиянды болуы мүмкін, ал сол болатта болатын басқа МЕҚ түрі болат өнімінің қажетті құрылымын қалыптастыру үшін пайдалы. Көбінесе металл емес қосындылардың болат сапасына әсері олардың құрамы, морфологиясы, өлшемі, саны, болаттағы орналасуы және т.б. Сондықтан қазіргі болат өндірісінің маңызды мақсаттарының бірі болат балқытудың әртүрлі кезеңдерінде әртүрлі қосындылардың түзілуін, өсуін және жойылуын зерттеу негізінде әртүрлі металл емес қосындылардың сипаттамаларын оңтайландыру болып табылады (зиянды МЕҚ-ды азайту және пайдалы МЕҚ-ды барынша арттыру). Сондықтан болат балқытудың әртүрлі кезеңдерінде болаттардағы металл емес қосындыларды (МЕҚ) бақылау жоғары сапалы болаттарды алудың маңызды мақсаттарының бірі болып табылады.

Негізгі сөздер. Титанды болат, МЕҚ, 2-D және 3-D модельдеу әдісі.

Кіріспе. Бүгінгі күні болаттар мен қорытпалардағы МЕҚ бағалаудың ең кең тараған әдісі металл үлгілерінің жылтыратылған беттеріндегі металл емес қосындыларды екі өлшемді (2-D) зерттеу болып табылады. Дегенмен, бұл 2-D зерттеу әдісінің кейбір маңызды кемшіліктері бар. Алдыңғы бірқатар зерттеулер [1-2] болат үлгілерін электролиттік ерітуден кейін пленка сүзгісінің бетіндегі және/немесе металл бетіндегі МЕҚ және кластерлердің үш өлшемді (3-D) зерттеулерінде бірқатар маңызды көрсеткіштер бар екенін көрсетті, 2-D әдістерімен салыстырғанда артықшылықтар бар. Бүгінгі таңда электролиттік экстракция (ЭЭ), одан кейін 3D МЕҚ және сканерлеуші электрондық микроскопты (СЭМ) пайдаланып кластерлік зерттеулер 0,05-тен 500 мкм-ге дейінгі өлшем диапазонындағы әртүрлі болат үлгілеріндегі қосындылар мен кластерлерді бағалаудың дәл әдістерінің бірі болып табылады. Сондықтан, ЭЭ әдісі болат өндірудің әртүрлі кезеңдерінде болаттың әртүрлі коммерциялық маркаларындағы әртүрлі МЕҚ және кластерлерді дәл 3D бағалау үшін әртүрлі ғылыми жұмыстар мен жобаларда кеңінен қолданылады. Алынған нәтижелерді болат балқытудың технологиялық процестерін оңтайландыру және модельдеу және әртүрлі өнеркәсіптік болаттар мен қорытпалардың соңғы қасиеттеріне (механикалық қасиеттері, өңдеуге қабілеттілігі, коррозияға төзімділігі және т.б.) әртүрлі МЕҚ және кластерлердің әсерін бағалау үшін пайдалануға болады.

Осыған байланысты, құйылған күйдегі болатта металл емес қосындылардың пішінін өзгерту, көлемі мен өлшемін азайтуға бағытталған шаралар, болат өндірісінде олардың сапасын арттыру үшін маңызды тәжірибелік мәнге ие.

Болаттың металл емес қосындылармен ластану дәрежесін аса сенімді сандық бағалау мен олардың сапасын бағалау үшін, соңғы жылдары үш өлшемді (3D) зерттеу әдістемесі ұсынылған. Бұл әдістеменің мәні, металл емес қосындыларды бөліп алу үшін электролиттік экстракцияны пайдалану, кейіннен тұнбаға түскен әртүрлі фазалы қосындылар бөлшектерін СЭМ-ге зерттеу болып келеді. Бейметалл қосындыларын зерттеудің мұндай әдістемесі аса келешегі зор, сенімді және Жапония, Оңтүстік Корея, АҚШ пен Швецияның жетекші зертханаларында кеңінен пайдаланылады [3].

Болаттың сапалық көрсеткіштеріне кері әсерін тигізетін металл емес қосындылар болат үлгілерінде әртүрлі пішінде болуы мүмкін. Олардың өлшемі бірнеше микроннан миллиметрдің бірнеше үлесіне дейін өзгереді.

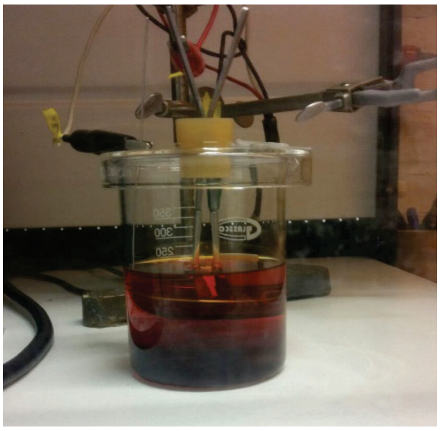
Тәжірибе. Сынақ алдында зерттелетін болат үлгісінің [4] бетін тотық қабатынан, кірден жәнешаңнан тазартылды. Металл үлгіні дыбыстық толқындар әсерімен ацетонмен, кейіннен бензинмен шайылды.

Электролиттік экстракцияда зерттеліп отырған үлгінің еру жылдамдығын анықтау үшін, үлгінің өлшемін (12,03 х 10,04 х 8,75 мм) өлшеп және үлгінің салмағы (6,7753 гр.) анықтадық. Металл сынамасын дәлдік үшін метанолмен шайдық, аспапқа бекітіп және ауа ағынымен кептірілді.

Сурет 1.1 - Құрамы 3D-әдісімен зерттелген балқытылып алынған болат үлгісі

Сурет 1.2– Үлгінің өлшемдерін өлшейтін аспап

Сурет 1.3 – Зерттелетін металл үлгісінің салмағы



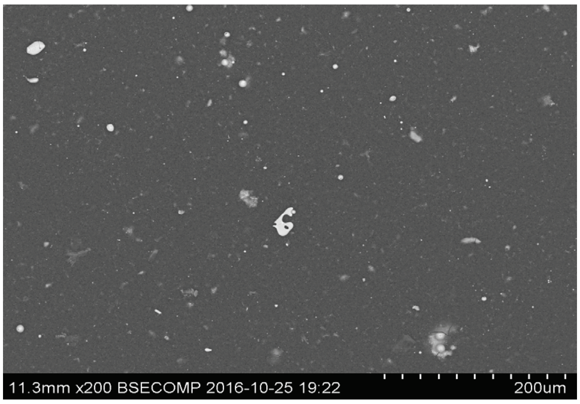
Сурет 1.4 - Еріту әдісімен металл үлгілерін зерттеуге арналған құрылғы

Қондырғыға бекітілген үлгіні арнайы электролит 10% АА сұйықтықтың беттік деңгейінен шамамен 3,5 мл төмен батырып және қондырғыны іске қостық (сурет 1.4). 45-60 mA тоқ пен 150 мВ кернеу (800 немесе 1200 кулон зарядымен) беру кезінде зерттеліп отырған үлгінің металдық негізінің еруі жүреді. Болат үлгісі ериді, ал қосындылар шыны ыдыстың түбіне шөгеді.

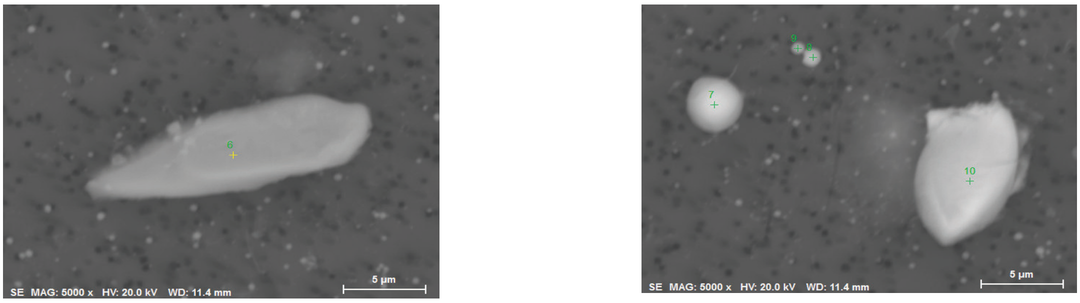
Нәтижелерді талдау. Электролиттік ерітуден кейін, тұнбаны поликарбонат сүзгіші (0,05-5мк өлшемді ашық кеуектері бар қабықша) арқылы өткізеді. Мембраналы қабықшалы сүзгіштер бетіне бөлініп алынған қосындыларды (металл және металл емес) СЭМ көмегімен 300 – 10000 дейін үлкейту кезінде зерттейміз (сурет 1.5). Зерттеліп отырған тұнба бетінің жалпы ауданы шамамен 0,6 м^2 дейін болады.

Зерттеліп отырған металл үлгісінің салмағы экстракция барысында 6,7753 гр-нан 6,6574 гр. дейін азайды, демек, 0,118 гр. металл еріді. Бұл кезде сүзгіш бетінде 0,118 гр бейметалл қосындыларының тұнбасы түзілді. Фазалық құрамы бойынша қосындылар легірлеуші металдарға жатады.

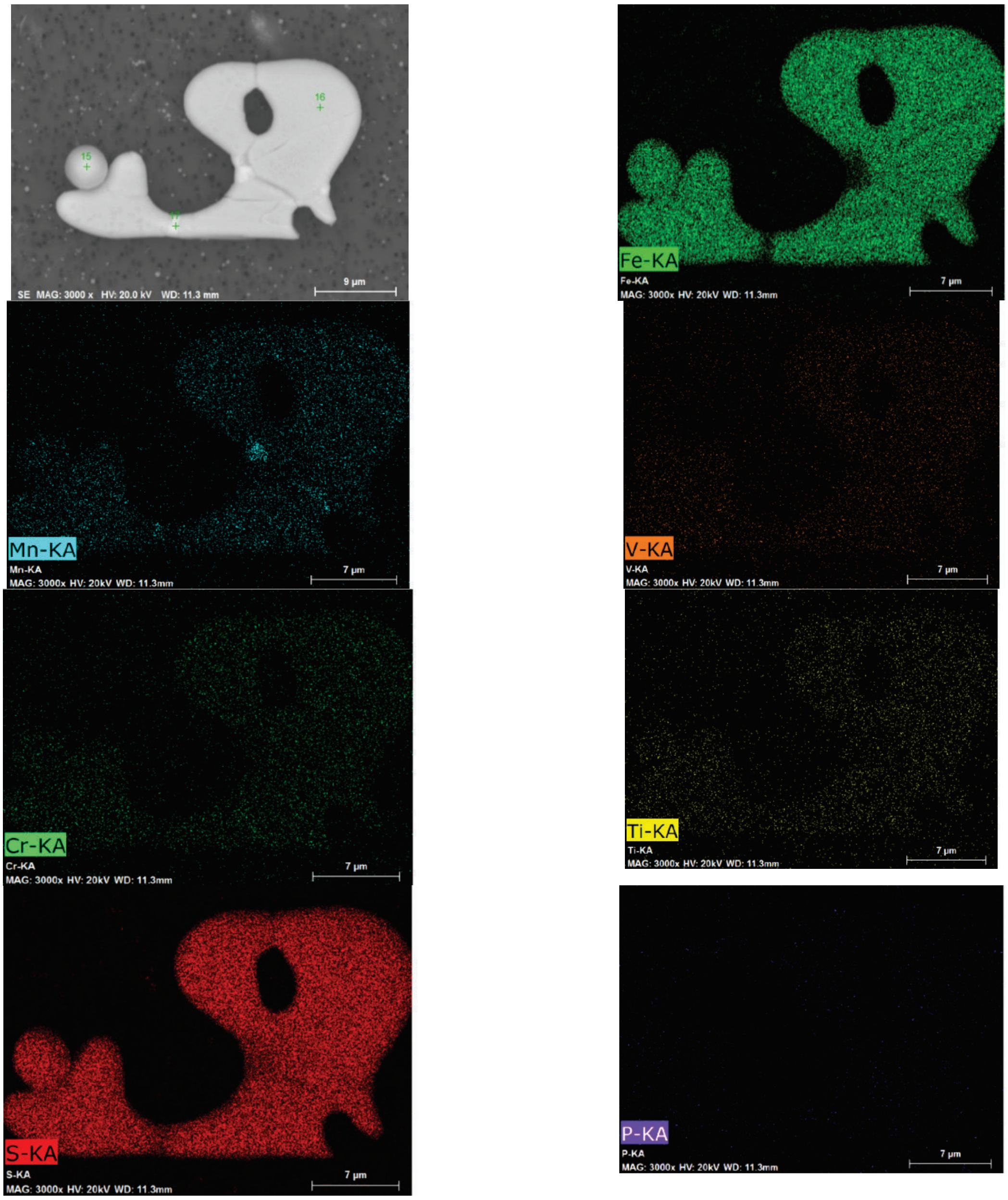
1.6 суретте табиғаты әртүрлі қосындылардың морфологиясын көруге болады.



Сурет 1.5 –Электролиттік экстракциялаудан кейін сүзгіш қағаз бетіндегі кластерлер қосындыларының типтік СЕМ-кескіні, х200



Сурет 1.6 – Болат үлгідегі морфалогиясы әртүрлі титан қосындыларының морфологиясы, х5000



Сурет 1.7 – Болат үлгідегі х3000 еседүлкейтілген қосындылар морфологиясы және қосындылардағы элементтердің таралуы

СЭМ деректерін өңдеу көрсетуі бойынша, зерттелген болат үлгісі металл және металл емес қосындыларымен ластанған. Пішіні әртүрлі көптеген, кейде дөрекі, тікенек тәрізді қосындылар кездеседі.

Зерттеу нәтижелері, сканерлеу электронды микроскопты қолданып үшөлшемді (3D) зерттеу әдісінің тиімділігін көрсетеді. Әдіс - ластану дәрежесінің сенімді сандық сипаттамаларын алуға және болат сапасын бағалауға мүмкіндік береді. Мұндай зерттеулер, қорытпаларды тазалау, легірлеу, балқыту және құю процестерінің технологиясын оңтайландырудың ғылыми-негізделген тәсілдемесі үшін негіз болып келеді.

Кешенді титан-магнетит және редукциялаушы реагенттен тұратын шихтадан балқытылып шығарылған, табиғи легірленген болаттың металдық негізін электролиттік еріту әдісімен анықталған металл емес қосындылардың химиялық құрамы, пішін, өлшемі мен салыстырмалы мөлшері зерттелді. Металдық материалдарда металл емес қосындыларды электролиттік бөліп алу әдісін және алған тұнбаны сканерлеу электронды микроскобында зерттеу әдісін үйлестіру тиімді.

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3D-исследования включений в титановых сталях

Ш. Галипаулы , Г.М. Койшина

Аннотация. Качество и свойства конечных изделий из различных сталей зависят от их состава, структуры и характеристик неметаллических включений (НМИ), представленных в сталях. Однако некоторые включения могут быть очень вредными для конечных свойств стали, в то время как другой тип НМИ, представленный в той же стали, полезен для формирования требуемой структуры стального изделия. Часто влияние неметаллических включений на качество стали зависит от их характеристик, таких как состав, морфология, размер, количество, расположение в стали и т.д. Поэтому одной из важных целей современного производства сталей является оптимизация характеристик различных неметаллических включений (минимизация вредных НМИ и максимизация полезных НМИ) на основе исследований образования, роста, удаления различных включений на разных стадиях процессов выплавки стали. Поэтому контроль неметаллических включений (NMI) в сталях на различных стадиях выплавки стали является одной из важных целей для производства высококачественных сталей.

Ключевые слова. Титановая сталь, НМВ, 2-D и 3-D модели

3D investigations of inclusions in titanium steels

Sh. Galipauly , G. M. Koishina

Abstract. The quality and properties of the final products made of various steels depend on their composition, structure and characteristics of non-metallic inclusions (NMI) present in the steels. However, some inclusions can be very harmful to the final properties of steel, while another type of NMI, represented in the same steel, is useful for forming the required structure of a steel product. Often the influence of non-metallic inclusions on the quality of steel depends on their characteristics, such as composition, morphology, size, quantity, location in steel, etc. Therefore, one of the important goals of modern steel production is to optimize the characteristics of various non-metallic inclusions (minimizing harmful NMIs and maximizing useful NMIs) based on studies of the formation, growth, and removal of various inclusions at different stages of steelmaking processes. Therefore, the control of non-metallic inclusions (NMI) in steels at various stages of steelmaking is one of the important goals for the production of high-quality steels.

Keywords. Titanium steel, NMI, 2-D and 3-D models

"МАТЕРИАЛТАНУ ЖӘНЕ ЖАҢА МАТЕРИАЛДАРДЫҢ ТЕХНОЛОГИЯСЫ" БӨЛІМІ

"МЕТАЛЛУРГИЯЛЫҚ ИНЖЕНЕРИЯ" БӨЛІМІ

Протокол

о проверке на наличие неавторизованных заимствований (плагиата)

Автор: Galipauly Shyngys

Соавтор (если имеется):

Тип работы: Магистерская диссертация

Название работы: Research and development of ways to improve the efficiency of processing of titanomagnetites

Научный руководитель: Гүлзада Қойшина

Коэффициент Подобия 1: 0.9

Коэффициент Подобия 2: 0

Микропробелы: 2

Знаки из здругих алфавитов: 14

Интервалы: 0

Белые Знаки: 0

После проверки Отчета Подобия было сделано следующее заключение:

☑ Заимствования, выявленные в работе, является законным и не является плагиатом. Уровень подобия не превышает допустимого предела. Таким образом работа независима и принимается.

☐ Заимствование не является плагиатом, но превышено пороговое значение уровня подобия. Таким образом работа возвращается на доработку.

☐ Выявлены заимствования и плагиат или преднамеренные текстовые искажения (манипуляции), как предполагаемые попытки укрытия плагиата, которые делают работу противоречащей требованиям приложения 5 приказа 595 МОН РК, закону об авторских и смежных правах РК, а также кодексу этики и процедурам. Таким образом работа не принимается.

☐ Обоснование:

Дата 31-05-2022

Заведующий кафедрой МиОПИ

Бәрменшинова М.Б.

Протокол

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☐ Обоснование:

Дата

25.05.2022 г.

Khoishina

проверяющий эксперт

Қойшина Г.М.

библиографический список также соответствуют всем требованиям, каждую главу сопровождают выводы.

5. Оценка внутреннего единства и направленности полученных результатов на решение соответствующей актуальной проблемы теоретической или прикладной задачи

Структура работы представляется оригинальной и логически обоснованной; соответствует заявленной теме и отвечает задачам исследования.

6. Замечания к работе

Существенных недостатков в данной работе не выявлено. Выявлены несущественные недостатки: недостаточно иллюстративных материалов, графиков и т.п. Однако найденные недостатки не влияют на качество исследования по данной проблеме.

ЗАКЛЮЧЕНИЕ И ОЦЕНКА РАБОТЫ

Данная работа является интересным научным исследованием по своей новизне, теоретической и практической значимости, уровню исследования; отвечает требованиям, предъявляемым магистерским диссертациям. Поставленные цели и задачи диссертантом успешно решены.

Выпускная диссертационная работа рекомендована к защите и заслуживает оценки «отлично» (95%).

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МИНИСТЕРСТВО ОБРАЗОВАНИЯ И НАУКИ РЕСПУБЛИКИ КАЗАХСТАН
СӘТБАЕВ УНИВЕРСИТЕТІ

ОТЗЫВ

НАУЧНОГО РУКОВОДИТЕЛЯ

на *магистерскую диссертацию*
(наименование вида работы)
Галипаулы Шынгыс
(Ф.И.О. обучающегося)
7М07204 – Металлургия и обогащение полезных ископаемых
(шифр и наименование специальности)

Тема: «Исследование и разработка способов повышения эффективности переработки титаномагнетита»

За время работы над магистерской диссертацией Галипаулы Шынгыс зарекомендовал себя в качестве трудолюбивого и заинтересованного исследователя, способного к научному труду и применению своих знаний на практике.

Титаномагнетиты достаточно распространенная разновидность рудного сырья. Переработка титаномагнетитов имеет ряд особенностей, которые необходимо учитывать при выплавке чугуна. Диссертационная работа содержит результаты мероприятий, направленных на совершенствование технологии выплавки чугуна с целью повышения производительности и снижения рисков, связанных с образованием карбидов титана. Актуальность заявленной темы не вызывает сомнений.

Структура работы представляется оригинальной и логически обоснованной; соответствует заявленной теме и отвечает задачам исследования.

Магистрант продемонстрировал достаточно высокий уровень навыков сбора и поиска информации. В работе использованы современные источники по исследуемой обстали, опубликованные ведущими специалистами в этой области.

Исследование выстроено с учетом всех аспектов заявленной темы, а именно: выполнен обзор подходов к определению сущности анализируемого понятия, предпринята попытка формулировки основных методологических проблем, представлен анализ данных реальной компании, осуществлен анализ перспектив. По каждой главе представлены выводы.

Знания, полученные при написании работы, могут иметь непосредственное применение в практической деятельности.

Работа выполнена на высоком уровне, поставленные цели достигнуты и задачи выполнены. Работа соответствует требованиям, предъявляемым к магистерским диссертациям

Замечания по магистерской диссертации отсутствуют. Отзыв о магистерской диссертации и предлагаемая оценка «отлично» (95%).

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