Various Recent Developments in the Sintering Field
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The present paper describes various recent developments in the sintering field.

Sinter machine size

In 1960 the first sintering machines of 3 and 4 meters width were started up. The biggest of these machines had a reaction surface of 225 m² (Jones & Laughlin, USA, Aliquippa Works). Only in 1968 a bigger sintering machine with a suction area of 288 m² was completed for Wendel-Sidélor, Usine de Joeuf (France). Subsequently sintering machines of ever increasing size were constructed at short intervals: in 1970 the first 5 meter wide sintering machine with a suction area of 400 m² for Wendel-Sidélor, Usine de Rombas (France), in 1972 the first 400 m² sintering machine of 4 meters width for Messrs. Fried. Krupp Hüttenwerke A.G., Werk Rheinhausen (Germany), in 1971 the first 500 m² sintering machine for Nippon Steel Corp. (Japan) Kimitsu Works and in 1973 a 550 m² sintering machine of 5 meters width for Nippon Kokan K.K. (Japan), Fukuyama Works. A 600 m² sintering machine of 5 meters width is already under construction for Nippon Steel Corp. (Japan), Ooita Works.
Nowadays we are able to offer sintering machines of a reaction surface of up to 1200 m² at a width of up to 6 and 6.5 meters.

**Improvements on maintenance**

A certain number of sintering plants are operating with a plant availability of above 90%. These high values were only attainable because the design of machines subject to wear was improved to ensure a long service life, easy maintenance and quick exchange.

The machines subject to wear comprise the spiked roll crusher with crushdeck and hot screens.

An important measure for achieving a high plant availability is to carry out the maintenance work together with the preventive maintenance at the workshop and not during a maintenance shift. This supposes that during a planned shutdown a complete unit is removed and replaced by an exchange unit overhauled at the workshop.

An optimum design of the buildings with operating floors, the incorporation of the necessary lifting tackle and the provision of transportation routes are required for a quick assembly and disassembly in order to ensure shortest possible maintenance periods.

Our design of sinter plants is based on the demand that after an operating time of 6 weeks the following mechanical parts are exchangeable within 8 hours: 2 hot screens - 1 spiked roll crusher - 1 mobile grizzly underneath
spiked roll crusher - 12 pallets - 1 ignition hood section - 1 plate covering of a vibro-conveyor.

To ensure these exchange times, new designs were developed such as steel sheet hood above the hot screens which can be exchanged by motor without any preparatory work and hot screens exchangeable by motor.

The exchange of complete 30 t - screens takes 10 minutes.

The spiked roll crusher shaft being driven through easily removable shaft mounted gearboxes is vertically removed with lifting tackle. The mobile grizzly can be removed in longitudinal direction of the burning machine, picture 1.

Application of a higher suction

In the technically interesting range of 400 to 2000 mm water gauge the waste gas volume increases with the square root of suction, as is shown in picture 2. On the contrary, when the sinter capacity is plotted against the suction, a lower increase than that corresponding to the square root is found, as shown in picture 3. The exponent is about 1/3 i.e., the sinter capacity rises with increasing suction at a lower speed than the waste gas rate so that the specific air volume has slightly to rise with the suction. The sinter product characteristics and the fuel consumption do not change or only change slightly at an increased suction and when high-grade ore mixtures are used.
The choice of the suction for the design of a sintering machine influences the capital and operating costs. If a specific quantity of sinter product is to be produced from predetermined raw materials in a new plant, the capital expenditure decreases when a higher suction is applied because a smaller sintering machine is sufficient to produce the desired sinter product quantity.

However, the electric energy costs increase. A typical example of the variation of the different parameters in such a case is given in picture 4. The specific sinter capacity at a suction of 78.5 mbar (800 mm WG) was assumed as 35 tons/sq.m. x 24 h and the influence of the suction was calculated according to the equation $L = c \cdot p^{1/3}$. The corresponding sintering machine surface was ascertained for a sinter capacity of 11,000 t/24 h. In the lower half of the picture the specific sinter production and electric energy as well as the sinter machine size and in the upper half the capital, electric energy and processing costs are plotted against the suction. The calculation of these costs was based on a capital expenditure of DM 35,000.-- per sq.m. suction area, on an amortization rate of 20 % and on energy costs of DM 0.05/kWh. As can be seen from the diagram, the processing costs continuously rise with an increasing suction. Therefore the question as to the optimum suction for the design of new plants arises anew.
Cooling of sinter on the sintering machine

It has been stated on several occasions that in the event of on-strand cooling the fuel consumption and return fines rate are lower than in case of the normal method. The sinter is said to have a higher degree of oxidation resulting in an improved reducibility, being of smaller and more uniform size and having a higher mechanical strength. These advantages are attributed to the different cooling method in the event of on-strand cooling in which case the cooling air is sucked through the existing cracks in the sinter and those additionally produced during cooling whilst in case of the normal method the air flows around the large hot sinter lumps. The statements on changes of the properties coincide with our own experience and results of the corresponding tests except for the improvement of the sinter strength. Quite to the contrary, the sinter strength experiences an impairment if the other changes of quality take place as described and this is also the case with the normal mode of operation when the return fines rate is increased or the stress exerted on the sinter or the return fines size are reduced. If, on the other hand, one tries to produce sinter of similar mechanical properties to those obtained by the normal method, no differences are found any longer as far as fuel consumption, return fines balance and oxidation degree are concerned.

The question of sinter cooling can therefore not be made
dependent on process conditions. The decisive factors are rather the capital and maintenance costs as well as environmental control.

The ratio of cooling surface to sintering surface $V$, the specific sinter capacity $F_s$, the cooling capacity $F_k$ and the total output $L$ are decisive for the design of a sintering machine on which the sinter product is also cooled. These parameters referring to commercial and pot sintering tests are shown on fig. 5. The tests were carried out with sinter mixtures as they are presently used: rich ore mixtures and minette ore mixtures as well as minette ore mixtures enriched with high-grade ore.

In the event of minette ores, the ratio of cooling surface to sinter surface is $V = 0.4$ to $0.5$. In the case of rich ore mixtures, $V$ shows very different values. Sinter machines for rich ore mixtures are presently designed in general for sintering capacities of $40 - 45 \text{ t/sq.m. x 24 h}$. Within the range of these values, $V$ varies from 0.4 to beyond 2. These figures illustrate the difficulty of determining the surface ratio for the design of a sintering plant. A high surface ratio results in a high capital expenditure; a low surface ratio is unsuitable because it impairs the operational flexibility and only certain ores meet the requirements.

To compare the capital costs of conventional plants and those with on-strand cooling, various projects were elaborated or taken as a basis. Picture 6 shows a comparison
of the capital costs in % for a sintering machine having a suction area of 200 sq.m. and a sintering machine with a suction of 256 sq.m. As can be seen from this picture, the plants with on-strand cooling are more expensive by 13 to 30 % than conventional plants according to the necessary cooling area, sintering machine widths and suction in the cooling section.

Cooling on the strand offers a distinct possibility of further improving plant availability and thus raising production by increasing the intervals between repair shifts and cutting down maintenance costs, because the hot screens, tray conveyor and cooler are eliminated. Furthermore, the discharge end of the machine, including the crusher, can be simplified because it is not exposed to the same degree of wear as with hot sinter. With rich ore practice, the hot section of the plant contributes to about 25 % of the total maintenance costs (wages and materials) which can be to a large extent minimised. However, since only two big sintering plants with on-strand cooling were recently started up, a comparison of the maintenance costs with conventional plants cannot yet be made.

The increase in the operating costs due to a higher consumption of electric energy has been investigated by us at various suctions in the cooling zone with cooling on the strand. Whereas in the case of a conventional cooler the consumption per ton of sinter product amounted to
about 3.5 kWh, for cooling on the strand it varied between 14.2 kWh with 68.7 mbar (700 mm WG) suction and 21.2 kWh with 98.1 mbar (1000 mm WG).

With regard to dedusting, the cooling on the strand is advantageous. Although there are no differences for the dedusting of the sinter waste gases, the room dedusting is simpler because the hot section is omitted.

Mechanical dedusting units are sufficient for cooling air dedusting. However, in contrast to be conventional method, the whole cooling air volume is dedusted.

**Automation**

A certain number of plants use a return fines balance control system developed by Siemens and Lurgi in 1962. In this process the variations of the return fines bin level are used as indicators for the return fines balance, the following variables being influenced: the return fines rate by the deviation between the real value and an eligible reference value of the return fines bin filling level (bin level deviation) and the return fines rate by the deviation of the return fines rate from a desired reference value for the return fines cycle.

Picture 7 shows on the left hand the principle of the return fines balance control.

This control design was taken over by a digital computer in one plant whereby numerous improvements were achieved.
Moreover, a permagnag measuring instrument of the Belgian Research Center CRM was used in this plant for the determination of the Fe\(^{++}\)-content of the sinter product. This instrument is using the magnetic susceptibility of the sinter product as a criterion for its content of bivalent iron. The permagnag instrument was installed in the cold screening station. A partial stream branched off from the return fines material flow is continuously fed into this instrument.

The permagnag gives a signal which at a given ore mixture shows a clear dependence on the magnetite content of the sinter product. However, as the measurement is also dependent on the mix composition and in permanent operation, a temporal drift of the curve is to be expected, the instrument has frequently to be recalibrated at least when a slightly changed mix is used (e.g. at a pile change).

This problem was solved in such a way that the instrument is automatically calibrated by the computer on the basis of the laboratory analysis.

Picture 8 shows the Fe\(^{++}\) curve (laboratory and permagnag) which was plotted over several days as an example of the results obtained by this control system. This period covers a reference value change from 7 to 6.4 \% Fe\(^{++}\). The picture shows the good monitoring behaviour of the controller. The lower Fe\(^{++}\) value is achieved by a higher return fines cycle reference value.
The pile change carried out at the end of the shown period adversely affected the Fe$^{++}$ content of the sinter product. After a few hours the control system brought the Fe$^{++}$ value again to the Fe$^{++}$ reference value by corresponding RK control steps.

For comparison, a prolonged operational period (one month) was statistically evaluated. Two sintering machines were operated by using the same mix. In the plant SI, the conventional return fines balance control system was in operation, the return fines cycle reference value having been adjusted by hand in accordance with the hourly Fe$^{++}$ laboratory analyses. The plant S2 was operated according to the digital control program described above.

Picture 9 shows the appreciably smaller variation ranges of plant S2.

As concerns Fe$^{++}$, the variation of $\sigma_{S1} = 0.91$ was reduced to $\sigma_{S2} = 0.56$. Consequently it was possible to operate the plant at a lower mean value (reference value) because the risk is correspondingly lower that a value is achieved by the Fe$^{++}$ control system which is below the minimum value. Thus, it was possible to attain at roughly the same minimum value an average Fe$^{++}$ value which, in the shown example, is by 0.5 % Fe$^{++}$ lower.

An Fe$^{++}$ content by 0.5 % lower of the sinter product means a fuel saving of 2 kg/t sinter in the plant - and 1.75 kg/t sinter in the blast furnace.
Use of charcoal as fuel

The use of charcoal as fuel for sintering is advantageous for Brazil. The corresponding tests have demonstrated that this charcoal can be used without difficulties, if a specific basicity and slag amount can be maintained in the Sinter product. Picture 10 gives the principal results of such a test series. As is shown on this picture, satisfactory results are obtained at a basicity exceeding 1.4, a slag amount of above 8.8 % and a return fines grain size -5 mm.

SO₂ removal

The sinter waste gas contains 0.2 - 2 g SO₂/Nm³. This sulphur originates from the raw materials, primarily from the fuel and some sulphur-bearing ores or cinders. The SO₂ and SO₃ concentrations in the various wind boxes of the sintering machine are different. Measurements carried out in a sinter plant showed the distribution of SO₂, SO₃, F and Cl represented on picture 11. This distribution curve suggests the exclusive purification of the partial gas stream with a high SO₂ content instead of the total sinter plant waste gas volume.

A commercial plant for SO₂ removal from sinter plant waste gases is under construction at the Triest works of Italsider. This plant will be operated according to the aforesaid partial gas scrubbing principle and will be commissioned before long. The waste gas from the central section of the 42 m² sintering machine will be conveyed to a radial flow scrubber with the gas inlet being positioned on top and a primary spraying zone. In the scrubbing zone proper the washing liquid is injected through a central nozzle and mixed with the gas. The washing liquid consists of a solution of Na₂SO₃ and NaOH. The scrubbed water-saturated waste gas is liberated from condensate in a droplet separator.
Subsequently the waste gas from the remaining wind boxes is mixed with the waste gas from the droplet separator and is discharged through existing facilities into the atmosphere.

The SO₂ containing washing liquid flowing off from the washing system is again treated in two cycles. In the first cycle a solids separation and in the second cycle the lye regeneration by gypsum decomposition in a partial stream are achieved.

The slurry obtained is dehydrated in a continuously operating rotary filter. In the second stage the sodium sulfite or sodium sulphate contained in the water is converted to calcium sulfite or calcium sulphate by the addition of lime hydrate. The contained iron oxide serves as catalyst. By the addition of lime hydrate and the conversion of sodium sulphate to calcium sulphate the soda lye is liberated and recycled. Consequently, the losses of soda lye are to be compensated only. The dehydrated gypsum slurry (about 2.5 tpd) has to be mixed with suitable additives before it can be dumped.

Sintering of pellet feed

In connection with the investigations concerning the liquid ore conveyance, sintering tests with the material, which in this case was present as pellet feed, were carried out. Some of the test results are given in fig. 12.

The used ore mixtures were composed of 43.8 % of calcareous minette ores and 56.2 % pellet feed or of 30 % calcareous minette ores and 70 % pellet feed. The grain size of the calcareous minette ores was 0 to 10 mm with the fines portion -0.045 mm being 5.1 %, while the grain size of the pellet feed was -0.125 mm with the fines portion -0.045 mm being 81.9 %. The specific surface of the pellet feed was 2200 cm²/g.
Production of calcium ferrite sinter

In an iron and steel works, in which mainly pellets are smelted, it is possible to mix the circulating materials such as flue dust, mill scale, LD dust etc. with the necessary basic materials and to sinter this mix. In this way it would not only be possible to remove the acid contained in the basic additives outside the blast furnace but also to convert the circulating materials into suitable burden constituents.

The sinter product rate required for a fully prepared burden is low so that a very high percentage of pellets can be used for the burden. When processing acid pellets, e.g. a calcium ferrite sinter product portion of as low as 20% would be required even at an SiO₂ content of up to 7% and a CaO content of below 1% in the pellets, to obtain a self-fluxing burden. Without this burden preparation about 350 kg limestone and dolomite per ton pig iron would have to be charged so that a production decrease by about 15% and an equal increase in the coke consumption would have to be expected.

Compositions of sinter mixtures for the production of calcium ferrite sinter are given in fig. 13. Fig. 14 shows sinter product characteristics. Due to the partially high mill scale and flue dust content of the mixtures a low fuel addition is required only. The sinter production is about 30 t/m² suction area and 24 hours.
FIGURE 4

FIGURE 5
LEGEND

SM = SINTERING MACHINE
SA = SINTERING AREA
CA = COOLING AREA
F = COOLING AREA / SINTERING AREA RATIO
SC = SINTER COOLER

TYPE 134 = CIRCULAR DIP RAIL COOLER 202 SQ M
TYPE 135 = CIRCULAR DIP RAIL COOLER 243 SQ M

INVESTMENT COSTS COMPARISON IN PER CENT
CONVENTIONAL PLANT / COOLING ON STRAND PLANTS

APRIL 1974
PRINCIPLE OF RETURN FINES - AND FE++ - CONTROL
CONTROL OF Fe** CONTENT

FE** SINTER (%)

RK SETPOINT (%)

PILE - CHANGE

18 22 6 10 14 18 22 2 6 10 14 18 22 22.6.74

Lab Permagagn

20.6.74 21.6.74 22.6.74
**S1: ANALOG CONTROL**

Period: May 1974

**S2: DIGITAL CONTROL**

Deviation from Setpoint

Comparison of Variation Ranges
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<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
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<td>Basicity CaO/SiO₂</td>
<td>0.9</td>
<td>0.9</td>
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<td>1.4</td>
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<tr>
<td>Slag content %</td>
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<td>6.9</td>
<td>8.8</td>
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<tr>
<td>Return fines size mm</td>
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<td>5</td>
<td>5</td>
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<td>5</td>
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<td>Return fines rate %</td>
<td>80</td>
<td>70</td>
<td>70</td>
<td>70</td>
<td>70</td>
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<td>Charcoal /t of sinter kg</td>
<td>152</td>
<td>112</td>
<td>113</td>
<td>87</td>
<td>80</td>
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<tr>
<td>Sinter output t/m²·24 h</td>
<td>33.0</td>
<td>44.1</td>
<td>39.5</td>
<td>41.9</td>
<td>43.5</td>
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<tr>
<td>ASTM-index +6.3 mm %</td>
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<td>42.8</td>
<td>47.0</td>
<td>45.7</td>
<td>46.7</td>
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<tr>
<td>FeO content in sinter %</td>
<td>21.4</td>
<td>15.7</td>
<td>10.5</td>
<td>6.1</td>
<td>5.2</td>
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</table>

**Figure 10**
FIGURE 11

GEHALT IM SINTERABGAS ÜBER DIE LANGE DER SINTERMASCHINE

C73-1103
<table>
<thead>
<tr>
<th>Test</th>
<th>Pellet feed rate</th>
<th>CaO SiO₂ ratio</th>
<th>Burnt lime add.</th>
<th>Moisture of sinter mix</th>
<th>Bed height</th>
<th>Time sintering cooling</th>
<th>Return fines balance</th>
<th>Spec. production sintering cooling</th>
<th>Sintering+cooling</th>
<th>ISO Tumble test +6.5 mm -0.5 mm</th>
<th>Coke consumption kg/t sinter</th>
<th>Sinter to cooling area ratio</th>
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</thead>
<tbody>
<tr>
<td>Nr.</td>
<td>%</td>
<td>%</td>
<td>%</td>
<td>cm</td>
<td>min</td>
<td>min</td>
<td>%</td>
<td>t/m²-24 h</td>
<td>%</td>
<td>%</td>
<td>kg/t sinter</td>
<td>ratio</td>
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<td>56.2</td>
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<td>-</td>
<td>9.2</td>
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<td>-</td>
<td>9.1</td>
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<td>6.7</td>
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<td>27.0</td>
<td>16.5</td>
<td>42.3</td>
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<td>1.75</td>
<td>-</td>
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<td>4.6</td>
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<td>27.9</td>
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<td>56.6</td>
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<td>6.8</td>
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<td>26.3</td>
<td>16.7</td>
<td>45.7</td>
<td>51.1</td>
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</table>

Constant conditions: Return fines 60%  
Recycling time 3 min  
Suction sintering 157 mbar  
Cooling 59 mbar

Figure 12
Figure 13: Sintermixtures for producing of calcium ferritsinter

<table>
<thead>
<tr>
<th>Composition of mixtures in % (dry basis)</th>
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<tbody>
<tr>
<td>Mixture</td>
</tr>
<tr>
<td>Flue dust / slurry</td>
</tr>
<tr>
<td>Dust of electrical precipitator</td>
</tr>
<tr>
<td>Mill scale</td>
</tr>
<tr>
<td>Fine ore</td>
</tr>
<tr>
<td>Limestone</td>
</tr>
<tr>
<td>Moisture of mixture</td>
</tr>
<tr>
<td>Loss on ignition</td>
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<tr>
<td>Basicity</td>
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Note: In the mixtures 1 to 3 the mill scale portion was raised at a lower dust percentage; The mixtures 4 to 6 correspond to compositions as they were found for a study based on the use of dumped dusts.
**Figure 14**: Data for producing of calcium ferrite sinter

<table>
<thead>
<tr>
<th>Mixture</th>
<th>Return fines rate&lt;sup&gt;x&lt;/sup&gt;</th>
<th>Coke breeze</th>
<th>Moisture of sintemix</th>
<th>Bed height</th>
<th>Suction</th>
<th>Hydration test&lt;sup&gt;xx&lt;/sup&gt;</th>
<th>ASTM tumble test</th>
<th>Return fines balance</th>
<th>Sinter output&lt;sup&gt;24 h&lt;/sup&gt;</th>
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</thead>
<tbody>
<tr>
<td>Nr.</td>
<td>%</td>
<td>%</td>
<td>%</td>
<td>cm</td>
<td>mbar</td>
<td>% -3 mm</td>
<td>% -3 mm</td>
<td>%</td>
<td>t/m² 24 h</td>
</tr>
<tr>
<td>1</td>
<td>30</td>
<td>1</td>
<td>6.7</td>
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<td>7.32</td>
<td>64.8</td>
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</table>

<sup>x</sup> referred to dusts, mill scale, ore and limestone

<sup>xx</sup>5 kg sinter of 8 - 20 mm are charged into a container. Furthermore, a vessel containing 90 g water is placed into the container. The container is closed and a temperature of 105°C is maintained for 16 hours. This test serves to evaluate the resistance of sinter product to storage. Values for self-fluxing sinter product: 3 to 6 %.