MANHEIS PROCESS OF COPPER CONVERTING.

GRADUATION THESIS.

W. J. WAKEFIELD

'02.
MANHE'S PROCESS OF COPPER CONVERTING.

As early as 1867 experiments were made in Russia and later in 1878 in England to try to treat cupriferous substances by a process similar to that applied by Bessemer to the refining of cast iron, without using any other fuel than the silica manganese and carbon or phosphorus in the pig.

The idea was undoubtedly originated by the apparent similarity of the properties of copper matte to those of pig iron. In the case of the iron the impurities are removed by simply blowing air through the fused mass. Consequently it would be reasonable to suppose that sulphur and iron could be removed from mattes by similar means since iron and sulphur both have a greater affinity for oxygen than copper. But notwithstanding the apparent similarity there are important differences.

In pig the amount of impurities to be oxidized never exceed 10% while copper mattes may contain as much as 70% of impurities, chiefly sulphur and iron. So that a much larger amount of impurities must not only be oxidized but separated from the copper. Since all the products of oxidation are not gaseous they must be eliminated by some other means than volatilization. Then again the impurities in pig iron develop a much greater amount of heat during the combustion as is shown by the following equations:

I. \[ \text{Si} + 2 \cdot \text{O} = \text{SiO}_2 + 7800 \text{ calories} \]

II. \[ \text{C} + 2 \cdot \text{O} = \text{CO}_2 + 8000 \text{ calories} \]

Total = 15800 calories.
As against,

I. \( \text{Fe} + 0 = \text{FeO} + 2200 \text{ calories} \) (1576 calories by Peters).

II. \( \text{S} + 2 \cdot 0 = \text{SO}_2 + 1500 \) " (2200 " " " ).

Total = 3700 "

Also there must be a considerable deduction of heat for the disassociation of the sulphides of iron and copper in the case of copper converting which does not occur in iron refining. The process is only made possible by the specific heat of copper being two-thirds less than that of iron and the weight of copper in the matte being much less than the weight of an equal amount of pig. So that the process requires less than the refining of cast iron.

The experiments of Pierre Manhes, a Frenchman, were commenced in 1880. The apparatus employed being a small Bessemer converter of the usual type, having an air vessel at the bottom an vertical tuyeres for injecting air upwards through the metal collected in the bottom of the converter. As was expected the iron and sulphur oxidized the former formed a liquid slag at the expense of the refractory lining.

At the commencement of each experiment the heat developed was sufficient to maintain the fluidity of the metal and slag, but at a certain period the progress of the operation suddenly changed. The regular bubbling of the mass, produced by the air blast, was followed by a rapid boiling accompanied by violent projections of the slag. This action becoming so intense as to empty entirely the vessel.

On the other hand as the copper had been liberated it accumulated in the bottom of the converter, where, from the disappearance of the combustible elements, it became chilled and gradually obstructed the tuyeres with solid copper.
In order to obviate the difficulty arising from the cooling of the copper the tuyeres were placed so as to be horizontal when in blast. This permitted the copper to settle in the bottom and not to be subjected to the chilling influence of the blast. At first there was great difficulty in disposing of large quantities of ferruginous slags. This was partially obviated by blowing twice. That is, a low grade matte was blown up to about 72% Cu then poured, and after breaking up it was refused and blown up to blister copper.

The experiments proved that the process was practicable, and led to the establishment of a converter plant at Éguillies France. Later the Manhes patents were purchased by the Parrot Silver and Copper Co., of Montana who put up the next converter plant and the first converters in the United States at Butte Montana. Here, as was the practice in France, the matte was first blown up to 75% - 80% Cu in one operation then it was remelted and blown up to blister copper, 99.15% pure. The original stationary converters, turned by hand and subsequently by steam were transformed into interchangeable converters of larger capacity which were tilted by hydraulic power.

In 1886 Schumacher demonstrated that mattes carrying as low as 45% Cu could be converted with ease in one operation. Since then mattes as low as 30% Cu have been successfully treated. This of course made the process cheaper and led to a more general use.

CONVERTERS.

There are two general shapes or types of converters now in use.

I. The Round, or Parrot converter which is used by the Parrot Silver and Copper Co. and the Montana Ore Purchasing Co. at Butte. This also includes the modified types used at Anaconda and Great Falls Montana.
2. The Cylindrical Trough, or modified Leghorne used at Bisbee, Cananea, and Jerome. This converter is similar to those of the first class in the general shape, but instead of having the nose at the end it is at the side. And it is made of two longitudinal instead of three transverse sections.

To these might be added another class known as the Stalmann, or Old Anaconda converter which resembles the Parrot somewhat in general appearance but has a square shape which is intended to lessen the corrosion of the lining and to prevent the blowing out of the molten metal. It has been discarded and is now not in use anywhere.

In the two types there is a decided difference in shape which may become an important matter in the future. The trough shape converter admits of a gradual tilling of the bowl during the operation by which means the blast may be forced only through the superincumbent layer of the matte and slag, thus requiring less blast pressure and preventing the blast from coming in contact with the reduced copper to any extent. While this would not seem to be an advantage, because the blast obviously will not be in contact with the matte for so long a time but owing to the decrease in amount of air which must necessarily result from reducing the blast from one-half to two-thirds, in this style of converter, the oxidizing power would not seem to be so great. The practical results however show that the oxidizing influence is sufficient to convert the matte fast enough when other factors are taken into consideration. This light blast makes a saving in three ways:

1. In compressing the air.

2. Much less dust is made and therefore it costs less to build dust chambers, since they may be much shorter to obtain the same results in
dust saving.

3. Saving in precious metals. Experience has shown that the loss of silver is much less when a light blast is used.

On the other hand the elimination of lead, zinc, arsenic, antimony, bismuth, selenium and tellurium might be less although probably not to any appreciable extent owing to the volatility and slaggability of these elements. Also, it would seem that this trough shape converter is a most suitable type since the Anaconda Copper Co. in their new reduction plant just finished, has put in eight stands for this style of converter. The converters are eight feet long by thirteen feet high and are fitted with eighteen L 1/4 inch tuyeres with automatic ball closing valves. This gives a very large converter having more tuyeres with much larger openings than any in use in Arizona. This converter was first used at Bisbee Arizona. It was introduced there by Dr. James Douglass and it has been mostly through his efforts that it has been perfected.

The following table taken from J. D. Peters Copper Smelting shows the sizes, capacities, blast pressure etc. generally used.

<table>
<thead>
<tr>
<th>Style of Converter</th>
<th>Outside Height Feet</th>
<th>Outside Diameter Feet</th>
<th>Blast Temperatures Approx.</th>
<th>Initial Charge Lbs.</th>
<th>Maximum Charge Lbs.</th>
<th>Weight Shell Baking Lbs.</th>
<th>Blow Air, zhekov</th>
<th>Number of tuyeres</th>
</tr>
</thead>
<tbody>
<tr>
<td>Harriet (New Hampshire)</td>
<td>8 5</td>
<td>5</td>
<td>11</td>
<td>2500</td>
<td>9000</td>
<td>16000</td>
<td>16</td>
<td>16</td>
</tr>
<tr>
<td>Harriet (Great Falls)</td>
<td>10 6</td>
<td>6</td>
<td>13</td>
<td>7000</td>
<td>17000</td>
<td>22000</td>
<td>12</td>
<td>16</td>
</tr>
<tr>
<td>St. Lawrence</td>
<td>13 7</td>
<td>7</td>
<td>16</td>
<td>10000</td>
<td>22000</td>
<td>26000</td>
<td>10</td>
<td>18</td>
</tr>
<tr>
<td>Copper Queen</td>
<td>7 25</td>
<td>5.67 x 8</td>
<td>5 2</td>
<td>4000</td>
<td>10000</td>
<td>18000</td>
<td>12</td>
<td>18</td>
</tr>
</tbody>
</table>
The Copper Queen converter which is barrel shape is made in two sections and held together by bolts through angle irons. The top piece which contains the opening or nose has four lugs rivetted to it in which chains can be hooked and the hood removed by a differential pulley or an overhead crane. Unlike the other converters it is not swung on trunnions but cradled on two pair of rollers and are revolved by a vertical rack and sector or in the later types to be used at Cananea and elsewhere by rack and pinion. Operated by a vertical cylinder. The blast pipe is centered opposite the axis of revolution and attached by a stuffing box to a pipe leading to a wind-box on the converter shell. The lower section is made of 5/16 - 7/16 inch boiler iron according to the size. The hood is made of slightly lighter material which is reinforced by plates around the nose. This form of converter is fitted with eleven tuyeres having 3/4 inch openings and automatic ball socket holes.

The Parrot converters are usually made in three sections which are of boiler iron with cast iron trimmings. The thickness varies with the size of the converter and is generally about 15/16 inch. These sections are held together with cast iron flanges bolted with 1/4 inch bolts. About 14 bolts are used to hold the top and middle sections and a somewhat larger number for the bottom joints. As the two lower sections are rarely taken apart they are sometimes made in one piece. A cast iron box is fastened to the bottom forming the wind-box which contains about 16 horizontal tuyeres. These tuyere holes are closed on the outside by wooden plugs which may be removed for punching. The converters are supported on stands which are fitted with journals to carry trunnions to which the body or shell is fastened. One of the trunnions is cast solid and one hollow to carry the blast to the wind-box. This style like the
Copper Queen converter is tilted by horizontal or vertical hydraulic cylinders to the piston of which is attached a tooth rack that meshes with a pinion fastened to the hollow trunnion. In the case of the Copper Queen converter the pinion is riveted to the shell. The rack is long enough and has enough motion to allow a complete revolution. The tilting is regulated by crutchlow valves and both the levers operating these valves and the levers regulating the blasts are placed together on a platform and the converters are controlled from there. The hydraulic pressure is usually furnished by a small plunger pump but at Bisbee static pressure from a reservoir 400 feet above the works is used. The pressure is 150 - 250 pounds to the square inch.

PLANT.

An ideal arrangement for a pneumatic reduction plant would be one in which the furnaces could be tapped into a reverberatory foreherth and the molten matte run from there into the converters. The chief advantage of this arrangement would be:--

1. The remelting of the matte is unnecessary.
2. Reduction in the cost of fuel for the furnace owing to the charge being melted only enough to be tapped into the foreherth.
3. Saving by discarding the ordinary foreherths which chill up and burn out.
4. Better separation of the matte from the slag and a more uniform product.
5. The reverberatory foreherths would furnish a storage vessel for the matte and thus make the converter and furnace department more independent of each other and allow irregular running of both the converters and furnaces which is often necessary.
6. The tenor of the matte may be regulated to suit the condition of the linings.

In the case the arrangement is such that the matte cannot be brought direct from the foreherths to the converters or the reverberatory foreherths it is necessary to cast the matte into moulds and after breaking it up to take it to the cupola furnace situated so that the molten material can be run directly into the converters through a silica lined trough. This remelting causes loss in three ways:

1. Use of 12% coke, extra labor, and lining for cupola.
2. Loss in flue dust of about 1%.
3. Loss of metals in the slag.

The cost of remelting will be about $4.00 per ton or 2/10 of a cent per pound when the coke costs $12.00 a ton and wages are high.

The converters are usually arranged in sets of three, and have three extra shells and two tops to each set or stand. Each converter discharges its gases into individual flues which are provided with heavy steeply inclined iron plates which admit of being cleaned of the matte and copper which are blown out of the converter upon them.

There are several ways of handling the converters, either by hand steam, hydraulic, or electric cranes or by cars. The cars are fitted with hydraulic jacks so that when they are run under the converter it is easily lifted off its supports and can then be run to the lining department. The most approved method is by electric cranes. At the new reduction works just finished at Anaconda in the converter department the converters and hot metal ladels are handled by two 60-ton electric cranes.
the repair parts by a 40-ton crane running 20 feet above the heavier cranes. In the casting house a 20-ton crane is used to handle the ladels and moulds.

The blast is furnished by horizontal compressors. In the latest type this consists of a compound condensing Corliss engine with 18-inch and 32-inch cylinders and a 42-inch stroke. The air cylinders are 42 x 42 inches and to supply a set of three converters, with two converters in use all the time, they must furnish 3000 cu. ft. of air at a pressure of 12-16 pounds per sq. inch per minute. An engine of the size mentioned must make about 50 revolutions per minute to furnish this amount of air. The trough shaped converters require a much lower blast pressure, about 5 \(\frac{1}{2}\) pounds, and are therefore much more economical in this respect than the other converters. 20\% on an average of the entire cost of converting with the Parrot style converter goes to pay for the blast.

**Linings.**

When the Bessemerizing of copper was first attempted at Montana considerable difficulty was experienced in getting a suitable lining. It was expected that a lining would last a week as is the case in steel making. When it was found that it would not, several attempts were made to protect the lining by water jacketing. First the lining above the tuyeres where the corrosion was the greatest was cooled by imbedding coils of pipe around the tuyeres. This showed by protecting the lining, at one point, the corrosion was transferred to another so that the whole interior was water jacketed. This attempt like the first was a failure for as soon as the pipes were exposed the charge ran cold because there was no SiO to flux the FeO and the mass became sticky and would not let the

# See Bixons Lead and Copper Smelting for more complete description of the experiments.
blast through it. A basic lining of CaO and MgO the same as used in the basic Bessemer process was next tried. The result was a pasty mixture of lining, copper and iron oxide from which the copper could not be separated. Then to prevent the corrosion of the lining silica was introduced into the charge with the blast. But this was not successful because the silica is only needed during a comparatively short period and it would be impossible to introduce enough to combine with the FeO in it would do so. Silica had been added to the charge in all sorts of forms and ways even as melted silicates and in every case it was found that it preferred to collect in little globules and float on the top of the charge. Then it seems that it is necessary that the lining should be corroded and that the problem of linings consists in relining the converters as quickly and as cheaply as possible and making a change of vessels in the least possible time when the lining becomes so thin that the ratte is in danger of breaking through the sides.

The lining universally employed is a mixture of quartz and clay. The clay is used to hold the particles of silica together and to help withstand the wearing effect of the blast. A highly graphitic clay found near Benson is used at Bisbee. At Anaconda clay containing much iron and lime is used. The clay need not necessarily be pure, but it must be plastic and sticky. On the other hand the quality of the quartz is quite an important matter. Experience shows that the purer the quartz the better and any having less than 98% silica will not do. The quartz at Bisbee is obtained from boulders picked up in the washes, while at Cananea a highly silicious gangue carrying values in copper will be used.

The quartz is first run through Blake crushers which will prepare the material for Cornish rolls. The pulverized silica is next mixed with the proper amount of clay, usually about 10-12%, which has been pulverized
in a Chilian mill. The mixture is next made into a paste of the required 
constituency and rammed or tamped into the converters. This tamping is 
usually done by hand but the Anaconda Co. now has a machine which it is 
claimed will make a better and more permanent lining than can be made by 
hand.

Chemistry of the Process.

Bessemerizing of copper matte as is made possible by the greater affini-
ity of sulphur for copper than for iron or by the great affinity of oxygen 
for iron and its slight affinity for copper. So when a copper matte, which 
for all practical purposes may be considered as a mixture of the cuprous 
sulphide of copper CuS and the ferrous sulphide of iron FeS, is subjected 
to the oxidizing influences the iron will begin to oxidize at once but the 
iron is oxidized and slagged off. Then the copper begins to oxidize 
rapidly and liberates large volumes of SO₂ as shown by the following:-

1. 

\[ \text{CuS}_\text{f} + \text{SO}_2 \rightarrow \text{CuS}_\text{f} \cdot \text{SO}_2 \]

The evolution of the gas is violent and if allowed to continue would 
completely empty the vessel. If the slag is skimmed off just before the 
copper begins to oxidize this would be avoided. After the skimming off 
the copper oxidizes as before but so long as there is sulphur present in 
the form of SO₂ the reduction of the copper will take place just as ra-
pidly as the oxidation. So there is a constant precipitation of copper 
and liberation of SO₂ according to the following equation

11. 

\[ 2\text{CuO} + \text{CuS}_\text{f} \rightarrow 6\text{Cu} + \text{SO}_2 \]

The precipitated copper settles to the bottom of the vessel and away from 
the oxidizing blast and remains there in a molten condition while the SO₂ 
escapes and the little remaining iron forms a slag, at the expense of the 
silica of the lining, and floats on the top and protects the copper from 
further oxidation near the end of the operation. The common impurities in 
the matte are lead, zinc, bismuth, arsenic, antimony, nickel, cobalt,
#12.

selenium and tellurium. Over 70% of each of these with the exception of the last two is either volatilized or goes into the slag. On an average 95% or more of the lead and zinc are eliminated.

The heat that keeps the charge molten is furnished in three ways.

1. By the oxidation of S. I.S 20 300 1500 calories
2. " " " Fe. II. Fe 0 FeO 2200 "
3. " combination FeO and SiO

The heat furnished by the combination of FeO and SiO is not known but is probably small compared with the heat furnished by the other two reactions.

As a large portion of the heat is furnished by the oxidation of the iron, a low grade matte will give a much higher amount of heat to the mass than a high grade matte which does not contain so much iron. This is borne out by experiment for a charge of low grade matte will smelt half or more of its weight of a chilled charge left in the converter while a charge of high grade matte would only make matters worse.

Then the total heat available to keep the charge molten will be the sum of the heat evolved by the three above reactions minus the heat lost by the disassociation of the several sulphides present.

Experience has shown that mattes lower than 45% contain so much iron and evolve so much heat that the lining is worn away too rapidly. On the other hand mattes containing more than 60% copper do not contain enough iron to keep the mass molten enough near the end of the operation.

So 45%-60% mattes are the limits within which it is economical to convert. That is as far as the process of converting alone is considered.

But the metallurgist has to determine in which of the operations, the

These figures are given just reversed by Peters. This is probably a mistake since the behavior of different grades seems in the converters to indicate such.
smelting, or the converting, the concentration of the copper is best
carried the farthest. In the case coke is very expensive, as it is in
many places in the West, it would pay to convert a lower grade of matte
and pay more for linings; here power is cheap and good quartz and clay are
easily accessible.

The solid products of a converter are slag and copper of different
degrees of purity depending on the point at which the process of oxidation
is stopped. The slag is essentially a silicate of iron which may be repre-
sented by the formula\[FeO \cdot SiO_2\]. The amount of iron of course depends on
the tenor of the matte used but strange as it seems a high grade matte
within certain limits gives a more ferruginous slag than a low grade matte.
The slags contain from 1½-5% copper and as they are generally useful as
fluxes, they are cast into bricks and added to the blast furnace charge
and in this way there copper contents are saved.

There are three kinds of copper made. The first is composed of about
95% copper and 5% slag. It has to be refined before being cast into
anodes and is only made when the charge is to cold or the lining too thin
to finish the blowing. The second is called gas finish copper, 96-97% pure
on account of the \(SO_2\) gas which being dissolved in the molten copper is
expelled on cooling. The third called blister copper, on account of the
blister like spots on the surface, is most often made. In order to make
this grade of copper it is necessary to over-blow slightly, thus forming
some

Blowing a Charge.

After the converter is refined it is dried by blowing a hot blast or
burning fuel in it with a light blast. It is then taken to the converter
stand and the blast connection is made, after which it is turned down
to get rid of the ashes and then tilted in such a position that when
the molten matte is poured in by the crane or run in from the cupola the tuyere holes will be above the level of the liquid material. The blast is turned on and the converter tilted up to its normal position.

The flame that then issues from the nose is dense white, due to the volatilization of the zinc, lead, antimony, arsenic, and etc. and somewhat tinged with green and red which gives way to blue when the iron is all oxidized. The converter is now turned down and the blast shut off as soon as all the tuyeres are exposed. Then the slag is either rabbled or poured off as much as possible. Cold matte rich in iron is now added to start the oxidation of the CuS and the converter tipped up after the blast is turned on again.

The flame color is now very variable and depends on the size and the temperature of the charge. The colored flame gradually decreases in volume and length towards the end of the operation, until, at the end there is only a bright red flickering.

These final changes of color are too faint and variable to rely on so the nose is watched for the appearance of spark-like metallic globules which at first stick to the inside of the hood but soon come in greater quantities and cease to glow, and rebound sharply from the walls of the hood. At this point the converter is turned down and the metal poured into moulds on the car which runs on a track underneath the converter. The moulds are made of soft cast iron and hold 200-300 pounds of copper. The amount of copper produced per 24 hours is about ten tons for one stand.

The converter immediately after pouring is ready for another charge. The second charge is poured in as before onto the slag which is left from the previous operation. The life of the lining depends on the grade of matte used and on the quality of the clay and quartz. If the clay is not plastic or the mixture poorly ground the lining might collapse during
The first charge. If the material and the work of the lining are first
class the lining will stand on the average about eight blows. It takes about one hour and a half per blow.

Elimination of Impurities.

In the converter process the several stages of the reverberatory process such as calcining the matte, smelting the resulting product for copper bottoms, retreating the resulting rich slag and matte, and refining the copper produced before its being cast into anodes for the electrolytic process. All these processes which require three or four days operation, are condensed into about an hour and a half. The object of the two processes is to remove the impurities and to reduce the copper. In each case the iron is practically all removed and the other impurities are in varying degrees of quantity. The following results of treating the same matte by the two different processes are given by Edward Keller of Montana.

Elimination of Impurities by Reverberatory Process.

<table>
<thead>
<tr>
<th>Element</th>
<th>Percentage</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pb</td>
<td>99%</td>
</tr>
<tr>
<td>Bi</td>
<td>54%</td>
</tr>
<tr>
<td>Sb</td>
<td>50%</td>
</tr>
<tr>
<td>As</td>
<td>21%</td>
</tr>
<tr>
<td>Se &amp; Te</td>
<td>60%</td>
</tr>
</tbody>
</table>

Elimination of Impurities by the Converter Process.

<table>
<thead>
<tr>
<th>Element</th>
<th>Percentage</th>
</tr>
</thead>
<tbody>
<tr>
<td>Pb</td>
<td>99%</td>
</tr>
<tr>
<td>Bi</td>
<td>96%</td>
</tr>
<tr>
<td>Sb</td>
<td>66%</td>
</tr>
<tr>
<td>As</td>
<td>91%</td>
</tr>
<tr>
<td>Se &amp; Te</td>
<td>71%</td>
</tr>
</tbody>
</table>

Varying amounts of these volatile elements will change the position of the elements with regard to the degree of elimination. But the order of lead
lead, zinc, bismuth, arsenic, antimony, selenium, tellurium is generally followed and in general it may be said that when the amounts of lead and arsenic are each greater than the amounts of bismuth and antimony respectively the elimination of the lead and arsenic will be the greatest, and when the order is reversed bismuth and antimony will be most eliminated.

The following table shows the total average elimination of impurities taken from all the results that could be found in the converter and the reverberatory processes of refining of impurities.

<table>
<thead>
<tr>
<th>Elements</th>
<th>Reverberatory</th>
<th>Converter</th>
</tr>
</thead>
<tbody>
<tr>
<td>Zn</td>
<td>98%</td>
<td>100%</td>
</tr>
<tr>
<td>Pb</td>
<td>99%</td>
<td>99%</td>
</tr>
<tr>
<td>Bi</td>
<td>54%</td>
<td>96%</td>
</tr>
<tr>
<td>Sb</td>
<td>50%</td>
<td>68%</td>
</tr>
<tr>
<td>As</td>
<td>21%</td>
<td>82%</td>
</tr>
<tr>
<td>Se &amp; Te</td>
<td>60%</td>
<td>50%</td>
</tr>
</tbody>
</table>

The behavior of selenium and tellurium is very variable in the converter. The results show an elimination of 2% - 90%.

Very little gold is lost but as silver is somewhat volatile and especially so in the presence in the volatile metals and metalloids and when a high pressure and violent oxidizing agents are used, the loss in volatilization is some times as high as 40% and although this loss is to a great degree prevented by dust chambers it constitutes about the only serious disadvantage of the process of converting.

Cost of Converting.

The cost of converting copper where reverberatory foreharthats are used and cheap power is available is about $6.00 per ton or three tenths of a cent per pound but ordinarily these figures are about one to three tenths of a cent too low as the plants as a rule are not arranged so that cranes or reverberatory foreharthats can be used.

As has been said, Butte Montana is the place where this great process has been perfected. This method has made it possible to treat in a compar-
itively short space of time the enormous output of great mines where labor is high, fuel expensive and the plant thousands of miles from where the copper is used.

The new reduction works just finished at Anaconda are the largest in the world, but they would have been unmanageably large had they been on the old English roasting and reverberatory furnace lined. The new methods now in use have made it possible for the copper mines to survive the steady decline in the price of the product. Formerly to treat 200 tons of easily fusible and well-roasted ore would have required 20 small brick furnaces. The matte produced was roasted at least five times in stalls and then remelted for black copper and white metal. The product was then brought up to blister copper in a reverberatory furnace. To-day the matte is run directly from the blast furnaces to a reverberatory former hearth and from there into the converters and within an hour and a half it is reduced to almost purity, without any handling or consumption of fuel except what is required to furnace the blast. It formerly took about 200 days, or one days labor to the ton of ore, to do what is now done in fifty days or one day's labor to four tons of ore. These vast advantages which have here been outlined are the result of scientific investigation and modern engineering and metallurgical skill of American metallurgists.

May 15, 1902.

Walter J. Wakefield