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PROCESSING OF COPPER-ZINC, COPPER-NICKEL AND OTHER SULPHIDE CONCENTRATES AND ORES BY THE OXYGEN SUSPENDED ELECTRO-THERMO- TECHNIQUE

by

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Theses

The modern pyrometallurgical methods of processing copper carrying sulphide ores boil down to matte smelting with its subsequent conversion to blister copper. The copper carrying ores are mainly pyrite and semipyrite smelted while the concentrates are processed in reflector furnaces (widely applied as of recent years).

The above methods have the following setbacks:

a - low efficiency in utilization of valuable constituents - due to loss of zinc, lead, cadmium and other metals contained in the ores and concentrates.

b - low efficiency in utilizing of sulphide concentrate heat producing properties, which causes the smelting process to require additional carbon fuel, thus augmenting the power consumption.

c - complicated multi-stage technological pattern.

d - difficulties in purifying large quantities of gas and in utilizing the sulphur due to low SO_2 content.

e - low and uncontrollable desulphurization which makes it practically impossible to regulate matte composition.

Matte smelting by the developed method of shaft pyrismelt allows considerable sulphured traction perse and in the form of H_2SO_4 but the technique is applicable only to ores with low zinc content.

To utilise more effectively the valuable components complexity is intensified by introduction of additional procedures. Thus multisole furnace roasting, fluid bed furnace treatment and sintering on machines with an upward blast are used to obtain gases for H_2SO_4 production.

Slags are subjected to fuming rotating Kilning and other procedures for cadmiun zinc lead and rare metal extraction. The technological cycle in this case is not accomplished and is significantly more complicated due to additional processes (conversion) the vilatile products demand.

Dispersion smelting is making considerable headway these recent years. Heat generating properties of sulphite concentrate are more effectively utilized on smelting copper and Cu-Zn concentrate dispersion in preheated blast. Fuel consumption is low and gases subsequently render H_2SO_4. The method has practically all the setbacks inherent to air-blast processing.

Cu-Zn concentrate processing is elaborated in the USSR and cyclon smelting with air and O_2 rich air blasting , torch smelting of air and O_2 suspended concentrate and converter pyroselection research is underway.

The cyclon smelting technique combined with sediment chamber permits roasting, smelting and zinc lead and other metal volatilisation, also slag and matte separation in a common aggregate. Volatilisation of Zn being 75% lead 85%. The method makes desulphurisation controllable and permits for a wide production of gases rehdering H_2SO_4.

The slags though are not of the dump metal content level which is a setback.

Cyclon chamber air clast smelting with electric furnace thermoprocessing of slag has increased technological efficiency of the process.

In an efficient utilisation of sulphide ores and concentrater heat generating properties, large volume of gases produced, purification problems and low SO_3 content are the common drawbacks of the air blast methods.

Intensification of the so-called classic processes (shaft, reflecter) of cu-sulphite and cupro-zinc and other material treatment by O_2 enriched airblast is underway since 1962 in the USSR.

At present, however, the enriching of blast with O_2 is limited by the construction of the aggregates which handicaps considerably - the use of this powerful intensificater of processes in the so-called "classical" conditions.

Autogeneous washing and smelting, on the other hand permit the heat generating properties of sulphide ores to be utilised to the full, which gives technical O_2 blasting or O_2 rich air blasting a special interest. The O_2 participation in the blast and technical O_2 blasting should in each case be determined by the nature of material processed and economical rationale.

In metallurgical processes O_2:

- augments its concentration in the reaction zone, which causes intensification of $V = K\, CO_2^n$
- causes decrease of reaction products, elevation of t° and acceleration of the reaction Arenias $\log V = B - \frac{A}{T}$
- promotes heat transfer from gases to charge $E = C\,(\frac{T}{100})^4$ Kcal/m²/hr
- brings about a decrease of gaseous products of the reaction and assists concentration of valuable constituents in gas.

Thus O_2 is a powerful factor in intensifying of metallurgical processes. In Canada Copper Cliff technical O_2 is used in suspension smelting of copper-nickel concentrates.

The setbacks of the method are: slags are not dump bare and require additional stripping in separate aggregate: the furnace construction is complicated, refractory elements life span is short and dust exhaust is high.

We have developed in the USSR an electrothermo-processing method of copper and zinc carrying oxidised material treatment with condensation of liquid zinc. It has definite advantages but on smelting, say, sulphide Cu-Zn material previous roasting or sintering of charge is required eliminating utilisation of heat generated by oxidation.

Moreculenium plant employs electrothermal-processing of liquid zinc carrying slags, which is more effective than hard charge electro-processing but the operations were periodical.

Semiindustrial bests are carried on in the USSR since 1963 and a combined aggregate and method are contemplated with consideration of all the assests and advantages of O_2 power smelting of complex ores and concentrates.

The aggregate and the method is called the KIVCET technique (oxygen suspended cyclono-electrothermo-technique.)

KIVSET implies combining autogenous roasting and smelting of charge dispersed and suspended in a cyclon chamber, fed by technical oxygen and with zinc volatalised from the smelt in the electrothermic part of the aggregate. The zinc is then condensed into liquid metal the slag is stripped of copper, lead and other metals. The operations described are a continuous interrupted process effected at one aggregate. One entire cycle affords: sulphur extraction to provide rich gases up to 90% anhydride; copper and precious metals to give matte; lead, germanium, selenium, renium, cadmium and other volatile compenents turn into volatants and zinc into liquid metal. Such an aggregate with a 50 ton charge capacity has been functioning under the all union mining and metallurgy non-ferrous metals research institute since 1965. Semi-industrial tests are carried on here to obtain optimal construction and unit performance and to develop processing techniques for various copper containing ores and concentrates.

<u>Rating of the KIVCET</u>

Capacity of aggregate up to 90 t charge unites

- Concentrate drying installations

- dosator and loading system for smelt-charge.
- Smelting part: cyclon chamber and gas-smelt separation chamber.
- Electrothermal part adjacent to smelting part and their gas space is separated by a partition immersed in smelt.
- Zinc irrigated condenser
- System of cooling, crude purification and refining of gases.
- Substation, transformer 750 KVa
- Control instruments and automatic control systems.

The main technological operations of the KIVSET.

- Charge preparation for smelting.
- Roast-smelting of charge in cyclon chamber.
- Slag and matte separation.
- Electrothermo stripping of smelt
- Condensation of zinc (in Zn processing)
- Cooling and purification of gases.

Charge preparation. The concentrate and crushed flux are mixed in preset proportions and dried to not more than 2% humidity. Then disintegrated to 1 mm - 100% size.

The drying may be effected in a rotating drum, in fluid bed furnaces or drying tunnel as was confirmed by tests in semi industrial and industrial scale. The charge is then pneumo-transferred to KIVSET hoppers.

Roast-smelting in cyclon chamber

The cyclon chamber was included into the KIVSET system due to: a)- Intensification of sulphides caused by high mixture forming properties (in small volume) and utilisation of highly developed and active surfaces of five ground materials.

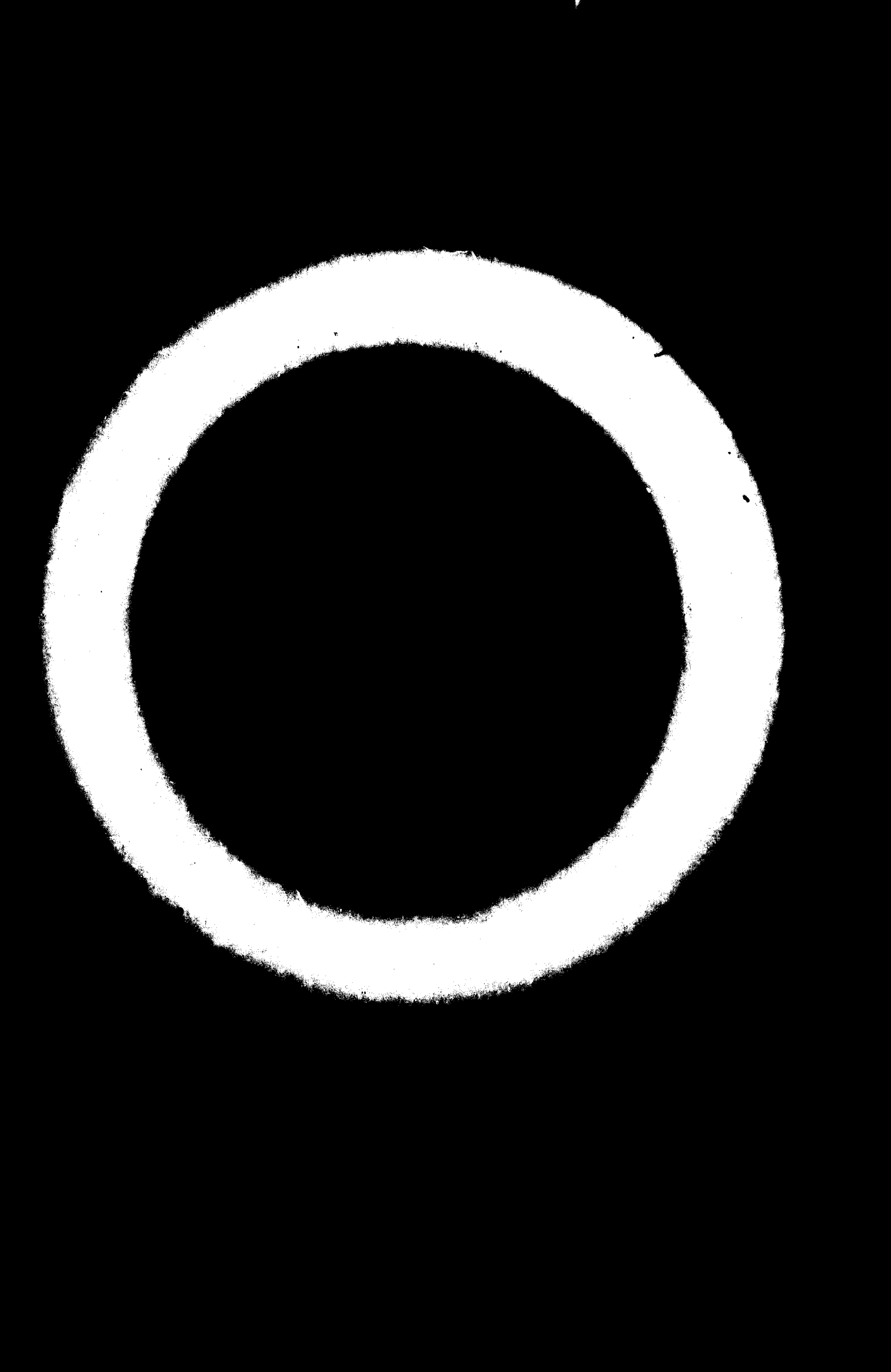
b)- Efficient charge-dust recovery and consequently low dust -exhaust.

c)- Efficient gas phase and smelt products control.

d)- Intense heat and mass exchange permitting high productivity for small volumes.

Effective cyclon principle and technical O_2 blast accelerated the process considerably. Various cyclon chambers were tested and stable performance has been achieved.

Slag and matte separation. The operation was effected in the three-electrode electrothermal part of the aggregate in conditions of continuous smelting. Height of smelt was maintained at 800-1200 mm. The output of smelting (slag matte) is periodical.



The modern pyrometallurgical methods of processing copper carrying sulphide ores boil down to matte smelting with its subsequent conversion to blister copper.The copper carrying ores are mainly pyrite and semipyrite smelted while the concentrates are processed in reflector furnaces (widely applied as of recent years). The above methods have the following setbacks:

a. low efficiency in utilisation of valuable constituents
b. low efficiency in utilising of sulphide concentrate heat producing properties which causes the smelting process to require additional carbon fuel, thus augmenting the power consumption
c. complicated multi-stage technological pattern
d. difficulties in purifying large quantities of gas and in utilising the sulphur due to low SO_2 content
e. low and uncontrollable desulphurisation which makes it practically impossible to regulate matte composition.

To increase the complexity of valuable component utilization new operations are introduced into the technological [illegible] roasting in multi-hole furnaces, in fluid bed furnaces, sinter roasting [illegible].

To extract zinc, lead- cadmium and rare metals from slags fuming rotary kiln and other techniques are applied. To intensify concentrate and complex ore processing and increase complexity in ore utilization electrothermo-techniques, cyclon blast and oxygen blast smelting, torch suspension smelting in air and oxygen blast stream, converter pyrosection methods are used in the USSR.

Intensification of the so-called classic processes (shaft and reflector smelting) in copper, copper-zinc and copper-nickel extraction was begun in 1962 in the USSR on the oxygen enriched blasting basis.

But oxygen enriching of the blast stream is limited by plant construction which hampers maximum efficiency of oxygen.

This efficiency can be achieved when (e.g.) roasting and smelting are autogenic. Technical oxygen and oxygen enriched air blasting present special interest here, as providing the autogenic conditions.

Semi-industrial tests for a combined technique and aggregate are conducted in the Soviet Union since 1963. They envisage the methods and techniques of ore processing with the application of oxygen and electric energy.

The aggregate and the method are called the KIVCET technique (oxygen suspended cyclono-electro-thermo mechanique) KIVCET implies combining autogenous roasting and smelting of charge dispersed and suspended in a cyclon chamber fed by technical oxygen and with zinc volatalized from the smelt in the electro thermic part of the aggregate. The zinc is then condensed into liquid metal, the slag is stripped of copper lead and other me

The operations described are a continuous uninterrupted process effected at one aggregate. One entire cycle affords sulphur extraction to provide rich gases up to 90% anhydride, turning of copper and precious metals into matte, turning of lead, germanium , selenium renium, cadmium and other volatile components into volatants, and zinc-into liquid metal. Such an aggregate with a 50 tons charge capacity has been functioning under the All Union Mining and Metallurgy Non-ferrous metals research institute since 1965. Semi-industrial tests are carried on here to obtain optimal construction and unit performance and to develop processing techniques for various copper containing ores and concentrates.

Copper-zinc concentrate processing - Composition in percents

16-20 copper, 7.3-10.5 zinc, 1.5-2.0 lead, 24-26 iron, 32-34 sulphur, 15% of quartz flux was blend charged to the concentrate. 1100 tons of charge were processed during the optimization and an uninterrupted long period-system of autogenous roast-smelting, smelt stripping, zinc condensation and gas purification were obtained. The main indices of the processing: 1 ton charge consumption : [illegible] m^3 of oxygen.

stripped charge reduction 1.5-2.0 %

cyclon chamber capacity 20 t/m^3 cyclon perhr.

zinc condensation in condenser [illegible]%

oxygen assimilation in cyclon [illegible]%

content of SO_2 in gases 75-85%.

Metal extraction (return into cycle considered) in %.

98.5% copper, 97.5% gold, 97.4% silver into matter,

62% zinc into metal, with 8.5% initial content in charge.

Comparison extraction and level of complex utilization of copper-zinc ore.

Processing pattern	Cu	Pb	Zn	S	Au	Ag	K_4	K_6
KIVCET	95.0	62.0	70.8	90.0	94.6	94.5	85.3	85.9
Electric smelting with previous sintering	95.0	68.0	70.8	90.0	93.0	93.0	87.6	88.0
Reflexion smelting	93.0	30.0	-	60.0	-	-	76.1	-
Charge smelting	93.4	9.4	60.8	-	96.5	95.3	68.3	70.0

Complex ore utilization coefficients are at one level for all but the charge and reflector smelting

Power consumption per 1 ton of wet charge

	Charge smelt with O_2 app	Reflector smelt with O_2	Electrothermy	KIVCET
Fuel	234	430	147	64
Electrics energy	277	19	840	210
Oxygen	56	55	-	100
Total kWh	567	504	987	374
%	100.0	89.0	174.0	66.0

Heat and electric power consumption per 1 ton extracted metal

	KIVCET	Reflector smelting	Charge smelt	Electrotherm.
Consumption (in kW+h)	1978	4027	3347	4542

As is readily seen the KIVCET proves to be the most power sparing technique due to efficient use of heat productive properties of sulphides. 1 t is also advantageou as to complexity. At present an experimental-industrial 500 ton a day KIVCET plant is pending start at a copper smelting plant and a 1200 t furnace is in project.

Copper nickel concentrate smelting:

Copper-nickel concentrates were smelted, containing: 3.9%-4.3% nickel

1.8-2.1 copper

0.1-0.15 cobalt

14.7-15.7 sulphur

23.2-24.6 iron

25.7-26.1 silicon dioxide

7.9-10.2 MnO_2

4.0-4.2 CaO

5.9-6.5 AlO (alumina)

The sulphur content being insufficient to provide autogenous smelting additional coal-dust fuel was fed into the cyclone chamber together with the concentrate.

Gas-contained 20% of SO_2, metal extraction into matte was in %: 97.6 nickel, 96.2 copper, 80.8 cobalt.

Losses in slags 1.4; 2.3; 11.3% respectively.

Overall power consumption in kwthrs
(oxygen fuel blasting electric energy)

Electrosmelting with sintering	Thompson plant	KIVCET
Drying - 52	Drying - 105	Drying - 105
Sintering - 88	Roast/blast - 7	Smelting with stripping - 390
Electric smelting - 567	Electric smelting skumig - 485	
Total: 707	597	495
%: 100.0	84.5	70.0

The KIVCET technique is the most economical as to power consumption.

Rich sulphide copper-nickel ore processing

Ore consumption: 8-10% copper, 3-4.9 % nickel, 0.08-0.12 cobalt, 40-45% iron, 28% sulphur, 6-8% SiO_2, 2% Al_2O_3. 2000 tons were test processed.

It has been established that autogenous process and stable performance are achieved in the following conditions

- desulphurisation not less than 60% with mattes containing Cu + nickel above 30%.
- slags with not more than 30% silica.

In these conditions oxygen consumption per 1 ton charge was 200-220 m^3 - 80% SO_2 and 4% oxygen in gases.

Extraction into matte 96.3% copper, 96.7% nickel, 77.1 % cobalt with 23.6, 12.5 and 0.2 respectively.

Losses with slags were 3.5% copper, 1.6 nickel, 19.2 cobalt with 0.4, 0.15, 0.035 respectively.

Semi-industrial KIVCET slag processing tests were carried out by the cementation method and cast iron was obtained.

After stripping slags of the following composition were obtained:

0.04% copper, 0.007% nickel, 0.0002% cobalt, 41.2% silica, 32.0% calcium dioxide, 8-10% iron.

Laboratory and industrial experiments prove that after deep stripping and elimination of iron slags can be used as construction material (casting).

Technological and economical assessing of the KIVCET processing of sulphide copper-nickel ore - have proved it to be highly effective as compared with other techniques.

Smelting of copper pyrite gold containing concentrates

Semi-industrial tests were carried out.

The composition of the concentrates: 3.8% copper, 38% sulphur, 30% iron, 8% silica, 10 gm/ton, gold, 40 gm/t silver.

Extraction of metals into matte: 93.1% copper, 90% gold, [illegible] silver, SO_2 in gases 7%.

KIVCET is thus effective in copper and pyrite concentrate processing.

Rich copper concentrate smelting

Semi-industrial tests were effected on charges of the following composition: 26-29% copper, 1.5-17% nickel, 0.04 cobalt, 29--32% iron, 30-32 sulphur, 8-10% silicon dioxide. The process was autogenous on smelting to matte with approximately 50% copper.

Slags contained 0.35-45% copper, 0.03-0.05 nickel, 0.008--0.012 cobalt.

In 1969 test KIVCET smelting of copper concentrates into blister copper was carried out. The result are encoraging and hold future short cuts in copper production, effective copper and precious metal extraction possibilities, and full transfer of sulphur into rich gases. Good results were obtained on lead -tin and other concentrates. 30.000 tons of various metals were test processed by the KIVCET technique. The technological indices of complex treatment of valuable ore constituents and low power consumption are all in favour of the KIVCET method and fully justifyits its description as highly commendable.

The technique and aggregate are patented in a number of countries.

Electrothermic smelt stripping

Slag stripping in conditions of KIVCET smelting of copper zinc concentrations has the following characteristic a) the slag being stripped is in constant contact with copper abundant matte. b) the process is uninterrupted as slag-matte smelt is continually fed into the stripping zone from the smelting zone

c) copper extraction takes place simultaneously with zinc volatilisation (and other constituents) smelt stripping (zinc volatilization and copper extraction from slags) was achieved by charging reducing coke or coal into the electrothermal part and maintaining the necessary t° of the smelt.

Zinc condensation. There is a continuous flow of zinc vapours and gases from the electrothermal part of the aggregate into the condenser. Condensation is effected on a jet condenser on sprayed zinc. The electrothermal part of the aggregate and condenser working pressure is 10 mm H_2O.

Gas cooling and purification:

KIVCET smelting gases have specific properties,

- high content of SO_2 and SO_3 90% and 5% respectively- hi
- high dust content up to 340 g/t
- presence of volatilised zinc lead and other volatile metals.

A corresponding gas purification scheme has been elaborated with consideration of the above factors.

They are passed through air cooling chambers or salvage boilers.

Processing of copper containing ores and concentrates. Results.

a) [illegible]

Composition: 16-20% copper, 7.3%-10.5% zinc, 1.5%-2.0% lead, [illegible] iron, 32%-34% sulphur, 1% quartz flux was added to the concentrate.

11000 tons of charge was test processed on a semi-industrial scale and a long period uninterrupted performance was achieved. As well as autogenous roast-smelting technology, smelt stripping, condensation of zinc and gas purification.

The following were characteristic:

Per 1 ton charge consumption

smelt oxygen - 210 m^3

reductor for stripping - 1.5-2.0%

specific efficiency of cyclon chamber 20 t/m^3 per hr

Zinc condensation in condenser 92%

Oxygen utilization in cyclon 98%-99%

Smelt product constituents.

	Cu	Zn	Pb	Fe	SiO_2	CaO	SO_2
Slag	0.49	2.0	0.19	28-32	36-38	7-9	-
Matte	50.0	2.5	4.20	21.8	-	-	-
Gas	-	-	-	-	-	-	75-85

Metal extraction with return into processing cycle.
[illegible]% gold, 97.5% silver and 97.4% zinc into matte.
[illegible]% with zinc content in concentrates 8.5%

Irrecoverable losses of zinc with slags 11%, of copper 1.[illegible]% lead 5.[illegible]%.

To assess economic efficiency of the [illegible] method a comparison chart is given below.

Extraction coefficients were for an identical product when **complexity was** considered.Blister copper zinc, lead and copper **in gases suitable** for sulphuric acid and perimar extraction.

Processing	Cu	Zn	Pb	S	Au	Ag	K_4	K_6
KIVCET	95.0	62.0	70.8	90.0	94.6	94.5	85.3	85.9
Electrosmelting with previous sintering	95.0	68.0	70.8	90.0	93.0	93.0	87.6	88.0
Reflector smelting	93.0	30.0	-	60.0	-	-	76.1	-
Shaft smelting	93.4	9.4	60.8	-	96.5	95.3	68.3	70.0

As evidenced by the chart we obtained approximately identical level coefficients for 4 and 6 main components. The shaft and reflector smelt coefficient is significantly lower due to considerable losses of sulphur and zinc.

__Power consumption__. Comparison chart per 1 ton wet charge and 1 ton crushed metal

	Shaft smelt with O_2	Reflector smelt + O_2	electrothermy	KIVCET
Fuel	234	430	147	64
Electric energy	277	19	840	210
O_2	56	55	-	100
Total kwthrs	567	504	987	374
%	100.0	89.0	174.0	66.0

Heat and power consumption per 1 ton extracted metal.

	KIVCET	Reflector smelt	Shaft smelt	Electrothermy
Consumption 1 ton/kwt/hr	1978	4027	3347	4542
%	100	203.6	169.2	229.6

Due to efficient use of heat generating properties of sulphides the KINSET method is most rational in thermo of power consumption.

It furnishes a high degree of complexity at relatively low power consumption. A 300 t capacity KIVCET aggregate is underway at present for copper-zinc ore processing.

A new 1200 ton charge perday KIVCET plant peaject is under construction in this country,

b) Copper-nickel ore and concentrate processing

Modern copper and nickel extraction from sulphide cupro-nickel ores and concentrates is effected by various pyrometallurgical means: smelting of roasted skimmings in reflector furnace skimmings smelting sinter smelting and ore smelting in electric furnaces and smelting of suspended concentrates.

The cupro-nickel skimmings are first roasted at 700°C in multiple furnaces then smelted in reflector furnaces.

The assets here - are autogenous smelting, low fuel consumption and the roast gases (4.5% SO_3 content) may be used for [illegible] production. The setbacks: additional labour demanding roast [illegible] of concentrates high cost of gas purification due to

excess of air air blast ($\alpha = 3.0$);
low specific output of reflector furnaces (4.4.5 t/m^2 high fuel consumption on smelt (12.5% high nickel content in slags (0.2%); this at relatively low nickel content in matte (cu-nickel 13%).

The Khariavalta (Finland) employs suspension hot blast smelting of cu-nickel concentrates. These are dried in a rotating drum to 1% humidity processed in a suspension smelting furnace smelting shaft. Slag -matte separation is effected in the same furnace. Gases are utilised for H_2SO_4 production.

The slags and converter slags are stripped by adding nickel concentrates in a separate electric furnace. The fuel consumption is insignificant in smelting (3.0%). The set back of the method is that the slags are rich in copper and nickel content and demand an additional treatment.

There is a recent tendency to use electric smelting of roasted concentrates (thompson plant, Canada). The technology is modern and compact as to equipment and pattern. The concentrates are roasted in fluid bed furnace with air blasting. The hot skimmings are smelted in furnaces. The process is thus autogenous and SO_2 content in gases is within 17-20%. But power consumption even when hot skimmings are smelted, is signifant (up to 440 [illegible]).

The Thompson technique is handicapped by high cu-nickel content in slags (0.28%) and low content in matte (16-17%). The charge of dustlike skimmings into the furnace results in high dust in exhaust gases and makes their purification imperative.

Nickel is electrosmelted in the Norilsk nickel plant. Th concentrates here have a low sulphur content (14-15) sintering of such concentrates usually does not permit utilisation of S - as the SO_3 content is low . The loss of methods is high in charge and sintering preparation. Cold sinter smelting in electric furnaces requires approximately 550 kwt hours per 1 ton. Electric smelting of cupro-nickel ores is effected too in Severonickel, USSR.

The overall power consumption is high here due to insufficient utilisation of heat generating properties sulphate oxidation. The technological pattern is complicated and the utilisation complexity is low.

We have effected cupro-nickel concentrate and rich sulphide cupro-nickel ore processing on a semi-industrial installation in AUNFMRI (all union non-ferrous metal reaearch institute) by the KIVSET technique. The aggregate is simpler as the condenser is canceled. The results are as follows.

Concentrate constituents:

3.9%-4.3% nickel, 1.8%-2.1% copper, 0.1%-0.15% cobalt, 14.7%-15.7% sulphur, 23.2%-24.6% iron, 25.7%-26.1% silicon dioxide 7.9-10.2% magnesium oxide. 4.0-4.2% calcium oxide 5.9-6.5 aluminium oxide. The concentrates are poor in sulphur and infusible (refractory).

Low sulphur content precludes autogenous smelting so - 14% of additional coal dust fuel was in produced with the concentrate into the cyclon chamber.

Products of the following composition were obtained during the balance period.

	Nickel	Copper	Cobalt	Sulphur
Matte	16.9	8.3	0.366	23.7
Slag	0.09	0.074	0.019	-

The gases contained up to 20% SO_3. Extraction of metals into matte was 97.8% nickel 96.2% copper, 80.8% cobalt. Losses with slag 1.4%, 2.3%, 11.3% respectively.

This at the following technological parameters.

Specific capacity of cyclon chamber

25 t/m^3 cyclon volume perhr

Desulphurisation - 60%

Period of smelt in furnace - 11 hrs

Slag t° ~ 1400°C

Stripping losses:

electric power - 220 kwt per/t charge

reductor - 1.8% charge weight

smelt-oxygen - 340 m^3t

A comparison chart is supplied for reference

	Thompson	KIVCET [illegible]
Power consumption kwt/hrs	440*	110
Capacity of furnace per m² per day	4.4	18.0
Content in slag		
nickel	0.2	0.09
copper	-	0.07
cobalt	-	0.[illegible]
Content in matte		
nickel	77.[illegible]	96.3
copper	1.[illegible]	0.3
cobalt	-	0.37
Content in gases on SO_2	17.[illegible] (on roasting)	16.[illegible] on smelting

*) power consumption on [illegible] [illegible] would require a power [illegible]

Overall kwt hr power consumption

Electrosmelting with sintering	Thompson	KIVCET
Drying in KC (50%) concentrate fuel blast 45+7= 52	Drying in KC(100%) fuel blast 90+15= 105	Drying 100% fuel blast 90+15 = 105
Sintering 3% coal, blast 73+15 = 88	Roast blast mixing	Smelting with slag stripping (O_2, fuel power reducer 18%) 145+110+90+45 = 390
Electrosmelting power consumption, 1.5% reducer for stripping 530 + 37 = 567	Electric smelting of skimmings 440+45 = 485	
707	597	495
% 100	84.5	70

As it is readily seen the KIVCET technique requires less power expenditure.

High sulphite cupro-nickel ore processing

The KIVCET technique will increase extraction efficiency as flotation and sintering are cancelled. The constituents of the ore treated were as follows 8-10% copper 3-4% nickel 0.08-0.12% cobalt, 40-45% iron, 28% sulphur, 6-8% silicon dioxide, 2.0 aluminium trioxide 2000 tons of ore were test-processed.

Semi-industrial tests of ore smelting showed the autogenous process of roast-smelting and stable performance of the cyclon chamber to be achieved at the following conditions

- size of ground ore particles 60%-80% minus 0.074 mm
- desulphurization not less than 60% and matte content of copper + nickel above 30%
- slag content not more than 30% silicon dioxide (corresponds to 10% quartz subcharge to ore)
- cyclon chamber production 19 t/m^3 of its volume

The oxygen consumption per 1 ton charge was 200-220 m^3 and SO_3 and O_2 content in gases was 80% and oxygen up to 4% respectively.

In smelt stripping satisfactory metal content in slags and their extraction into matte was achieved on following indices:

consumption of reducer - 1.2%

consumption of cupro-nickel ore 2%

electric power 300 kwthrs

capacity of 1.0 t/m^2 of reflector surface of electrothermal section and height smelt up to 1100 mm.

The following results were achieved

extraction into matte: 96.9% copper

96.7% nickel

77.1% cobalt

with 83.6%, 12.5%, 0.2% content respectively.

Slag losses: 3.5% copper 1.6% nickel 19.2% cobalt.

with 0.6% 0.1% 0.03% content respectively.

Investigation results indicate means of further slag loss elimination through increase of matte metallization and slag acidity.

Semi-industrial aggregate thermo-balance data served as basis for slag stripping power consumption calculations in an aggregate of 300 ton charge perday capacity. It is assessed that 150 kwthrs per one ton charge will be required.

Semi-industrial tests of the cementation method of KIVCET slag processing with subsequent cast iron produce were performed. After stripping the following slags were obtained. 0.045 copper 0.007 nickel 0.0002 cobalt 41.2 silicon dioxide 32.0 calcium oxide 8-10 iron. Laboratory and industrial data prove the slags, useful for construction materials manufacture after stripping and iron extraction. Cast items may be produced of slags after their corresponding modification. Demand should in every case be the criterion for rational deep stripping and iron elimination where subsequent cast iron produce and building material production are contemplated.

The KIVCET technology compares with other patterns as highly efficient where technical and economic construction are reckoned with.

Copper pyrite gold carrying concentrates smelting

Semi-industrial tests of KIVCET smelting of cupro-pyrite gold carrying concentrates were performed, on the assumption that the KIVCET technique will provide high extraction of copper and precious metals into matte and S into gases and that the more complicated pyrite concentrates (lead and zinc carrying) may be treated with the extraction of their main constituents

into metal. Approximately 300 tons were processed during the test period it has been established, that:
The process is autogenous and stable at cyclon camera capacity 16 ton/m^3 per 1 hr. There was 71% SO_2 and 2% O_2 in gases at an 37% desulphurization.

The test period gave the following data.

Metal extraction into matte 93.1% copper.

90.0% gold, 95.2% silver.

Losses with slago 5.4% Cu 0.9% Au 4.[illegible]% Ag.

Composition of smelt products

Matte: 18.8-21.0 Cu, 50.5-53.9 Fe, 23.3-24.6 S, 58.6-66 g/ton. Au, 176-204 g/ton silver.

Slag: 0.29-0.36 Cu 36.5 Fe 37.6 SiO_2

0.1-0.4 g /ton Au, 2.6-3.3 g.t. silver.

Consumption per 1 ton charge: power - 100-140

oxygen - 270 m^3

small coke - 1.6%.

It has also been established that germanium is mainly accumulated in dusts which may serve as extraction material. The KIVCET method is thus satisfactorily efficient in processing gold and copper carrying pyrite concentrates and in extraction of valuable constituents.

High copper concentrate smelting

Semi-industrial tests were carried out on charge consisting of: 26-[illegible]% Cu, 1.5-1.7 nickel, 0.04% cobalt, 29-32 iron, 30-32% sulphur, 8-19% silica. 90% Cu content matte smelting ensur autogenous operation of the semi-industrial installation. Metal content in slags was 0.35-0.45% Cu, 0.03-0.05 nickel, 0.008-0.012 cobalt.

Consumption per 1 ton charge

power - 80-120 kwt hrs

oxygen - 200 nm^3

small coke (slag stripping) - 2.5-40%

In 1969 test KIVCET smelting of pure copper concentrates to blister copper was begun . The results are encouraging and development of the technique will permit:

- to shorten the technological cycle
- intensify the extraction of Cu and precious metals because such operations as roasting and conversion are cancelled
- to almost entirely extract sulphur into rich sulphuric gases etc.

The investigation is underway at present.

KIVCET copper-tin and other concentrate processing tests were effected and satisfactory results obtained.

A metallurgical remodelling of Cu-tin concentrate processing is envisaged as a result of these tests, 30000 tons of various materials were processed over the KIVCET semi-industrial testperiod.

We have obtained positive technological data on complex utilization of valuable constituents of raw materials and the decrease of power consumption. This makes possible to recommend the KIVCET technology. for processing different monometallic and polymetal ores and concentrates. The aggregate and method are patented in several countries.

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