All That Glitters: Readings in Historical Metallurgy

Edited by
Michael L. Wayman

THE METALLURGICAL SOCIETY OF THE CANADIAN INSTITUTE OF MINING AND METALLURGY
Preface

ON A FAVOURITE CAT,
DROWNED IN
A TUB OF GOLD FISHES

'Twas on a lofty vase's side
Where China's gayest art had dyed
The azure flowers that blow,
Demurest of the tabby kind
The pensive Selima, reclined,
Gazed on the lake below.

Her conscious tail her joy declared:
The fair round face, the snowy beard,
The velvet of her paws,
Her coat that with the tortoise vies,
Her ears of jet, and emerald eyes -
She saw, and purr'd applause.

Still she gazed, but 'midst the tide
Two angel forms were seen to glide,
The Genii of the stream:
Their scaly armour's Tyrian hue
Through richest purple, to the view
Betray'd a golden gleam.

The hapless Nymph with wonder saw:
A whisker first, and then a claw
With many an ardent wish
She stretch'd, in vain, to reach the prize -
What female heart can gold despise?
What Cat's averse to Fish?

Presumptuous maid! with looks intent
Again she stretch'd, again she bent,
Nor knew the gulf between -
Malignant Fate sat by and smiled.
The slippery verge her feet beguiled;
She tumbled headlong in!

Eight times emerging from the flood
She mew'd to every watery God
Some speedy aid to send: -
No Dolphin came, no Nereid stirr'd,
Nor cruel Tom nor Susan heard -
A favourite has no friend!

From hence, ye Beauties! undeceived
Know one false step is ne'er retrieved,
And be with caution bold:
Not all that attempts your wandering eyes
And heedless hearts, is lawful prize,
Not all that glisters, gold!

Thomas Gray (1716 - 1771)
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Introduction

Since November 1979, the Bulletin of The Canadian Institute of Mining and Metallurgy (CIM) has been publishing articles on historical and archaeological metallurgy in a regular monthly feature entitled "Historical Metallurgy Notes". The publication of these articles has been organized by the Historical Metallurgy Committee of The Metallurgical Society of CIM, a committee formed in 1978-79 at the suggestion of Peter Tarassoff to provide a forum for historical metallurgical activities within CIM. Since its inception, this group has operated under the successive chairmanships of P.J. Mackey, W.J. Thoburn and S. Zador. Contributions to Historical Metallurgy Notes have been provided not only by Committee members but also by many others with interests in the field; some 70 different persons have authored or co-authored the Notes, which have covered a wide range of topics. Now, after some ten years of articles, it is appropriate to look back over these contributions and to publish a selection in book form so that they will be available to an audience much broader than the readers of the CIM Bulletin.

The authors whose works are presented in this volume represent an enormous diversity of backgrounds and technical interests. Here are not only the expected professors of metallurgy and retirees from mining and metallurgical companies, but also leading scientists and engineers in our industry, as well as archaeologists, historians with interests in the history of technology, people connected with museums of mining and metallurgical history and professional archaeometallurgists. In recent years the application of the natural sciences to studies of the past, has made many significant contributions to our understanding of the ancient world in general and the development of technology in particular. As a result of chemical and metallurgical analyses of archaeological artifacts of metallurgical significance (including metals, ores, fuels, fluxes, slags, furnace and crucible materials, etc.) it is becoming possible to trace the history of the use of metals from earliest times.

The first metals to be used, likely as early as the ninth millennium B.C. in the Middle East, were those which occur in nature in metallic form: gold, silver, copper, platinum, meteoritic iron and the naturally occurring alloys electrum (gold-silver) and tumbaga (copper-gold). The earliest smelted metals of which we are aware come from Anatolia, south of the Black Sea in present-day Turkey. Here copper and probably lead smelting appeared first about the sixth millennium B.C. Later, artificially produced alloys became more common, beginning with arsenical copper and tin bronze. The Bronze Age of the Old World is most commonly associated with the period beginning about 3000 B.C., reaching its peak during the second millennium B.C. Soon after, it was succeeded, for reasons which are still in dispute, by the Iron Age. Our knowledge of the early smelting and use of iron is hindered by its tendency to corrode and thus disappear from the archaeological record, however, the widespread occurrence of iron ores made its eventual use inevitable. As time passed, other metals became known and used until we arrive in our present century with an enormous range of metallic materials found in an even wider range of applications.

Of course, metals and alloys are only useful if they can be produced in useful shapes and forms. As has been pointed out by Dr. Ursula Franklin, the noted archaeometallurgist and member of CIM's Historical Metallurgy Committee, the study of ancient metal artifacts reveals that metals as new materials were initially processed by technologies which were already in use for other materials. Early native metals were shaped by grinding and abrasion just as was stone. The early Chinese cast bronzes resemble in some respects products of the pottery industry which preceded them. Similarly in several parts of the world metal in the form of strip or wire was woven using pre-existing textile technologies. The development of the technologies used in the fabrication of metal and alloy objects is also considered in this volume.

From different parts of the world at different times, there also exists written records, which in some cases include details of technological processes. Hence, the later stages of the evolution of technology can be followed without having to rely completely on the archaeological record. Aspects of historical, as opposed to archaeological, metallurgy are topics of the majority of articles included in this book.

This CIM volume on historical metallurgy naturally has a strong emphasis on Canadian developments. Canada has, in fact, had an outstanding record in the development of modern metallurgy and metal extraction processes. Specifically Canadian innovations this century include the zinc suspension roaster (Cominco), the ammonia leaching of sulphide ores (Sherritt Gordon), the Inco flash smelting furnace, the Noranda continuous copper smelting process, and many others. However, the foundation of Canada's mining and metallurgical industry can be traced to earlier developments in the last century. Many of these developments are dealt with in this volume including descriptions of early goldcyanide plants, copper smelters, and the first lead, zinc, aluminum and nickel operations.

This volume is not intended to provide complete coverage of the history of metallurgy but rather, as its subtitle states, selected aspects of this history. The collection is subdivided into two major sections. The first
deals with the history of metallurgy world-wide, including early developments in the Mediterranean area, in Asia, and in the New World, presented in approximate chronological order. The second section deals specifically with Canadian developments beginning with the use of metal by the native peoples of the north and the "gold" exploitation by the Frobisher expeditions of the late 16th century, and continuing through the gradual development of the Canadian metallurgical industry. Here, following the introductory articles, the topics are grouped chronologically for each of the specific metals considered, beginning with iron and steel.

The Historical Metallurgy Notes enters its second decade of publication in the CIM Bulletin at a time when there is increasing activity in those fields of research which concern the history of metallurgy. With more and more people participating, the number of relevant publications growing, and more historical contributions at regular meetings of CIM and other technical societies, aspects of history are clearly playing an increasingly important role in our lives. It is expected that the future will see continued improvements in our understanding of the past in general, and the history of metallurgy in particular. It is hoped that the articles contained in this volume are able to convey a sense of the fascination associated with these developments in our understanding.

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The Development of Metallurgy
Native copper: humanity’s introduction to metallurgy?

Michael L. Wayman

Occurrence

Native copper is copper which exists in nature in elemental, metallic form, rather than being chemically combined as is the case with the more usual oxides, sulphides, carbonates, etc. Several other metals, including gold, silver and platinum also occur in native form as do, rarely, iron, lead and mercury; a few naturally occurring native metal alloys, notably electrum (gold-silver) and tumbaga (gold-copper or gold-silver-copper) are also known. An analysis of the relative availabilities of native metals as nuggets in a useful size range shows native copper to be by far the most abundant of the native metals, exceeding both gold and silver by at least two orders of magnitude. While such an estimate is crude, it serves to illustrate why copper was one of the first metals to have been used by ancient peoples.

Native copper is known to occur on all inhabited continents, although its exact occurrence and abundance in ancient times cannot be unambiguously determined because many surface deposits may have been totally explored and are therefore no longer identifiable. It is clear, however, that native copper does or did occur in all areas inhabited by the early metal-using cultures, for example in the Andes, in Central America and in the Near East. It is remarkable that most of the world’s native copper occurs in the deposits of the Lake Superior region, primarily on the Keweenaw Peninsula of northern Michigan and on Isle Royale, an island in Lake Superior some 50 miles north of the Keweenaw deposits. Canadian deposits of native copper are found in Newfoundland and Labrador, Nova Scotia, Quebec, Ontario, British Columbia, the Yukon and the Northwest Territories (on Victoria Island, Bathurst Inlet and the Coppermine River area). It was to locate the latter deposits that the explorer Samuel Hearne was sent overland from Hudson’s Bay in 1770; they were also surveyed by Sir John Franklin’s expeditions in the 1820s.

Formation

Several mechanisms have been proposed for the formation of native copper. Primary native copper forms directly from a copper-bearing hydrothermal solution. While the exact mechanisms remain uncertain, it appears that the presence of iron is important, the reduction of copper being associated with the reaction of haematite, or alternatively, ferrous sulphate. One proposed reaction is

\[ \text{Cu}_2\text{S} + 3\text{Fe}_2\text{O}_3 + 5\text{H}_2\text{SO}_4 \rightarrow 2\text{Cu} + 6\text{FeSO}_4 + 5\text{H}_2\text{O} \]

The native copper found in the basaltic lavas of the Lake Superior region and the Canadian Arctic are examples of primary native copper.

Native copper can also form by the reaction of copper-bearing solutions with organic matter in swamps containing peat and bog iron, and also by the replacement of wood in old mine timbers exposed to copper-rich mine waters.

Secondary native copper forms typically as the result of the weathering of copper-iron sulphide deposits. Here pyrite dissolves in ground water to form iron sulphate and sulphuric acid, which dissolve the copper. The upper, weathered parts of the deposit thus become depleted in copper, which is carried down and deposited just below the water table, so that a secondary enrichment of copper sulphides occurs in that region. Just above the water table, native copper forms by reactions such as

\[ \text{Cu}_2\text{S} + 3\text{Fe}_2\text{O}_3 + 4\text{H}_2\text{O} \rightarrow 2\text{Cu} + 6\text{FeSO}_4 + 4\text{H}_2\text{SO}_4 \]

Deposits of secondary native copper were recognizable to ancient peoples because above the native copper, the weathered, depleted zone was capped with a brightly coloured gossan, an iron-stained deposit of silicate clays and hydroxycarbonates. The gossan became even more important to metallurgy with the development of smelting because it made an appropriate flux for the smelting of the oxidized ores of copper.

The native copper of the Lake Superior region occurs in several forms. Geologically, these deposits occur in a series of lava flows, separated by layers of conglomerate. Copper occurs in amygdaloid form in the blow holes near the top surface of each lava flow. Copper is also found replacing some or all of the cement which forms the matrix of the conglomerate as well as in fissures and veins which lie parallel to or cut across the bedding planes of the lava flows.

In addition, large quantities of native copper, removed from the deposit by the action of glaciers and other erosional forces were carried tens or hundreds of miles from the original lodes and deposited as “float copper” in stream beds and other alluvial and glacial deposits. Float copper was a prime source of native copper for ancient peoples. However in some areas, including the Lake Superior district, true hard-rock mining operations, much more difficult than the gathering of float copper, were carried out.

Prehistoric Copper Mining in the Lake Superior District

In the Old World, native copper artifacts have been found which date back to the 8th or 9th millenium B.C. In North America, it is believed that native copper was being used at least as long ago as the 4th millenium B.C. although well dated copper artifacts of this age are rare. It is clear, however, that large amounts of native copper were mined between 4000 B.C. and 1000 A.D. from the Keweenaw Peninsula and Isle Royale. These regions contain the remains of ancient mines, in the form of pits and trenches. Typically, an individual pit was 20 to 50 feet deep and 10 to 30 feet in diameter, containing about 1000 tons of rock and copper. In some cases, pits were mined adjacent to each other, forming trenches, with ledges of

rock left between individual pits for control of water. It is estimated that at least 5000 such pits were dug by ancient peoples in the Lake Superior area with total copper production estimates ranging upward from 5000 tons over the 5000-year period. This was clearly a large-scale operation for the population of the time.

The mining was carried out by firesetting, a technique used worldwide prior to the introduction of explosives, and one which survived well into the 19th Century in Europe. Wood fires were built directly on the rock, then when a high temperature was achieved, the fire was swept away. The rock fractured due to thermal contraction stresses as it cooled, aided in some cases by the quenching of the hot rock with water. The fractured rock was then broken further by pounding with stone mauls, or hammerstones, sometimes with the aid of copper chisels and wedges. When a piece of copper was extracted from the rock, mineral matter which adhered to it was removed by further pounding with hammerstones.

Archaeological excavations of these mine pits have revealed that since abandonment they have become filled with organic matter, leaves, tree roots, etc. Embedded in the organic matter are found hammerstones, charcoal and ash from firesetting, as well as, occasionally, wooden, bark and leather implements for water and rock removal. The rock walls are often blackened from the fires.

Hammerstones, broken and unbroken, are found in profusion in the mining areas. Some ten wagonsloads of hammerstones were taken from one single pit and an estimated 1000 tons of them were found in an intensely mined area on Isle Royale. They range in size from 1 to 8 inches in diameter, 3 to 12 inches in length and up to 60 pounds in weight. Most of those found on Isle Royale are naturally occurring shapes and are not grooved, while on the mainland there are indications of intentional shaping and many are grooved for the attachment of handles.

Also found occasionally in the ancient pits are large masses of native copper which the prehistoric miners were unable or unwilling to remove from the pit. In one case, a mass of copper, 10 feet by 3 feet by 2 feet, weighing nearly 6000 pounds was found 16 feet below ground level, supported on wooden cribbing some 5 feet above the floor of the pit. Typically such large masses had had all projections and irregularities removed by hammering, so that their surfaces were left by the ancient miners in a smoothed, beaten condition.

No evidence has yet been found that such mining operations were being carried out in the 16th and 17th centuries when the early European explorers reached the area, although copper tools and ornaments were in widespread use. Trees growing on abandoned mining pits frequently are many hundreds of years old. It is possible that these mining operations were for some reason discontinued, although the gathering of float copper may well have continued.

The Keweenaw Peninsula of northern Michigan became a major copper-producing region during the period 1850 to 1930. In fact, in 1905, some 15% of world copper production came from the native copper of this area. The ancient peoples, however, proved themselves to have been as good at mine exploration as were their modern descendants; no major modern mine was located in an area that was not on the site of prehistoric workings! Furthermore, the removal of large masses of native copper created problems for modern miners as well. The largest mass of copper reported was 46 feet long by an average of 12 feet wide and 4 feet thick, weighing 420 tons. In the late 1850s, 20 men worked for 15 months with chisels to cut this mass into pieces of a few tons, small enough to be removed from the mine. A total of 27 tons of copper chips were produced from cutting up this one mass. The copper as mined was melted in a reverberatory furnace, but large masses which could not fit through the charging doors of the furnace had to be stockpiled and charged when the furnace roof was removed for re-bricking.

The Metallurgy of Native Copper

Elemental Composition

Native copper is characterized by a high purity, typically in excess of 99.9% Cu. The most commonly occurring trace impurities are Ag, As and Fe with lesser amounts of Ni, Zn, Sn, Co, Cr, In, and Sb. Except for the silver content it is of the same order of purity as modern high-purity copper.

The trace element contents of native coppers have been extensively studied by neutron activation analysis, most notably by Professor G. Rapp at the University of Minnesota at Duluth, who has built up a data bank of hundreds of native copper analyses. Characterizing the trace element content of native coppers from various geological and geographica regions is useful for two reasons. First, the elemental composition can be a reasonable guide for determining whether an ancient artifact was produced from native or from smelted copper (in most, but not all, cases, ancient smelted copper is of lower purity than native copper). This information is important for example in studies of the development and spread of ancient metallur-
gical technologies. In northern North America, where smelting was not carried out in pre-Columbian times, it is possible to use the existence of smelted copper in archaeological sites as an indication of contact with outside civilizations, such as the Norse settlements on Greenland.

Secondly, trace element studies have the potential of permitting the “fingerprinting” of native copper deposits so that artifacts can be traced back to the source of raw material from which they were manufactured. Such identification would be useful for example in tracing the trade patterns of prehistoric peoples. The extent to which such fingerprinting will ultimately be successful remains unclear, although promising. Required are both data banks of trace element contents of native copper deposits and the application of appropriate statistical methods. The situation is complex, however, being affected by such problems as arsenic zoning in the Lake Superior deposits\(^\text{[10]}\). The application of techniques such as lead isotope ratio analysis may ultimately prove of assistance to the successful fingerprinting of native copper.

**Microstructure**

The microstructure of unworked (as-mined) native copper is characterized by:\(^{[11-15]}\)

i) a very coarse grain size, with grain diameters ranging from 0.1 mm to upwards of 1 mm.

ii) the presence of long thin twins. These are presumably the result of local grain boundary migration (partial recrystallization) which occurs in response to geological strains in the native copper deposit. Local regions of especially high purity would be especially susceptible to such strains as well as to grain boundary migration at ambient temperature. Grain boundary migration would be assisted if the native copper deposit had been heated, for example by local volcanism, by rock motion during faulting, or by forest fires.

iii) a relative absence of second-phase particles. Native copper has an exceptionally clean microstructure, as clean as or cleaner than modern high-purity refined copper. Non-metallic inclusions usually consist of entrapped minerals such as quartz or calcite. Occasional particles of sulphides or oxides of copper have been reported but these are relatively rare. Some native coppers, such as those of the Lake Superior and Coppermine River districts contain inclusions of native silver which can on occasion be visible to the naked eye.

**Native Copper Artifacts**

When native copper is worked mechanically it behaves as would be expected from copper, exhibiting ductility and work hardening. The deformation geometries can be understood by interpreting the grain flow patterns revealed by etching polished metallographic sections through the artifacts. On annealing cold-worked native copper, recrystallization occurs, with the consequent formation of an equiaxed grain structure and annealing twins. These phenomena are all utilized in the studies of the metallurgy and manufacturing techniques used in the production of useful objects by ancient peoples (see e.g. ref. 16). However, in such studies it is first necessary to ensure that changes have not occurred in the microstructure since the artifact was formed, i.e. in use or since use. For example, if an artifact is discovered in a recrystallized condition, the possibility must be considered that it was used in the cold-worked condition and has since recrystallized. Copper if very pure and sufficiently heavily cold-worked can recrystallize at ambient temperature over long periods of time. Such cases are known to have occurred with artifacts but only rarely: in most cases heavily cold-worked artifacts are found to have survived thousands of years without recrystallizing. Two situations which are not uncommon can cause recrystallization, however. Firstly, some cultures practiced cremation rituals and copper grave goods associated with such burials will have crystallized. Secondly, after use or burial some artifacts are exposed to high temperatures as a result of forest fires. Thus, for example, many North American spear points from the Old Copper Culture (3000 - 1500 B.C.)\(^{[17]}\) are found to be in the annealed condition despite the fact that this is not functionally desirable: obviously because forest fires are a frequent occurrence such a condition is not necessarily significant.

Melted and cast copper can normally be distinguished from unmelted native copper by the presence of porosity, a dendritic or columnar grain structure and the copper-copper oxide eutectic constituent in the microstructure. There is no evidence for the use of melting and casting technologies in the production of copper artifacts by the pre-Columbian North Americans. However in Central and South America, as well as in the Old World, native copper was probably melted and cast. Unfortunately, the melting process normally results in contamination of the pure copper, so that its identification as native copper becomes less possible. Furthermore, once a civilization begins to make use of melting technologies, recycling by remelting, as well as alloying, becomes practical so that there is no longer any possibility of identifying native copper as one of the raw materials from which an artifact was made.

Native copper has been used by many cultures in all parts of the world to produce artifacts (Figs. 3 and 4) for use as:
a) tools and implements, e.g. chisels, axes, adzes, knife-blades, awls, pikes, punches, needles, pins, fish hooks, etc.
b) fasteners, e.g. rivets for joining blades to handles, staples for repairing broken thin objects
c) ornaments and ceremonial objects, e.g. headdresses, pendants, ear-spools, rings, bracelets, breast plates, ceremonial plates decorated by embossing or repoussé work.
d) weapons e.g. spears, arrow and harpoon points, swords, knives, etc.

The basic technological process used for the production of all of these was hammering, folding or forging. In most cases periodic annealing was necessary to recover the ductility lost during cold-working. Alternatively, the hammering could have been done hot, although this is generally held to be unlikely in most cases due to the consequent handling difficulties. In some cases the object was left in the annealed condition while in others the object was cold-worked generally or locally after the final annealing in order to increase strength.

Particularly striking examples of elaborate artifacts produced from native copper are the thin embossed copper plates of the South-East Ceremonial Complex (1200-1400 A.D.) which have been excavated from burial mounds in a number of southeastern U.S. states. Studies, including experimental replication of these plates and other copper sheet artifacts, have revealed that techniques used for dressing rawhide were probably adapted to allow successful hammering of native copper (with periodic anneals) to thin sheets using stone hammers on stone anvils. The sheets were then ground to uniform thickness between flat stones, using sand or sandstone as the abrasive. The outline of the ceremonial plates was cut from the sheet by pressing a groove into the sheet, then grinding away the corresponding ridge on the back of the sheet. Alternatively the sheet could have been simply fractured at the groove. The outlines of figures such as dancing hawkmen were then pressed into the sheets using bone or horn tools while the sheet was supported on rawhide (repoussé). Other sheet artifacts such as ear-spools were manufactured by pressing the sheet into shaped molds or forms. Many of these artifacts are of striking design and craftsmanship.

While it is not possible to be sure that native copper was the first metal to be used by ancient peoples, it clearly played an important role in those cultures which had not yet developed smelting technologies. It is also possible that familiarity with the desirable properties of native copper was a factor in allowing ancient peoples to recognize the importance of the accidental production of smelted copper, for example by the reduction of malachite in a pottery kiln. In any case the path of development is believed to have led from hammered native copper to smelted/melted copper, to copper-arsenic and copper-tin alloys and ultimately to the thousands of metallic alloys now in daily use in the late second millennium A.D.

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Man's first use of jarosite: the pre-Roman mining-metallurgical operations at Rio Tinto, Spain

J.E. Dutrizac, J.L. Jambor and J.B. O'Reilly

Introduction

The present-day metallurgical community is quite familiar with the use of jarosites \([MFe_2(SO_4)_{2n}](OH)_{2n}\), where \(M = K, Na, NH_4, Ag, 1/2Pb, \text{ etc.}\) as a means of controlling iron, alkalis and sulphate in hydrometallurgical circuits. This modern precipitation technology has found an increasingly widespread application in the zinc industry\[(1)\] and is gradually also being extended into the hydrometallurgical processing of copper and cobalt\[(2)\]. The chief advantage of jarosite precipitation is that the iron precipitate settles and filters well, and gives low losses of divalent metals in the washed residue.

Although this jarosite precipitation technology was developed only within the past twenty years, man's involvement with jarosites extends back to antiquity, and jarosites have played an important role as a source of silver in several ancient metallurgical centres. In the early days of this century, argentojarosite was an important local source of silver at the Tintic Standard mine, Dividend, Utah, where over 200 000 oz of silver were recovered from high-grade natural jarosite over a three-month period\[(3)\]. At the Matagente mine in Peru, pockets of argentiferous plumbogaljojarosite up to 40 m thick were mined and processed by the Indians prior to the arrival of the Spaniards\[(4)\]. In Cyprus there are vast jarosite deposits associated with the ancient mining sites\[(5)\], but exploitation of these jarosites has been insignificant because of their low silver values.

The most extensive mining of silver-bearing jarosite, and certainly the oldest documented instance of such jarosite exploitation, was in the vast pre-Roman workings at Rio Tinto, Spain. This mine, the greatest single mining-metallurgical site currently known from the ancient world and also the oldest site of more or less continuous metal production, has witnessed many metallurgical firsts. Aside from the initial processing of jasotic ores for their silver values, the development of large-scale hydrometallurgy also seems to have occurred at Rio Tinto. As early as 1737, copper was recovered by cementation on scrap iron from the mine waters at Rio Tinto\[(6)\]. Large-scale heap leaching of calcined ore commenced with the British operations in 1876 and, by 1900, raw pyrite was heap leached to recover associated copper values\[(7)\].

Modern mining at Rio Tinto has been for copper and pyrite, and this has created the popular impression that copper always has been the dominant metal exploited at this site. In fact, recent archeological evidence suggests that serious copper mining began only with the Roman occupancy of southwestern Spain after the Second Punic War. Prior to that, however, there had been an entire millennium of silver production from jasotic ores. The ensuing wealth had a profound influence on the development of the ancient world and some authors even suggest that Rio Tinto was the site of King Solomon's Mines whose silver, traded by Phoenicians, stimulated development in the eastern Mediterranean during the period 971-931 B.C.\[(8)\]. Whether Rio Tinto was the location of these legendary mines is impossible to prove, but the early importance of silver-bearing jarosites at Rio Tinto is now well established and deserves some commentary in the metallurgical literature.

The Jarosites at Rio Tinto

Massive superferous pyrite currently is the principal ore at Rio Tinto; the pyrite reserves are thought to be the world's largest. An average analysis of the ore\[(9)\] is stated to be, in weight %: Fe 42.16, S 48.51, Cu 2.45, Zn 2.19, Pb 1.17 and As 0.43; the silver content of the pyritic ore is low and ranges from about 30-50 ppm\[(10)\]. The massive pyrite is generally capped by a gossan or “iron hat” (Fig. 1) consisting mostly of goethite and hematite and containing about 50% Fe. The gossan averages about 30 m in thickness and was presumably at least this deep in the pre-Roman period. The base of the gossan is flat lying and the contact with the massive sulphides is fairly sharp. The gossan itself is metallurgically of some interest; it contains negligible copper but has about 50 ppm Ag and ~ 3 ppm Au. The recovery of gold from the gossan, which had to be stripped prior to the open-pit mining of the underlying low grade copper stockwork, was a critical factor in the economical development of the Cerro Colorado part of the deposit\[(10)\]. Lead (~ 0.6% Pb) is present mainly as the sulphate with only traces of the carbonate. The high As level (~ 1% As) of the gossan has prevented its use as an iron ore\[(11)\], although areas with relatively low As have been utilized in past world shortages.

Jarosites do not become plentiful until the base of the gossan is approached. There, an earthy layer ranging in thickness from a few centimetres up to 1.5 m contains concentrations of the jarosite minerals\[(12)\]. The jarosite layer does not form a continuous sheet at the bottom of the gossan; in places it is patchy and in others it is totally absent\[(12)\]. Occasionally the lower limit of the gossan apparently was marked by a layer of native sulphur\[(12)\]. The jarosite layer is often colour banded in various shades of yellow, red, grey or black although the yellow jarosite colour predominates often to the complete exclusion of the others. Ironically, the resemblance of the jarosite layer to clay sometimes resulted in its rejection with the waste gossan during the initial years of modern mining of the massive sulphides\[(13)\].

It is estimated that the jarositic layer at the base of the gossan originally amounted to 3 000 000 short tons with over 2 000 000 tons having been extracted in pre-Roman or early Roman times. In addition to this layer of jarositic material, it is also inferred\[(14)\] that large surface or near-surface pockets of the jarosite ore were present, as in Peru\[(15)\], but that these pockets have long since been

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mined. Early reports on Rio Tinto cited the presence of large depressions over the lodes that apparently represented the pre-Roman extraction of such pockets of jarosite ore. There is also ample historical documentation of large “caves” cut into the gossan to recover ores. Although most evidence of such deposits has been removed by large-scale mining, one recently discovered “cave” about 50 m long still contained traces of jarosites on its walls. Of course, the jarositic ore at the base of the gossan has been conclusively documented by modern studies, by frequent reports from the 19th century, and even by several Roman writers such as Pliny the Elder.

Although neither the massive pyrite nor the overlying gossan are particularly attractive as a silver ore, the jarositic ore located at the base of gossan or occurring as pockets is very much enriched in both silver and gold. Surprisingly few analyses of this valuable material have been published, but the available analytical data are summarized in Table 1. Clearly the jarositic layer is variable in composition but contains significant values of Ag (avg ~ 1600 ppm or 50 oz/ton) and Au (avg ~ 30 ppm or 1 oz/ton).

Mineralogically, the jarositic ore consists of major quartz and iron oxides together with minor and variable amounts of barite (BaSO₄), cerussite (PbCO₃) and anglesite (PbSO₄). The only common arsenate mineral is the hydrous ferric arsenate (scorodite) although arsenate substitution in the jarosites, especially lead jarosite, may also occur. A variety of jarosite-family minerals has been identified as a result of recent mineralogical studies. Plumbojarosite (Pb₉Fe₃(Fe₃Si₃O₁₀)(OH)₆), argentojarosite (AgFe₉(Fe₃Si₃O₁₀)(OH)₆) and natrojarosite (NaFe₅(Fe₃Si₃O₁₀)(OH)₆) were recently found and jarosite (KFe₅(SO₄)(OH)₆) has been reported in older work as well as in the present study. In addition to the various end member species cited above, extensive solid solutions among the jarosite species, including hydronium jarosite, are quite likely. Silver probably occurs as argentojarosite, as a solid solution in other jarosites, as a Ag₂S precipitate (?) and as cerargyrite (AgCl) which has been observed to form the core (10-15 μm) of plumbojarosite crystals. Based on the cyanidation behaviour of select samples, it is estimated that 50% to 70% of the silver is present in the various jarosites, and that the balance occurs as halides or sulphides.

**Early Mining and Metallurgical Techniques for Processing Jarositic Ores**

One great archaeological problem associated with mining metallurgical sites is that current operations tend to obliterate past traces of human activity. The extensive operations at Rio Tinto are somewhat unusual in that so much evidence of the past remains in spite of the massive scale of modern mining and processing. To understand the chronology of metallurgical processing at Rio Tinto, it is first necessary to consider the famous and extensive slag deposits existing in the region. Forbes notes that some 15 to 20 million short tons of ancient slag exist at Rio Tinto. Studies have shown that two kinds of slag are present although the types are difficult to distinguish by eye. The newer slag is black, well fused and contains 0.7-1.0% Cu,
a trace of Pb as well as \( \sim 5 \) ppm Ag. This slag, amounting to only \( \sim 1,000,000 \) tons, represents the processing of copper ores which were likely rich secondary sulphides concentrated at the interface with the massive pyrite (Fig. 1). The presence of numerous Roman artifacts and datable coins, together with extensive ancient descriptions, all indicate that this relatively small quantity of slag was produced during the Roman occupation when both copper and silver were being extracted. The greater tonnage of older slag, underlying that from the Roman era, is somewhat less well fused and contains fragments of silica. As the data of Table 2 indicate, these slags are relatively low in copper, but rich in lead and silver. 

It is improbable that such slags resulted from copper smelting and it is now felt that this vast tonnage reflects the accumulated output from the processing of silver-bearing jarosite ores from at least 1200 B.C. to the Roman era. The presence of colourful secondary copper minerals together with a conspicuous gossan must have drawn early man's attention to the Rio Tinto deposits. Recent studies have indicated that the processing of secondary copper minerals took place at Rio Tinto as early as the third or fourth millennium B.C. Since the rich silver-bearing jarosite ores occurred both as surface pockets in the gossan and at the base of the gossan where exposed by erosion, it would have been only a matter of time until smelting tests were carried out on the jarosite earths. Recent archeological evidence has demonstrated that the native people (the Tartessians) were mining the pockets of jarosite ore as early as 1200 B.C. and processing it to recover metallic silver. By 1000 B.C., Phoenician merchants were trading with the Tartessians, exchanging silver for the manufactured and agricultural products of the era. There is extensive evidence of widespread Phoenician activity in the area during the period 700-800 B.C. Traders from the Eastern Mediterranean dominated the silver trade until about 510 B.C. when the Carthaginians conquered Spain and destroyed the city of Tartessos. Throughout the early era, it is likely that the local inhabitants were responsible for most of the mining and metallurgical operations, and that the Phoenicians served largely as traders. The silver these wide-ranging merchants transported to the eastern Mediterranean helped to fuel the flourishing economy there and created the legends of fabulous mineral wealth as exemplified by King Solomon's mines.

Initial mining efforts must have amounted to little more than the digging of the soft jarosite ores from surface pits or the exposed edges of the gossan. With the progression to deeper mining, it became necessary to address ore haulage, dewatering, ventilation and all the other problems which confront mining operations. At some point, it was realized that additional ore reserves might be encountered by the sinking of shafts, either to strike new subsurface pockets of jarosite ore or to locate ore at the base of the gossan in inaccessible parts of the orebody. Such shafts were circular and 0.8 m in diameter with footholes cut into the sides; the miners descended using the holes and pressing their backs against the shaft walls. These shafts had to penetrate through the 30 m of barren gossan to reach the jarosites; later, when the Romans mined the rich secondary copper ores, shafts as deep as 115 m were sunk. If the opening encountered ore, lateral development work took place from the bottom of the shaft; if no ore were encountered, the shaft was abandoned. Literally hundreds of such openings have been discovered and many more must have existed in antiquity. We can only marvel at the ability of the ancient miners to extract so much ore from such narrow shafts, using the light from flickering oil lamps and hauling the ore to the surface in reed sacks. Fortunately for the early miners, both the jarosite ore and the gossan are relatively soft and friable, and would have permitted easy extraction using antler or bone picks and stone or bronze tools. Later, during the Roman and Carthaginian periods, iron picks, hammers and shovels were used extensively.

Mining development around the exposed edges of the gossan gradually expanded into an extensive adit system. These adits were dug for the dual purpose of mining the jarosite layer at the base of the gossan and of draining water from operating sites. Figure 2 illustrates a typical adit at Rio Tinto showing the narrow bottom and enlarged top to permit the easy passage of the miner's torso and broader shoulders. Commonly, internal galleries were 0.7 m across, 1 m to 1.3 m high and had small drainage channels cut along the bottoms to eliminate seepage water. Richer seams or pockets of ore resulted in the development of more expansive galleries, up to 3 m in width. The adits themselves were often long, with some extending over 1000 m. Several of the adits revealed during the modern mining operations were clearly designed and built to drain the gossan, and it is interesting that the Romans developed pumping systems to raise the acidic mine water from depth to the adits whence gravity flow occurred. Forbes notes the use of screw pumps and, most remarkably, of large under ground water wheels, 4.5 m in diameter that were arranged in "nests" and were capable of raising water over 27 m to the level of an existing adit. More than 30 such wheels have been recovered, often in an excellent state of preservation because of iron and copper sulphate impregnation of the timbers.

FIGURE 2. Early adit at Rio Tinto clearly illustrating the enlarged upper portion (Ref. 16).
\textbf{a section; b plan; c conjectural reconstruction}

\begin{figure}[h]
\centering
\includegraphics[width=\textwidth]{figure3}
\caption{Simple bowl furnace used for smelting as found buried beneath a layer of top soil; this particular example is from the United Kingdom (Ref. 18).}
\end{figure}

The jarosite ore is not metallic in appearance and one can only speculate on its original recognition as a silver source. Once the value of the jarosite was recognized, however, its exploitation began on a large scale as the presence of some 15,000,000 short tons of lead-silver slag indicates. Early mining methods were highly selective and it is likely that the ore brought to the surface was high grade. There is also some evidence\cite{15,16} that the ore was crushed and coarsely ground; the presence of crude hard porphyry mortars and pestles throughout the Rio Tinto area testifies to the wide extent of this activity prior to the Roman period. The ground ore was sometimes washed to reduce its high silica content\cite{16}; the washed ore was also probably mixed with gossan to provide enough iron to flux the remaining silica. The jarosite ore was often lacking in lead (Table 1) and there is evidence that galena, litharge or metallic lead had to be added to the smelting charge to collect the silver\cite{15,16}. Pliny the Elder (~ A.D. 70) indicated that the silver ores at Rio Tinto required the addition of galena\cite{16}, and legend notes that the Phoenician traders exchanged their lead anchors for ones of pure silver, with the lead presumably being used for silver smelting\cite{16}. Jones\cite{16}, however, considered that much of the jarosite ore, especially after concentration by washing, would require no flux or added lead because of the presence of alkalis in some of the jarosites and the presence of lead in others.

Pre-Roman smelting was done in small bowl furnaces cut into porphyry bedrock\cite{16} and lined with clay; Figure 3 illustrates such a simple smelting furnace. The ground ore, flux and galena would be mixed with charcoal and ignited. The charge would be fanned by a bellows with the blast being directed through a primitive clay tuyere; extra charcoal and ore would be added as required\cite{17,18}. The charcoal requirements to produce 15,000,000 tons of slag would have been considerable even assuming that the output occurred over a thousand-year period. Charcoal supply must have been a constant concern to the silver smelters\cite{17,18}. Smelting would have continued in the bowl furnace until a complete charge of lead bullion and slag had been produced. At this point, the furnace would be cooled to solidify the two phases\cite{17} which were then mechanically separated. Later practice permitted the tapping of slag and bullion to allow more continuous furnace operations\cite{11}. The silver was recovered from the bullion by cupellation, and the common presence of litharge indicates that this technology was known from the earliest days of mining and smelting at Rio Tinto\cite{17,18}. The silver would be melted and cast into ingots. The litharge was likely used to collect more silver, lead metal itself was of little value until the Roman era. This continuous small scale smelting of jarosite ore produced a vast tonnage of silver slag and enormous quantities of silver. By the time the Romans arrived at Rio Tinto, most of the readily accessible jarositic ore had been exhausted, and the Roman metallurgists turned more and more to the supergene-enriched copper ores concentrated along the upper surface of the pyrite (Fig. 1). The Romans were also interested in silver, of course, and treated any jarosite ores encountered. There is also considerable evidence that the Romans reprocessed much of the earlier slag to recover its silver values (up to 600 ppm Ag). With cessation of the Roman operations around 200 A.D., Rio Tinto experienced a major decline which lasted until the modern period. Today, silver and gold are still recovered from small pockets of jarosite and from the gossan itself\cite{18}, but the amounts pale in comparison to the wealth removed by the pre-Roman metallurgists.

\textbf{Examination of Rio Tinto Jarosite Ore}

Despite the historical and economic importance of jarosites at Rio Tinto, apparently little attention has been devoted to mineralogical studies to define precisely the nature of the silver-rich jarosite horizon. To the writers' knowledge, the only two publications which give explicit evidence, rather than inference, about the presence of multiple jarosite minerals at Rio Tinto appeared in 1950 and 1981\cite{9,12}; the latter authors identified plumboglaucite, argentojarosite and natrijarosite by X-ray diffraction. From time to time, current mining operations encounter remnants of the jarosite ore and ancient mine workings. Recently the authors obtained four specimens of such material, kindly made available through the courtesy of J.P. Hunt, Company Secretary, Rio Tinto Minera S.A. The largest of the specimens (Fig. 4) is about 15 cm by 12 cm by 4 cm, and the smallest is about a quarter this size. All
are otherwise similar in appearance in that each consists of a dark brown, highly porous and somewhat friable mass of granulose quartz that is cemented mainly by goethite. The predominant feature is a rough, box-work appearance typically characteristic of gossan. In the largest specimen are several randomly oriented, straight quartz veinlets that average about 1 mm in width; their presence indicates that the specimen was probably taken from an oxidized part of the silicified stockwork zones that commonly underlie the associated massive sulphide deposits.

Polished sections of the oxidized material show that occasional specks of anhedral pyrite occur in the quartz which typically has highly irregular boundaries with the surrounding iron oxides. Although barite has been reported to occur sparingly scattered throughout the gossan, the mineral is abundant in the specimens studied here and is evident megascopically as gray, striated, euhedral crystals up to 1 mm long. Some of the crystals appear to be zoned, but microscopic examination revealed only cracks that simulate zoning; X-ray powder-diffraction patterns and backscattered-electron images of the barite did not reveal any compositional variations.

Some of the goethite in the hand specimens has minutely botryoidal surfaces. Where concentrated, the iron oxides show good colloform textures in polished sections (Fig. 5). In a previous study of Rio Tinto gossan, it was found that some of the colloform iron oxide layers are hematite which alternates with goethite and layers of jarosite-group minerals. The specimens studied here differ slightly in that hematite and buff-coloured jarosite occur mainly as powdery coatings on the goethite. Ragged-edged platelets of gypsum have been found isolated among the thicker coatings of jarosite. Also found within the colloform goethite are inclusions of rutile and cassiterite (Fig. 6), both of which have been confirmed by X-ray powder patterns. In previous work, tin oxide was found to occur as hydroxomarchite (Sn₂O₃(OH)₂).

Several X-ray powder patterns of the jarosite coatings examined here gave identical results indicating that compositional variations are minimal in this suite. Least-squares refinement of measurements from one of the patterns (114.6 mm diameter camera, CoKα radiation) gave a = 7.304 Å, c = 34.36 Å. These cell dimensions indicate that the mineral is probably potassium jarosite, with minor substitution causing a slightly enlarged c relative to that of end-member jarosite. Atomic absorption analysis of a composite sample of the jarosite, handpicked under a binocular microscope and weighing 3.6 mg, gave K₂O 6.94%, Na₂O 0.92%, PbO 0.59%, Fe₂O₃ 42.2 wt%, Ag 300 ppm. These results support the identification of the mineral as jarosite and also indicate that minor substitution of various elements for potassium has occurred; of particular relevance is the detection of 300 ppm Ag as this indicates that the jarosite is an important silver carrier.

It has been reported that substantial replacement of sulphate by arsenate occurs in some of the Rio Tinto jarosite. As the presence of significant amounts of arsenate in jarosite would seriously reduce the reliability of X-ray powder diffraction identifications, a composite sample of the jarosite was examined using a scanning electron microscope and energy-dispersion analysis. The analyses confirmed that slight variations in the minor elements (Pb, Ag) are present, and indicated that arsenic is at or below the detection limits.

Conclusion

The various mineralogical studies of jarosite material from Rio Tinto have persistently pointed to the jarosite minerals as being the most important carriers of silver. Specific jarosite-group minerals that have been identified are natrojarosite, jarosite, plumbogarosite and argentojarosite; the abundance and composition of these minerals are variable from site to site, but even the alkali-rich jarosites can bear appreciable silver as has been found in this investigation of material apparently collected from a site above or below the main silver-bearing horizon. The mineralogical studies continue to reinforce the archeological conclusion that the Rio Tinto oxide zones were the source of enormous silver wealth that was widely exploited from at least 1200 B.C. to the Roman era.

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Readings in Historical Metallurgy 11


Roman lead plumbing: did it really contribute to the decline and fall of the Empire?

J.E. Dutrizac, J.B. O’Reilly and R.J.C. MacDonald

Introduction

Sometime around 3000 B.C., primitive metallurgists working in the mountainous regions of Asia Minor bordering on the Black Sea (Pontus) produced a soft, easily fused metal today known as lead. It appears certain that the first metal was made by reductive smelting of galena, but whether this recovery arose by the accidental inclusion of galena in the stones of a fire hearth or whether it occurred as a consequence of simple pyrometallurgical experiments on a series of interesting “stones” will likely never be known with certainty. The properties of galena, high metallic lustre, great density and facile cleavage, must have excited the ancients, and such “stones” would have been the object of primitive testing. Shortly after the discovery of lead, the ancients found that the controlled oxidation of this metal at elevated temperatures yielded smaller amounts of a more interesting material, silver. Again, it is uncertain if the production of silver by primitive coppersmelting techniques was the result of a fireside accident or the consequence of a deliberate experiment, but the important fact is that silver was produced shortly after the discovery of lead. The techniques for making lead and silver spread gradually throughout the entire civilized world, likely as a result of both technology diffusion and independent rediscovery.

The physical properties of lead could not have been overly attractive to the ancients, but silver was a different matter altogether. The gleaming white lustre and relative immunity to oxidation quickly assigned silver to the precious materials category, a status it still enjoys. In early times, argentiferous galena was inevitably processed only for its silver values; lead was generally an unwanted by-product and was frequently discarded as litharge (PbO) during the coppersmelting process. The silver was eagerly sought, however, and resulted in considerable mining activity culminating in the famous Athenian silver mines at Laurium at the southern end of the Athenian peninsula.

Lead as a valued article of commerce and an industrial staple was very much a Roman metal. Although previous civilizations had produced lead and had found limited uses for the metal, it was the Romans who developed an extensive industrial market for it, notably for such applications as water pipes, linings for cisterns and aqueducts, roofing, water catchment basins, pot linings, coffins, dishes, net weights, etc. The Romans searched their empire for galena deposits and developed a broadly based trade network for both lead and silver. Unlike their predecessors, the Roman demand for lead itself was so great that they worked some deposits for their lead values only, and recovered no silver at many of the sites. In the Roman period, it was uneconomical to recover silver from lead if it contained less than 100 g/tonne, and the metal produced from the British mining camps of Derbyshire, Yorkshire and Shropshire fell into this category. Of course, the Romans were only too happy to process rich argentiferous ores, but inevitably also produced lead from the silver-free litharge formed during the coppersmelting process.

Roman Lead Metallurgy

In Roman times, galena was the principal ore mineral of both lead and silver and the galena ore, whether silver-bearing or not, would be subjected to various concentration techniques prior to smelting. These would commence with the selective mining of the rich ore veins and would be followed by hand copping to recover lumps of high-grade material. The pieces of ore, probably always very rich by modern standards, would be crushed and coarsely ground in a muscle-powered mill and upgraded by various combinations of sieving and washing. By the Roman era, the use of core washing tables to eliminate less dense gangue was well developed and was especially effective for dense galena ores. The washery generally consisted of a large reservoir of water which could be released into a distribution tank and thence into a series of jets where the ground ore was hand washed to eliminate the lighter gangue. Much of the richest lead ore would collect in the corrugated troughs directly below the washing jets, but some of the finer particles would be recuperated in the water collection launders, whence they might be recirculated for additional washing. When the water had drained from the troughs, the coarse-grained concentrate would be spread onto the central and slightly sloping table for final drainage and solar drying, which would also tend to agglomerate the concentrate.

Two general methods appear to have been used to make lead from the coarse galena concentrates. In the simplest operation, the galena would be interlayered with wood or charcoal in a very low shaft furnace. As the charged moved down the shaft, the galena would undergo partial oxidation followed by reaction of the oxidic lead with residual sulphide to form crude lead metal and SO₂ gas. In the second general procedure, the galena would be pre-roasted in trenches, and the broken calcine would then be mixed with wood or charcoal and smelted in a low shaft furnace to yield crude lead. Both processes ejected SO₂, and lead fumes at or near ground level, and the atmospheric pollution coupled with the great demand for wood quickly devastated the mining area, creating an unhealthy environment for the slaves who ran the operations. Roman slags contained 10-30 wt% PbO, but were extremely low in silver (0.0004-0.06 wt% Ag). The lead bullion was tapped from the furnace and run into clay-lined moulds to yield 100- to 200-lb pigs. These moulds commonly bore inscriptions which have permitted the accurate date and origin of the lead to be determined. The crude lead was often subjected to low-temperature dressing in a reverberatory-type furnace to eliminate copper and to oxidize reactive impurities, very much as is done today. Finally, the lead was desilvered by coppersmelting. In this method, the metal was oxidized in a shallow hearth lined

(wt%) of lead pipes indicate:

<table>
<thead>
<tr>
<th>Element</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ag</td>
<td>0.0008%</td>
</tr>
<tr>
<td>Sb</td>
<td>trace</td>
</tr>
<tr>
<td>Bi</td>
<td>0.033%</td>
</tr>
<tr>
<td>Cu</td>
<td>0.006%</td>
</tr>
<tr>
<td>Fe</td>
<td>0.0004%</td>
</tr>
<tr>
<td>As, Sn, Cd</td>
<td>not detected</td>
</tr>
</tbody>
</table>

The silver content of Roman lead varied somewhat depending on the skill of the various metallurgists, the initial silver levels, etc. but generally ranged from 0.002 to 0.008 wt% Ag.

In Roman times, lead sheet was prepared by casting onto flat sand beds fitted with clay-on-wood sides; rolling apparently was not used until fairly modern times. Large lead cisterns and lead aqueduct linings, etc. were made by fusing the edges of the sheets together. To achieve this, the sheets would be cut carefully to form butt joints and a temporary mould would be positioned on the joint. Pure lead or lead alloy would then be poured into the joint to seal it. The trick was clearly to have the added metal sufficiently hot to flow and just melt the walls of the sheet. Too little heat and the vessel would leak; too much and there was a danger of melting a hole through the lead. Most welds seem to have been effected using pure lead, although there are examples of Pb-Sn alloys similar to modern solder having been used. The outer melting point of the solder would have been of some assistance during welding, but likely the principal purpose of the tin was to provide a stronger joint.

Most cities in the Roman era were supplied with water by an extensive network of aqueducts and water piping made from both clay and lead metal. Water pipes and drains consumed a large proportion of Roman lead, and these pipes were also made by casting. The initial step was to cast a long, narrow sheet of lead on a sand table; the inclusion of manufacturers’ data in the sand moulds was common, as is evident from the final pipe section shown in Figure 1. The strip could be as long as 7 m and this produced equally long pipe sections; because the wall thickness was often 1/4 to 1/2 in., such sections must have been difficult to handle and this was possibly the reason that pipe lengths seem to have been standardized at about 3 m. The long, flat sheets were cut into suitable widths and were then formed into a long tube; both butt joints and flared joints have been reported, but the flared type seems more common. The tube was filled with sand or clay, and strips of clay or clay-faced wood were then placed on either side of the joint to form a shallow trough, as shown in Figure 2. Molten lead was poured into the trough to weld the pipe tight. There is no evidence that Roman pipes were ever made by extrusion or direct tubular casting. Pipe manufacture received the fullest attention of Roman engineering genius and pipe sizes were carefully standardized, with approximately 2- and 4-in. diameters being common. The efforts made to standardize the size of lead piping is notable in that it is probably the first example of the standardization of a widely produced manufactured article. Wall thicknesses were also specified (via the weight of the pipe) to ensure pressure-withstanding capabilities although Roman engineers often played it doubly safe by encasing the entire pipe in concrete. Taps and delivery pipes were usually made of bronze and these were attached to the lead water pipes by a tightly fitting joint sealed with lead solder.

Lengths of pipe were welded together using a box joint. The pipes were cut diagonally and were inserted one into the other. A piece of lead sheet was placed over the joint,
and the entire assembly was covered with a three-piece mould which left a \( \frac{1}{2} \) in. gap between the mould walls and the pipe. Finally, lead was poured into the box to seal the pipe. Figure 3 shows a photograph of such a box joint on a Roman pipe from Britain. Although rare today, thousands of such box joints linking hundreds of miles of piping must have been used in the Roman era.

**Lead Poisoning Among the Romans**

The above description has indicated that the ancient Romans knew how to smelt and refine lead, and that they had developed several industrial applications for the metal, notably in the areas of water piping and storage. The Roman appetite for lead was enormous and for a period individual consumption reached 4 kg/y, a level comparable to current usage in the industrialized nations. A single six-hundred ton unit on the aqueduct of Lyons consumed 12,000 tons of lead, and equally vast amounts of metal were employed throughout the Empire. Rome itself, then a city of about one million inhabitants, was supplied with water delivered in lead pipes and the per-capita water consumption exceeded that of most modern cities. Given the great output of lead and the uses to which it was put, it is natural to speculate on the extent of lead poisoning and its effects, if any, on the collapse of Roman civilization.

Leaving aside the unfortunate plight of the slaves assigned to mine and smelt the lead, the average urban Roman was faced with four potential sources of lead contamination. The first and most obvious of these is the water plumbing system which would deliver both drinking and cooking water to the household. The pickup of lead by water flowing in lead pipes is a complex problem related to the “hardness” and the pH of the water as well as its temperature. In general, the plumbosolvency rises rapidly with either increasing or decreasing pH, with the effect being particularly pronounced below pH 5.5. The plumbosolvency also increases dramatically with decreasing Ca concentration, as is evident from Figure 4. Similar results have been observed for natural waters, and it is evident that lead contamination from water pipes is very dependent on the water quality. In hard-water systems rich in calcium, lead is a fairly acceptable piping material and it continued to be used safely (?) until the mid twentieth century. In soft-water areas, lead is dangerous to use and problems are still being noted in soft-water areas where lead piping or cisterns were previously installed. The absence of calcium is doubly injurious: first, it encourages lead dissolutions (Fig. 4), and secondly its absence encourages lead uptake by the gastrointestinal tract. A modest concentration of dissolved calcium will effectively block lead uptake by the body and deliberate addition of CaCl\(_2\) (2-4 ppm Ca) to soft-water systems is practised to reduce lead poisoning. Calcium reacts with dissolved species to form a Ca-Pb-CO\(_3\)-SO\(_4\) scale which prevents Pb dissolution from the piping.

Presumably, the Roman water supply, originating in mountain lakes, was fairly hard (rich in calcium) and presented little immediate threat as concerned chronic lead dissolution from the piping. Even today, Roman water is very hard by any standard (78-222 ppm Ca) and could probably be safely contained in lead pipes, as suggested by Figure 4. Ancient Roman pipes and aqueducts were frequently heavily encrusted with calcareous deposits that necessitated periodic cleaning and descaling efforts. The high calcium contents must have reduced the plumbosolvency and possibly the scales would have entirely prevented contact between the lead and the flowing water. This statement would have to be modified for those Roman houses using soft rain water which would likely be stored in lead cisterns. Also, each city in the Empire would have had its own water supply and, consequently, would face varying degrees of water hardness and, hence, lead dissolution. This may be one reason for the large variations in the reported lead content of bone from different Roman cities. In some of these cities (Roman Bath and Verulamium), the bones contain ~ 30 µg/g of Pb, a level similar to that found in ancient, non-lead-using societies. In other cities (e.g. Cirencester), the Pb levels are an order of magnitude higher, suggesting some source of water contamination. Unfortunately, little information is available on the water quality of the different Roman-era cities and the relative degree of contamination must remain speculative.

If lead poisoning were a factor in imperial Rome, it is likely that the lead intake was associated more with lead-lined cooking pots than with lead piping. To avoid imparting a bitter copper taste to food, the Romans lined bronze and copper ware with lead sheet, and significant amounts of lead must have been transferred to the food therein prepared. Obviously the uptake would depend on the nature of the food, the temperature, whether the pot was scraped, etc. Compounding the problem was the Roman penchant for repeatedly boiling wine in lead-lined pots to prepare sugar of lead (Pb-acetate), which was then

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**Figure 3.** Length of lead pipe from Red House, Corbridge, Northumberland, U.K., showing box joint; the length of the pipe is 1 m (Ref. 2).

**Figure 4.** The effect of calcium ion (as CaCl\(_2\)) on the plumbosolvency of lead from lead water pipe (Ref. 10).
added to wine or food to stop fermentation and decay\cite{1}. The sugar of lead, as the name implies, has a sweet taste which was apparently much favoured by the ancients, especially by the connoisseurs of fine food and drink\cite{2}. Where cases of chronic lead poisoning occurred among the nobility of Rome, they are likely attributable more to this cause than to water pipe corrosion. It is also interesting to note that lead was added to food and beverages well into the 19th century\cite{3}, in spite of the well-established hazards of the practice. Apparently, the immediate concern of making beer or wine drinkable outweighed a vague future of lead poisoning.

 Pewterware was in common use in Roman times and was probably widely employed by the "middle" class; i.e., those rich enough to avoid wood or clay dishes, but not sufficiently affluent to afford silver. Roman pewter was variable in Pb concentration; presumably the more lead-rich products were cheaper, and were employed by the lower classes. It is easy to imagine a constant uptake of lead and lead corrosion products as the food was cut and scraped from the pewter dishes. Lead poisoning from pewter continued into the early 19th century until rigid specifications were established; modern pewter is essentially Pb-free.

 Like ourselves, the ancient Romans were also exposed to a multitude of other sources of lead\cite{4}, including deliberate ingestion in the form of prescribed medicine, lead-based paints and lead-bearing cosmetics. Of these sources, only lead-based paints and dyes are a continuing source of lead contamination; as late as 1980, some 78,000 ton/yr of lead were used to prepare paints and dyes in the United States alone\cite{5}.

 The potential for chronic lead poisoning in the Roman Empire certainly existed, and many such cases must have occurred. Given their familiarity with lead smelting, the Romans were certainly well aware of lead poisoning (plumbism) and its obvious symptoms\cite{6}. Written medical reports from the 2nd century B.C. clearly spell out the hazards of lead and its ingestion; both the chronic poisoning evident in lead smelters and the more subtle effects caused by sugar of lead were well known. One can only assume that the Romans viewed sugar of lead much as we view alcohol consumption or cigarette smoking; a danger may exist, but the immediate pleasures outweigh any long-term health hazards. By the 1st century B.C., the potentially hazardous use of lead plumbing with certain types of water was also well established\cite{7}. Presumably, any general outbreaks of chronic lead poisoning in the Empire would be investigated to see if the plumbing were at fault; clay or wooden pipes were often used to overcome such difficulties\cite{8}.

 Chronic lead poisoning must have occurred from time to time in the Empire and, in fact, there were epidemics of lead poisoning caused by lead-treated wine reported from the Roman period and well into the middle ages. Significantly, however, the little available evidence on lead concentrations in ancient bone suggests that the Romans did not suffer any more generally from chronic lead poisoning than we do. Analyzed Roman bones\cite{9} show the same range of variables as do bones found throughout the Anglo-Saxon, the Medieval or the relatively modern industrial eras (30-400 μg Pb/g bone). Actually, there have been very few lead analyses performed on Roman bone, and even fewer attempts to correlate the findings with the social-political status of the deceased individuals. There is simply no evidence to support the contention that the Roman nobility was preferentially afflicted with lead poisoning which caused the Empire to decay. Lead poisoning was widespread and severe in 18th and 19th century industrialized Britain and lead plumbing was widely used in regions of extremely soft water. Is there anyone, however, who would attribute the collapse of the British Empire to lead poisoning?

 Waldron and Wells\cite{10} observed lead bone analyses as high as 540 μg/g in modern (living) British patients who showed no apparent signs of plumbism. Most ancient bone analyses (and ~ 90% of the lead in an adult is contained in the bone) are considerably less than this value, suggesting that plumbism was certainly not a universal chronic ailment produced by general consumption of lead-contaminated water. The Romans (and indeed ourselves) inhabited a lead-polluted world and all Roman bone analyses show significantly more lead than would be expected from a pristine environment (~ 0.7 μg Pb/g bone). Part of the difficulty with assessing the impact of lead poisoning is that the early symptoms are subtle and not easily diagnosed. Before modern instrumentation, the conditions could have been ascribed to many common ailments\cite{11}. The Romans were readily able to diagnose the acute forms of lead poisoning and would presumably have taken remedial action, including the removal of the offending lead source. They would likely have been unable to detect the subtle forms of low-level lead ingestion, including minor losses in fertility, logic, agility, etc., that would be caused by minor uptake of lead from pipes carrying hard water. To some extent, modern man faces the same problem. The combustion of leaded gasoline creates a persistent, easily measured, but low level of lead contamination, and we really don’t know if these low levels of airborne lead pollution are producing subtle changes in our ability to cope with the complexities of life. If the Roman watchword concerning lead was: “don’t drink the water,” ours must be: “don’t breathe the air.”

### Lead and the Fall of the Roman Empire

The Roman Empire did eventually fall, but the contribution, if any, of lead poisoning to this decay is open to speculation. Gillian\cite{12} has recently postulated that lead intake among the Roman nobility (in wine and food) produced a decline in fertility and mental capacity that affected their ability to govern effectively. The article claims to have conclusive data from bone analyses to support the contention, but no figures are given. Most historians seem to reject the role of lead poisoning in the Empire’s decay and, in place, advance a multitude of complex socio-economic-military factors both within and without the Empire, and this latter approach seems to be quite valid.

### REFERENCES

Old iron nails are one of the most common objects which entice the general public to ask questions about the history of metallurgy. There is a lot of history in old nails — history of the manufacturing industry, and social and economic history. Nails, therefore, are fine teaching tools; whether students have to acquire metallurgical skills or explore the structure and properties of wrought iron, whether they want to look into the history of inventions and the development of machinery, studying nails tells a story better than any teacher or any book. I have used this approach often and the following note is based on several student projects, particularly the work of Carmina Herrero and Vicky Marchant.

Nails, those essential metallic fasteners, have long been with us. Their size, shape and design is basically a compromise between the makers' ability to control and manipulate material and the demands of the intended application.

There were copper and bronze nails in antiquity, but in this paper I want to deal only with iron nails.

In Rome, iron nails were common, and Cleere carried out the fine metallographic investigations on a large stock of nails recovered from a Roman fort site in Scotland. Those nails were hand forged; sorting them at the site on the basis of length, shape of head and shape of shank (in cross section) indicated that there seemed to have been some nail standardization. It appears that the Roman smiths used a series of heading dies starting with a 1/8-in. size. Each successive size increased by approximately 1/16-in. The metallographic inspection of the microstructures of the nails showed that the smaller nails were made in the one operation; i.e. both the tapering and the formation were accomplished on the heated rod without reheating. However, in the case of the larger nails, the nail had to be reheated for the tapering after the head had been forged. Most nails were tapered on all four sides of the shank and the shanks were square in cross section; only one group had rounded shanks. Most heads were flat disks, but in one group the heads were pyramids and another group had conic heads. Figure 1 shows the procedure used to produce the pyramidal heads. Obviously, Roman nail making was sophisticated and each nail design was meant for a particular use.

Throughout the Middle Ages, we have reports on and illustrations of nail making. We also know the elaborate system of guilds which organized those engaged in nail-making activities. Houses and ships, armour and horse shoes, carts, carriages and almost all trades needed nails — and each nail was made individually by hand from a rod of wrought iron. Like the Roman nails, most Medieval nails were tapered on all four sides, although round nails have been found. The nails were headed in an operation separate from the tapering, and the heading tools changed little through time and culture. Hand-forged nails varied widely in quality, likely because of variations in the raw material. They could not be driven directly into the wood; holes had to be prepared first, using an awl. Special punches developed for this purpose remained in use until fairly recent times.

The increased use of water and horse power that began in the Middle Ages improved the ways of working and forging iron, but the actual nail-making process did not really change until the 18th century. The improvements brought about by the use of water power and related developments, particularly in terms of machines and bellows, were reflected in the better raw materials available for nail making. The quality and reproducibility of the wrought iron changed considerably when water-driven bellows made reverberatory firing a common practice. Easier control of the nail rod dimension was achieved when slitting and rolling could be mechanized. In England, rolling mills and slitting mills came into common use in the 16th and 17th centuries. The development of slitting mills reduced the amount of labour involved in nail making. Although water-powered hammers could not forge bars smaller than about 3/4-in. square, a nail rod of appropriate size could be cut off from sheet using a slitting mill. This metal strip would then be heated to a temperature at which the shank, point and head could be shaped by the nail maker by manual hammering. Earlier workers probably followed the same procedure, except that the nail blanks were cut out completely by hand.

Up to the end of the 18th century, the British nail-making industry was typical of the household-centred cottage industry so common prior to the establishment of large factories. We get a detailed description of what nail manufacture was like from Reeve's "Manufacturing Industry," a compendium originally published in 1819.

Nails, Manufacture of. The immense consumption of these articles, in all the mechanic arts and trades, cause their fabrication to be a considerable branch of national manufacture. It is chiefly carried on in Staffordshire, in the neighbourhood of Dudley, Wolverhampton and Birmingham; indeed it is the principal consumption of the malleable iron made in that part of England. The iron made in the Puddling furnaces, anc worked by rollers instead of a forge-hammer. This metal is, by repeated rolling, reduced into small thin bars, which are then passed through the grooves of the slitting-rollars, and thus divided or slit into three, four, five or six small square rods, of a proper size to make nails. These, which are called nail rods, are very extensive article of trade. The nailers reside chiefly in the cottages, where the women and children assist in the labour. They employ forceps such as are used by smiths; but the bellows are very lightly loaded, so that a very small motion given to them now and then will blow sufficiently to heat the rods; two, three or four of which, according to their size, are always kept in the fire together; and when any one has a good red heat, the nailer takes it out of the fire, and, battering it on the anvil, brings it to a sharp square point at two or four strokes; he then applies it over the edge of a chisel, fixed on the anvil, and by striking a single blow on the rod, cuts off a sufficient length to make a nail, which falls into a tin pan; then he makes another, cutting it off likewise, returns the end of the rod to the fire.
for another heat; now, with a pair of tweezers, like sugar-tongs, he takes up the nail, and introduces its point into a square tapering hole, made across through the end of an iron moulc; by striking a blow or two on the end of the iron, he flattens it down, and forms a head, the figure of which is determined by the number and direction of the blows given it.”

It has been estimated that the weekly output of a woman nail maker would have been about 7000 nails. A 16th century manuscript, dealing with the material used in the construction of an inn, states that 53,000 nails were required for fixing the roof, 15,000 for fixing the floorboards, 200 for fixing the lead dressings and joining windows and latches, and 400 spike nails for fastening the large timbers. The original financial account opens with the purchase of nails, showing that these were ordered well in advance of any actual construction. Today it is estimated that 65,000 nails are required in the building of a small traditional house.

For Britain, in the 18th century and later, export to the colonies, particularly North America, was the main source of trade in iron nails. Nevertheless, nails were a scarce commodity in the American colonies. It reached the point where buildings were being delayed because of the lack of nails, and abandoned structures were burned down in order to retrieve and re-use nails. In some localities, this scarcity persisted well into the 1800s. Figure 2 presents some of the many kinds of nails used in North American building construction.

The steady demand for nails of dependable quality, the availability of steam-powered machines and the desire to abandon an old-fashioned manufacturing system combined to produce a great deal of incentive to change the method of nail manufacturing. Three basic technical problems had to be solved before nails could be manufactured satisfactorily in quantity by a modern industrial process:

1. the question of how to taper a nail in machine operation had to be solved;
2. a way to mechanize the heading operation had to be thought of and proven out; and
3. a materials problem, that only emerged gradually, had to be solved.

It became apparent that the texture of a rolled sheet, which essentially consists of grain deformation and alignment parallel to the direction of rolling, resulted in uneven and undesirable mechanical properties of nails when that sheet was cut perpendicular to the direction of rolling.

Before 1800, a variety of processes, devices and patents appeared showing the widespread interest in mechanizing the manufacture of iron nails. Some of these inventions were short-lived; others were the precursors of modern technologies. One attempt was to use cast iron, or malleable cast iron, for nail manufacture. Although it was recognized that these nails were more brittle and mechanically not as satisfactory as wrought iron nails, they were nevertheless used fairly extensively between 1770 and 1820. In North America, their use seemed to have been limited to the Canadian Maritime provinces and the New England coast.

There were a variety of attempts to combine rolling and cutting as well as some shaping operations. One of those patented in 1835 involved slitting a plate of malleable cast iron into rods and passing these rods through rollers, with the rollers pressing the metal into nail shapes. The metal was then cut to separate the shapes. Finally, mechanization came through the use of cutting a rolled plate to form cut nails. These cut nails were symmetrically tapered on two sides of the shank only. Using machines with a fixed position for the blade, this taper could be accomplished by either moving the sheet or flipping it over. In this manner, the nails were cut in wedge-like shapes, with the length of the nail determined by the width of the nail plate. Figure 3 shows the early procedure used to produce nail blanks. These nails were of uniform thickness and were first headed in a separate operation using a tool similar to that.
used for hand forging of nails. Later, they were cut and headed entirely by machine. Once the basic technology of nail-cutting machines had been worked out, the cut nail took over the trade very rapidly. One can summarize the chronology of these changes using several principal stages:

1790-1810
In this period, nails were machine cut, the nail plate being reversed under alternate blows of the cutter. A few stamp-headed nails occurred, but most were headed by a single hand-driven hammer blow. L-headed nails made from headless nails were to continue in use even until the 1850s for floors and clapboards.

1810-1825
Machines were invented to make cut nails that obviated the necessity of having to turn the nail plate. Until 1825, such nails continued largely to be headed simply by being struck with a hammer.

1825-1830
The cutting of nails continued, but water-powered machines were developed that headed them automatically. The heads of these nails, however, are rather thin and lopsided.

1830 - about 1855
Wire nails were invented in France (1835) that were ground to a point and headed by hand.

1830 - about 1890
Cut nails were produced in machines which cut and headed them uniformly. Heads were less thin, more uniform and were comparatively square. These heads were extra heavy on large nails. Cross sections of shanks, on virtually all nails, tended to have three sides at right angles, with one concave side (Fig. 4).

1855
Machines were invented in France to make complete wire nails automatically by 1855. It was about 1890, however, when wire nails outnumbered cut nails.

1870
In or about 1870, cut nails were annealed to prevent their rupturing when clinched.

For the purpose of dating, the external shape of the shank, the shape of the head and the microstructure of the nail are witnesses to the mode of manufacture, which, in turn, can be fixed in time.

Although cutting machines provided a big step toward the mass production of nails, it took some time before the relationship between the structure of a rolled sheet (Fig. 5) and the structure of the nail were clear. Thus, early machine-cut nails show a preferential grain orientation perpendicular to the axis of the nail, and later nails show that the nails were cut parallel to the direction of rolling. It appears that this occurred only after about 1870. The mechanization of nail manufacture took root first in Britain’s American colonies and continued to proceed there at a fast pace. There are many good reasons for such a development. North America was not hampered by a traditional, household-based manufacturing system. Labour was scarce and there was a great deal of incentive to use machines whenever possible. There was also a great desire to be independent of the unreliable trade with Britain and to supply the basic needs domestically. Already, in 1810, 410 nail factories were producing 15 million pounds of nails annually in North America. At that time, two-thirds of the iron coming from rolling mills was used in the manufacture of cut nails in the United States. Nails provided from one-third to three-quarters of the profits of the many companies involved in the manufacture of iron objects.

In Canada, one of the first forges known to make nails and small hardware was set up by De La Salle at Niagara, where his ill-fated ship, The Griffon, was built in 1678. The bar stock was still imported from France, as were the drawings for the ship and the skilled workers.

About the time that the first nail-making machine was invented in the United States, John Bigelow opened a
small workshop in the St. Laurent suburb of Montreal to become Canada’s first nail manufacturer and founder of a company which later became part of The Steel Company of Canada, Limited.

John Bigelow survived the competition of his counterparts by using one of the primitive American machines himself. It was run by horse power and cut the nails from the iron automatically, but the nails still had to be headed by hand and he still brought in his raw material in iron hoops from the wholesalers.

A different process for making nails was introduced in the middle of the 19th century. These were wire nails and the original invention was French. The earliest wire nails were small, but when North American production began in the 1850s, and when it was perfected in the following decades, the sizes became larger.

One should not look at wire nails as a successor to cut nails, but rather as another response to the problem of the mechanization of nail production. In France, where the use of wire for nails began, the approach was patterned essentially after the manufacture of pins, with a wire-wound head that dates back to the 16th century. Commercial production of wire nails began in France about 1834, but the first patents were issued in 1806. This was only a few years later than the date of the first cut nail machine patent.

We all know that in the end wire nails won out over cut nails, only to be superseded again by the screw type nail, a development in which The Steel Company of Canada had such a massive influence. The over-all knowledge of A. B. Dove, former product development metallurgist in Stelco’s Wire and Fastener Divisions, is a legend among all those who are “hooked” on the history of nails. Some of the information in this note was developed in conversation with him.

The standard histories of technology, such as Singer et al. (Oxford U. Press), and the histories of iron and steel making contain details on nail making; particularly interesting are the accounts from the 18th and 19th centuries in the encyclopedias and journals such as Technology and Culture or Historical Metallurgy. Cleere’s paper on Roman nails was published in 1962 in the Journal of the West Scotland Iron and Steel Institute.

The line drawings in this note are adapted from Lee H. Nelson’s “Nail chronology as an aid to dating old buildings” (Am. Assoc. for State and Local History, Techn. Leaflet 48, 1968).
Dotaku: an example of metallurgy in prehistoric Japan

K.T. Aust

Introduction

The Yayoi period, from about 300 B.C. to A.D. 300, represents a time of important changes in Japanese history. During this period the advanced rice culture of the Asian continent was introduced to Japan, along with the techniques of animal husbandry, spinning, bronze casting and metal working(1). As was the case in the Jomon period (11000 B.C. to 300 B.C.), the Yayoi period is traditionally defined by a particular type of pottery, characterized by the use of the potter's wheel and an extreme simplicity of decoration and form(2). Pieces of this pottery were found in 1884 in a district of Tokyo, known as Yayoi-Cho, not far from Tokyo University. Within a decade the name Yayoi had come to be used for the period which also saw the first casting of metals in Japan(3).

During the Yayoi period, the foundation of the ancient Japanese state was laid as the individual small communities were unified and as labour specialization and social levels developed. In fact, the Yayoi may be the true ancestral culture of Japan(4). However, the origin of the Yayoi is not definitely known. One theory indicates that the Yayoi people may have been driven from the Asian mainland (via Korea) by the dynastic wars that accompanied the unification of China under the Ch'in and Han and that these settlers rapidly displaced the Jomon. Bronze objects and the cultivation of rice may have arrived about the same time — in the third century B.C. Iron was in Japan within another century, but since it rusts badly and disintegrates, only a few small pieces are known from such an early date(5). After about 50 B.C., the Yayoi cultures of western Japan were in fairly close and regular contact with the continent via Korea(6).

Little is known of the significance of bronze goods in Yayoi society and history. Agricultural and wood-working tools continued to be made of stone, wood or iron during the Yayoi period. Bronze appears to have been used largely for decorative or ritual purposes, although their full and true function is still somewhat of a mystery(6). Toward the end of the Yayoi period the Japanese began to make some of their own bronze objects(7). One of these objects, the bronze bell or “dotaku” depicted in Figure 1, shows Japanese innovation based on continental technology. Unlike Yayoi bronze weapons, these bells appeared to be completely indigenous Japanese products(8). However, the original prototype for these large bells may have been a small bell for horses found in Korean archaeological sites(9). As the Yayoi people developed their metal industry, the bell increased in size and became strikingly different from the smaller Korean bells(10). Many examples of dotaku appear to have been made solely for burials or ceremonial purposes rather than for actual use. The bells may have served as ritual objects used by leaders as symbols of authority(11).

As seen in Figure 1, the dotaku were objects of an elongated cone form with an opening at the top by which they might be hung. They have an oval section, a fin-like border running down the sides, and are usually heavily ornamented both with abstract designs and with crudely sketched pictures. Bells of a different form appear to have been used in China during the Zhou dynasty (11th century to 256 B.C.). However, illustrations of decorations from old Chinese and Japanese sources, given in Figure 2, indicate some stylistic similarity between the Yayoi and the Han dynasty(12). Figure 2a shows the ornamentation of a bronze drum of the Han dynasty (206 B.C. to A.D. 220) and Figure 2b shows one from a Japanese bronze bell.

Archaeological Sites

During the Yayoi period, a bronze weapons culture was centred in north Kyushu, at the west end of the Inland Sea.

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FIGURE 1. Several dotaku photographed by the author in the Tokyo National Museum during a recent visit.

FIGURE 2. (a) Patterns from Chinese Drum (Han dynasty). (b) Patterns from a Japanese bronze bell (reference 8).
Overlapping it somewhat in both area and time was the bronze bells culture, centred in the east end of the Inland Sea and in the Kansai region. This is illustrated in the map of archaeological sites of the Yayoi period (Fig. 3), showing the distribution of bronze objects. The bells are concentrated in the eastern edges of the Inland Sea and within the Kyoto-Osaka region, i.e. away from the area adjacent to the continent where Yayoi imports entered and where most imports underwent their transformation to insular types. This distribution of the bells gives some support to the theory that the bells are fully indigenous. The bronze blades — swords, spears, and halberds — are concentrated in Kyushu, southern Honshu and Shikoku.

The centralization of bronze bell production in the Kyoto-Osaka region has been established not only from the concentration of bell finds but also from finds of the moulds used in the casting of the bells. At Higashinara, Osaka Prefecture, a site occupied from early Yayoi (300 B.C. to 100 B.C.) into Kamakura times (A.D. 1192 to A.D. 1333), excavation in 1974 uncovered the remains of two middle Yayoi burials, each surrounded by a square ditch and a number of semisubterranean pit dwellings. The excavated materials found in association with these features included six types of stone moulds made of tufaceous sandstone for the casting of bells, ceramic replicas of bronze halberds, and moulds for the manufacture of comma-shaped beads. The excavation did not uncover any evidence of a workshop area for bronze casting. The location of the tufaceous sandstone used for the moulds was a 25 km² area to the northwest of the site. The absence of references to kiln parts and crucible shards in the Yayoi period is puzzling but may be due to subsequent problems or removal by continental craftsmen when they left Japan.

**Bronze Metallurgy of Dotaku**

Bronze-working technology, as well as artifacts, was introduced from the Asian continent to Japan. The continental bronze imports included Western and Eastern Han mirrors, Chinese bronze coins and Chinese weapons. The initial entry of these items into Japan is placed at the end of the early Yayoi period. Later in the Yayoi period, bronze bells appeared at Yayoi sites and these are believed to have been cast in Japan. The art of casting the dotaku may have been learned from China, but the designing has been ascertained to be original to Japan. Therefore, some Japanese archaeologists regarded the dotaku as a unique development peculiar to the Japanese, although their origin may have been from Korea (via China) where they had practical use.

The bells were cast in piece moulds of clay or sandstone. An example of the latter is the mould from the Higashinara site near Osaka (Fig. 4). The mould is 43 cm long, 29 cm wide, 12.5 cm thick and 28 kg in weight. The bronze, heated to about 900°C or higher, was poured into the mould which consisted of two outer halves with an inner core. For the great majority the casting required ties for fastening the sections of the mould together; these were inserted through the top, across the upper part of the wall and at the base, leaving holes in the finished product. Frequently, early and middle bells of mediocre workmanship have patched places where bronze was added in an attempt at smoothing out imperfections. It seems that the Yayoi metallurgists were probably part-time specialists who fashioned bronze objects at the time of the year when agricultural duties did not demand all their time.

The bronze bells served the same purpose as the bronze weapons, which were large, wide-bladed types in thin, brittle metal, quite impractical for use, intended entirely for ceremonial and ritual purposes. The earliest bells, found in the Inland Sea region, are rather badly cast, but subsequent generations improved their skills and were able later to produce excellent pieces of workmanship. It seems unlikely that the thin and fragile dotaku could have been used as percussion instruments, although their shape implies they may have been.

Several problems related to the introduction of bronze...
metallurgy to Japan were discussed by Hitchins\textsuperscript{10}. One of these problems concerns the acquisition of the necessary knowledge of the alloying elements and the actual casting process. Independent development in Japan appears unlikely because there is no archaeological evidence of any earlier experimentation with metallurgy. In addition, this was a time period when there was contact with the metallurgically knowledgeable continent. Therefore, the introduction of the technology by continental craftsmen after the initial appearance of continental metal appears more likely. The fact that North Kyushu appears to have served as the initial region for bronze-casting (of weapons) in Japan is significant because of its geographical proximity to the Korean peninsula.

To cast bronze requires temperatures of 900°C or higher. The ability to achieve such temperatures arises in some cases, such as in the Shang dynasty of China (16th to 11th century B.C.), from experience in the firing of ceramics. However, a limited study of Yayoi ceramics by Sakara indicates that it is unlikely that Yayoi potters had knowledge of the production of temperatures high enough to melt the alloy and to produce crucibles to withstand the temperatures required for the melting process\textsuperscript{10}. It appears, therefore, that continental craftsmen were involved in the early bronze casting in Japan. In order to account for the concentration of bronze-bell casting in the Kyoto-Osaka region of central Japan beginning in the middle Yayoi period, it has been postulated that bronze casting was introduced directly into this region.

Another problem considered by Hitchins concerns the possible sources of the raw materials required to produce the bronze alloys. There appears to be no indisputable evidence of mining of the native ores until the later Nara period (A.D. 710 to A.D. 794). Also, there is no archaeological or historical evidence to indicate the importation of continental ores. Another possibility involves the remelting of imported continental bronze artifacts. This could be tested chemically to some extent. For example, if bronze has been remelted, it could be recognized in a decreased tin content (due to oxidation) and a tendency toward a per cent increase in the trace elements. One study along these lines by Tanabe\textsuperscript{12} has indicated that the Japanese bells were probably not made from remelted continental (Chinese) bronze artifacts.

\textbf{Lead Isotope Studies}

A study based on the pioneer work of 3rill and Wampler\textsuperscript{13} was conducted by Yamasaki \textit{et al.}\textsuperscript{14} on the lead isotope ratios in various Japanese and Chinese archaeological bronze objects. The lead isotope method is useful for studying ancient objects because it is insensitive to chemical history of the objects studied. For example, the lead isotope ratios associated with an ore are not changed when the ore is smelted. Yamasaki \textit{et al.} found that Japanese dotaku and arrowheads show similar isotope ratios to those of early Han mirrors, suggesting raw materials with the similar isotope ratios. This result, together with the similarity of ornamentation noted previously in Figure 2 appears to relate the dotaku to the bronze metallurgy of the Han dynasty.

Mabuchi \textit{et al.}\textsuperscript{15} have recently reviewed their work relating early Japanese bronze culture to Chinese and Korean cultures through lead isotope studies. They conclude that bronzes in Japan's prehistoric ages were without exception of continental Asian origin. For example, lead isotope ratios of two earlier style dotaku indicate that these materials may have come from Korea, as shown in Figure 5. The isotope ratios of both samples fall on the Korean line. Also, two tongues which were excavated together with the latter dotaku fall on the same line (Fig. 5). Analysis of 33 dotaku objects of middle style (Fig. 6) clearly show that these contain the same kind of lead as do the Western Han (202 B.C. to A.D. 8) mirrors (Fig. 5). Lead isotope data for specimens of 18 dotaku of later style are given in Figure 7. This figure shows that these cata concentrate in a small area, the dispersion being only about three times the standard deviation of their measurements. It was concluded that the lead of the later style dotaku came from the same mine as that of the middle style but homogenized Chinese bronzes were used to produce the dotaku of later style.

These results of Mabuchi \textit{et al.}\textsuperscript{15} indicate that the dotaku in the earlier period were made with Korean bronzes, but soon these were replaced by Chinese bronzes. Most of the dotaku were made with such materials, and in the later stages a number of homogenized Chinese bronzes were used for casting the dotaku. The dotaku in the earlier styles are generally small, 20 cm to 30 cm in height, and contain more than 10% tin. The dotaku
TABLE 1. Chemical composition of the dotaku (after Mabuchi et al.)

<table>
<thead>
<tr>
<th>No.</th>
<th>Lead Isotope Type</th>
<th>% Cu</th>
<th>% Sn</th>
<th>% Pb</th>
<th>Analyst</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>earlier</td>
<td>74.46</td>
<td>15.25</td>
<td>7.65</td>
<td>Kamei</td>
</tr>
<tr>
<td>2</td>
<td>style</td>
<td>75.9</td>
<td>12.5</td>
<td>5.92</td>
<td>Yamaski</td>
</tr>
<tr>
<td>3</td>
<td>middle</td>
<td>83.84</td>
<td>7.62</td>
<td>7.63</td>
<td>Chikashige</td>
</tr>
<tr>
<td>4</td>
<td>style</td>
<td>85.88</td>
<td>3.49</td>
<td>8.89</td>
<td>Kamei</td>
</tr>
<tr>
<td>5</td>
<td>style</td>
<td>83.54</td>
<td>7.68</td>
<td>8.26</td>
<td>Chikashige</td>
</tr>
<tr>
<td>6</td>
<td>style</td>
<td>82.87</td>
<td>6.17</td>
<td>7.46</td>
<td>Tanabe</td>
</tr>
<tr>
<td>7</td>
<td>later</td>
<td>90.87</td>
<td>4.17</td>
<td>4.27</td>
<td>Chikashige</td>
</tr>
<tr>
<td>8</td>
<td>style</td>
<td>87.91</td>
<td>4.52</td>
<td>3.38</td>
<td>Kamei</td>
</tr>
<tr>
<td>9</td>
<td>style</td>
<td>89.14</td>
<td>4.93</td>
<td>2.66</td>
<td>Tanabe</td>
</tr>
<tr>
<td>10</td>
<td>style</td>
<td>90.63</td>
<td>5.22</td>
<td>3.90</td>
<td>Chikashige</td>
</tr>
</tbody>
</table>

in the later styles are 70 cm to 140 cm in height and contain only 4% to 5% tin.

Mabuchi et al. also found that there exists a relationship between the isotope data and the chemical analysis reported by others (Table 1). High tin, middle tin and low tin groups in chronological order appear to coincide with the isotope early, middle and late style classification respectively, of Mabuchi et al. This indicates that neither random remelting of broken bronze objects nor gradual addition of Japanese copper occurred. Instead, some specific materials such as bronze ingots were perhaps imported in three stages and served for making the dotaku.

It is evident that studies of Japanese archaeometallurgy, such as the metallurgy of the dotaku, may provide further understanding of Japanese prehistory during the Yayoi period and the role of this period in the emergence of the present nation of Japan.

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The origins of zinc and brass

J.E. Dutrizac and J.B. O'Reilly

Introduction

Zinc is a relatively abundant element which occurs as both high-grade sulphide ores (sphalerite ZnS) and oxide ores which the ancients collectively termed “calamine” (smithsonite ZnCO3 and/or hemimorphone ZnSiO3(OH)2·H2O and/or hydrozincite Zn5(CO3)2(OH)6). Although oxidic zinc is readily reduced by carbon, the process becomes thermodynamically viable only at ~1000°C, and at this temperature the zinc is released as a vapour (B.P. 907°C). Casual cooling of the vapour results in its re-oxidation by excess combustion air or even the carbon monoxide product of the reduction reaction.

\[ \text{ZnO} + \text{C} = \text{CO}_{\text{gas}} + \text{Zn}_\text{vapour} \]

Only fairly rapid quenching of the zinc vapour and the isolation of the reduction products from the combustion gases will yield metallic zinc; these conditions were not produced deliberately until the advent of distillation technology. Until quite recently, the principal use of zinc was in brass manufacture: galvanized iron was not discovered until the 19th century and zinc casting alloys are a relatively recent invention. As will be seen, brass could be produced without recourse to a pure zinc intermediary, and the discovery of brass preceded that of zinc by many centuries. The purpose of this paper is to outline the discovery and early development of both zinc and brass with special reference to the European scene.

Accidental Zinc Production

Although the conditions for deliberate zinc production were not known to European metallurgists until the close of the Mediaeval period, there is some evidence that small amounts of metallic zinc were made accidentally in ancient times and that this metal was used in spite of its poor mechanical properties. The archaeological evidence for the early use of zinc is not, however, very convincing. In part the difficulty lies with the ease of reaction of zinc with weak acids or bases, making its preservation unlikely. Furthermore ancient chroniclers could have confused zinc with a number of other white metals which were known in ancient times. Zinc continued to be confused with other white metals until the dawn of the modern era, and this makes the early written record for zinc most unreliable.

A few ancient zinc artifacts have been discovered although none is unequivocal proof of the ancients’ knowledge of zinc. A silver bracelet inlaid with zinc was found on the Isle of Rhodes in the ruins of Camiros, destroyed in 500 B.C. The bracelet appears to be authentic, but it is not at all certain how the zinc was identified. Caley suggests that the presence of the metal may have been based on visual appearance only and that the “zinc” was actually corroded silver14. A zinc-covered fountain was allegedly found in Pompeii. The zinc was apparently identified only by visual means and could equally have been As- or Sn-covered copper. A subsequent search for this fountain in the museum of Naples was unsuccessful15. A fragment of zinc with 1.3% Pb was found together with 4th century B.C. coins in a dig at the Athenian Acropolis16. The corroded sheet was originally thought to be lead, but on cleaning and subsequent chemical analysis was found to be zinc with a greater variety of impurities than modern metal11. Forbes11 suggests that this sheet may be a modern product, but the metal seems to have been hammered and was found badly corroded and surrounded by genuine artifacts. An idol found in Dacia, and possibly of Roman age, was found to consist of 87.5% Zn, 11.4% Pb and 1.1% Fe. Unfortunately this and a second similar zinc artifact were not obtained in a closely supervised archaeological excavation and, consequently, there is an element of uncertainty concerning their origin.

If zinc was produced at this early date, it was almost certainly of accidental origin and was infrequently encountered. Although native zinc14,15 and brass are known15,16, both metals are exceedingly rare in the natural state and generally occur as tiny grains in placers, oxidized ore cappings or in basalt. The exception is a ~100 g Zn nodule found in a cavity in basalt in 19th century Australia16. Even assuming such natural material could be collected, the amounts of native zinc recuperated would be inadequate to prepare useful quantities of the metal, and ancient man was likely unaware of the existence of the native element. It is known, however, that small amounts of metallic zinc can be “quenched” in the cracks of the brickwork of iron or lead-smelting furnaces.

John Percy17 had in his possession a zinc sample obtained from the cranks of a 19th century blast furnace treating zinciferous iron ores. Presumably the zinc vapours were formed in the strongly reducing zone and then penetrated into the larger openings between the refractories where they were quenched. An impure zinc containing ~10% Pb was occasionally produced in the lead blast furnaces of the lower Harz at least as early as 161718 and possibly as early as 1300 A.D. It was observed to form in crevices of the wall where the brickwork was not well plastered, and in porous charcoal. The zinc had little value at the time and was collected only in special circumstances.

Agricola in 154612 also reported that a white metal was condensed in the walls of the smelting furnaces in the Harz and that this metal had been known for centuries. He observed, furthermore, that a similar metal called “zincum” was being produced under similar circumstances in Silesia. Paracelsus (1493-1541) was the first European to state clearly that “zincum” was a new metal (element) and that it had properties distinct from other known metals19. Thus, by about 1600, European scientists were aware of the existence of zinc and had reached the conclusion that it was a new element; all the metal they had examined, however, was likely of accidental origin or had been imported from the East.

Deliberate Zinc Production

The deliberate pyrometallurgical production of zinc neces-

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sitated an effective condensation system. It is not surprising, therefore, that the first intentional production of metal occurred in India where, as in many areas of the Mid-East, distillation technology was being developed. The systematic production of zinc was first described in the Hindu book Rasarnava which was written sometime between 1100 and 1300 A.D. The 14th century Hindu work Rasaratnasamuchchaya describes how the new "tin-like" metal was made by indirectly heating calamine with organic matter in a covered crucible fitted with a condenser (Fig. 1). Zinc vapor was evolved by the indirect heating of oxidic zinc with carbon, and the vapor was quenched by air cooling in the condenser located below the refractory crucible. Presumably the metal was recovered as a solid which required remelting. By 1374, the Hindus had recognized that zinc was a new metal, the seventh known to man at that time, and a limited amount of commercial zinc production was underway.

At Zawar, in Rajasthan, great heaps of small retorts bear testimony to extensive zinc production from the 12th to 16th centuries. The tubular retorts are about 25 cm long and 15 cm in diameter with walls about 1 cm thick. A small diameter tube was sealed onto the open end and the zinc vapors likely condensed in this. The retorts were closely spaced in a furnace which was probably heated with charcoal fanned by a bellows. The retorts at Zawar likely made both zinc metal and zinc oxide. By this time in India, zinc was used to make brass whereas the oxide was used medicinally. Over 130,000 tons of residue remain at Zawar and this is felt to represent the extraction of the equivalent of 1,000,000 tons of metallic zinc (Zn + ZnO).

From India, zinc manufacture moved to China where it developed as an industry to supply the needs of brass manufacture. The Chinese apparently learned about zinc production sometime around 1600 A.D. A scientific encyclopedia composed by over 800 authors and issued in the latter half of the 16th century makes no mention of zinc, however, the book Tien-kong-kai-ou published early in the 17th century and reissued in 1637 clearly related a procedure for zinc manufacture. Calamine ore, mixed with powdered charcoal, was placed in clay jars and heated to evolve zinc vapor. There is some uncertainty concerning the nature of the condensers used by the Chinese with some authors describing equipment similar to conventional condensers and others showing simply a ceramic dome sealed onto the reduction pot. Zinc production in China expanded rapidly, and metal began to be exported. Despite the economic importance of this early production from China, and the still earlier production from India, there does not seem to be a comprehensive review of zinc production localities or methods in either China or India.

As noted above, the Europeans were likely aware of the existence of zinc by 1600 A.D. A small, probably discontinuous, supply of impure zinc was being made accidentally in lead furnaces in Germany, and the first trickles of higher grade Oriental zinc were beginning to arrive in northern ports. It had already been recognized that metallic zinc additions to copper gave brass, although most brass was still being made by the well-established calamine process. After 1600, zinc became more and more an object of commerce with virtually all the metal destined for brass manufacture to support a growing industrial economy (galvanizing with zinc was patented only in 1836). Most of the early zinc was imported into Europe from India or China. The early trade was carried on by the Portuguese and Dutch, and there is a record of the capture and sale of a quantity of zinc taken by the Dutch from a Portuguese carrier in 1620. Analysis of a lot of Chinese zinc brought to Europe in 1745 gave 98.99% Zn, 0.76% Fe and 0.25% Sb with Cu, Ni, Ag, As and Pb not being detected. This commercial purity zinc was sold as speaultre or spalier
and the British scientist Robert Boyle latinized the term to spelter in 1690 from which originates spelter, the commercial term for zinc\[8\]. The spelter trade with the East flourished throughout the 17th and first half of the 18th centuries although there seem to be no records concerning the tonnages involved.

Tradition maintains that the knowledge of deliberate zinc smelting in a retort-type apparatus was acquired by an Englishman on a visit to China just prior to 1740. A vertical retort procedure was developed by W. Champion in 1738 and by 1743 a zinc smelter had been established at Bristol in the U.K.; this was certainly the site of the first deliberately produced commercial zinc in Europe. Champion’s apparatus was in principle similar to that used by the early Hindu metallurgists (Fig. 1). A charge of calamine and carbon were sealed into a clay crucible having a hole in the bottom. This was luted onto an iron tube extending below the crucible furnace into a cool chamber below (Fig. 2). The closed end of the iron tube sat in a tub of water and it was here that the metallic zinc was collected. The distillation took a total of about 70 h to yield 400 kg of metal from all 6 crucibles positioned in the furnace. An annual production rate of 200 t has been suggested for the entire Bristol works at that time\[13\].

This type of apparatus continued to be employed until 1851 although it was very fuel inefficient, consuming 24 t of coal for every ton of spelter produced. In 1758, William Champion’s brother, John, patented the calcination of zinc sulphide to oxide for use in the retort process, thereby laying the foundation for the commercial zinc practice which continued well into the 20th century.

By combining Oriental knowledge with Western large-scale technology, Champion had brought commercial zinc production to Europe. His process (Fig. 2) was a batch operation which required withdrawing the crucible and retort after each cycle; it was labour intensive and fuel inefficient. A major technological improvement came with the development of the Belgian process about 1815\[14\]. The principal advantage of this technique (Fig. 3) is that the retorts were fixed horizontally into the furnace allowing them to be charged and discharged without cooling. The condensers were luted onto the open ends and this also simplified the task of metal recovery. By placing the retorts in large banks, fuel efficiency was greatly increased.

With the advent of the Belgian furnace concept the modern era in zinc production had arrived. Although major improvements in retort and furnace design were to be made, the horizontal retort process remained, in principle, unchanged for a century and a half. As late as 1950 it accounted for 50% of the world’s zinc output, but is clearly now obsolete with respect to the electrolytic process. Horizontal zinc retorting came to an end in North American in 1976\[15\].

The Origins of Brass

Although the deliberate production of zinc is a relatively modern event, the manufacture of brass (Cu-Zn alloys) is an ancient process. That brass could be systematically produced in large tonnages at least 1500 years before Zn was produced commercially is a tribute to the skills of the early metallurgists and an indication that the lack of fundamental knowledge (e.g. the elemental composition of brass) need not be a barrier to technological advancement.

Bronze (Cu-Sn alloy) has been known since at least 3500 B.C. Most bronzes have only a trace of zinc (<0.5%), but some contain over 20% of this metal\[16\]. Many of the early high zinc “bronzes” would best be described as brass or possibly gumnmetal (Cu-Zn-Pb-Sn). For example, a prehistoric object from central Europe was found to contain 75.0% Cu, 22.6% Zn and only 1.1% Sn. Many other objects similar in composition have been reported from prehistoric northern and central Europe; such alloys are considered to have been produced accidentally by the smelting of copper ores containing significant concentrations of zinc. Similar “bronzes” containing over 23% Zn were produced at Gezer, Palestine, around 1400 B.C. to 1200 B.C., and these are also considered to be accidental\[17\]. Arsenical coppers were produced in antiquity in the same manner and zinc is no more volatile than arsenic. The activity of the zinc is significantly lowered by dissolution in the copper and the kinetics of its re-oxidation are also greatly reduced.

The gold-like colour of the accidentally produced high-zinc bronzes must have excited the ancient metallurgists and it is quite reasonable to expect that efforts would be made to produce such alloys on a systematic basis. Although there are various early references to brass, it is never certain whether brass or bronze’s actually meant (a confusion which continues today among the non-metalurgical population) or whether the brass was simply a metallurgical accident resulting from the treatment of a high-zinc copper ore. Forbes\[18\] concludes that brass was systematically made in Pontus around 1000 B.C., but the evidence for this is not convincing especially as it concerns the existence of ancient brass objects. There is some indication that the calamine process for brass was known to pseudo-Aristotle of Athens in 320 B.C.\[19\] and to the Romans by 200 B.C.\[20\], but if this knowledge was available, neither of those early societies produced a great outpouring of brass goods which have survived for archaeological study.

In fact, the systematic manufacture of true brass seems to have sprung up quite abruptly about 50 B.C. in the Roman Empire although there must have been a long period of experimentation leading up to this time. The result of the flourishing of this technology has been a wealth of brass coins and other true brass articles (ceremonial armour, jewellery, etc.) containing 20% to 30% Zn with the balance being mainly copper. Possibly because brass superficially resembles gold, the Romans developed
a liking for brass coins and literally millions of these must have been produced of which thousands have survived to the present day. Figure 4 shows two brass coins from the National Currency Collection of the Bank of Canada. The coin on the left is a sestertius from the reign of the Emperor Trajan (98-117 A.D.) and it contains 82.0% Cu, 13.7% Zn, 3.76% Sn, 0.27% Fe, 0.07% Pb and about 0.1% In. On the right is a sestertius from the period of Caligula (37-41 A.D.) and it consists of 79.9% Cu, 18.5% Zn, 0.33% Sn, 0.65% Fe and also some indium. SEM-EDX analysis of the patina on this coin also indicated Cl, S and Si. The presence of indium in the brass is unexpected and might shed some light on the origin of the zinc and/or copper.

Because of its method of manufacture (see below), Roman brass was 6 to 8 times more costly than copper, and Zn-rich brasses were more expensive than low-Zn brasses. Thus the zinc contents of Roman coins offer some insight into the inflationary problems of that empire. Table 1 illustrates the Zn content of two types of Roman coin during the first 200 years of the Christian era. The zinc content of the coinage metal fell steadily with time. Because coins were the only form of money used in ancient civilizations, there is a close relationship between the value of the coinage and its face value. Clearly, inflation was as serious a problem for the ancient Romans as it is for us today.

The calamine process by which the ancient Romans prepared brass has been extensively documented. Crushed “calamine” was mixed with powdered charcoal and granulated copper and then heated in acovered crucible to about 1000°C. As the zinc vapour was evolved at elevated temperature, it was absorbed by the copper, resulting in brass. The alloy melted more easily than pure copper and the charge eventually fused and was cast into molds. The literature suggests that additions of calamine and charcoal were sometimes made during the heating cycle, presumably to yield high-zinc brasses. Although brass could be made using natural calamine, the Romans soon discovered that better quality metal was made using sublimed ZnO, and industries were established to react zinc ores (often natural calamine) with carbon and to collect ZnO (with some metallic zinc?) in the cooler parts of the furnace. This ZnO was used for both brass manufacture and medicinal purposes. Thus the ancient metallurgists achieved some degree of quality control firstly by carefully selecting the source of the copper and secondly by the use of sublimed ZnO as the zinc source.

The calamine process continued to be used for brass manufacture throughout the Middle Ages and into the modern era. This process yielded all the monumental brasses set into churches throughout Europe. Over 10,000 of these brasses still exist although countless others were stolen for their metal content. Monumental brasses are true brasses containing 23% to 30% Zn together with 1% to 3% Sn and 2 to 7% Pb to enhance fluidity. Brass cannons were made in England as early as 1385 and brass wire, used for wool carding, helped to create the industrial revolution. Most brass made in Europe before 1700 was produced by the calamine process. Although brass had long been made in the Orient by the addition of zinc metal to copper, the method was developed in Europe only after 1600 A.D. The calamine process, which claimed to yield a brass with superior mechanical properties, continued to be used in England until 1861 when it was replaced by the modern method of adding pure zinc to copper under carefully controlled conditions.

Acknowledgments

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Zinc and brass in ancient India

H.V. Paliwal, L.K. Gurjar and P.T. Craddock

Introduction
The Indian cultural heritage of mining, smelting and the use of minerals and metals dates back to ancient times. Although the exact date of the commencement of such activities in India is not certain, Indians are known to have been familiar with gold, silver, copper, tin and lead and to a lesser extent with iron at least since the Vedic period (Table 1).

Zinc and Brass in Ancient Literature
The Vedas are encyclopedic compilations of knowledge written in Sanskrit, which in origin may go back to 5000-2500 B.C. One of these, the Atharva Veda describes numerous uses of gold, silver, copper and tin. Detailed references to mining and smelting activities are available in the later epic, Ramayana; this gives an authentic social, cultural, moral, political and economic life of the country during the reign of "Rama" who ruled the country about 1600 B.C. The Ramayana describes the occurrences of metallic ores and minerals as well as smelting. The descriptions suggest that not only bronze but also brass may have been used at this time[1,2]. This is more than a millennium earlier than the periods normally associated with the production of brass in Europe.

A record of various uses of brass and "rasak" (zinc ore - zinc oxide?) appears in the Charak Sahita written by Maharshi Charak around 1000 B.C. Brass was used in the manufacture of various medicinal apparatus, while zinc oxide (pushpanjan), prepared by heating pure zinc metal in air, was used as eye ointment. Various surgical instruments described in the Sushrut Sahita (500 B.C.) were made of brass.

The earliest detailed references to zinc ore, its varieties, smelting process and uses are found in Rasarnavam Rastarama which is believed to have been written between 500 and 100 B.C. According to this work, zinc ore, "rasak", is of three types. "Matta rasaka", the best quality, is like yellow-coloured soil and does not lose weight on heating, suggesting that it may be zinc oxide. "Gud (jaggery) rasak", the medium quality, is brown in colour and seems to be sphalerite (zinc sulphide). "Pashan rasak", the poorest quality, has an earthy colour and is hard like stone. This seems to be silicate of zinc.

A process of zinc extraction referred to as "Satva Padan" (Satva = extract, Padan = process) is described in the Rasarnavam Rastarama as follows:

"The rasak (zinc ore) is powdered by the wise "Vaidya" (practitioner of the Indian system of medicine) and put into a cloth bag which is dipped into a container of woman's urine for seven nights. The soaked powder is then treated with juices of yellow and red flowers (saffron, acacia, catechu, lac tree, termaralia and dhak) and then treated repeatedly with alkaline, neutral and acidic solutions, respectively. After this, "wood", "lakh" (lac tree), "haldi" (curcuma roxor), "harre" (termanalia chaubula), "kenchua" (earth worm), "suhaga" (borax) and "grahdhoorn" (kitchen carbon flowers) are mixed together and added to the treated powder. The product is then kept in the crucible or retort ("mooshya yantra") and heated strongly. The extract released on strong heating has a diamond-like shine which is undoubtedly zinc".

It is normally assumed that because of the volatility and reactivity of zinc at smelting temperatures, brass was produced by cementation (co-smelting of zinc and copper ores) long before metallic zinc was produced, so the suggestion that metallic zinc was available in this period is of great interest.

Acharya Chanakya, the Prime Minister of the Maurya Dynasty (327 B.C.) who is known as the Indian Machiavelli, gave very systematic guidance for the search, identification, mining, extraction and uses of various metals as medicines, coins and war materials in his book Kautilyas Arthasastra. He placed great importance on the mining and smelting industry. According to him "Mines are the source of the treasury; from the treasury comes the power of government, and the earth, whose ornament is the treasury, is acquired by means of the army and the treasury".

In Hellenistic Greek literature there are references to unusual copper alloys in the lands which were invaded by Alexander's army. Archaeological excavations at Takshashila, now in North Pakistan, have provided firm evidence for the contemporary use of brass and probably zinc as well. In the deposit dating from the 5th to the 1st century B.C. were found many brass objects, at least one of which contained 34% to 35% Zn. This suggests that brass may have been manufactured by mixing copper with zinc metal since its zinc content is too high to have been attained by the cementation process.

The ancient Ayurvedic (Indian system of medicine) literature suggests that zinc was used in the manufacture of brass but the exact process adopted remains unclear. The earliest brasses so far recovered from excavating are from Taxilla, and date from the 4th century B.C. – 2nd century A.D.; as well, brass articles of the first century A.D. have been unearthed from ancient Buddhist stupas. A beautifully turned brass casket of the 2nd century A.D. was found in a stupa at Manikyalaya in 1830 by General Ventura. Brass and bronze were used for coinage in ancient India. Circular punch marked brass coins of Dhandeva and Aryaverma of Ayodhya (first century B.C.) have been collected and preserved in the Indian Museum. Brass was also used in the making of statues of Gods and Goddesses. A 30 cm by 13.5 cm Buddha statue of the 6th century A.D. was found at Fathpur near Kangurakot. Huien Tsang has described another remarkable use of brass in the construction of an unfinished Vihar at Nalanda during the reign of Harshavardhana (Table 1). It is clear, however, that from the 7th century A.D. onward, brass was in common use in India for ornaments and household utensils as well as for medicinal purposes.

In medieval times, many books described the character-

TABLE 1. Indian chronology*

<table>
<thead>
<tr>
<th>Date</th>
<th>Event/Period</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Ca. 5000-2500 B.C.</strong></td>
<td>Vedic culture. Arrival of Aryans (Rig Veda)</td>
</tr>
<tr>
<td>Ca. 3200-2500 B.C.</td>
<td>Indus Valley Civilization</td>
</tr>
<tr>
<td><strong>Ca. 1600 B.C.</strong></td>
<td>Ramayana War</td>
</tr>
<tr>
<td>600 B.C.</td>
<td>Rise of Magadha</td>
</tr>
<tr>
<td>500 B.C.</td>
<td>Persian Conquest. Rise of Buddhism</td>
</tr>
<tr>
<td>321-184 B.C.</td>
<td>Maurya Dynasty</td>
</tr>
<tr>
<td>200 B.C.-300 A.D.</td>
<td>Sung and Kushan period</td>
</tr>
<tr>
<td>319-606 A.D.</td>
<td>Gupta Dynasty</td>
</tr>
<tr>
<td>600 A.D.</td>
<td>Invasion of Huns</td>
</tr>
<tr>
<td>608 A.D.</td>
<td>Harsh Vardhan’s rule</td>
</tr>
<tr>
<td>1000 A.D.</td>
<td>Afgan raids</td>
</tr>
<tr>
<td>1336-1527 A.D.</td>
<td>Muslim Sultans</td>
</tr>
<tr>
<td>1488 A.D.</td>
<td>Portuguese in India</td>
</tr>
<tr>
<td>1527-1707</td>
<td>Moughals</td>
</tr>
<tr>
<td>1650-1761</td>
<td>British and French in India</td>
</tr>
</tbody>
</table>

*Modified after Tylecote, reference 5
** According to the Hindu way of reckoning time these periods would be much earlier than the dates given here.

istics and properties of brass. Notable among these are the Rasratnasamuchchaya which was compiled in the late 13th century A.D.

A detailed account of smelting to produce metallic zinc by what are clearly distillation processes is also given in the Rasratnasamuchchaya. First, the crucible or retort must be made ready. Prepare a “vrantak” (egg plant) shaped crucible of soil, put a twelve “angul” (1 angul = 1 inch) long tube in it which protrudes like a “dhatura” (thorn apple) flower. The tube should be hollow with one end having a periphery of eight anguls. Such crucibles are known as “vrantak moosha” (Moosha = crucible) and are used for the distillation of “kharpar” (Zn ore), etc.

Four processes of zinc extraction from zinc ore are described in great detail in the Rasratnasamuchchaya. The basic process and ingredients used are similar to those described earlier in Rasarnavam and in Rasratnakar, a work of the 2nd century A.D., but suitably modified in light of the knowledge gained during the course of time. The book says “take equal parts of “lac” (lac tree), bark of “pipal” (ficus tree), “harad” (termalalia chaubula), “haldi” (curculongalang roxob), “rasien”, rock salt, “suhaga” (borax) and “kharpar” (zinc ore), grind them together and bake them with cow milk and purified butter. Then make the whole thing into a ball and put into a vrantak shape crucible or retort. Dig a hole in the bottom of the kostihi (furnace) and place a water-filled vessel in it, with a perforated cup or saucer over its mouth. Then, fix the filled crucible in an inverted position over the perforated saucer or cup. Put charcoal of the “jijube” (zyzypad jujube) over and around the crucible and heat it strongly with the help of bellows. On heating, zinc extract goes down and gets collected in the vessel bottom. Archaeological evidence for the use of this process is described below.

Lexicographic works like Amarkosha, Tilakmanjir,
97.5% lead and 150 ppm silver. Thus, the indications are that the scale of operation was enormous and the smelting technology advanced.

It is of interest to note that copper ore is also found at Rajpura-Dariba, suggesting the possibility that brass may have been locally produced.

**Rampura-Agucha**

The Rampura-Agucha lead-zinc deposit located 270 km northeast of Udaipur, provides similar evidence of early lead-zinc mining and smelting in India. Here, the ancient mine workings are preferentially located at the northern limit and in the hanging wall of the deposit. Slag dumps occupy approximately 1800 m² area; excavations done at random spots indicate that their depth is not more than one metre. A piece of wood from the mine workings has been carbon-dated at 2240 years b.p. (Table 2).

 Petrographic thin sections of slag indicate the presence of quartz and feldspar (kaolinized) which have not undergone much change, suggesting incomplete smelting. Furthermore, the slag was found to contain pieces of charred wood entrapped with a vitreous and scoriferous matrix. Some of the slag samples analyzed contained up to 6% lead, 3% zinc and up to 15 ppm silver. Since the present-day ore contains 13.5% zinc, as compared to 3% in the slag, it appears that an appreciable fraction of the zinc was separated in the processing. The implication is that zinc or zinc oxide was being deliberately produced there 2000 years ago.

Near the slag dump area several retort-like pieces were reported. When assembled their appearance suggested a cylinder approximately 20 cm long with walls about 4 cm to 5 cm thick and an innermost pipe-like feature with a coating of dirty white material, mainly zinc sulphate. They could be mistaken for tuyeres but for their closed pointed ends. This is highly suggestive of a used retort. Along with this, some thin-walled tube-like material containing a thin coating of blister type material was also found.

In the vicinity of the Rampura-Agucha deposit are mounds in which are buried archaeological remains, glassy slag and brick wall structures. These throw some light on the ancient civilization and the antiquity of the mining, giving it a date earlier than 600 A.D. The slag recovered from these mounds is the same type as that found in the vicinity of the deposit.

**Zawar Mines**

At Zawar, 45 km south of Udaipur city, Rajasthan, in hilly, sparsely vegetated terrain, has been found solid evidence for early zinc mining and smelting in India. The rocks of the Zawar area constitute a part of the Precambrian Aravalli system and include an assemblage of low-grade metamorphosed calcareous, arnaeous and argillaceous sediments. Dooomite which is the principal host for lead and zinc is sharply inclined into steep ridges above the valley bottom. At the dolomite ridges are extensively honeycombed with ancient workings which follow the plunge of the orebody downward from the outcrop for several hundred metres with no access from the valley sides. The presence of smooth faces and considerable depths of charcoal, ash and calcined rocks suggest that fire setting was the principal means of breaking rock and ventilating the mine workings. A few iron chisels and unusual pestle-like hammers have been recovered from the ancient workings within the modern Mochia mine of Hindustan Zinc Limited.

The remains of wooden stairways, incline haulage shafts, staging and drainage leats survive in the Zawar-mala mine (Fig. 2). Samples taken here for carbon-14 dating from a haulage shaft and a drainage leat reveal that these are approximately 2000 years old.11 12 (Table 2).

Enormous heaps of used retort debris, sometimes reused to build the walls of houses lying in and around old Zawar, indicate that a highly advanced and prosperous zinc mining and smelting industry thrived here in the past. The evidence suggests that this industry was at its peak during 13th to early 18th century A.D.

The discovery of intact ancient zinc smelting furnaces at Zawar13 14 has confirmed the descriptions of the distillation process given in the Rasratnasamuchchaya and described above. The furnaces are extremely sophisticated structures and were used to carry out one of the most advanced pyrometallurgical processes in use prior to the industrial revolution. Furthermore, the presence of hundreds of thousands of tons of debris suggests that many tens of thousands of tons of zinc were produced.

Most of the zinc was used in the production of brass for which there has long been a prodigious demand. The copper for this brass may have been obtained locally as there are numerous abandoned copper mines and heaps of slag within a 20 km radius of Udaipur. These mines and smelters seem to have been in production before and during the period when Zawar was in production 10 (Table 2). It is also possible that copper might have been supplied from the copper district of Khetri.

The history of Zawar is interwoven with the colorful past

<p>| TABLE 2. Table of radio-carbon dates in years b.p. (before the present) for remains found associated with ancient mining and smelting at Rajpura-Dariba, Rampura-Agucha, Zawar, and Ingaldhal, Karnatakak |</p>
<table>
<thead>
<tr>
<th>Sample</th>
<th>Description</th>
<th>Laboratory No.</th>
<th>Date (bp)</th>
<th>Identification</th>
</tr>
</thead>
<tbody>
<tr>
<td>DARIBA</td>
<td>East Lode 4 m to 5 m depth, wood sample</td>
<td>NPL Ahemadabad</td>
<td>2215 ± 85</td>
<td>Acacia sp</td>
</tr>
<tr>
<td></td>
<td>South Lode 100 m depth, timber support</td>
<td>NPL Teledyne</td>
<td>3040 ± 150</td>
<td>Aogeissus sp</td>
</tr>
<tr>
<td></td>
<td>East Lode 263 m depth, timber support</td>
<td>NPL Teledyne</td>
<td>2365 ± 85</td>
<td>Bamboo</td>
</tr>
<tr>
<td></td>
<td>South Lode 100 m depth, rope</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>AGUCHA</td>
<td>Wood support from 30 m depth</td>
<td>BM 2356</td>
<td>2240 ± 60</td>
<td>Terminalia sp</td>
</tr>
<tr>
<td></td>
<td>Balaria mine 10 West stope, 1st level</td>
<td>BM 2338</td>
<td>170 ± 50</td>
<td>Terminalia sp</td>
</tr>
<tr>
<td></td>
<td>415 MRL timber</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>Mochia mine</td>
<td>BM 2381</td>
<td>2360 ± 50</td>
<td>Terminalia sp</td>
</tr>
<tr>
<td></td>
<td>Zawarzama mine</td>
<td>PRL 932</td>
<td>2410 ± 100</td>
<td>—</td>
</tr>
<tr>
<td></td>
<td>Zawarzama mine</td>
<td>BM 2149</td>
<td>1920 ± 50</td>
<td>—</td>
</tr>
<tr>
<td></td>
<td>Timber from scaffold</td>
<td>BM 2148</td>
<td>2120 ± 60</td>
<td>—</td>
</tr>
<tr>
<td>Base of debris</td>
<td>charcoal</td>
<td>BM 2223</td>
<td>230 ± 60</td>
<td>Artocarpus sp</td>
</tr>
<tr>
<td>INGALDHAL</td>
<td>Zawar site 34 charcoal</td>
<td>BM 2243</td>
<td>80 ± 60</td>
<td>—</td>
</tr>
<tr>
<td></td>
<td>Copper mine 30 m to 40 m depth, wood</td>
<td>BM 2364</td>
<td>1810 ± 35</td>
<td>—</td>
</tr>
</tbody>
</table>
of the former state of Mewar (Udaipur). An inscription dating to the 7th century A.D. found at Samoli in southern Mewar refers to the arrival of a merchant in the area and his starting of a mine at Zawar, then known as Arandyagire. Inscriptions on old temples constructed in and around old Zawar tell of mining and working of the deposits in the 7th century A.D. and of the full flourishing of the industry between the 15th and 17th centuries. During this period Zawar became a fairly large town. According to local tradition Rana Pratap Singh (1572-1597) took shelter in the old mine openings during the struggle for freedom against the Mughals. Recently, a group of 29 silver coins (Fig. 3) freshly minted, dated firmly to 1593 A.D. in the reign of Akbar, were recovered from soil burying an ancient furnace, demonstrating that furnaces in this area were in full operation prior to this date. During the time of Maharana Bhim Singh (1778-1828) the State of Mewar was plundered by Maratha invaders. Mewar rapidly approached dissolution and with every sign of civilization fast disappearing, the mines of Zawar were abandoned. In the 1940s, they were reopened by the Utilization Branch of the Geological Survey of India and at present are producing around one million tonnes per year of lead and zinc ore with reserves of 25 years proven.

It is clear from the evidence presented above that for many centuries in the past, long before the industrial revolution in the west, India has been a land vigorously involved in flourishing zinc mining and smelting activities.

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Brass making in medieval western Europe

Sandra K. Zacharias

Brass, a deliberate alloy of copper and zinc, has been known since before Roman times. There are two main methods of making brass:

1. Cementation or calamine process — crushed zinc ore (calamine ZnCO₃, or calcined calamine, ZnO), is mixed with charcoal and small pieces of metallic copper and heated to about 900°C in a closed crucible.
2. Alloying — metallic copper and zinc are melted together in a reverberatory furnace.

According to most modern authorities (e.g. Smith and Forbes 1955), the calamine process was the only one used in western Europe prior to the 18th century. At that time, the alloying technique began to take over, because metallic zinc was successfully produced on a large scale for the first time. Previously, this had been a problem because zinc vaporizes at 906°C or about the same temperature at which it can be reduced from its ore. Thus, some means had to be devised to collect and condense the zinc vapour (Tytecote 1976).

Metallic zinc was known, however, earlier than the 18th century A.D. Several early finds have been made, including a rolled sheet of metallic zinc found in excavations at the Athens Agora dating to the 3rd to 2nd century B.C. (Farnsworth et al. 1949).

The Chinese were producing zinc by at least the 10th century A.D. and, by the 17th century, Chinese zinc ingots were being imported into Europe by the Dutch East India Company (Needham 1974). Metallic zinc is also mentioned in several European documentary sources from the 16th century (e.g. Paracelsus, Agricola).

This study traces the changes in Western brass-making methods from Classical to Renaissance times by examining documentary and archaeological evidence.

Brass: Identification and Nomenclature

The names for brass and other copper-based alloys reflect their origins, properties and uses. Prior to the 20th century, the terms for brass and bronze were used interchangeably in most documents (e.g. the King James Bible), resulting in conflicting interpretations by modern scholars.

The ancient Greek name for brass, used from the 7th century B.C. onward, has been identified as orichalcos or “mountain copper” (Caley 1964; Craddock 1978), possibly referring to its original place of manufacture, or to a naturally occurring copper-zinc deposit (as is found in Cyprus or Persia).

In the Roman period both copper and copper-based alloys were called aes. There was also a specific name for brass, Aurichalcum or “golden copper”, which is phonetically similar to the Greek orichalkos. According to Pliny (XXXIV, 2) Aurichalcum was used to make sesterteri, or brass coins.

In the Latin writings of the Medieval period, beginning with Theophillus (ca. 1100), aes and aurichalcum are used more or less interchangeably. Theophillus generally uses aes to mean an impure copper-based alloy (e.g. containing lead) and aurichalcum, a refined copper-based alloy — each with either tin or zinc as the major alloying element.

Albertus Magnus (ca. 1248) uses aurichalcum to refer to any copper-zinc alloy, and aes to mean “copper”. Cyprium (from Cyprium = Cyprus, a major ancient copper source) is also used by both authors to mean “copper”.

In Renaissance times, latten (or laten, laton, lattyn) became the common English word for brass, akin to the French laiton (= brass) and Italian latta (= sheet brass), and probably based on the Latin latte or lathe (= sheet) (Cameron 1974). Usually small amounts of tin and lead were present in latten, in addition to copper and zinc (Table 1).

Brass was not manufactured in England until the late 16th century (Donald 1961). Forbes (1966) suggests that the term latten referred to the flat cast sheets of brass which were traded from the continent.

The origins of the German word for brass, messing, are obscure. Forbes (1966) suggests that it may be related to the Latin massa (= lump of metal). This would be a good descriptive term for the initial product of the cementation process.

The modern English brass may be related to the French brasier (= braze or solder).

Copper-zinc alloys in Renaissance times and later had different names based on their composition and uses. The major ones are summarized in Table 1.

Zinc: Indentification and nomenclature

When zinc ingots from China were imported into Europe beginning about A.D. 1600, there was some confusion about the nature and identity of this metal. This resulted in a profusion of names quite unrelated to the local names for zinc ores. These included tutenag (derived from the Middle Persian tutiy, calamine, which became the English tatum, zinc oxide), also called “toothanage” or “tooth and egg” metal (Needham 1974), and spelter (likely from the similar coloured lead-tin alloy, pewter, or the Dutch equivalent, spiautert (Dawkins 1950).

The Chinese had begun to produce metallic zinc in quantity by at least the 9th to 10th Century A.D. In the Ming Dynasty, A.D. 1402 and later, almost pure zinc coins were in use (Needham 1974). There is some suggestion of an Indian or Persian origin for zinc making (Werner 1972)*. European zinc, in contrast, was not produced on

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*See also Falluval et al., this volume.
**The archaeological evidence for metallic zinc at pre-Christian Era sites has been well summarized by Forbes (1966) and Craddock (1978). Many of the finds were made in the late 19th century and are now lost. Two of the best documented finds are from the Agora at Athens, dating to the 3rd and 2nd Centuries B.C. (Farnsworth et al. 1949) — a rolled sheet of zinc and a small zinc statue. The sheet is 99% pure but not of a modern composition. It has been well hammered to 0.5 mm thick. As zinc is brittle from room temperature to 100°C and above 250°C, this sheet must have been worked between 100°C and 250°C. Forbes (1966) believes this find is out of context, and from a later Chinese tea chest. However, the composition is unlike the normal Chinese alloy (Needham 1974).
TABLE 1

<table>
<thead>
<tr>
<th>Name</th>
<th>Cu</th>
<th>Zn</th>
<th>Sn</th>
<th>Pb</th>
<th>Remarks</th>
</tr>
</thead>
<tbody>
<tr>
<td>pinchbeck</td>
<td>88-93</td>
<td>7-12</td>
<td></td>
<td></td>
<td>after Christopher Pinchbeck, London jeweller</td>
</tr>
<tr>
<td>gilding metal</td>
<td>80-90</td>
<td>10-20</td>
<td></td>
<td></td>
<td>alpha-brass, easily cold worked, imitates gold in colour</td>
</tr>
<tr>
<td>tombac</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Tournay's metal</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Mannheim gold</td>
<td>84</td>
<td>5</td>
<td>7</td>
<td></td>
<td>“bronze” with similar properties to gilding metal</td>
</tr>
<tr>
<td>Prince’s metal</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Best brass</td>
<td>76-84</td>
<td>16-24</td>
<td></td>
<td></td>
<td>yellow alpha-brass, easily cold worked</td>
</tr>
<tr>
<td>Bath metal</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Dutch gold</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Latten</td>
<td>75-85</td>
<td>5-20</td>
<td>3-8</td>
<td>0.2</td>
<td>as above</td>
</tr>
<tr>
<td>Standard brass</td>
<td>70</td>
<td>30</td>
<td></td>
<td></td>
<td>greenish-yellow alpha-brass, cold workable</td>
</tr>
<tr>
<td>cartridge brass</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>practical upper zinc limit of the calamine process</td>
</tr>
<tr>
<td>Muntz metal</td>
<td>60-62</td>
<td>38-40</td>
<td></td>
<td></td>
<td>greenish-yellow, alpha and beta brass, easily hot worked, hard,</td>
</tr>
<tr>
<td>naval brass</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>corrosion resistant (e.g. for ships)</td>
</tr>
<tr>
<td>Macht metal</td>
<td>48-57</td>
<td>43-52</td>
<td></td>
<td></td>
<td>reddish beta-brass, hard and brittle</td>
</tr>
<tr>
<td>White brass</td>
<td>34-40</td>
<td>60-66</td>
<td></td>
<td></td>
<td>white, gamma-brass, very brittle</td>
</tr>
</tbody>
</table>

(after Alexander and Street 1944; Needham 1974; Cameron 1974; Craddock 1978)

an industrial scale until after A.D. 1743, when William Champion (who may have learned the process from the Germans or from further East) set up a smelting operation in England (Day 1973). Prior to this time, metallic zinc was not unknown in the West **, however there seems to have been a problem in correlating the European alchemical view of zinc as a semi- or false-metal with this new metallic substance, of unknown manufacture, from China.

Even the early Greek writers view zinc as an impostor or semi-metal. Strabo (XIII,1, 56) mentions that “droplets of false silver (pseudargyrum)” form on the interior furnace roof when iron is smelted with a certain earth found in Asia Minor. He then describes how this false silver can be added to copper to make oreichalkos, strongly suggesting that false silver is zinc.

Dioscorides (V,46) and, later Pliny (XXXIV, 33) both describe how pompholyx, or pure white zinc oxide (identified by its suggested use as an eye salve), condenses and adheres to the furnace interior during copper smelting. Depending on the zinc content of the ore and on the atmosphere inside the furnace, either metallic zinc (reducing atmosphere) or zinc oxide (oxidizing conditions) could be formed as a smelting by-product. The “magical” formation and time-consuming collection of these substances would surely increase their potency for healing and alchemical purposes.

In the medieval period, Paracelsus (ca. 1540) (cited in Dawkins 1950) is credited with the first description of zinc (= zincke), “a bastard of copper, a peculiar metal, but often adulterated by foreign metals. It is of itself fusible, but does not admit of hammering, but only fusion. It does not allow itself to be mixed by art with other metals, but remains by itself.”

This was an accurate description of zinc’s properties as far as they were known. Zinc is too brittle to be cold worked to any extent, and its low melting point (419°C) and boiling point (906°C) prevent easy mixing with other metals.

In De Re Metallica (ca. 1556), Agricola refers to conterfei or metallic zinc, as an accidental furnace product. In De Natura Fossilium (ca. 1560), Agricola uses the word zincum to mean the metal or, more likely, the ore: “cognate with . . . cadmia and pyrites is a compound which the Noricians and Rhetians call Zinicum”.

Forbes (1966) and Dawkins (1950) suggest that the term zink referred to the needle-like form of either the condensed zinc crystals or of the local calamine (ZnCO₃) ore: from the German zinke, zacke = point, jagged part, tooth (Dawkins 1950). Agricola apparently did not identify the furnace product with zinicum.

Löhneiss (ca. 1617) (cited in Agricola 1950) was the first to state that the furnace wall condensate is “a metal called zink or counterfei . . . [which] greatly resembles tin, but is harder and less malleable” and greatly esteemed by the Alchemists.

Finally in 1656, Glauber (cited in Agricola 1950) correlated metallic zinc and its ore, calamine: “zinc might be called a fusible lapis calaminaris inasmuch as both of them partake of the same nature” (such as the ability to transform copper into brass).

Manufacturing Techniques

Calamine Process

The first description of brass making by the cementation of calamine (ZnCO₃), or calcined calamine (ZnO), with copper is an indirect reference by the Greek physician, Dioscorides (V, 46) who states that pompholyx (zinc oxide) is produced as a “soot which flies up when the copper refiners sprinkle powdered cadmia [= calamine (Agricola 1950)] over the molten metal”.

Pliny’s (XXXIV, 33) descriptions of metallurgical processes are often confused, and probably not first hand. He does note, however, that Spanish and Livanian copper “most readily absorb cadmia” and are used for making sterceres, or brass coins.

A passage in Pseudo-Aristotle (cited in Forbes 1966) describes how “the bronze of the Mosynnoeci (a tribe in Asia Minor) is very shiny and light in colour though tin is not mixed with the copper but a certain earth which occurs there is smelted with it”. Craddock (1978) points out that both zinc and arsenical ores are found in Asia Minor. This passage could refer to either zinc-cementation or to smelting arsenical ores with copper.

The next source is Theophilus (ca. 1100) who describes brass making step-by-step, from crucible manufacture
onward. The calamine (ZnCO₃) is first calcined to zinc oxide by heating red hot in an open fire. Then it is mixed with charcoal and small copper pieces and heated in a closed crucible.

He goes on to describe how the copper is heated to liquid and more calamine is stirred in. Translators Smith and Hawthorne (Theophilis 1963) suggest that this is poor metallurgical practice: if the copper were liquid (1) it would sink to the bottom of the crucible, so there would be far less surface area for the zinc-cementation process to take place, and (2) the temperature would be high enough to rapidly vapourize the zinc formed by adding calamine, which would then escape from the open crucible. Zinc-cementation is a surface phenomenon: for highest zinc content the copper pieces must be tiny and the temperature kept between 900°C and 1000°C (above 906°C the zinc vapourization temperature and below 1083°C, the copper melting point) (Craddock 1978). After cementation, the product must be melted to homogenize the alloy. Theophilis may be combining the two operations in his description.

Albertus Magnus (ca. 1248) describes how either calamine or furnace tutty may be used to colour copper gold. He suggests that a more golden lustre may be obtained by sprinkling crushed glass on top of the mixture in the crucible to form a slag which will help prevent the escape of the zinc vapour; in other words, increase the zinc content of the brass.

Biringuccio (ca. 1540) has the next most complete description of brass making. He describes how either calamine or furnace tutty can be mixed with broken up pieces of copper, and sprinkled with a layer of powdered glass, then heated in a closed crucible for 24 hours. Translators Smith and Gnudi (Biringuccio 1942) point out that crushed charcoal must be added to this mixture for the process to succeed. The powdered glass would also act as a flux to aid in the reduction of the calamine to zinc.

Ecker (ca. 1565) (Fig. 1) describes an operation similar to those already noted: a mixture of 46 pounds of powdered calamine (= galmey), 64 pounds of broken up copper, and powdered coal (= kohlen), dampened with water, is heated in 8 closed crucibles for 9 hours, then stirred with an iron rod and left one “good” hour more. The resulting brass can be reheated with calamine to achieve a higher “colour” or zinc concentration, but he considers this no great advantage. In fact, “die Messing-Brenner sagen, wenn anstatt des Wassers Urin und Alun genommen wird, so wird der Messing im ersten Feuer schön genug”.

Löhneiss (ca. 1617) (cited in Biringuccio 1942) describes how in about 1550 at Rammelsberg, Germany, Erasmus Ebener began to make brass from ofenbruch or tutty, impure zinc oxide accretions from lead smelting that had previously been piled in the slag heaps. After this time several authors note that devices were set up in furnaces to collect the tutty (e.g. Swedenborg 1734 cited in Smith and Forbes 1955).

In the Landesarchiv Salzburg from the 17th century (Bauer 1974) a two-step brass making operation, to achieve a higher zinc content, is described: (1) cementation of calamine with broken up copper, and (2) cementation of calamine with the broken brass from step (1).

Diderot’s (ca. 1765) description of brass making recalls Theophilis. The calamine is calcined to zinc oxide in an open fire made with wood and straw. Then it is sieved, mixed with charcoal, and heated with copper in a crucible.

*The Brass-workers say: If urine and alum are used instead of water, the brass from the first fire will be beautiful enough*.
TABLE 2

<table>
<thead>
<tr>
<th>Period</th>
<th>Cu</th>
<th>Zn</th>
<th>Sn</th>
<th>Pb</th>
<th>Remarks</th>
</tr>
</thead>
<tbody>
<tr>
<td>Greek</td>
<td>~97</td>
<td>~3</td>
<td>n.a.</td>
<td>n.a.</td>
<td>coins, accidental alloy*</td>
</tr>
<tr>
<td></td>
<td>~94</td>
<td>~6</td>
<td>n.a.</td>
<td>n.a.</td>
<td>6th c. B.C. statue, alloy deliberate? *</td>
</tr>
<tr>
<td>Etruscan</td>
<td>84-87</td>
<td>12</td>
<td>1-3</td>
<td>n.a.</td>
<td>3-2nd c. B.C. statues, delib. alloy*</td>
</tr>
<tr>
<td>Roman I</td>
<td>~67-70</td>
<td>22-28</td>
<td>0.1-3</td>
<td></td>
<td>coins, decorative work; 1st-4th c. A.D.</td>
</tr>
<tr>
<td>II</td>
<td>~70-80</td>
<td>10-20</td>
<td>0.1-5</td>
<td>0.1-6</td>
<td>trend to lower Zn alloys</td>
</tr>
<tr>
<td>Roman period</td>
<td>77</td>
<td>17</td>
<td>4-5</td>
<td>0.5</td>
<td>c. 150-200 A.D.</td>
</tr>
<tr>
<td>N.W. Europe</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>&quot;Hemmoor&quot; bowls **</td>
</tr>
<tr>
<td>Medieval West</td>
<td>~83</td>
<td>11</td>
<td>3</td>
<td>3</td>
<td>11-13th c. (German) doors, cast objects</td>
</tr>
<tr>
<td></td>
<td>~77</td>
<td>17</td>
<td>2</td>
<td>4</td>
<td>(Werner 1970)</td>
</tr>
<tr>
<td></td>
<td>77-81</td>
<td>20</td>
<td>~3</td>
<td>~4</td>
<td>14th-17th c. (as above)</td>
</tr>
<tr>
<td></td>
<td>70-80</td>
<td>17-23</td>
<td>~3</td>
<td>2-8</td>
<td>14th c. mem. brass plates (latten) **</td>
</tr>
<tr>
<td></td>
<td>70-80</td>
<td>11-28</td>
<td>~3</td>
<td>1-13</td>
<td>15th c. (as above)</td>
</tr>
<tr>
<td></td>
<td>65-73</td>
<td>27-33</td>
<td>~1</td>
<td>~2</td>
<td>1500-1550 (as above)</td>
</tr>
<tr>
<td></td>
<td>67.5-81</td>
<td>15-31.5</td>
<td>0.1-3</td>
<td></td>
<td>1550 (as above); corrodes more easily than</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>high Sn-Brasses</td>
</tr>
<tr>
<td>Modern alpha</td>
<td>66-76.5</td>
<td>21.5-34</td>
<td>0.1-0.5</td>
<td>1-1.5</td>
<td>1500-1600 astronomica instruments (Hedges 1979)</td>
</tr>
<tr>
<td>brass (also see</td>
<td>71</td>
<td>27</td>
<td>~1</td>
<td>~1</td>
<td>U.S. NBS 37D (mihel and Asaro 1979)</td>
</tr>
<tr>
<td>Table 1</td>
<td>70</td>
<td>30</td>
<td></td>
<td></td>
<td>Brit. Standard Coldworking Brass **</td>
</tr>
</tbody>
</table>

* Craddock (1978)  
** Cameron (1974)


Alloying Process

The earliest reference to brass production by the direct alloying of metallic copper and zinc is, possibly, Strabo (XIII, 1, 56). He states that "droplets of false silver" formed on the furnace walls as an accidental smelting by-product can be added to copper to make oreichalkos. Industrial scale brass alloying did not take place in Europe until the late 18th century, following Champion's 1743 patent for zinc smelting (Day 1973).

The alloying process is a two-step operation: (1) reduction of zinc from its ore, and (2) copper and zinc melted together into an alloy. It has advantages over the calamine process: greater purity of zinc; accuracy of alloy proportions (with higher zinc content), and ability to have a larger scale operation. To its disadvantage, it requires more time (2 steps) and more fuel than the calamine process.

The German chemist Marggraf is credited with the first description, in 1746, of zinc smelting from calamine ore. There are other earlier unsubstantiated claims including those by Henkel (1721), and a Dr. Lane (1717) (Cocks and Walters 1968).

Champion's zinc smelting apparatus consisted of 6 to 8 large crucibles (3 feet high with a rim diameter of 21 inches) in a furnace, each with a 4-inch iron drain pipe protruding from its base. A charge of calcined calamine (and charcoal?) was added to each crucible through a small opening in the lid. When heat (~900°C) was applied, the zinc vapour ran down into the pipes where it cooled. The condensed liquid zinc dropped into a pan of water under the pipes (Cocks and Walters 1968).

Earlier, an apparatus operating on a similar principle was used in India (Forbes 1966) (Fig. 2). The Chinese method (ca. 1637) (Fig. 3) was somewhat different. Calamine was put in tightly sealed crucibles that were dried slowly to prevent cracking when heated. A pile of crucibles was fired with charcoal to fuse the ore in each into a mass. Zinc was apparently produced at a 20% loss. Zinc vapour would likely coat the crucible walls (Dawkins 1950). A crucible in use in early 20th century China (Fig. 4) suggests a mechanism for more efficient zinc production using small crucibles (Werner 1970).

Once zinc is distilled and cast into small ingots, it can be alloyed with copper in a reverberatory furnace. A certain portion of the zinc is lost, due to its low boiling point.

Archaeological Considerations

Alloy Composition

Brass is a deliberate alloy of copper and zinc. Zinc that is...
naturally present in a copper ore may be partly retained by the smelted copper depending on the ore type and smelting process. Estimates range up to 7% zinc for accidental alloying resulting from a high zinc copper deposit (examples of such deposits are in Cyprus, Asia Minor, Persia) (Forbes 1966; Werner 1972).

Most of the zinc in copper carbonate and oxide ores, or “dead roasted” copper sulphide ores (sulphides roasted in an open fire at less than 800°C to entirely convert them to oxides) is retained by the copper if reducing conditions are maintained during smelting. However, zinc is lost when copper sulphide ores are reduced by matte smelting, a process first used by the Romans (i.e. smelt to matte [CuFe sulphides], at 1200°C, then refine) (Tylecote et al. 1978).

Up to the late 17th Century analyzed European brasses do not exceed 35% zinc, and usually contain less than 28% zinc (Craddock 1978; Hedges 1979). Werner (1970) has made experiments which show that the likely zinc composition of a brass made by cementation at 1000°C is between 23% and 28% zinc. He suggests that 30% zinc is the “magic” number for the calamine process.

Grothe (1971) disagrees with Werner. He points out that zinc-cementation can take place over a range of temperatures. In ideal conditions* up to 50.5% zinc can be cemented with the copper at 950°C. However, the practical upper limit of the calamine process appears to be about 30% to 33% zinc.

Biringuccio (ca. 1540) describes a brass that weighs 8% more than the copper from which it was produced — about an 8% zinc-brass. This may have been a special purpose alloy similar to pinchbeck (Table 1). From the charge proportions that Ercker (ca. 1565) describes, the product is a 27% zinc-brass. Later, Wm. Champion (Day 1973) achieved a high of 33.3% zinc by granulating the copper before cementation — molten copper was poured through a sieve into cold, running water. These and the brasses presented in Table 2 are all alpha-brasses which can be easily cold worked.

In direct alloying of zinc and copper, the zinc content can be much higher, limited only by the volatile nature of zinc and the melting conditions. The presence of small quantities of tin and lead, from metal re-use or deliberate additions, act to improve the alloy (tin increases strength and lead, fluidity for casting) and also to depress the amount of zinc that can be absorbed (Needham 1974). Lead excludes zinc by 1:1, tin by 2:1 (i.e. 3% tin will exclude 6% zinc) both by their physical presence and by lowering the copper-alloy melting point and, thus, decreasing the zinc cementation range. More than 5% tin lowers the melting point below 900°C or below the temperature at which zinc vapourizes (Craddock 1978).

Approximately 10% zinc loss also occurs each time the alloy is remelted (Craddock 1978).

**Product Control**

The main method in use beginning in Biringuccio’s time or earlier, was to weigh the copper before and after cementation or alloying (the similarity in atomic weights means that the weight % increase in copper is approximately the weight % zinc in the brass).

The approximately 30% practical upper limit for zinc in the calamine process creates another kind of product control. Craddock (1978) found that Roman brasses usually contained either 22% to 28% or 10% to 20% zinc. The latter suggests dilution of the original brass by adding about an equal amount of copper or bronze.

Colour change from “imitation gold” (under 20% zinc) to a deeper greenish yellow (20% to 40% zinc), as noted by Ercker, could also have been an indication of zinc content.

Table 2 summarizes the average changes in brass composition in analyzed brasses from Classical to Modern times.

**Discussion**

The general trends in brass composition over time (Table 2) are toward a higher zinc content and, especially after 1550, a higher purity brass. These changes occurred during the time that the calamine process was, according to all sources, still the major brass making technique; evidently technical changes were taking place which affected the alloy composition.

In general, less than 33% zinc brass, with a higher level of impurities than the modern, was produced by the calamine process. Impurity levels could vary, however, if (1) furnace tuffery were used instead of calcined calamine, (2) scrap bronze or brass were used instead of copper, or (3) improved copper refining methods were employed.

The brasses produced from the 99% pure Chinese zinc ingots introduced into Europe about A.D. 1600 may also have had low impurities and, possibly, higher zinc contents. A possible example of Chinese-zinc brass is a 1634
English quadrant containing 34% zinc and only 0.1% tin (Hedges 1979).

The change from the calamine to the alloying process in Renaissance Europe was slow in spite of earlier knowledge of metallic zinc. Several factors may be involved in this reluctance to change:

(1) The desired properties or intended uses of the alloy — brasses of less than 20% zinc look like gold and may well have been considered alchemically (and aesthetically) superior to greenish-yellow high zinc-brass. Brass in the high alpha range is also not as easily cold worked as lower zinc-brass. In addition, small amounts of impurities such as tin and lead can be beneficial, for example in the production of large castings such as memorial brass plates. Fine instruments, in contrast, require a higher purity brass. Compounds, in particular, cannot be made from an alloy with many iron impurities. Day (1973) suggests that Chinese zinc was used for instrument brass.

(2) The increased initial expense of the alloying process over the calamine process — more time, fuel, and new equipment were required for the two-step alloying technique.

(3) The European alchemical tradition — the calamine process followed the precepts of Alchemy: copper was transformed into a kind of gold by mixing it with an Earth. Zinc, on the other hand, was a “bastard of copper”, “false silver”, “counterfeit”, an accidental and undesirable byproduct of copper or lead smelting. It did not fit into the alchemical concept of seven planets with their corresponding seven metals (Paracelsus, cited in Dawkins 1950). For zinc (and bismuth, cadmium, etc.) to attain “metal” status, a major restructuring of scientific theory was required.

“Having considered what I have told you concerning brass, it seems to me impossible to deny that it is one of the works of alchemy, since copper is red by nature and this redness is taken from it by art and converted into yellowness” (Biringuccio).

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The Farga Catalana

P. Molera

Introduction

In Catalonia, between the 10th century and 1878, some installations were developed which at first were called “factories” and later “fargas”, since they manufactured iron and steel of low carbon content. Because of its excellent quality, the fame of this metal spread from Catalonia to many countries. Fargas were started in Corsica, the Low Countries, Germany, Oriental India, Madagascar, Finland and along the rivers Orinoco and Mississippi among others. The extractive process used in these fargas has passed into history as the “Catalan Farga Process”, “Catalan Process”, or just “Farga Catalana”. With this technology, raw material was obtained for the fabrication of tools, nails, weapons, grilles, etc. to supply the national and export markets.

Aside from meteorites and native iron, found as metallic nodules in basalts in Greenland and Palestine, the ancient civilizations, which had no means to reach the fusion temperature of iron (1535°C), obtained iron by a direct process. Similar processes were used quite recently (1930s) by certain tribes in Central Africa; holes in the ground were filled with iron ore and coal, and combustion encouraged by blowing air by mouth or with bellows made of animal skins. Thus without melting the iron it is possible to reduce the mineral and obtain a porous and impure metallic mass, which is purified on forging by mechanical squeezing. With a process very likely similar, the famous column at Dehli weighing 6.4 m.t. was built about 300 B.C.

Description

The Farga Catalana process consisted of feeding a mixture of iron minerals (oxides), sufficiently triturated with crushed charcoal, into a furnace into which air was blown under pressure through a tuyere ending less than 30 cm above the furnace floor. By the combustion of the charcoal, temperatures of some 1200°C could be obtained, sufficient to convert the mineral into a pasty mass consisting of iron nodules and slag. This mass was compacted and purified by forging with drop hammers [1].

It is somewhat difficult to give an exact description of the elements which make up the Farga Catalana, because each installation depended on circumstances, such as location, economic means of the owner, and the creativity of the planners. However, in all the Fargas there was a furnace with its water nozzle (“trompa d’aigua”) and two drop hammers.

Furnace

The furnace of the Farga Catalana was the most important element of this technology. It consisted of a cavity in the shape of a truncated pyramid, the dimensions of the base being about 60 cm by 50 cm, with the height some 80 cm. The furnace was placed next to a main wall, called “piec del foc”. All the walls of the furnace were flat, except the “l’ore-contravent” (wind wall opening) which, to facilitate the extraction of the reduced product, had a convex surface and was partially covered with steel plates. The wall of the furnace next to the “piec del foc” and opposite to “l’ore” was called “les porques” and contained the tuyere which injected air into the furnace. In a side wall was the “lieieiro”, a hole for removal of the liquid slag. The front of the lieieiro was the “cava” (cellar), a stone wall, without surfacing and slightly inclined.

The characteristics which differentiated this from other contemporary iron-making processes was the system used to inject air into the furnace — the water nozzle (Fig. 1). This made up of what is now known as the Venturi effect. To accomplish this, water from the river was passed into a reservoir, from which it fell through a vertical tube, which in its upper part had a reduced cross section and holes, named “espirals”. The decreased pressure in the reduced cross section resulted in the suction of atmospheric air through the “espirals” into a wind box at the end of the tube (“la caixa de vents”) from which it was injected into the furnace through the tuyere.

The great secret in the construction of the furnace of the Farga Catalana was the inclination of the tuyere with respect to the wall of the furnace, and in the distance between the tuyere and the bottom of the furnace. The incidence angle of the tuyere was between 35 and 45 degrees and the distance from the bottom of the furnace was between 10 cm and 30 cm.

Forge

The drop hammer was the element of the Farga Catalana used to hot deform (forge) and consisted of the hammer (“mall”), the anvil and the woodwork. The hammer was made up of a mass of steel weighing some 500 kg, at the end of a large shank made from a tree trunk, generally oak, 4 m long (Fig. 2).

A water wheel rotated a shaft (“calaibre”), at the end of which was a steel crown (“bota”) with four lugs (“palmes”). These lugs depressed one end of the shank, raising the hammer at the other end; further rotation of the calaibre allowed the hammer to fall.

The anvil was made up of the “pedra”, the “demet” and the “demet”. The “pedra” was a large stone, almost completely buried in the ground, very shock-resistant, on the upper central part of which a rectangular hollow had been made. Into this a rectangular steel slab (“demet”) was inserted held by means of steel wedges (“demet”).

Raw Materials

The Farga Catalana needed mineral and coal as raw material. The mineral, or more correctly, the concentrated ore suitable for the Farga Catalana process, was limonite, or hydrated oxide of iron, found in the Eastern Pyrenees. If this type of mineral was not available, ferric oxide, olistost, was used, after a hydrating treatment consisting of concentration, crushing and exposure outdoors for a few days, spread out on the ground. The orebodies exploited by the Fargas were rather poor. Information on the beneficiation of iron ores is available for numerous Catalan zones[2].


References:

The fuel and reductant used in the Farga process was charcoal made from pine and beech wood. The carbonization of the wood was accomplished by burning it with insufficient air. Tree trunks, cut in short lengths, were piled up and covered with a layer of soil, with holes made in the lower part and on top for the entrance and exit of air. Once the fire was lit with very dry branches, the carbonizing process proceeded, burning small amounts of lignite and cellulose. The evolution of this process could be followed from the outside by taking note of the fumes coming off. Experimentally, it may be observed that on heating wood at temperatures above 100°C water vapour comes off in the form of white fumes. On heating wood in the temperature range 120°C to 500°C, the fumes are blue, because at these temperatures pyrogenic acid, a mixture of phenol and guaiacol, is given off. The end of the process is when no more fumes are given off, because then the charcoal will start to burn. The charcoal is, then, the product resulting from this incomplete combustion of wood. It is a dark substance, brittle, fibrous, with few salts and high in carbon content.

The importance of charcoal to the Farga Catalana is obvious from the placement of these installations near large woods rather than near the orebodies. This is logical, if one considers that for each ton of iron produced, larger amounts of charcoal were needed than of mineral. On the other hand, the price was the inverse; a ton of mineral was more expensive than a ton of charcoal. A document of the 17th century, signed in Ripoli, quotes he use for 15 years of a pine woods at 6 gold “doblas”.

All Fargas which we have found were located next to a river, which supplied water, via a reservoir, to feed both the water nozzle and the water wheel.

**Operation**

The Fargas operated (Fig. 3) with a specialized personnel team under an “administrador”, a position equivalent to that of a manager in today’s firms; he was in charge of purchasing raw materials, the sale of the metal produced and employment of personnel[10]. Generally, he was also the owner of the Farga. The technical part was the responsibility of the “foguer”, a position equivalent to that of the superintendent of today’s steel plants. He took care of the construction and repair of the furnace and tuyeres, and he controlled the quality of the products made. The Farga personnel rested on Sundays, although on this day toward evening the “escola” and his helper took care to light the charcoal in the furnace so that on Monday it was sufficiently warm to be charged with mineral. The person in charge of mineral preparation was the “picamera”.

The operation of ore reduction was started by covering the bottom of the furnace with charcoal powder, which was lit. Then, once the furnace was hot, a layer of larger pieces of charcoal was added. Immediately, a steel plate was placed parallel to the “porgues” in the centre of the
furnace; this permitted placement of a vertical layer of carbon on the tuyere side and a layer of mineral on the other side of the plate. Then the plate was removed and the upper part of the mineral bed was covered with a layer of small pieces of moist carbon and slag, so that the outside surface had a spherical shape and would concentrate the flames. Immediately air was blown in through the tuyeres and after one and a half hours of operation it was possible to obtain temperatures above 1200°C. Over a period of 3 or 4 hours mineral and carbon were added, while in the meantime, through the “ileteiral”, liquid slag was removed. If the slag was very viscous and therefore rich in iron, it was reintroduced into the furnace. At the end of the operation an irregular ball of about 100 kg of iron or low-carbon steel was found at the bottom of the furnace, with slag inclusions and pores — “el masser” (Fig. 4). At this moment the most spectacular operation at the Farga started: the removal of the “masser” from the furnace and its transport to the drop hammer. This operation required the use of all the personnel of the Farga. The purpose of the drop hammer equipment, which operated under the direction of the “mallor” was threefold: to eliminate the slag from the “masser”, to make it more compact and to shape it. The “masser” had to be forged while it was still hot from the furnace. Normally the compaction was done in one drop hammer and the shaping in another.

The output of the operation was low because to obtain 100 kg of metal more than 300 kg of mineral and more than 400 kg of charcoal were required.

The metal obtained was classified according to quality and the extent of the shaping. As to quality it was divided into “ferro comà” (iron) and “ferro bò” (steel). The latter was a low-carbon steel, which was harder, of higher strength and of lower ductility. It was obtained by allowing the “ferro comà” to stay longer in the furnace surrounded by incandescent carbon, with the precaution that the iron mass would not be in contact with the atmosphere (Fig. 5 and 6).

According to whether or not it was forged, the product was classified as “batut” and “buidat”. Depending on the dimension of the forged section, the “batut” was classified as:
(a) “Verga o vergalina”: a round section used for weapons and nails.
(b) “Barrot”: a square section with a three-finger cross section.
(c) “Llaunes”: a rectangular section, 3 fingers wide and one finger thick.

(d) “Caira”: a square section, one finger thick.

**Theory**

The Farga Catalana process, judged by the quality of the products obtained, was a fairly high technology, although it had been developed by metallurgical art transmitted from masters to apprentices through the Middle Ages and modern times. The following is a scientific interpretation of that technology. The great intuition of the artisans of that time surprises us. We shall comment on the major concepts.

The operation done by the “picamera” (ore crusher) consisted of grinding the concentrate and then heating it in a carbon furnace placed outside the Farga building. The objective of this calcination was to transform the limonite or the hydrated oligist into solid, very porous iron oxide with large specific surface. Water vapour was eliminated into the air. Siderite ore, very rare in the Oriental Pyrenees, was changed into ferrous oxide, a porous solid, and carbon dioxide gas which went into the atmosphere. Porous solids were desirable since the speed of reduction depended on the specific surface of the ore.

The particular relative position, in vertical layers, of carbon and the ore on starting the operation in the Farga Catalana furnace was to obtain the maximum possible
amount of carbon monoxide in front of the entrance of the tuyere and then to direct this toward the ore. In this way the ferric oxide was reduced to ferrous oxide, and finally to iron. The reductant was oxidized to carbon dioxide, which escaped into the atmosphere.

Simultaneously with the reduction process, carburizing and slagging reactions took place. At about 910°C, face-centred cubic iron is able to dissolve carbon from the dissociation of carbon monoxide into carbon and carbon dioxide, transforming iron into steel. At the same time impurities in the iron ore, such as silicates and carbonates, were slagged off. When the furnace reached a temperature between 700°C and 800°C the ferrous oxide and the manganese oxide in the ore reacted with silica to form a fairly fluid iron and manganese silicate slag which was separated from the iron, because it was less dense and could easily be poured through the "ileteiro". At temperatures above 900°C calcium carbonate, a gangue very frequent in the iron ores of the Catalan Pyrenees, underwent decomposition into calcium oxide and carbon dioxide. The calcium oxide reacted with the silica to produce a calcium silicate slag, more stable than the iron and manganese silicates. For this reason when calcium silicate was present in the slag, ferrous and manganese oxides remained in part, and in part were reduced so that iron and manganese were incorporated into the "masser" (ingot).

Sulphur is usually an undesirable element in steel because it causes embrittlement. Sulphur present in the "masser" originated in the carbon cinders and was concentrated in the non-metallic inclusions of the microstructure.

Phosphorus is another chemical element which embrittles steel. In the iron products of the Farga Catalana there is little of this element present, although calcium orthophosphate usually is present in the iron ores of the Oriental Pyrenees. At the temperature of the furnace of the Farga Catalana the calcium orthophosphate can decompose into diphosphorus pentoxide which is reduced by carbon monoxide, with phosphorus dissolving in the iron. However, in the slag of this type of furnace, there are sufficient amounts of calcium oxide to displace the reaction to the formation of calcium orthophosphate which goes into the slag. As a result the reduction of the diphosphorus pentoxide and the phosphorization of the iron is minimized.

Slag found in the neighbourhood of the meeting of the rivers Ter and Costabona, the location of the "Farga Setcases" (Gerona), has been found to contain 63% iron oxide and 1.2% calcium fluoride. Such a high percentage of iron in the slag was normal for the Farga Catalana process, and was one of the decisive factors in its decline. However, the calcium fluoride content of the slag improved the quality of the end-products of the Farga Catalana, because it behaved as a flux, reducing considerably the melting point of the slag.

At present the Farga Catalana belongs to the glorious past of Catalonia, and is an important constituent of the historic introductions which usually are present in the first pages of modern steel-making books. This process was a great innovation in its time because of the introduction of the water nozzle to inject air into the furnace, producing a more constant blast than that produced in other processes. The Farga Catalana process was gradually replaced by the blast furnace, so that by 1878 the Fargas had disappeared altogether. However, at present new processes are being developed to produce iron directly, which, of course, is a logical way to proceed. Their product, sponge iron, is very similar to the "masser" from the Farga Catalana.

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Cast iron in late medieval Europe: a re-examination

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Introduction

The purpose of this paper is to review the evidence at hand concerning the earliest centuries of cast iron in Europe, from roughly 1400 A.D. until the second half of the sixteenth century. For the most part it is concerned with the relationship between two technologies which emerged first in China and then later appear in Western lands, gunpowder weapons and iron founding. We should not assume, as I hope to show later in this discussion, that weapons constituted the only product for which the iron founder had any market, but it is clearly true that attempts to produce weapons constituted the most severe technical challenge for the early founder. Firearms appear for the first time in Europe in the 1320s and develop into significant weapons on the field of battle by the end of the century. Theirs is a complex history, subject neither to easy summary treatment nor to simplified models of development. I hope to be able, through the analysis of the evidence concerning cast iron, to show that one important strand in the fabric of gunpowder's history concerns the materials from which cannon could be made. This question is subject to a number of shades of meaning, because "cannon" can be made for many purposes under a profusion of different names, and under sharply varying conditions of use. As a result, the question of materials is frequently a question of competing materials, and involves a complex set of factors, of which production costs and the ultimate safety of the user seem the most important. Judged in this manner, cast iron was an important but unsatisfactory metal for weapons purposes until 1543 at which time English founders learned to make iron muzzle-loading cannon which could rival bronze ordnance in most important performance criteria, but which cost much less to produce. With that event, the history of cast iron and the history of weapons entered into the close relationship that persisted until the advent of rifled artillery made of steel in the middle of the nineteenth century.

History

Our view of the earliest appearance of cast iron in Europe rests on two rather brief documents. The first, published in 1913 as part of Otto Johannsen's systematic survey of early references to cast iron[1], is dated 1400 and records the payment made by the city of Wesel (near the Dutch border) to craftsmen for a "cast iron gun." Nothing further can be derived from this document — neither the size of the weapon, nor its method of manufacture. Another very early reference to cast iron is the subject of some misunderstanding and needs a momentary digression. The archives of Frankfurt am Main contain a statement made by one Mercklin Gast, a gunner, attesting to his abilities. He claims to be able to make gunpowder, to purify saltpeter, to shoot with all types of weapons, and to be able to cast "small hand guns and other guns." Unfortunately, this statement is not securely dated, and can be assigned on palaeographic grounds only to the period around 1400. B. Rathgen, who published the document in 1920[2], sought to place it somewhat earlier, in 1390-91, but his case involves too many assumptions to be accepted at face value. Rathgen further argued that an error in the document indicates that already by this time large weapons could be cast. The last line reads in German,

* Item, er kan klein hant bussen uz jsen g und andere bussen uz jsen gyeszen. *

Statements of this type were generally dictated to notaries, as they have the character of contractual promises. Both the handwriting and the endorsement of this specimen indicate it was recorded from dictation. The scribe had begun to write Gast's original claim, i.e. that he was competent to cast small hand guns from iron; then, at the behest of the speaker, he struck part of the sentence and added the phrase "and other guns." This is a fairly obvious indication of the opposite to what Rathgen sought to make it mean. Gast claims to be able to cast small weapons and other weapons perhaps slightly too large to be held by hand. By about 1400, therefore, we may safely assume from the documentary evidence that Europeans were able to found with iron, but only in a few regions, notably the Rhine valley, and to produce objects of relatively small sizes, up to about 60 cm or so in any dimension.

In examining the subsequent documentary evidence, it is convenient to divide it into two chronological periods, before and after 1450, for it is only since the middle of the 15th century that our record becomes abundant in references to cast iron. For the first half of the century, virtually all references are to cast iron cannon. In 1414 there begins a series of Burgundian references to cast iron cannon, some of them rather small (intended for a projectile weighing 8 pounds), but some of quite respectable size, e.g. destined for a 20 pound ball. This Burgundian series is interrupted after 1417, but then it resumes again in 1431 and lasts until 1445. From that same year we have an account book from Siegen which records the commissioning of 10 breech-loading cannon to be procured from a single producer.

Evidence for the use of iron founding to make objects other than weapons is extremely rare for the period before 1450. Only two references seem to have survived, one to cast iron fireplace andirons in the region around Siegen in 1445[3], the other to cast iron frying pans in Brescia around 1442[4]. Unfortunately, it is impossible to supplement this meagre record with artifactual evidence, for there are no pieces of cast iron work that can be firmly dated to before 1450, despite numerous claims and assertions to that effect[5]. The result is that up to the middle of the fifteenth century, our picture of the emergence of iron casting is rather limited.

In evaluating what little evidence we have, we should keep in mind two general aspects of the ways in which

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documents survive. First, the fifteenth century is not the “Dark Ages.” It is hardly devoid of written records. On the contrary, it was a period which witnessed the growth of the earliest version of modern bureaucratic government, with all that such bureaucracies imply about generating and storing copious records. Second, the documents most likely to be retained (and thus become available to the modern historian’s investigations) are rather systematically biased so as to over-represent weapons and to underrepresent other artifacts. Weapons, after all, tend to be commissioned and paid for by governments, whether municipal, noble, or regal. They also tend to be costly items, and expenditure of considerable sums of public money is yet another activity that generates records, whether of decisions to expend funds, or of the actual disbursement. The result is that weapons, particularly large and expensive weapons, are more likely to leave traces in the documentary record than are purely private artifacts or more humble possessions.

Seen in this light, assertions about the speed with which Europeans adopted iron founding or their motives for doing so both lose some of their force. Indeed, the number of firearms attested for the entire fifty years from 1400-1450 is less than 60 pieces, and that figure includes 30 guns from a single account. The geographic range of the references is highly restricted, chiefly to Burgundy and to the Rhine with its lower tributaries. (One isolated document from 1429 in Pavia mentions a cast iron cannon as war booty; one wonders if it might have been Burgundian in origin?) We need not quibble over the figure of approximately 60; certainly there may have been more cast iron weapons for which all traces have vanished. The appearance of records attesting to a market for civilian objects of cast iron indicates that the new techniques were becoming part of the repertoire of the ordinary iron producer interested in something other than large-scale production for military purposes. We shall return to the relative balance between the military and civilian markets shortly. On balance, it would seem wisest to speculate that the first half of the fifteenth century was a period in which iron founding established its earliest markets for both commercial and military products.

Cast Iron Processes

From the period before 1450, we have no documentary evidence at all concerning the processes employed by the iron founder. To the historian accustomed to medieval and renaissance customs, this comes as no surprise.

Process technologies tend to be as under-represented in the written record as weapons are over-represented. The reasons are obvious enough. Late medieval craftsmen were not prone to write about their arts; many were illiterate; none wished to imperil his livelihood by revealing secrets; most were perfectly competent to remember what rules and procedures their art demanded of them. It is not surprising, therefore, that we have no description of iron casting from the pen of a foundryman. By great good fortune, we do possess two brief descriptions of iron founding from the middle 1400s written by non-specialists. Gunners form a modest exception to the rule regarding the willingness of craftsmen to write about their arts.

One text in particular, the so-called Feuerwerkbuch, containing many recipes of use to gunners, was widely circulated in many manuscripts from the 1420s onward. One manuscript dated 1454 contains recipes not found in other manuscript copies concerning the preparation of iron for casting. The anonymous compiler describes two methods, an indirect process in which low-carbon iron is carburized and melted, and a direct process in which fluid iron is produced from ore. He does not tell us what products may be cast, but he does indicate that the liquid can be run from the furnace directly into moulds, and he recommends the addition of 5% to 6% bismuth to increase fluidity. In one recipe, he refers to the furnace as being similar to that employed by bell founders.

Our second description of process is that of the noted Italian architect Antonio di Piero Averlino, better known as “il Filarete.” In his untitled treatise on architecture, Filarete takes the reader on a journey into the mountains to the iron smelter at Ferriere. He describes in detail how the ore is roasted with lime, ground, sifted and charged into the furnace in layers alternating with charcoal. He gives the first glimpse of the physical layout of a smelter, including the odd water-powered bellows the process required. Most interesting is his reference to the manner in which the furnace was tapped so as to allow the molten iron to flow into a water-filled pit, where it solidified into small lumps. C.S. Smith has interpreted this as a form of granulation and thus as an indication that the furnace’s output was destined for a finery to be converted into wrought iron.

These descriptions tell us a good deal about the level of sophistication that iron-masters had attained by the middle of the fifteenth century, but they do not substantiate a fraction of the speculations built up on such a slim documentary base. In particular, we have no reason to assume that bronze casting played a particularly crucial role, or that the Umschmelz or secondary melting of bloomery iron was the historical predecessor of direct casting at the smelter. Filarete’s description as explicated by Smith makes it clear that in some regions the amount of bloomery-produced wrought iron was being supplemented by the product of the chafery, but we have no warrant to speculate on how widespread the practice had become.

The best known description of the process of casting stems from Vannoccio Biringuccio’s Protechnia first published in 1540. Biringuccio assumes that the only products one would want to cast are cannonballs or exploding shells, and he bases his entire description on this assumption. He acknowledges that some founders put in small amounts of arsenic or realgar in order to make the iron alloy more fluid and more easily cast, but he recommends against this owing to the increased brittleness of the finished shot. Instead, Biringuccio recommends the melting of that “crude, corrupted kind [of iron] that has been sent through the furnace in order to purify it,” perhaps mixed with a little scrap. Thus Biringuccio’s techniques are both variations of the Umschmelz method, and this would seem appropriate in view of the products he has in mind for the foundry. What is striking by contrast is that the only analyses of cast iron artifacts from the fifteenth century of which I have any knowledge indicate that there is no evidence indicating addition of other elements to increase fluidity. Because the artifacts were cannon or cannon cores, it may well be that the early iron founder used two techniques, direct casting for larger objects whose successful completion was more doubtful, and indirect casting for smaller objects wherein the founder had more control over the end product. It would be most interesting to explore this hypothesis through metallurgical examination of the extant artifacts.
Products and Uses

The wider field of documentation available concerning cast-iron products from the period after 1450 displays some interesting features when examined critically. Johannsen’s survey mentioned above contains mention of 151 identifiable objects from the period 1450-1530. Of these 44 are items for use in the hearth or oven. These include cast iron ovens and ovenplates, grills or graters, apparatus to suspend pots, and fire-backs [Kaminplatten], frequently decorated specimens. This is not a controlled sample, and it may not be at all representative, yet the number of hearth and oven-related pieces is impressive, especially in light of our observations above concerning the biases inherent in the sources. It would seem that from the 1440s onward, and perhaps considerably earlier, the lack of plasticity at red heat that is cast iron’s most conspicuous physical feature made it a useful metal for any implement exposed to high levels of heat for prolonged periods of time. We might well speculate that for the ordinary producer of iron goods, the small but steady market made up of humble items such as these represented a far more important aspect of their trade than did the occasional commission for large weapons or war materials.

Goods for war do make up a significant amount of the market for cast iron objects as it is manifest in the documentary record. The single most frequently cited category of object in Johannsen’s survey is cast shot or shell. The former is solid and non-explooding; the latter hollow and filled with gunpowder. Lack of good fuses made expending shells difficult to use, and thus they are less numerous in the inventories than solid shot. Forty-eight different references to these items appear in Johannsen, many of them inventories or statements of account. The quantities contained in these references are staggering; in many instances several thousands are mentioned, and in one imperial document over 100,000 iron shot are claimed to be on hand.

Shot had traditionally been made of stone, with metal (usually lead, though sometimes bronze) playing a secondary role. Indeed, so entrenched had the habit of thinking of projectiles as stones become that many of the inventories record iron shot as “Eisenstein” or “pièces de fer” for want of any other term. Along with shot should be recorded 13 references in Johannsen to Klötzte or “plugs”. These were small shims or wedges used to help hold the cannonball in place until the volume and pressure of gases released by the combustion of gunpowder was sufficient to force the ball out of the barrel. Such objects found very little use in the gunnery practice of later times, for the late sixteenth-century or seventeenth-century gunner could rely on a gap between the shot and the bore of his cannon to remain in the range of approximately 2% of bore diameter. At this range, “windage” was not as significant in reducing the velocity of the projectile as was the case earlier. In the fifteenth century, on the other hand, tolerances must have been much greater, for Klötzte were routinely employed to seat and hold the shot within the piece.

Evidence for the production and use of cast iron ordnance in the period after 1450 does not yield the impress so of a rising curve. Johannsen’s survey contains 13 references to cast iron ordnance for the period ca. 1400-1450, while for the period 1451-1500 there are still only 14; the period from 1501-1530 (when Johannsen arbitrarily terminates his work) still shows only 6 refer-

ences. Admittedly, counting the number of times an activity or object is mentioned in a highly biased body of source materials represents a crude way of practising historical research; yet it is elementary doctrine in any course on research methods that, assuming an activity remains roughly constant over time, we expect to encounter it with increasing frequency in our documents as we approach the present, because the number of documents to survive grows larger. For most western European countries during the period under consideration, the total amount of documentary survivals is increasing at a rapid rate as we move from 1400 toward the mid-sixteenth century. Consequently, when one does not encounter an activity with increasing frequency in the sources, one is puzzled and seeks to explain its relative decline. Particularly since Johannsen’s evidence for production of all other objects made of cast iron, from cannon shot to grave markers, shows an increasing frequency of appearance as we approach the present, then no matter how poor Johannsen’s data may be, we must still explain why the casting of cannon does not also increase over time. In order to do so, we must look briefly at what we know about the history of early firearms.

History of Early Firearms

From the middle of the fourteenth century onward, we have evidence that guns could be made in two different ways. They could be cast of a suitable metal, some alloy of bronze, or they could be made of iron staves welded together along their edges. Forged iron barrels were made on a mandrel by a smith working with great technical skill. It was no mean feat to achieve a series of welds strong enough to resist the pressures of a gunpowder explosion, and in all but the smallest pieces, hoops were generally placed around the forged barrel and allowed to shrink on as they cooled in order to increase the strength of the piece. From the point-of-view of costs, iron was a far cheaper metal than bronze, although the added labour of forging must have added very significantly to the final cost of the finished piece. From the standpoint of operations, evidence from the fifteenth century suggests that bronze weapons were generally preferred. Here we must begin to distinguish between smaller hand-held weapons of the sort that would eventually evolve into the arquebus and the musket, and larger weapons which came to be known by a variety of names, from which we moderns have selected “cannon” to serve as the generic term. Muskets need not concern us here. At the levels of force and ranges of size involved, forging worked well enough; virtually all hand-held weapons have forged iron barrels until the advent of steel in the 19th century. Cannon form a much more complex problem.

Wrought iron cannon worked well enough provided that certain specific conditions were met. As a rule, they were breech loaders, which gave them a certain margin of safety not otherwise available. Breech loading weapons in the fifteenth century were made with removable powder chambers which could be locked into place using iron wedges and a stirrup-shaped strap around the breech. The seal or obturation between the end of the gun and the leading edge of the powder chamber was always problematical, but the relief of stress provided by this leakage must have been beneficial to the barrel’s longevity. For similar reasons, wrought iron cannon fired stone projectiles, because the lesser density of stone meant that the powder
charge could be reduced to levels considered safe without at the same time making inordinate sacrifices in muzzle velocity. Finally, wrought iron weapons were, at least toward the end of the century, considered appropriate only for smaller calibre weapons, swivel guns, carrying lesser charges. By the early years of the sixteenth century, wrought iron pieces seem to be falling into disfavour in the face of increasingly superior performance capabilities on the part of bronze ordnance.

Bronze was, of course, the metal with the longest history of being founded, and by the 1300s Europeans had become adept at making very large roughly cylindrical objects in the form of church bells. Slight changes in casting techniques were all that was required to produce at least primitive cannon in the form of tubes sealed at one end. From the second decade of the fifteenth century onward, bronze cannon became the ever-more strongly-favoured variety. Virtually nothing has been demonstrated conclusively about the reasons for bronze's triumph, but it would seem that a variety of changes in technology and in tactics interacted so as to force change in a particular direction, and as is so often the case, what began as an attempt to obtain an advantage over one's enemy ended as a scramble to maintain rough equality in a shifting technological environment. Corned gunpowder, for example, was produced for the first time in the 1420s; it consisted of granulated rather than pulverized gunpowder. By enhancing the packing qualities and the combustion properties of gunpowder (as well as allowing it to be standardized as to strength), corned powder encouraged gunners to strive for maximum power in their shots. Bronze, the strongest type of cannon, was the beneficiary.

Breech loaders were dropped from favour (except where cramped firing platforms made them a necessity) in favour of the safer and more efficient muzzle-loading weapon with a breech cast as an integral part of the barrel. Only bronze could achieve this shape, given existing foundry techniques. The transformation of shot from stone to iron once more emphasized the ultimate barrel strength of the weapon, to the detriment of all other ordnance but bronze. Tactically, western armies shifted from reliance on a few large pieces firing immense shot at a slow rate to batteries of smaller individual pieces firing smaller (but denser) shot at more rapid rates. These lighter pieces could be more easily manoeuvred, but their dependence on denser shot made them highly inclined toward bronze cannon.

The shapes of cannon were changing as well, or rather, the morphology of cannon was splitting into two distinct families. For the sake of convenience, we can roughly categorize these as the large calibre, or bombard line and the smaller calibre, or veuglaire group. The former were generally stone-throwing guns, pedreros in the Spanish *lingua franca* of the Mediterranean, while the latter included the "long cannon" known as the culverins, serpentina, falcons and a host of other names. The relationship between the length of a cannon and its bore eventually came to dominate development outside Portugal and Turkey, which retained the pedreros longer than any other nations. Long barrels in an artillery piece act to increase the muzzle velocity of the projectile up to a certain maximal value of about 2000 ft/sec. (610 m/sec.) in the case of black powder propellant. This is because the barrel must contain the pressures of the expanding volume of heated gases liberated by combustion even as the projectile moves down the barrel, thus expanding the total volume of the containing vessel from instant to instant. With black powder (and in contrast to modern nitrocellulose propellants) the maximum rate of gas evolution is reached rather quickly and remains essentially constant until combustion ceases; there is a maximum barrel length beyond which the projectile is expanding the volume of the containing vessel faster than the evolution of gases fills it. The resulting pressure drop will actually retard the projectile's ultimate muzzle velocity. In practice, the range of marginal change in performance is rather generous, perhaps fortunately for early gunners: it varies between about :2 calibres of barrel length (i.e. a barrel 12 times its diameter), which results in a loss of only about 12% in terminal velocity, and 21 calibres, which is the absolute upper range under ideal combustion conditions. Above 21, retardation sets in.

If one could show that early gunners were somehow seeking to optimize for muzzle velocity in some crude empirical way, our story would be simple. They also had another concern, safety. The gunner lived a hazardous and somewhat grimy life, not without its aspects of glamour, but always with the danger of accidental explosion. He insisted, as men often do who expose themselves to occupational hazards, that his equipment be made as safe as the technology of his age would permit. One can easily show that the charges employed, the wall thicknesses of cannon, and the handling procedures used (or recommended) were designed to ensure an extra margin of safety. The length of the barrels of many fifteenth and sixteenth century bronze pieces of the veuglaire family can be shown to be very far out on the curve of efficiency, and in some cases beyond the maximum length, consistent with optimal performance. Why would gunners bother to load themselves down with excess barrel length weighing thousands of extra pounds? The answer is that in casting the cannon, longer barrels translated into safer cannon at the breech.

In his brilliant study of Mediterranean warfare in the sixteenth century, J.F. Guillemart, Jr. has shown that the thickness of the breech of bronze cannon, as well as the exact composition of the alloy at that point in the casting, both depended on the fluid pressures generated by casting such pieces in a breech-downward position. The bronze cannon founder took care to make the barrel of his artillery good and long, and he went even further by providing his mould with an exceptionally large casting bell, an enormous sprue containing upward of two tons of metal. The result was a reasonably safe cannon, one which, if it passed proof firing (at a charge double that employed in practice), could be used in the field with safety by any skilled gunner.

Bombard development, by contrast, remained on a technically more elementary plane. Bombards with their relatively large calibres could hardly hope to reach the lower limit of roughly 12, and there was no possibility of ever obtaining an upper figure close to 21. Attempts to make bombards safe for iron cannonballs succeeded only in making them heavy and cumbersome. This was not a crippling disadvantage, but it tended to reduce the number of instances in which a bombard was appropriate to some of the more leisurely instances of siege warfare. Bombards evolve shorter and shorter varieties as the fifteenth century wears on, reaching one terminus of development as a squat mortar firing a rather large ball in a high trajectory over the walls of intervening fortifications. It was as if gunners realized that they could not achieve in this family the goals set by the veuglaire type and instead chose to
optimize safety by having a barrel of minimum length, thereby allowing much of the force of the blast to escape uselessly. Another possibility exists: explosive shells, as mentioned above, do exist from the fifteenth century. We know that in some cases they were placed in very short mortars with the fuse leading to their internal charge placed upward. The gunner approached his piece with two matches, one for the propellant and the other for the shell’s fuse. We do not know how frequently this hair-raising procedure was employed, but it does mark the logical end point of bombard development.

Returning to the evidence for cast iron in light of the above digression, we can see that to all intents and purposes, the new material would have to compete with the best available casting metal of the day and would have to make complex shapes, essentially heavy-walled closed tubes, that could withstand high stresses under adverse conditions of use. It is apparent that this could not be achieved in the fifteenth and early sixteenth centuries. References to cast iron ordnance are interspersed with clues as to their limitations, inventory reports listing burst, fractured, or split pieces, indications of a search for some craftsman from another city who can make good weapons of cast iron, references (especially late in the period) to cast iron mortars or bombardelles.

Extant pieces of cast iron ordnance (which cannot generally be dated with precision, but which nevertheless must stem from the fifteenth century) are all short, squat, thick mortars. Its very shape testifies to its peripheral status in the changing armaments technology of late medieval Europe. The only other definite indication of the morphology of cast iron weapons comes in references which specify separate powder chambers. We are then confronted with cast iron breech loaders, a type which was also peripheral to the main lines of development. To be sure, it must have been a tempting prospect to the early generations of iron founders to believe that they could produce weapons the equal of the best bronze cannon, and it may well be the case that in some localities such as Burgundy, local iron founders did for some inexplicable reason, produce guns which could stand comparison to bronze. If so, then their success was due to factors over which they had no control, for their work was not replicable. Had it been, the history of both cast iron and firearms would have been quite different.

It is, I must confess, frustrative to look over the early history of iron founding and find myself unable to account in detail for the pattern presented by the evidence. In plain fact, however, we do not know with certainty what prevented the earliest founders from achieving successful iron cannon. Authorities such as the compiler of Feuerwerkbuch and Birinuccio treat founding as an art applied only to small simple objects such as shot; we have no discussion at all of casting complex or large objects in iron so that we might gain some deeper insight into why the era saw so few cast iron cannon. Given the fact that successful castings were made of rather large grave monuments and ovens, mere size cannot have been the issue. One clue lies in the relative paucity of references to cast iron pipes; Janssen’s survey includes only three references. This could hardly reflect the absence of a market, for we know that many cities in the fifteenth century had elaborate water systems depending on either rolled lead pipes (which were expensive) or bored out log piping (which was short-lived). Many exaggerated town councils debating the cost of waterworks in the fifteenth century would have welcomed iron pipe had it been available. Was the casting of cylinders a problem that early founders could not completely resolve? The historian must at this point seek the counsel of the metallurgist in order to proceed with any rational investigation on this matter.

The task of making iron cannon which could equal the operating characteristics of bronze weapons fell to the people who have until this point in our story been conspicuous by their absence, the English. Whatever England’s early contributions to firearms, the English were plainly overtaken by superior French technology and tactics in the closing years of the Hundred Years’ War. Retiring from the continent to fight only among themselves for decades, Anglo-Hibernians cut themselves off from the main lines of artillery evolution for some time. Likewise, England shows no record to indicate any cast iron wares were being produced in the British Isles before 1490, when iron founders are mentioned south of Ashdown Forest. Gun shot was cast from at least 1496 onward in response to royal commission, and we know that King Henry VII sought to encourage founding as part of his plans to revitalize England’s artillery, for he encouraged foreign craftsmen to settle in England. One of these men, Paunclelet Symart, cast breech loaders of iron in 1508-1509. Rather typically, these guns exploded on proof firing, a circumstance which resulted in a term in the Tower for Symart and his apprentice. A second attempt, again with breech-loaders, was successful, and the resulting pieces seem to have been roughly equivalent to the smaller swivel guns known elsewhere as versos. Development continued for some time but was once again cut short by the ambitions of King Henry VIII, whose policies of naval expansion led logically to concentration on bronze weapons. Never a man to work by halves, the king spent lavishly on orders for Continental bronze cannon (perhaps he was dissatisfied with the rather feeble cast iron pieces), and he, like his father, sought to induce foreign artillery-founders to settle in England.

When in 1543 a nearly bankrupt king turned to English sources for badly needed ordnance, he was able to assemble a rather unusual complement of technical skills in the persons of Parson William Levett, a group of unnamed French founders (in bronze), the artilleryman Peter Baude, and the cast iron master Ralph Hogge. Working at Buxted south of Ashdown Forest, this team completed the first muzzle-loading weapon made of cast iron on English soil. So impressive was the performance that 120 more were ordered from Levett in 1545, and thereafter still more were poured. In 1547 we learn of the size of these new weapons; they were “sakers” (sacres on the Continent) of about 1400 lbs average weight. This rather limited type was apparently the result of small furnaces, not of casting techniques, for the erection of a double furnace in 1547-48 resulted in successful castings of culverins and demiculverins in 1549. England had become at a stroke the most advanced producer of ordnance in the world.

The effects of this alteration in the methods of producing artillery were quite dramatic. Cast iron guns were, let us admit, still not quite the equal of the best bronze ordnance, but they had one compelling advantage; they cost one-third as much. England tried, of course, to monopolize trade in the new artillery, but to no avail. Ultimately others were able to imitate English methods and increase the flood of cheap guns from the northern countries of Europe that ultimately swept around the world. It is worth recalling
briefly that the sixteenth century was a period of inflation unparalleled until our own day. Rising prices in general forced change in many traditional activities, including even warfare. Bronze ordnance was rising in price particularly rapidly, as was the labour to cut stone balls for pederos or to forge the older style of wrought iron pieces. Cast iron ordnance was, in a malicious way, what made the gunner cheap enough to use in war. Inexpensive and plentiful ordnance at sea underlined the superiority of European galleons over Muslims in the Indian ocean and the Mediterranean. On land, the effects were manifest in the feverish attempts to build fortifications capable of protecting their inhabitants from attack by cannon, efforts which were matched by more and better guns more effectively employed. Bronze ordnance survived — indeed it remained always the most princely form of artillery — but the cast iron cannon dominated from the Sea of Japan to the bloody sieges of the Thirty Years War.

We recognize that the English achievement in artillery manufacture was epoch-making, yet we are still at a loss as to how to account for it in technical terms. We know only that it was a replicable achievement. This implies that it did not depend on any specific local factor which remained hidden from the teams of experts, for when others learned of England’s ways, they too began to produce the new dark cannon. Wertime seeks to account for the English success in terms of local ores (phosphorous-bearing limonite, low in sulphur), and as a recognition of the importance of grey iron’s behaviour under properly controlled conditions of mould size and temperature. Speculation such as this may well be true, but it lacks evidence to substantiate it. Some crude empirical factors also come into play, for example, the habit of making cast iron ordnance with walls approximately two inches thicker for any given calibre than was the case with bronze; still, this cannot answer the entire problem. Given the successful casting of bronze cannon, it is tempting to assume that similar methods of founding iron would, with perhaps slight modifications, have produced similar results. Did the moulds used by iron founders for cannon have the large “bells” common in bronze work? Did they seek to improve the resistance to rupture by increasing the pressure under which the iron cooled, while at the same time slowing the rate of cooling? My suspicion is that if the English success came from close adherence to the model presented by bronze casting, then others would have stumbled onto it long before 1543. Perhaps then the essence of this shift lies in some critical breakaway from the false model presented by bronze founding? But I am now merely spinning out hypotheses. What is needed is a new attack on this problem with the tools available to the metallurgist.

Conclusion

Our review has pointed out the essential problem and also the limits of documentary history. It is quite clear that in Europe the ability to found with iron stems from the time around 1400. Artillery may or may not have played the most important role in its arrival in Europe, but artillery figures prominently in its later history. Cast iron was used predominantly for small objects in the fifteenth century, developing both a civilian and a military market for cast products. Despite repeated efforts to make iron ordnance, it seems to have been a failure; shot, not cannon, represents the military importance of fifteenth-century iron founding. As artillery changed during the course of the fifteenth century, iron became, if anything, even less a rival to bronze as a serious metal for ordnance. With the English breakthrough in iron founding in 1543, cheaper artillery on land and at sea had literally global consequences. We know as little about the technical reasons for England’s success as we know about the failure of earlier efforts, and it is impossible to believe that further research based on documents will ever reveal to us the essential core of this change. Is it naive to hope that the ability we have achieved in the analysis of metals in this century could be applied to gain answers where the documentary methods of conventional historiography leave only questions?

NOTES

(1) Johannsen, “Quellen” (1913; 1915; 1918).
(2) Rathgen (1920); Johannsen (1953), 203-204.
(3) Johannsen, “Quellen”.
(5) For purposes of this article, objects are considered only for purposes of corroborating documentary data; no attention has been given to claims for specific dates for objects unless the date appears as part of the casting.
(6) Johannsen (1910).
(7) Johannsen (1911); Spencer (1963).
(8) Smith (1964).
(9) Birunguccio (1424), 319-321.
(12) Tylecote (1976), 71-75.
(16) Johannsen (1953), 203-205. Some dated specimens also survive from the sixteenth century.
(17) Schubert (1943); Tylecote (1976). 92 misleadingly bases his discussion of casting iron cannon on Birunguccio’s discussion of bronze cannon founding.
(18) Cipolla (1965).

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The smelting industry in the Lower Swansea Valley of South Wales – a brief history

P.J. Mackey

The metal producing industries in Canada and in the U.S. in many ways owe their technical beginnings to Welsh technology. The first copper smelter in Canada started in 1849-50 with a Welsh reverberatory furnace and skilled furnacemen brought across from Swansea to operate the equipment. The Rocky Mountain smelting industry began with the installation of furnaces resembling the Swansea smelters. It was only later that the North American industry developed newer and improved methods. This article briefly describes the metallurgical history of the Lower Swansea Valley, being that section of land stretching northward from Swansea along the banks of the River Tawe, and bordered by hillsides east and west of the river.

The Valley has had a classic cycle of development and decline common to industrial areas of Britain which grew in the Industrial Revolution. Perhaps more than any other area today, the Swansea Valley represents a complete cycle of use from agricultural to the peak of industrial development through to decline and dereliction.

The conditions which made the Tawe Valley attractive to certain industries were laid down nearly 300 million years ago with the Carboniferous swamps and forests which covered the area, later forming the south Wales coalfield. As a result of glacial action and the deposit of glacial outwater and alluvial gravels, the valley adopted the traditional configuration, with its rocky sides occasionally remaining exposed.

Up to probably about the 16th century, the Lower Swansea Valley was similar in use and appearance to many other valleys in the region. From the 14th century, the natural oak woodland had provided a continuous source of timber to produce charcoal for iron making as well as for structural purposes. Agriculture was the main activity in the valley in this period, but it was likely that the farmers also mined small amounts of coal by way of “bell pits”, and adits into the coal seams.

The 17th and 18th centuries saw the origins of industrialization characterized in the case of the Lower Swansea Valley by the growth of small-scale workshops, collieries and mills alongside agriculture and forestry. This development was probably typical of the first stages of the Industrial Revolution. Coal was replacing wood for smelting iron and other ores, and the easily accessible coal was being increasingly exploited from a number of new pits and exported by sea. Such was the scale of coal production in the 18th century that Swansea quickly became the premier port in South Wales for coal export and was not overtaken until as late as 1828 by Newport, some miles east of Cardiff. In 1717, Dr. J. Lane established the Llanyfylach Copper Works, the first in the valley. It is generally considered that the techniques for the first smelters were imported from German operations. The earliest iron forges and tin plate works in Glamorgan was working at Ynys Penllwch, near Clydach, and another forge was working at Upper Forest both using water power.

Tramroads, railways and canals were being built in the valley toward the end of this period and about 1780 a toll road (A48) from Llanamas to Morriston was opened, with a new bridge over the River Tawe at Wyctree.

The Metallurgical Centre of the World

From about 1800 onward, the valley entered the major phase of industrialization with the extensive growth of firms operating large works and employing hundreds of people. Copper and other non-ferrous metals were produced in increasing amounts. By about 1850, the copper industry had developed to the extent that 17 out of 19 copper works in Britain were concentrated in the Lower Swansea Valley. To transport the ore and the coal, canals and railways were constructed that followed closely the course of the river. Tips and slag dumps associated with the industry began to cover a major part of the landscape. The Morriston Spelter Works was one of the earliest to produce zinc on a considerable scale. In the early 19th century the Hafod and Gwyrch Copper Works each employed hundreds of people, operated their own transport on the river canal and railway, and had their own coa. pits. The scale of production of copper, zinc and nickel as well as iron, silver, arsenic and associated chemicals such as sulphuric acid made the valley the non-ferrous metallurgical centre of the world. The extent and scale of these operations were unequalled in the world and they were the largest single concentration of industry in Britain and probably in the world. The trade resulting from this led to the growth of Swansea as a commercial centre and the creation of the North Dock, by diverting the River Tawe. This was followed by the opening of the South Dock, and late in the 19th century a new complex of docks to the east of the river in Fabian Bay.

From the middle of the 19th century many technical advances were associated with the industries in the valley, but these went parallel with a gradual decline of copper smelting. The third quarter of the nineteenth century saw lead, zinc and silver production rise to partially compensate, with some former copper works converted for this purpose.

Up to the 1914-18 war, Swansea was the main centre of zinc smelting in Britain, after which this activity also declined in relative importance while production grew in Canada, the U.S.A. and Australia.

The Mond Nickel Company was built at Clydach in the upper reaches of the Swansea Valley in 1900 for the extraction of nickel from partly refined sulphide matte from Canada. This was the second major metallurgical link with Canada and specifically Sudbury, the initial one being the establishment of the first smelter in the area set up by H.H. Vivian near the site of the Murray mine, just west of Sudbury. This plant was plagued by problems associated with the separation of copper and nickel and eventually closed, causing the loss of large sums of money for the


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Welsh investors. However, to return to the development of the Mond process, Sir William Grove, a notable Welsh scientist, had invented the Grove gas-cell in which hydrogen and oxygen are combined to produce electricity. Mond became interested in this and set about to commercialize Grove’s Cell. The accidental invention of the Mond Nickel carbonyl process is described by J.H. Beynon and D. Betteridge as follows:

“A first step for Mond was to find a cheap source of hydrogen and Mond proposed to start with a mixture of hydrogen and carbon monoxide, obtained by passing steam over hot coke. Obviously, the carbon monoxide had to be removed, and Mond proposed to do this by passing the mixture of gases over nickel, because he had found earlier that nickel valves on the soda plant became corroded as a consequence of reaction with trace amounts of carbon monoxide.

Thus, on one October evening in 1889 in the laboratory at ‘The Poplars’ (his London home), Langer, the chemist in charge of the laboratory, was carrying out the experiment of passing carbon monoxide over heated nickel. The apparatus was a tube surrounded by a furnace and packed with finely divided nickel. The gas emerging at the end was burnt, and what he expected to see was a residue of carbon and nickel oxide and the disappearance of the carbon monoxide.

“On this particular day, Langer sent the laboratory technician home early, and set about closing off the experiment himself. He put out the furnace and as the tube cooled down he noted a change in the colour of the flame from blue to smoky yellow. Being a trained chemist, he sensed the significance of this chance observation and rushed up to the house to bring Mond down to the laboratory. The observation was repeated, and then they heated the empty tube beyond the nickel and noted that the blue flame was restored and a mirror left on the tube. Chemical tests soon showed that the mirror was metallic nickel. They deduced, and soon proved, that the nickel and carbon monoxide reacted to form volatile nickel tetracarbonyl, the reaction being easily reversed. As Lord Kelvin put it, they had ‘given wings to a heavy metal’.

“Further experimentation showed that under easily controlled conditions only nickel was volatilized by carbon monoxide. Because gentle heating released the nickel and carbon monoxide, Mond saw the possibility of a process of separating nickel in a pure state from impure mixtures in a process in which the carbon monoxide could be continuously recycled.

“Within four years of the discovery, and after trial runs at Birmingham, the ‘Mond’ was in operation at Clydach, in the Swansea Valley.”

The largest copper smelter in Swansea was the Hafod Works (Fig. 1). These works were owned and operated by the Vivians, first by John Henry Vivian and later his son, Henry Hussey Vivian. H.H. Vivian was a highly qualified chemist who had been educated in Britain and Germany. As a businessman, he wielded immense personal power. A special train was always on hand in case he required to go to London. Each day, from his office in Landore, he would formally declare the world price of copper. The Vivians built many houses for their workers and built the first laboratory to be located within a metallurgical works. To protect their industrial secrets, the Vivians built a 10-ft stone wall between their factory and the adjacent site owned by Robert Morris. The men and women from the two factories were forbidden to speak to one another on pain of instant dismissal. The secret mode of operating was no doubt a factor in bringing about the eventual decline of the Welsh copper works. As a contemporary observer in the late nineteenth century, James Douglas noted: “At Swansea, every gate to the smelting-works is guarded, and as a result it has been as difficult for ignorance to escape out as for suggestions to find their way in.” Douglas regarded secrecy as a definite barrier to progress.

North of the main Swansea Valley region in the mid-nineteenth century, the first trials of the Bessemer process for steel production were being made at Merthyr Tydfil. No doubt the news of the success of this process and its possible application to copper would have become known to the operators of the copper works, and in fact the major families who owned the enterprises knew each other socially. For what reason we do not know, however, Vivian considered the Bessemer process quite inapplicable to copper smelting, a decision which coupled with other changes was to lead to the eventual decline and abandonment of the copper works in Wales. It is also worth noting
that C.W. Siemens and his brother Frederick developed the regenerative open-hearth steel furnace, which was first operated on a commercial scale at Landore in the Swansea Valley in 1868.

Air pollution problems had risen with copper smelting, but were made considerably worse with zinc processing. From both, large quantities of fumes consisting of sulphur dioxide, smoke and steam were produced, causing sulphuric acid to denude the remaining vegetation in the valley. The Vivians were assisted by Sir Humphrey Davy and Michael Faraday in experiments with various methods of treating the fumes, stimulated by the annual loss of an estimated 200,000 tons of sulphur into the atmosphere. The first sulphuric acid plant treating smelter gas was operated by the Vivians in 1865. Pollution was and still is not so much a question of scientific uncertainty, but largely a matter of money. However, as an environmental problem, fumes continued to be produced until the closure of the last works, the Swansea Vale Zinc Plant, in 1974, where separation and refining of zinc and lead took place in a very advanced process.

After about 1880, there was a gradual decline in the copper smelting business. It was becoming economical and technically feasible to smelt copper using imported coal rather than export the ore to South Wales. With the development of the copper mines in Montana particularly, and in Chile, the Americans began to emerge as the largest producer of copper. Technical developments played their role here, first with the introduction of the Manhès copper converter at the Parrot Smelter in Butte in 1884 and then with improvements in the size and efficiency of reverberatory smelting furnaces. The tinplate industry also went into decline, with tariff restrictions on the importing of tin plate to the United States. The opening of the steel plant at Port Talbot in 1947 and strip mills outside Swansea were made at the cost of closing the remaining tin works in the valley. Copper and brass casting operations on small scale were carried out by Yorkshire Imperial Metals Ltd. at the site of the Hafod Works until 1980, when it was also closed down.

Unfortunately, for the communities around the valley, the decline and abandonment of the chemical and metallurgical industries were not compensated by the growth of new industries. The Swansea municipal authorities, in conjunction with other bodies and guided by members of University College, launched the Lower Swansea Valley Project aimed at the redevelopment and reclamation of the devastated areas, slag dumps and dereliction. The former metallurgical centre of the world probably represents now one of the largest environmental pollution and planning challenges in Britain if not in the world. As we advance into the twenty-first century, are we not to avoid more such metallurgical wastelands — could a Sudbury, Ontario, a Butte or Anaconda, Montana, or other great metallurgical center today ever decline to similar conditions? The challenge and destiny remain ours.

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Acknowledgements
Much of the information here has been taken directly from the Lower Swansea Valley Facts Sheet (Ref. 6), and this is gratefully acknowledged. This publication and the additional references will provide the reader with far more information on the valley. My appreciation of the former metallurgical centre of the world was assisted by Professor D.W. Hopkins, whose kindness is sincerely acknowledged.
The Tredegar iron works of Richmond, Virginia

R.E. Johnson

Introduction

Seventy years after the start of the Industrial Revolution in Great Britain, as reckoned from Arkwright’s water frame patented in 1769, there was a significant development of the metallurgical industry in Virginia. In Britain, crucible steel had been rediscovered by Benjamin Huntsman in 1740, and this made spring and tool steel more generally available. The metal-turning, screw-cutting lathe, including the basic principles recognized in today's engine lathe, was developed by Henry Maudslay about 1797. Wrought iron, as manufactured by Henry Cort's process patented in 1784, was readily available. Richard Trevithick's high-pressure steam locomotive had met its test on the Pen-y-darren tramroad in 1804, and George Stephenson had built his famous Rocket, along with several other locomotives, at Newcastle around 1828. Puddling furnaces and a rolling mill had been placed in operation at the Tredegar Iron Wcrks in Glyn Sirhowy, Monmouthshire, Wales, in 1807.

In the United States, expanding commerce and a growing population resulted in continuing development from the Chesapeake to Virginia's Blue Ridge, and this demanded improved means of transportation. Ore deposits rich in hematite had been found in Virginia's Piedmont during the Colonial period, and a number of blast furnaces now dotted that area. The products of these operations were mostly pigs, kettles, stove parts and like items for local trade. The James River and Kanawha Canal was incorporated in January 1785. Its eastern leg, winding along the slope of the high bluffs overlooking the river, was completed about 1800. About two dozen railroads were chartered in Virginia between 1830 and 1837.

With expanding population and steady industrial development came a need for metallurgical products. Throughout the eastern United States, numerous metallurgical enterprises sprang up, and many of these flourished. The purpose of this article is to outline the development of one Southern metallurgical company, the Tredegar Iron Works, which was to have a significant impact on the industrial development of Virginia and the fighting of the American Civil War. Although the Tredegar Iron Works no longer exists, remnants of the early operations have been restored, and the current status of the site is also presented.

The Founding of the Tredegar Iron Company

The Tredegar Iron Company was chartered by act of the Virginia State Legislature on February 27, 1837; its principals were Francis Deane, the brothers John and Edward Cunningham, from the local cotton business, and Isaac Davenport, associated with the Franklin Paper Mill. Deane, an experienced blast furnace operator engaged in the iron business, was the only member with iron-associated experience.

The site selected for the new mill was a narrow strip of land between the James River and Kanawha Canal and the river itself, about half a mile west of the city and just below the dam bounding the southern edge of Harvie's Pond. This impoundment, contained in a ravine formed between two bluffs, had been incorporated into the canal waterway as a minor turning basin. At this site, the canal water level is a bit over 40 ft above the river. The location provided hydro-power for driving the mill rolls. Built as a transportation means, the canal reached into Virginia's western counties and was used to transport pig iron from the blast furnaces in the Piedmont to the Richmond mill site. It also served as a mill race for several industrial users. Another essential, fuel for the puddling and reheat furnaces, also favoured the Richmond site. Coal had been mined at Midlothian in Chesterfield County, just 14 miles southwest of the city, since colonial times. The new company, it was thought, would serve the fledgling railroad materials and equipment supply trade.

The reduction of a red-heated coarse billet into an elongated shaped bar of smaller and uniform section, through the nips of a pair of driven rolls, was then a rather new process, and for the company it was a completely new adventure. Little is presently known of the communications exchanged between the Virginia company and the Tredegar Works in Wales. It is known, however, that help was received from the Welsh firm in the person of Mr. Rees Davis. In appreciation and in honour of the great Welsh works, the name Tredegar was incorporated into and made part of the Richmond company's name.

Another company, the Virginia Foundry Company, was chartered the same day in February 1837; its location was between the rolling mill and the river. Neither company wasted time getting into production. The foundry actually began casting in December 1836 before its formal incorporation. By early 1838 the Tredegar Iron Company had absorbed its neighbour, forming a stronger, more versatile production facility. Deane was elected president, and continued in this position until 1842.

Principal among the Tredegar Company's earliest products were rails, ship spikes and rail chairs (this latter term descended from the period when short cast iron rail sections were joined and rested upon cast iron plates or "chairs" mounted upon stone blocks). By this time, however, rail design had evolved so that spans were longer, and consisted of either wrought iron "I" sections mounted upon stone blocks or strap iron on wood sills. The rail chairs had evolved more closely toward what we now recognize as tie plates. At this time, running northward from the city, the Richmond, Fredericksburg and Potomac Railroad was chartered in 1834, and southward the Richmond and Petersburg laid its first track from Richmond to Weldon, North Carolina, in 1836. Shipbuilding on the Chesapeake, the Baltimore clippers, may have accounted for some of the spikes trade.

The new company was beset with financial troubles nearly from its beginning. The panic of 1837 and the following depression caused prices to tumble and caught the company with a large stock of finished products on hand.


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The railroad boom ended. The principal market was lost. Moreover, the railroad business continued to languish. The company's debt began to grow.

In March 1840, a young engineering graduate of the U.S. Military Academy, Joseph Reid Anderson, was hired as the company's commercial agent. By the terms of his contract, Anderson was to receive 5% on sales and 2 1/2% of the company's debt if he settled this successfully.

The new commercial agent quickly exhibited energy and ability. Soon contracts with the navy for chain cable, shot and shell had successfully been negotiated. Impressed by this initial success, the Tredegar owners accepted their agent's recommendation and authorized the erection of cranes, lathes and other equipment necessary for the casting and machining of heavy ordnance. Again Anderson went to Washington, and in 1842, he came back with an order for one hundred naval guns. Problems were encountered with this gun order. The Navy Department, after accepting sixty of the guns, rejected the remaining forty when five of the pieces burst upon test firing. Experimentation was tried. Test castings were made using various combinations of different coals and pig iron from different blast furnaces.

The bulk of the coal consumed by the smithys, the puddling furnaces, and for melting gun iron came from the Chesterfield mines. The Chesterfield coal was of variable quality though, and the company was now importing Lehigh Valley coal, shipped by water from Philadelphia, for certain special purposes such as spike furnaces and some foundry use. Pig iron was being produced at a number of charcoal blast furnaces, all located in Virginia's western counties, but principally in Botetourt. The ideal gun iron seemed to be produced when Black Heath coal (from Chesterfield County) was used in combination with iron from either the Cloverdale or Grace furnaces, both located in Botetourt County.

Efforts were made to secure other government orders, and other avenues were tried as well. Great effort was made to secure the contract to build one of four iron steamers that were to be furnished. A rift had occurred, however, between the commercial agent and the owners over the matter of the burst guns. Anderson acted by leasing the entire plant for a period of five years at the rate of $8,000 per annum.

In December 1844, Army Ordnance ordered 60 heavy guns and the Navy followed with an order for 112 pieces the next year. Giving close attention to his casting methods and using only Cloverdale and Grace iron, Anderson re-established and maintained a high reputation for his cannon. Between 1844 and 1860, the company cast and delivered a total of 881 ordnance pieces to the American Federal Government. A contract for the revenue cutter Polk, an iron steamer, was awarded to the Tredegar in 1844. Prior to the Civil War, the company also constructed engines and boilers for two Norfolk-built navy frigates. A few months before his five-year lease expired, Anderson moved to obtain permanent control of the Tredegar works. On April 4, 1848, he purchased the entire plant from the stockholders for $125,000, the payments being staggered over six years.

Over the next decade Joseph Anderson, through a succession of partnerships, satisfied all financial obligations. In 1859 he gained control of the Armory Rolling Mill, the neighboring works to the east. The Armory Rolling Mill, built as a rail mill in the mid-1840s, had converted to merchant bar iron production since the availability of cheaper British and Northern-produced rails made manufacture of rails at the Richmond works unprofitable.

Business remained brisk over the period. Revival of railroad building in Virginia restored an important market, and Anderson developed new customers for his rolling mill products in New England, New York and Pennsylvania. Business was cultivated south of Virginia as well: bar iron, spikes and rail fastenings for southern railroads, and steam engines and mills for Louisiana sugar planters. During this period, the Tredegar also cast a total of sixty-four ordnance pieces for the State of South Carolina. The various Tredegar facilities were merged under the heading Joseph R. Anderson and Company. (From this period ordnance pieces produced at the works were stamped "J.R.A. & Co.".)

The Civil War Years

The war years, 1860 to 1865, provided the most severe test of the proprietor, his partners in management, and his company. Joseph R. Anderson and Co. came close to bankruptcy in 1860-1861 when railroads in the deep south defaulted in payment of substantial sums owed, and orders were cancelled. Increasing demand for ordnance from several southern states and from the newly formed Confederate Government, however, encouraged construction of a new and larger gun foundry. Then on May 15, 1863, a disastrous fire occurred. The locomotive and engine shops, where some forty American-type steam locomotives had been built, were destroyed along with an old blacksmith shop, the pattern shop, and a new gun boring mill; large sections of the machine shops and two old foundries were badly damaged. Despite the fire, the new gun foundry was completed in 1863; Figure 1 depicts the interior of this building at its likely existed during the Civil War. This painting is an artist's conception based on the existing building, available research and the operation of existing foundries. There were recurrent labour and wage problems, and the supply of essential materials became increasingly difficult as the war progressed. Maintenance of an adequate work force became all but impossible because of the Confederate State's conscription law and because of desertions, particularly those of the more skilled northern-born mechanics. In 1864, a large part of the work force, "The Tredegar Battalion," was pressed into the
trenches for the defence of Richmond, the capital.

In 1864, the proportion of slave workers increased, food supplies available became extremely scarce and inflation of the Confederacy’s currency ran rampant. The Tredegar sent agents far into Kentucky and the rest of the south to obtain corn, rice, cotton, bacon, cattle, horses and mules. Tredegar bartered with farmers, trading nails, spikes and bar iron for food supplies. Railroads were assured repair materials and rolling stock in return for transport of the supplies to Richmond. Within the works a tannery was set up together with a plant for making neat’s-foot oil. It was found that Chesterfield coal could be satisfactorily coked to replace the severely supply of Pennsylvanian anthracite. Such adversity notwithstanding, during this same period J.R. Anderson and Co. cast 626 field guns (iron and bronze) and 473 siege and artillery pieces for the Confederate States.

By far the most successful ordnance innovation was the banded and rifled Brooke gun developed in the early part of the war. Armor plating for the C.S.S. Virginia, of Monitor and Merrimack fame in the first naval engagement between ironclad ships, was rolled at the Armory Rolling Mill, the mill roll stand here being larger than the stand in Tredegar’s older mill. Armor plating was also provided for the C.S.S. Mississippi being built near New Orleans, and the C.S.S. Arkansas under construction in Memphis. The company also furnished machinery and sheathing for other vessels.

The world’s first railroad artillery battery, from plans drawn by John Mercer Brooke, was assembled by Tredegar carpenters and mechanics. A light-weight mountain howitzer which could be transported on the backs of pack animals was developed and twenty light-weight, rapid fire breechloading guns, designed by D.R. Williams, were built. Not until late in 1864 was an attempt made to cast a cored gun tube. Anderson had resisted the Rodman plan, first tried at the Fort Pitt Foundry (now Pittsburgh) in 1859, where larger guns were cast, breech down, with a water cooled core made slightly smaller than the intended finished bore. By this method, stronger and larger cannon could be cast because of the stress relief gained through cooling the tube casting from the inside. Also, the walls of the bore were harder and more wear-resistant, but more difficult to machine. On November 14, 1864, the founders poured 41,000 lb of molten iron into a specially designed 12-inch gun flask with water circulating through the core. The casting proved successful, and a second 12-inch Rodman gun was cast in February 1865. Tooling needed to finish the weapons was not ready before the end of the war, however, and neither gun saw service.

The Tredegar Works was spared from Richmond’s great fire which raged as Confederate forces evacuated the city. Some windows were shattered when the arsenal at the foot of Seventh Street, about 720 ft distant, exploded. The foundry’s guarding force held fast, turning away a vandalizing, looting, arsonist mob; the Tredegar survived essentially unscathed. Union troops immediately occupied the works and remained on the property for some time. Figure 2 shows the works as they appeared shortly after the fall of Richmond.

The Reconstruction Years

By the U.S. Confiscation Act of 1862, Anderson’s property was liable to Federal seizure. Only after pleas to Washington and probably more importantly the support of General Henry Halleck, Richmond’s Union military commander, was Anderson pardoned and his property retained. Halleck was aware of the tremendous need for rebuilding throughout the South. Its railroads were in shambles, and its tools and equipment had been worn out or destroyed. Anderson’s supporters argued that the Tredegar Works was needed to rebuild the South’s industry and that a man of Anderson’s energy and ability was necessary to head this industrial works. Anderson’s pardon, and the restoration of his property, was granted on September 21, 1865.

Though his properties remained intact, the proprietor was in debt and without funds. He began by liquidating his Henrico County mining properties, and there were other approaches to refinancing and reorganization. These were successful, and the new Tredegar company was capitalized at $1,000,000, with J.R. Anderson being elected president. Large numbers of skilled former slaves continued to work at Tredegar, and this greatly eased the resumption of in-plant operations.

The immediate post-war business was primarily with Southern railroad and bridge builders: spikes, fish plate, round and square bar stock from the rolling mill, and car-
wheels from the foundry. In this early period, large numbers of vats and other vessels also were cast for the chemical industry; these were cast in the 1861 gun foundry, and Figure 3 illustrates the interior of the foundry at this time. The company used a ladle car to transport molten iron from a large cupola located at the north end of the carwheel foundry, which was in fact, Tredegar’s 1837 foundry, now expanded.

Business improved and remained brisk until 1872. An order totaling 400 tons of rails, along with spikes, tie plates and fish plate, for the Richmond street railway system was completed in 1867. Wrought iron continued to be used for rails following the Civil War (Tredegar still had its puddling furnaces) but this practice was soon abandoned as cheap, hard, durable Bessemer steel became abundantly available. The company received a major order to supply bridge work and rails for Chesapeake and Ohio’s entire line. In 1871, it contracted to supply ironwork for 1670 box cars for the N.Y. and O.M. railroads. There were other contracts with the New Jersey Midland and the Montclair railroads as well. Unfortunately, all these railroads went into bankruptcy in the depression of 1873; following this, Tredegar went into receivership in 1876.

Anderson acted as receiver until 1879 when successful financing of the company’s debt was accomplished. The company slowly regained economic health and respectable profitability. From 1843 until his death in 1892, through times of peace and a severe war, through the range of business extremes and through a time of intensive industrial and technological advancement, Joseph Reid Anderson remained President, Chief Executive Officer and Manager of the Tredegar Iron Works.

By the turn of the century, the chimney which had served the 1861 gun foundry had been taken down. Accumulations of burned foundry sand, slag and like wastes had changed the land contours within and about the works. On the lowest part of the property, just above the high water level of the river in the area of the 1861 gun foundry, the ground level had been raised about 4½ ft.

Retaining walls, in segments, had been erected, thereby creating a terrace level at about mid-height between the foundries floors and that of the puddling furnaces located at canal level. The furnaces themselves, however, were gone.

Early in the century, Tredegar developed a good business manufacturing a range of sizes of horse and mule shoes, and there remained a continuing market for rail spikes and fish plate. The rolling mill provided the first step in each of these products, and Figure 4 shows the “new” bar mill completed in 1868. The age of wrought iron was now past, and steel billets would have been used. The sources of such stock are not presently known. In the carwheel foundry (the now expanded 1837 foundry building), there were three 40-ton Whiting cupolas. In the 1940s, Tredegar had become one of only two principal manufacturers of chilled iron railroad carwheels in the U.S., and Figure 5 shows the pouring of one such wheel. The 1861 gun foundry building, no longer being used as a foundry, served as a carwheel cleaning shed. Throughout the period following the Civil War the Tredegar Company, though no longer casting guns, continued producing artillery shells for the U.S. Army and Navy. A new shell foundry was built in 1917, and Figure 6 illustrates many shells of one type which were being made.

A heavy blow was struck when legislation was passed requiring that cast railroad carwheels be steel rather than cast iron. A try was made at investment casting and at rolling concrete reinforcing rod, but these efforts were unsuccessful. By now, the plant found itself in the heart of downtown Richmond and faced all the problems attendant to inner city operations. An opportunity for economic salvage appeared in 1957. The neighbouring Albemarle Paper Mfg. Co., its Richmond Division also locked into the central city environment, saw an opportunity for expansion of its physical plant. In a merger/purchase agreement, Albemarle Paper acquired the historic Tredegar property, the real estate and standing buildings. The Tredegar Company retained all machinery and metal not part of the building structures.

Albemarle made some limited use of the property. The bar mill shed was used for pulp storage, and the carwheel foundry for the storage of paper rolls. Perhaps the most significant investment was an addition to the office, the composite structure being used as a papermaking research laboratory. In an exceptional business masterstroke completed in the fall of 1962, Albemarle Paper purchased the measurably larger Ethyl Corporation, including the name.
Restoration of the Tredegar Iron Works

The roof of the old 1861 gun foundry failed in January 1972 when the bottom chord of a middle position truss parted. A crane was used in an attempt to relieve the stress on the building’s side walls, by placing a lifting strain upon the rafter chords at the roof peak. Untied across the bottom, the upper chords were acting like opening scissors. There was a slight jolt, then the whole of the old original roof collapsed in a cloud of dust into the building shell. A bow also had been pushed into the building’s south wall. Finally, in June, when the most severe flood on record caused the James River to exceed its banks to the level of the foundry floor, the City of Richmond Building Inspector visited the flood area and ordered the building razed.

Tredegar’s 1861 gun foundry is the most prominent of the buildings seen in Matthew Brady’s 1865 photo taken of the works after Richmond’s fall to the Union Forces. There had been insignificant modification of the structure through the years. The 26-inch thick walls were sound, with the one exception, and the building was of reasonable size, well proportioned and with its corbel and dental work in place. There was a genuine and general interest in Tredegar’s long and extraordinary history, and it could be argued that this quaint industrial relic (with a spot of green around it) would offer a welcome relief to the surrounding box-like high rise buildings of Richmond’s downtown business section. The Bureau of Building Inspection was cooperative, and Ethyl Corporation’s Board of Directors, in December 1972, approved the restoration of the 1861 gun foundry. The project was carried out in several steps over a period of about seven years.

When finished, the shell of the foundry had been restored with its monitor and dormers. The 86-ft high round brick chimney and the symmetrically arranged fire furnaces that tied into the chimney base were replicated. New timber trusses had to be built, but castings salvaged from the old trusses were used on the new. The roof slate came from the same Buckingham County quarry as that furnished for the building in 1861. The large opening in the Building’s east end where Tredegar’s steam switcher used to go through was closed. When approaching from the east, one sees the foundry building as Brady and Gardener did when they took their photographs of the works in 1865 (Fig. 2) and this point is clearly illustrated in Figure 7 which shows the restored structure as it exists today.

By 1978 most of the buildings around the property, exclusive of the restored foundry and the new shell foundry, had been removed. The area around the 1861 gun foundry, now known as the Tredegar Historic Area, was beginning to take on a park-like appearance. Early in 1986, the new shell foundry which was built on the street line was demolished. In the following year, the Tredegar Company in Chesterfield shut down for good after 150 years of business continuity.

On the Tredegar historic site in downtown Richmond, the old Pattern Storage building had been cleaned up and the structure repaired. In late 1987, planning and preparation were under way for upgrading the 1861 gun foundry into an attractive structure suitable for semi-public functions. With its 58 ft by 120 ft interior plan and 27 ft clear height to the “A” roof truss bottoms, it was determined that this room might be tastefully transformed into a place of assembly for meetings of relatively large size. To accommodate such usage, a stylized replication of the sand shed that once spanned the space between the gun and carwheel foundries was erected to provide toilet facilities and a small office. Heating and fire protection systems have been installed inside the foundry. Air conditioning is being installed together with an appropriate lighting system. In all of these improvements, where modern appliances must be incorporated, effort has been made to maintain the building’s historic appearance and integrity. The Tredegar Iron Works is now a Virginia Historic Landmark, and is listed in the National Registry of Historic Places.

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Manganese and its importance to nineteenth-century metallurgy

W.M. Williams

Although manganese is a metal virtually unknown to the layman, it is hardly necessary to point out to the engineer the importance of manganese in modern metallurgy, an importance intimately linked with iron (and steel), the basic materials of engineering construction for hundreds of years and certain to remain so for long into the future. World steel production will approach and perhaps reach 1 billion tons/year in the 1980s; with a manganese content between 0.5% and 1% in ordinary steels, and more in specialty steels, manganese use worldwide will soon approach 15 million tons annually.

Manganese was discovered and isolated in Sweden in 1774 by Karl Wilhelm Scheele (who also discovered molybdenum), but the element was only of scientific interest for several decades. By the early part of the nineteenth century, however, manganese oxide had become commercially available and was in use for colouring glass and for the production of chlorine. Commercial application in metallurgy appears to have been limited to the manufacture of crucible (or “cast”) steel, a high-quality product made by carburizing wrought iron, and melting it in small batches to make ingots which could be rolled and used for special applications, e.g. for the manufacture of fine tools or for “steeling” the cutting edges of implements such as wrought iron axes and chisels.* However, it was not until Bessemer’s epoch-making invention of cheap steel in 1856, and the dramatic expansion of the steel industry thereafter, that manganese assumed the importance that it holds in modern steelmaking.

The Bessemer Process

The year 1856 is a good reference point for a discussion of manganese, as well as ferromanganese, an alloy which was developed about twenty years later. The Industrial Revolution, which was basically a revolution in motive power (from waterwheel to steam engine) and human efficiency (though the use of machines instead of manual labour), was well advanced by 1856; large factories had been built and country people had been flocking to the rapidly expanding cities since the middle of the previous century, particularly on the European continent and to some extent in the United States, but above all in Great Britain, where the Industrial Revolution had begun and which at that time was approaching the zenith of its industrial and political power. Railroads were being constructed — with wrought iron rails — at great speed, iron bridges were replacing wooden ones and steam-powered iron ships were beginning to replace wooden sailing

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*In North America the process of forge welding a steel edge onto an iron part was also known as “jumping”.

vessels. Every new scientific and engineering marvel—gas lighting, plastics, photography, the common match, high-speed trains*—was cheerfully welcomed, and proudly displayed and demonstrated at international expositions in the main European capitals. It was in the middle of these hectic developments that Henry Bessemer (Fig. 1), a successful inventor and hard-headed businessman, decided to try blowing cold air through molten pig iron in an attempt to produce a soft, malleable, low-carbon iron.

Bessemer’s immediate aim was to develop an improved material for large guns, which were at that time made of bronze or cast iron. These materials were unreliable, even dangerous, and Bessemer saw an opportunity to make a lot of money, particularly from contracts with the French military, who were actively seeking an improved alloy for ordnance. After a visit to France in December 1854, to observe some gunnery tests, Bessemer returned to London to begin his experiments with cast iron. According to his own account, it was while supervising the melting of iron pigs in one of his experimental runs that he realized the possibility of decarburizing pig iron with an air blast.

He pressed on with his experiments, blowing cold air into molten pig iron contained in small crucibles (Fig. 2). Success came quickly; within a few months he had progressed from crucibles to large-scale furnaces (Figs. 3 and 4), the designs being based originally on a glass-melting furnace with which he was familiar. The result of these experiments was the famous Bessemer converter, mounted on trunnions for ease of blast control and for convenient charging and pouring (Fig. 5). Decarburization to malleable iron—or mild steel as it is called today—took place in less than 30 minutes, resulting in a material which had all the properties of the best wrought iron (Fig. 6), but which was many times cheaper.

However, Bessemer, whose knowledge of metallurgy was probably no greater than that of the average engineer of the mid-nineteenth century, had been singularly lucky. Purely by chance the pig iron he had used for his experiments was of a particularly high-grade iron, almost free of phosphorus and sulphur, and the quality of his experimental steels reflected the absence of these undesirable elements. It was not long before his luck ran out.

Soon after the dramatic announcement of the process*, licences were taken up (at £50,000 each) by iron masters all over England. However, within a few weeks there were angry complaints of steel of unacceptable quality, brittle and unworkable cold or hot. It must have been a desperate blow to Bessemer’s pride, but he was a rich man, with a store of resilience and enviable perseverance. He was convinced that, given time, he would solve the severe chemical problems which the attempted large-scale application of his process had brought to light.

There were two main difficulties: firstly, the process as described by Bessemer could not be applied to ores containing phosphorus; secondly, it was difficult to cast large, sound ingots, free of blow-holes. In short, a low-

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* In 1848, the London-to-Exeter express train regularly maintained an average speed of about 60 m.p.h., a figure scarcely exceeded a hundred years later even on the fastest British trains.

* At a British Association meeting in Cheltenham, England, in August 1856. There was no ‘publication’ as we recognize the term today. However, a very complete report appeared in the columns of The Times (14 August, 1856).
carbon steel, having high ductility and malleability, could not be produced from the common grades of pig iron. The phosphorus problem had to wait another 25 years before a young man, Sidney Gilchrist Thomas, a police court clerk with an interest in chemistry, thought of using basic refractories, so that until 1880 the Bessemer process could only treat pig iron made from non-phosphoric ores. The problem of incomplete deoxidation, however, was quickly solved with the help of manganese.

The Contribution of Robert Mushet
The idea of using manganese to deoxidize a steel and thus to ensure sound ingots belongs to Robert Mushet (Fig. 7), when a sample of brittle and burnt Bessemer steel fell into his hands soon after the disastrous introduction of the process into industry, Mushet tried remelting the steel and adding spiegeleisen in an attempt to produce a better quality ingot. In his own words:

"I had part of (a defective Bessemer ingot) cut into small pieces and I placed 16 oz in a small clay crucible, and placed the crucible with a lid upon it in a small assay furnace capable of fusing wrought iron. Into another smaller crucible I put 1 oz of pure Siegen spiegel, and placed the crucible in the flue of the melting furnace, born in the Forest of Dean, England, in 1811. His father, David Mushet, was a well-known metallurgist and an expert in the iron and steel technology of the period. Robert Mushet grew up helping his father in his successful researches into iron and steel and; by the time the Bessemer process was announced, was a skilled practising metallurgist 45 years old and, by all accounts, a gentle, trusting and somewhat unbusinesslike scientist. He was familiar with the value of manganese in steelmaking from a great many experiments with spiegeleisen, an imported pig iron containing about 8.5% Mn and 5% C. Thus,

"When the contents of the crucibles were melted, I withdrew quickly both crucibles from the furnace, and poured the melted spiegel into the melted Bessemer metal, and then emptied the mixture into a small ingot-mould. The ingot was smooth and piped and had all the appearance of good cast steel. I heated this ingot to a fair cast steel heat. Mrs. Mushet held the ingot in a pair of tongs, and with a sledge hammer I drew one half of it into a flat bar. This bar I heated, and twisted in a vice at a white
heat, a red heat and a low red heat. It remained perfectly sound, and clear in the edges, and not a trace of red shortness remained. I next doubled the bar, welded it and drew it into a chisel, which I hardened, tempered and tested severely, on hard cast iron. It stood well, and was in fact cast steel, worth 42 shillings per hundredweight.

"I saw then," says Mushet, "that the Bessemer process was perfected and that with fair play, untold wealth could reward Mr. Bessemer and myself."

On September 16, 1856, Mushet took out three patents for improving the quality of iron and on September 22, he took out two more. Twenty-first-century metallurgists will marvel at the speed of developments; less than six weeks after the publication of his results, the confident forecasts made by Bessemer about his process had been found wanting, the process was being ridiculed across the length and breadth of the country, and a solution to one of his major problems had been successfully tried and a patent sought.

Slowly, but much more cautiously now, the Bessemer (or perhaps more correctly the Bessemer-Mushet) process began to be accepted, although not before Bessemer himself had become a steelmaker, establishing a steelworks in Sheffield and forcing the iron masters to recognize the qualities of this cheap new material. Eventually, his process (and the open-hearth process which was developed some years later) would eliminate wrought iron, a material which had been man's most important metallic material for thousands of years*, in favour of low-carbon steel.

Bessemer always refused to recognize Mushet's contribution to his eventual success, maintaining that the value of manganese in steelmaking had been well known for several decades; in particular, Josiah Heath (an official with the East India Company and talented amateur metallurgist) had been selling packets of powdered manganese oxide and carbon to crucible steelmakers in the 1830s. This was true, but it was Robert Mushet who recognized one of the major difficulties of Bessemer's process as originally invented and it was Mushet who pointed out the cure.

Mushet was, however, unfortunate in his choice of patent agents and he was a poor businessman. None of his patents contain any direct reference specifically to the Bessemer process and by an unfortunate oversight he omitted to pay the stamp duty on the patents, so that all his rights were extinguished. Bessemer and his licensees were then free to use spiegeleisen without paying royalties of any kind. It is worth recording, however, that in response to a desperate personal appeal by Mushet's daughter in December 1866, Bessemer arranged for an annuity of £300 (a generous sum by the standards of the time), which continued until Mushet's death. Bessemer clearly felt some indebtedness to Mushet.

**Mushet's Air-Hardening Steel**

To Robert Mushet also belongs the credit for inventing the first commercial alloy steel. Although the material depended only in a minor way for its special properties on manganese (its basic composition contained 8% W, 1.2% Mn and 1.5% C), the discovery was an important one and

*Although the production of wrought iron was declining rapidly by the early 1900s, its use for special applications lingered on, particularly for chain and pipe, until about 1960.
certainly worthy of passing mention in any historical survey of the use of manganese in steelmaking.

A number of people, including Michael Faraday, had experimented with alloys of iron, but it was Mushet who first brought to market, in 1868, a successful and useful tool steel, the forerunner of what was to become an important class of steel in the twentieth century — steels which are today responsible for millions of critical manufacturing operations.

Mushet's researches were originally directed at improved cutting edges for tools, the work being carried out for a Scottish manufacturer who was dissatisfied with the plain carbon tool steels, which were all that were available at that time. During his experiments, however, Mushet found that by alloying with manganese and tungsten he could produce a tool steel that was air-hardening and capable of retaining its strength and hardness at higher temperatures than plain carbon steel. This steel, the first of the high-speed steels, increased the allowable cutting speeds by more than 50%, a substantial improvement at a time when the Industrial Revolution was spreading from Europe to other parts of the world, and notably to North America. It was, in fact, an American metallurgist, Maunsell White, who found that by suitable heat treatment the cutting power of the Mushet alloy could be greatly increased and it was White who introduced the dramatically improved high-tungsten high-speed steels in the early years of this century, steels which retained their strength and hardness even when used at red heat. (These steels, which could be used at four or five times the cutting speed of Mushet steel, caused a sensation when they were introduced and rapidly superseded Mushet's alloy.)

Robert Mushet, perhaps bruised by his lack of recognition in the development of the Bessemer process, did not patent his tungsten-manganese steel or his method of producing it. Instead, he attempted to go into tool steel production himself, but his business soon failed, apparently because of poor financing and quarrels among the directors. The company was liquidated in 1871 and

arrangements were made with a Sheffield steel company to manufacture and market his steel (Fig. 8). A royalty was paid on every ton of steel sold so that between this income and the annuity allowed him by Bessemer, Robert Mushet was able to live out his days in reasonable comfort. He died in 1891 at the age of eighty.

Hadfield and the Invention of Manganese Steel

Although many scientists had experimented with iron alloys from about the beginning of the nineteenth century, no commercial applications had resulted until Mushet's success with his air-hardening tool steel. However, the first really major invention in the field of alloy steels was made by Robert Hadfield about 100 years ago. Hadfield's steel (and it is often so referred to today) is a high-manganese steel containing 1% to 1.5% carbon and 11% to 14% manganese. The steel is austenitic and non-magnetic, it has a low thermal conductivity and, most remarkably, it is tough and extremely resistant to abrasion. For the latter reason, it is widely used today where good wear-resistance is required.

Hadfield (1859-1940) (Fig. 9) was a rich man's son, his father being a successful manufacturer of steel castings in Sheffield. Soon after leaving school in his native city, his father had a laboratory fitted out for him and encouraged his natural inclination toward metallurgical research. Hadfield was still only nineteen years of age when he visited the Paris Universal Exhibition in 1878 and was struck by the exhibit of the Terre Noire Company, which had recently been successful in producing blast-furnace ferromanganese and introducing the new material to the steel industry. (Before this time manganese additions were made as spiegeleisen or as manganese oxide.) The Terre Noire Company was naturally interested in publicizing ferromanganese and was glad to make available to the young Englishman not only pamphlets describing their product, but also the results of their own tests on the properties of steel containing up to several per cent manganese.

The results of the Terre Noire researches showed that steels containing more than about 3% manganese were
brittle. The company had not investigated higher manganese levels; Hadfield, having time, money and opportunity, decided to investigate further. He meticulously translated the pamphlets and data supplied by the Terre Noire Company and began his experiments on manganese additions to carbon steels. On September 7, 1882, Hadfield wrote in his records book that "the experiments have led to some very curious, perhaps most momentous, results that may to some extent entirely revolutionise metallurgical opinions as regards alloys of iron and steel" — a large claim, but perhaps excusable in a youth of 23 years, working at a time when routine microscopic examination of metals had not begun, phase diagrams had not been thought of and testing facilities were rudimentary. However, whatever the theoretical implications, Hadfield had, in any case, invented a new material which after quenching (a very puzzling feature of his experiments), was tough, non-magnetic and highly abrasion resistant, the hardness reaching 600 Brinell under impact/wear service.

Hadfield patented his alloy in 1883. It has been a widely used material ever since, finding application in crushing equipment, rod and ball mills, railroad trackage, dredgers and earth-moving and mining equipment. Hadfield encouraged all kinds of applications for his alloy. He emphasized its non-magnetic characteristics; he made a virtue of its poor thermal conductivity and suggested its use for fireside pokers; he pointed out the superior casting properties and the ability of manganese steels to reproduce fine detail (Fig. 10).

Less well known was the application of manganese steel to soldiers' helmets. In the extensive trench warfare of World War I, impact-resistant helmets were perhaps the most important piece of defensive equipment issued to the infantrymen. Figure 11 illustrates the results of a contemporary test of French, British and German helmets. The French helmet was "easily perforated by shrapnel bullets at 350 feet per second," the German helmet, although 12 oz heavier than the British, cracked badly upon impact, while the British helmet resisted shrapnel up to 900 feet per second"**.

Hadfield constantly pointed out, and supported with extensive tests, the superior abrasion resistance of manganese steel (Fig. 12). This property and its toughness are its most valuable characteristics and account for the widespread use of this alloy a century after its invention.

Robert Hadfield became a prominent and enlightened industrialist, and a keen supporter of metallurgical research and development. He went on to invent silicon steel, which has become, with its sister material grain-oriented silicon steel, a widely used material for the cores of transformers and motors. He died in 1940 at the age of eighty-one.

*Hadfield has recorded that he was "horrified" one day to see in an illustrated paper a picture of a British infantryman frying bacon in his helmet. He lost no time in informing the authorities.

FIGURE 11. Comparative tests on French, British and German helmets.

FIGURE 12. Typical high-wear application of Hadfield's manganese steel.
The early history of the electrolytic zinc process

D.J. MacKinnon

Introduction

Zinc metal was first produced in India around 1100 A.D. and a zinc industry based on retorting gradually developed in both India and China\(^1\). The vertical retort method for smelting zinc ores was developed in England in the 1730s; this was followed by the more fuel efficient horizontal retort process developed by the Belgians in the 19th century\(^2\). The horizontal retort process was the standard method of zinc manufacture until the 1950s when it became obvious that it could not compete with electrolytic zinc technology\(^3\).

The present day extractive metallurgy of zinc is dominated by the electrolytic process which now accounts for almost 80% of the world’s primary zinc production. In recent years the electrolytic zinc process has seen significant improvements: the fluid-bed roasting of zinc concentrates, improved leaching and residue treatment techniques such as the jarosite and goethite processes, improved solution purification procedures, mechanization of the zinc electroylysis step and of the zinc melting and casting operation. In addition, new leaching processes have been implemented for the treatment of oxidized ores, particularly silicates, and the pressure leaching of zinc sulphide concentrates is now a commercial reality.

Although the electrolytic zinc process was first used on a commercial scale in 1916, its history can be traced back to the 1860s. The purpose of the present paper is to outline the early history of the electrolytic zinc process and to trace the significant developments which led to the first successful commercial electrolytic zinc plants.

The Létrange Process

It is not certain who first prepared zinc electrolytically in the laboratory but the first United States patent involving electrolytic zinc deposition was issued to Charles Beasley of Paris, France in 1862\(^4\). According to Ralston\(^5\), the first patents involving the commercial-scale production of electrolytic zinc were applied for in Germany almost simultaneously by C. Luchow of Dantz and L. Létrange of Paris in 1881.

The Létrange process, patented in 1883\(^6\), is particularly interesting and worth describing because it substantially anticipated modern electrolytic zinc practice. According to Létrange\(^7\), the ore is pulverized and mixed with calamine and blende (sphalerite) so that during roasting a maximum of zinc sulphate is formed. The calcin is leached with either water or sulphuric acid. Létrange did not describe the purification of the leach solution. Apparently, the preparation of zinc sulphate solutions for making lithopone\(^8\) was a well known art at that time and he merely refers to purification “by known means”. The electrolysis cell was a wooden tank lined with sheets of glass or lead.

The cathode was a zinc sheet and the anodes were usually carbon. The only major differences between the Létrange process and modern practice are in the electrode materials: lead is now used for anodes rather than carbon, which deteriorates in sulphuric acid, and zinc sheet cathodes have been replaced by aluminum sheet to facilitate stripping.

Early Difficulties

In view of the fact that the Létrange process provided the basis for modern electrolytic zinc practice, it is surprising that so many years elapsed before the commercial implementation of the technique in 1916. Since 1880, thousands of workers have contributed to the development of electrolytic methods for recovering zinc. Many patents were issued and many papers were published during the period 1880 to 1916 and these have been documented by E.P. Mathewson\(^9\). However, the importance of solution purity was not sufficiently recognized prior to 1914 with the result that many of the early deposits were spongy and did not adhere well to the cathode.

Another factor which hindered the commercial development of the electrolytic zinc process was that the electroylysis of zinc consumes more energy than the deposition of almost any other major metal. The energy requirement for zinc electroylysis is almost twice that for copper, and as long as the economics of the recovery of copper by electrolytic methods were disputed, the electrolytic zinc process met opposition.

It was also found that certain ores could not be leached successfully by the methods available at this time. Further work was necessary before the problems associated with ferrites and other impurities were solved.

Further Developments

In the 1890s both C. Hoepfner in Germany and E.A. Ashcroft in England were developing electrolytic processes for the treatment of complex zinc sulphide ores\(^10\). A semi-commercial demonstration plant based on Hoepfner’s process was built in Furfurt in 1894. This plant, consisting of sixty electrolytic cells, treated pyrite cinder from a nearby sulphuric acid plant to produce zinc cathodes and chlorine gas which was used to make bleaching powder. The Furfurt plant was built with V-shaped wooden tanks to accommodate the rotating disc cathodes; the lower halves of the cathodes dipped into the electrolytic tank between the anodes. Individual anode compartments, constructed from wood with nitrated cotton diaphragms for sides, were submerged in the tank and enclosed the anodes. The anodes consisted of slabs of gas-retort carbon and had a life of only six to eight months.

The Furfurt plant operated for only two years but its success led to the construction of three more plants, two in Germany and one in England. Little is known about the German operations, but the plant in England was located at the works of Brunner, Mond & Co. in Winnington and operated for over twenty years. At Winnington, zinc oxide was treated with by-product calcium chloride liquor and

\(^*\) a white pigment consisting essentially of a precipitated zinc sulphide and barium sulphate.


Readings in Historical Metallurgy
carbon dioxide flue-gas, forming a zinc chloride solution and a calcium carbonate precipitate. The zinc chloride solution was purified and electrolyzed to form 99.96% pure zinc and chlorine gas. Problems with obtaining suitable materials for anodes and diaphragms and the emergence of the sulphate process in 1916, led to the demise of the chloride route.

At the same time, further gains were being made in the development of the electrolytic zinc process by E.A. Ashcroft who was experimenting with both aqueous chloride and sulphate media and with molten chloride electrolyte. In 1896 the Sulphide Corporation built a 1500-lb test plant at Cockle Creek, New South Wales to try the Ashcroft process on the Broken Hill complex sulphide ore. The ore was roasted, zinc was removed by leaching, and lead and silver were recovered from the residue by smelting. Zinc was recovered by electrolysis, with both sulphate and chloride solutions being used at different times. Although significant amounts of zinc were produced, the process was a technical and economic failure.

While Hoepfner and Ashcroft were developing their chloride processes, the German firm of Siemens and Halske was experimenting with the Lérange sulphate process. Their work produced numerous patents and resulted in improved techniques for purifying zinc sulphate solutions.

During the period between 1907 and 1914, several inventors improved parts of the sulphuric acid leach-electrowinning process that paved the way for the economic success of the first generation electrolytic zinc plants at Trail, British Columbia and at Anaconda, Montana. Among these were P.C.C. Isherwood who applied pressure leaching to refractory zinc-lead ores, U.C. Tainton who improved the technique of electrolyzing zinc sulphate solutions by using higher current densities, pure solution and a new additive, gum tragacanth, and A.G. French who developed a process for electrolyzing zinc sulphate solutions containing manganese sulphate whereby coherent zinc was formed at the cathode and hydrated manganese dioxide at the anode.

**First-Generation Electrolytic Zinc Plants**

The First World War provided an impetus to the commercial development of the electrolytic zinc process. Metallurgists at the Anaconda Copper Mining Co., Anaconda, Montana and at the Consolidated Mining and Smelting Co., Trail, British Columbia began intensive work on the process in 1913-1914. Both groups eventually developed processes which were technically feasible and economically sound.

Commercial production of electrolytic zinc in the United States began in the 10-ton per day plant at Anaconda late in 1915. Briefly, the process consisted of roasting a zinc sulphide flotation concentrate in two of the furnaces of the copper leaching plant. The calcine was leached with sulphuric acid to which a small amount of manganese dioxide was added to oxidize the iron. Powdered limestone was added to precipitate calcium and any arsenic or antimony present was carried down with the iron precipitate. The filtered solution was treated with zinc dust to precipitate copper and cadmium. The solution was then pumped through a clarifying filter-press into a storage tank and from there to the electrolytic cells where the zinc was deposited on aluminum cathodes.

The cell room contained forty-two tanks with pure lead anodes. The solution was electrolyzed at a current density of 20-30 amp/sq. ft. and the deposits were stripped every 48 hours. The current efficiency was 93% to 94%. A partial view of the Anaconda cell room is shown in Figure 1.

This plant was so successful that Anaconda began construction of a 35,000 ton per annum plant at Great Falls, Montana in 1915. This site was chosen because of its cheap power and the desirability of locating the plant as closely as possible to the busbar of the power station.

Apart from the Anaconda work, the most ambitious plans carried out in 1915 were those of the Consolidated Mining and Smelting Co. of Canada (later Cominco Ltd.) which continued the experimental work begun several years previously. Experimental work, which had begun at Trail in 1912, led to the erection of a 1000-lb per day experimental zinc plant in 1913 and to its expansion to 35 tons per day in 1915. The treatment of the lead-rich zinc ore was similar to that practised by Anaconda. The original tank room at Trail contained 448 cells, each of which was 27 in. wide by 36 in. deep by 83 in. long. The skinning of
cathode zinc at Trail is shown in Figure 2. The history of the Trail zinc operation has been reviewed in a recent publica-


Prior to 1914, the "zinc hydrate" process for treating the ores of the Bully Hill Mine, Shasta County, California was
developed on the assumption that a good zinc deposit could not be electrowon from a solution containing much sulphuric acid; hence every effort was made to neutralize the acid as fast as it formed at the lead anodes. In the
treatment of the Bully Hill ore, lime was used to precipitate zinc hydroxide and calcium sulphate from the zinc sulphate
solution. This precipitate was suspended in the zinc sulphate electrolyte and the sulphuric acid formed at the anodes was neutralized in situ by the zinc hydroxide.

About two years of effort were expended on the "zinc hydrate" process and some excellent zinc cathodes were
produced during this period. It was concluded, however, that the "zinc hydrate" process could not compete with the
direct electrolysis of zinc sulphate with acid regeneration, and the company eventually switched its attention to the
latter technique. An experimental plant came on stream in March 1915 producing 300 - 400 lb zinc per day. Figure 3
shows a 2 to 3-week output of Bully Hill Electrolytic Zinc, and Figure 4 gives some indication of the cathode quality.

At Welland, Ontario, the Weedon Mining Company was meeting success in operating the Watts process. This
process used a zinc sulphate electrolyte and also attempted to neutralize sulphuric acid formed at the anodes by
adding any solid zinc compound (such as zinc oxide) which did not contain objectionable impurities.

At Silverton, British Columbia, the Standard Silver Lead Mining Company was experimenting with the pro-
duction of electrolytic zinc by the French process[12]. French was probably the first to use manganese in the electrolysis
of zinc sulphate solution although its beneficial effects had been noted by other workers. French found that man-
ganese deposited as the dioxide on the anode and could be redissolved in sulphuric acid for reuse. His process operat-
ed with considerable manganese sulphate in the electrolyte and used a solution of by-product sodium bisulphate
rather than sulphuric acid for leaching.

The outbreak of war in 1914 was instrumental in accelerating the development of Australia's electrolytic zinc
industry. In July 1916, an agreement was signed between Amalgamated Zinc and the Tasmanian Government that
gave the company (to be known as Electrolytic Zinc Company) a lease on fifty acres of crown land at Risdon and a supply of electrical energy for the treatment of zinc
concentrates.

Construction of an experimental plant on the Risdon site soon followed; this began operation in December 1916
with a number of small glass cells each having a capacity of 2 lbs of zinc per day. Construction of a larger pilot-scale
plant to produce 250 pounds of zinc per day proceeded simultaneously with the operation of the glass cells. This
larger plant, consisting of one cascade of five lead-lined cells, each approximately 48 in. long by 30 in. wide by 42
in. deep and containing four cathodes, began electrolysis in March 1917. In January 1918, a plant designed to
produce 22 400 lb. of zinc per day was set in operation. A detailed history of the development of the Electrolytic Zinc
Company can be found elsewhere[13].

A summary of some of the more important zinc plants which were operating during the period 1907-1919 is
given in Table 1. The list, taken from Ralston[14], is incomplete for countries other than the United States, but
<table>
<thead>
<tr>
<th>Company</th>
<th>Location</th>
<th>Daily Tons Zinc</th>
<th>Remarks</th>
</tr>
</thead>
<tbody>
<tr>
<td>American Smelting &amp; Refining Co.</td>
<td>Murry, Utah</td>
<td>Experimental</td>
<td>Operated in 1916</td>
</tr>
<tr>
<td>American Smelting &amp; Refining Co.</td>
<td>Omaha, Nebraska</td>
<td>Experimental</td>
<td>Treated zinc crusts (1907-1916)</td>
</tr>
<tr>
<td>Anaconda Copper Co.</td>
<td>Anaconda, Montana</td>
<td>25-ton experimental</td>
<td>Operated 1915-1916</td>
</tr>
<tr>
<td>Basin Salvage Co.</td>
<td>Basin, Montana</td>
<td>150 tons</td>
<td>Built in 1916</td>
</tr>
<tr>
<td>Bully Hill Cooper Co.</td>
<td>Bully Hill, California</td>
<td>Experimental</td>
<td>Operated in 1915-1919</td>
</tr>
<tr>
<td>Chas. Butters</td>
<td>San Francisco, California</td>
<td>3-ton experimental</td>
<td>Built 1918</td>
</tr>
<tr>
<td>Electrolytic Zinc Co.</td>
<td>Baltimore, Maryland</td>
<td>10 tons</td>
<td>Dismantled 1919</td>
</tr>
<tr>
<td>Judge Mining &amp; Smelting Co.</td>
<td>Park City, Utah</td>
<td>7 tons</td>
<td>Built 1916-1917</td>
</tr>
<tr>
<td>Mammoth Copper Mfg. Co.</td>
<td>Kennett, California</td>
<td>25 tons</td>
<td>Operated 1917-1919</td>
</tr>
<tr>
<td>Northwestern Metals Co.</td>
<td>Helena, Montana</td>
<td>Experimental</td>
<td>Operated 1913-1914</td>
</tr>
<tr>
<td>Reed Zinc Co.</td>
<td>Palo Alto, California</td>
<td>Experimental</td>
<td>Operated 1914-1915</td>
</tr>
<tr>
<td>River Smelting &amp; Refining Co.</td>
<td>Keokuk, Iowa</td>
<td>10 tons</td>
<td>Operated 1916-1919</td>
</tr>
<tr>
<td>Tennessee Copper Co.</td>
<td>Copperhill, Tennessee</td>
<td>Experimental</td>
<td>Operated 1917</td>
</tr>
<tr>
<td>Western Chemical Co.</td>
<td>Denver, Colorado</td>
<td>19 tons</td>
<td>Built in 1918</td>
</tr>
<tr>
<td>Consolidated Mining &amp; Smelting Co.</td>
<td>Trail, British Columbia</td>
<td>50-75 tons</td>
<td>First unit 1914</td>
</tr>
<tr>
<td>Weedon Mining Co.</td>
<td>Welland, Ontario</td>
<td>25-ton experimental</td>
<td>Operated 1915</td>
</tr>
<tr>
<td>Brunner, Mond &amp; Co.</td>
<td>Winnington, England</td>
<td>5 tons</td>
<td>Operated 20 years</td>
</tr>
<tr>
<td>British Electrolytic Zinc Co.</td>
<td>Widnes, Lancashire, England</td>
<td>Experimental</td>
<td>Test Isherwood process</td>
</tr>
<tr>
<td>British Metale Extraction Co.</td>
<td>Liamsamlet, Wales</td>
<td>25-ton experimental</td>
<td>Bisulphite process, not electrolytic</td>
</tr>
<tr>
<td>Electrolytic Zinc Co. of Australasia</td>
<td>Hobart, Tasmania</td>
<td>15 tons</td>
<td>Operated 1917-1919</td>
</tr>
<tr>
<td>Tasmanian Metals Extraction Co.</td>
<td>Mt. Read, Tasmania</td>
<td>Experimental</td>
<td>Expanded to 100 tons, Abandoned 1915</td>
</tr>
<tr>
<td>Takada &amp; Co.</td>
<td>Japan</td>
<td>2 tons</td>
<td>Operated 1917</td>
</tr>
<tr>
<td>Mitsui &amp; Co.</td>
<td>Nagasaki, Japan</td>
<td>Experimental</td>
<td>Operating 1919</td>
</tr>
<tr>
<td>Anon.</td>
<td>Duisburg, Germany</td>
<td>Commercial</td>
<td>Dieffenbach process</td>
</tr>
<tr>
<td>Anon.</td>
<td>Hruschau, Austria</td>
<td>Commercial</td>
<td>Hoepfner process</td>
</tr>
<tr>
<td>Anon.</td>
<td>Finland</td>
<td>1 ton</td>
<td></td>
</tr>
</tbody>
</table>

* from Ralston(4)

indicates the great breadth of early developmental activity that helped to create the modern electrolytic zinc industry.

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The end of horizontal zinc retorting in the United States

J.E. Dutrizac

The end of an era in zinc metallurgy took place on July 31, 1976 when the horizontal retort furnaces of the National Zinc Company in Bartlesville, Oklahoma were permanently shut down\(^{15}\). For more than a century, horizontal retorting had been the most common method of producing zinc in the United States; it was a thriving industry in many parts of that country, especially in the natural gas belts of Oklahoma and Texas. With the Bartlesville closure, yet another metallurgical process has become an object of historical interest for North American metallurgists.

Although shrouded in controversy, the origins of zinc metallurgy seem to date back at least 2000 years. Pyrometallurgical methods with carbon reductants are capable of producing zinc from oxidative zinc ores, but only above the boiling point of the metal. Although slow cooling results in re-oxidation of the zinc vapour by air or carbon monoxide, quenching yields liquid zinc; some form of rapid cooling, either accidental or intentional, was likely used by ancient metallurgists to liquify the metal. Zinc metal found in Athenian objects of the Roman era was likely of accidental origin. The Romans employed brass extensively for coins and ornaments\(^{20}\), and this was intentionally produced by the calamine process whereby oxidative zinc ores were added to molten copper under strongly reducing conditions. Zinc appears to have been known in China and India as early as 1000-1300 A.D., and was probably smelted commercially in those countries from the 14th century.

Although the early metallurgical procedures were probably inefficient, they were able to produce zinc metal in significant amounts. The large heaps of zinc residues with clay retorts bear testimony to an early smelting industry of impressive magnitude. It is estimated that the heaps at Zawar, India alone represent the extraction of 100,000 tons of metallic zinc\(^{11}\). In the 1730s the smelting of oxidative zinc ores was developed in England using a vertical retort method. In 1788 a British patent was granted for making zinc from zincblends (ZnS) by roasting the ore in a coal-fired reverberatory furnace, mixing the calcine with charcoal and smelting the mixture with suitable zinc condensers. The English process was very energy intensive, however, and early in the 19th century the Belgians developed a horizontal retort process for the production of distillation of zinc that was more fuel efficient. This technique was gradually developed into what came to be known as the Belgian process, the forerunner of the "modern" horizontal retort furnace.

In 1836, the U.S. Congress ordered standard weights and measures to be fabricated from brass produced in the United States\(^{15}\). To make the zinc necessary for the brass, skilled workers were imported from Belgium and a small furnace was erected to treat the oxide ores (ZnO and ZnFeO\(_2\)) from northern New Jersey. Although the output was small, this was the first zinc production in North America. The first successful commercial zinc smelter was built in La Salle, Illinois by Frederick Mattiessen and Edward Hegler to take advantage of Wisconsin ore and Illinois coal. Their experimentation and development work had a significant influence on the course of American zinc smelting. To increase capacity, row after row of retorts were added in a long tunnel-like furnace; the size of the furnace helped to reduce the heating requirements. By 1872, they had developed a furnace containing 408 retorts that was eventually adopted by all smelters operating in the Oklahoma gas belt.

The National Zinc Company was incorporated in New York in March 1907, eight months before Oklahoma was admitted as the 46th state of the American Union; 1982 marks the company's 75th anniversary. The newly-formed company authorized construction of a plant in Bartlesville, then in Indian Territory, to take advantage of the plentiful supplies of natural gas and the vast tonnages of zinc ores pouring from the mines in the nearby Tri-state area. It is interesting to note that the authorized expenditure was only $180,000 for a plant of six furnace blocks. As an afterthought, an additional $10,000 was allocated for construction of a power plant and the pottery which was vitally important for the fabrication of clay retorts and condensers. Capital costs were certainly attractive by current standards and environmental regulations were then minimal. Unlike today when development can be delayed for years for want of an environmental impact statement, construction in 1907 began immediately. In a few months, work was completed on one of the six furnace blocks and the first slab zinc was poured in early December, 1907, the same year the company was founded. During World War I, the smelter employed about 2000 workers and contributed greatly to both the war effort and the economy of the region. Operations continued until a depression caused a temporary closure in 1921; production resumed in 1922. Although output was again curtailed in the early 1930s, the company has since operated continuously aside from a brief labour strike in 1958. The continuity of production at Bartlesville is in marked contrast to that at many of the other zinc plants in the region which operated only until the local natural gas supply became exhausted. At that time the smelter would be shut down and rebuilt at some new gas discovery site. Clearly, fuel costs were a major factor in the successful operation of a horizontal retort plant.

The horizontal retort process was the standard method of zinc manufacture until the 1950s and was the process employed at National Zinc. In this method, roasted zinc concentrates were mixed with carbon, either coal or coke, placed in a small diameter clay retort and then heated to about 1150°C in a gas-fired furnace. The reaction between the zinc oxide and carbon yielded carbon monoxide and zinc vapour which collected in a clay condenser fitted to the open end of the retort. Figure 1 illustrates a bank of the retorts fitted with their clay condensers during active operation; the carbon monoxide gas is burning at the clay-plugged lips of the condensers. The wide aisle required to charge and/or to service the retorts is quite evident. Molten zinc metal was periodically drawn from the clay condensers and was hand-cast into slabs, as illustrated in Figure 2. The small casting ladle with its direct manual control contrasts sharply with modern large-tonnage casting lines.

During the life of National Zinc’s horizontal retort smelter, several process improvements were effected. In 1923, a sintering machine was installed to agglomerate the fines generated during roasting, and this made the ore easier to smelt. In 1928, multiple hearth roasters were added, and the SO₂ gas which had been vented to the atmosphere since 1907 was used to produce sulphuric acid in a newly constructed acid plant. After several years of experimentation, the smelting furnaces were rebuilt in 1937 to accommodate larger retorts, and the firing cycle was changed from 24 to 48 hours, a practice which was quickly adopted by the other Oklahoma smelters since it reduced labour costs. The charging and cleaning of the retorts were mechanized in 1952, thereby relieving the furnacemen of much of the hot and difficult work while simultaneously increasing furnace throughput. In 1953, the company installed the first fluid bed reactors in the United States for the roasting of zinc concentrates. The new roasters reduced significantly the sulphur dioxide emissions from the plant, and in 1969, a new acid plant was built.

By the early 1960s it was becoming quite clear that the horizontal zinc retort process could not compete with the roast-leach-electrowin route to zinc. Labour and fuel costs were simply too high, and there were problems with in-plant hygiene and metal purity. During the early 1970s one by one the retort operations, such as those at Henryetta and Blackwell, Oklahoma, and at Amarillo and Dumas, Texas, were closed never to open again. With the cessation of retorting operations at Bartlesville in 1976, an era in American metallurgy had come to an end.

Although retorting at National Zinc ended in 1976, zinc production has continued. Today National Zinc produces 56,000 ton/y of zinc and 85,000 ton/y of sulphuric acid from a new $43,000,000 electrolytic refinery which still uses the roasters and acid plant of the old horizontal retort facility. The new leaching plant is notable for its use of modern goethite technology for iron control; the electrolytic tankhouse features automatic cathode stripping. The old retort furnaces (often fully charged with retorts) together with many of their ancillaries still stand, however, and can be viewed by metallurgists with an interest in the history of their profession. Some of the retorts and condensers have been preserved in local museums and the possibility exists for the conservation of still more of this unique and fascinating facility. To its credit, the company has prepared lengthy, if unedited, film footage which illustrates the last days of the horizontal retort operation and which is available under certain conditions.

Acknowledgments
The author would like to thank T.L. Vogt of the National Zinc Company for his support of this project and for providing the photographs.

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A hundred years of the Bayer Process
for alumina production

Fathi Habashi

On August 3, 1888, German Patent No. 43977 entitled “A Process for the Production of Aluminum Hydroxide” was issued⁴. The discovery which led to the patent was made by the Austrian chemist Karl Josef Bayer (1847-1904) (Fig. 1) who was at that time in Russia, and the process became known as the Bayer Process in his honour.¹,² The process immediately achieved industrial success, displacing the pyrometallurgical process that had been used until that time to produce alumina. The Bayer Process involved the pressure leaching of bauxite with NaOH solution to obtain sodium aluminate solution from which aluminum hydroxide was precipitated by seedling.

Today bauxite is not only the principal source of aluminum but also of gallium, both metals being effectively recovered by the Bayer Process. The purpose of this paper is to outline the historical background of this discovery and its importance to hydrometallurgy.

Background

Bauxite was discovered in 1821 by the French chemist P. Berthier who called it alumine hydratée des Beaux after the village of Les Beaux near Arles in southern France. The red colour of the deposit had attracted Berthier’s interest as a possible iron ore for the blast furnace in the district. However, he found that it contained too much Al₂O₃ to be of value for this purpose. The name was changed to bauxite in 1847, by A. Dufrénoy and to bauxite in 1861 by the French metallurgist Henri Sainte-Claire Deville.¹³ Bauxite was first thought to contain alumina dihydrate, Al₂O₃·2H₂O; however, later it was shown by thermal analysis to contain a mixture of the hydroxides Al(OH)₃ and AIOOH. Discoveries of bauxite were later reported in many parts of the world including Jamaica, Surinam, Guyana, Australia, U.S.S.R., U.S.A., and Hungary. It now represents one of the highest tonnage mineral raw materials treated chemically — about 88 million tons annually worldwide (Fig. 2).

Bayer’s Process relies on the technologies of three unrelated industries: The recently developed aluminum industry, the old textile industry, and the pressure technology of the rising synthetic dyestuffs industry.

The Aluminium Industry

After the 1854 discovery by Sainte-Claire Deville of a method for separating kilogram amounts of aluminum from its oxide, attempts were made to produce the metal on a large scale. To this end a plant was constructed at Salindres in France which combined two processes: the Le Chatelier Process for the production of aluminum oxide and the Sainte-Claire Deville Process for the production of aluminum chloride and its reduction to the metal by metallic sodium. These processes are now described.

The Le Chatelier Process

Aluminum can be solubilized readily from bauxite by acids but on adding an alkali to the solution a basic salt, rather than a hydroxide is precipitated. Furthermore, because iron and titanium are also dissolved in acid the precipitate will be contaminated, creating a separation problem. As a result this route was never used to prepare pure Al₂O₃. The French mining engineer Louis Le Chatelier (1815-1873) whose son the chemist Henri Le Chatelier (1850-1936)¹⁴ is best known for the thermodynamic principle which bears his name, obtained alumina from bauxite in 1855 by a thermal route in three steps:

1. An intimate mixture of powdered bauxite and sodium carbonate was heated in a reverberatory furnace to transform aluminum hydroxide into sodium aluminate: 2Al(OH)₃ + Na₂CO₃ → 2NaAlO₂ + CO₂ + 3H₂O.

2. Sodium aluminate was then leached from the insoluble ferric oxide with a dilute solution of aluminate from the preceding step.

3. The concentrated liquor was then treated with carbon dioxide to precipitate aluminum hydroxide analyzing typically 47.5% Al₂O₃, 50% H₂O and 2.5% Na₂CO₃. Carbon dioxide was obtained from the furnace exit gases.

A drawback of this process was the excessive reagent consumption because any silica in the bauxite reacted with soda to form sodium aluminum silicate which represented a serious loss.

The Sainte-Claire Deville Process

Henri Sainte-Claire Deville (1818-1881), a professor of chemistry at Ecole Normale supérieure in Paris, laid the foundation of the aluminum industry in 1854 when he prepared the metal by making use of the reaction between metallic sodium and the double chloride of aluminum and sodium: AlCl₃ + 3Na→ Al + 3NaCl, the sodium chloride acting as a protective flux. In 1856 a small plant constructed at Glaciére near Paris produced about 25 kg of the metal. Aluminum chloride-sodium chloride was prepared by heating Al₂O₃ with a mixture of NaCl and carbon in a

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¹³ This invention was made one year after another invention of equal importance to hydrometallurgy, the so-called Cyanidation Process for obtaining Silver from Ores”, British Patent No. 14174 issued October 19, 1887.


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stream of chlorine; the double chloride was then distilled off and condensed. Metallic sodium was produced by reducing Na₂CO₃ with carbon at high temperature and condensing the vapours.

Hamilton Y. Castner (1859-1899) of Columbia University in New York was responsible for reducing the cost of aluminum by his invention of two cheap methods for producing sodium from molten NaOH, the first in 1886 by reduction with an iron-carbon mixture, and the second in 1890 by electrolysis. However, this was too late for the Sainte-Clair Deville Process. Beginning in 1886 the Sainte-Clair Deville Process for the manufacture of aluminum was rapidly displaced by a new process invented simultaneously by Paul Héroult (1863-1914) in France and Charles Martin Hall (1863-1914) in the U.S.A. The new process was based on the electrolytic reduction of Al₂O₃ dissolved in fused cryolite (Na₂AlF₆) found in Greenland.

The Textile Industry

In 1885 the Austrian-born chemist Bayer emigrated to Russia to work in the Tentelev chemical Plant in Saint Petersburg (now known as Red Chemist). Russia at that time was open to all foreigners with technical and artistic skills. This plant used the Le Chatelier Process to produce aluminum hydroxide which was used as a mordant* for dyeing cotton, wool, and silk. The textiles to be dyed were soaked in a solution of the hydroxide dissolved in a weak acid, then squeezed, dried and steamed whereupon the hydroxide precipitated on the fibers. Thus treated, the textiles could be immersed in a dye solution to form a coloured "fake". This was a standard method of dyeing at that time. For example, Turkey Red, a popular red colour was prepared by dyeing with alizarine on aluminum hydroxide mordant.

During this period, with the aluminum industry in its infancy, the chemistry of aluminum hydroxide was under intensive investigation. Early investigators were aware of two types of aluminum hydroxide: the gelatinous and the crystalline forms. The gelatinous hydroxide was formed by rapid precipitation from an aluminate solution for example, by CO₂ and it was soluble in dilute acid. The crystalline hydroxide was formed by slow precipitation and was slightly soluble in acid. The gelatinous hydroxide was insoluble in ammonium hydroxide but soluble in sodium hydroxide. It was also known that if a saturated solution of the gelatinous hydroxide in NaOH was kept in a closed vessel for many days, a crystalline hydroxide was deposited. Bayer assumed that a solution saturated with the gelatinous hydroxide must be supersaturated with respect to the crystalline form. Consequently, if the solution of the gelatinous hydroxide were seeded with the crystalline form, precipitation of the crystalline hydroxide should slowly occur. Bayer succeeded in precipitating the crystalline Al(OH)₃ using the agitated tanks shown in Figure 3. Precipitation took place in 2 to 3 days.

In modern terms the precipitation of crystalline aluminum hydroxide from alkaline solution can be described as follows: The aluminate ion undergoes hydrolysis according to the equilibrium:

\[ \text{AlO(OH)}_2^- + \text{H}_2\text{O} \rightleftharpoons \text{Al(OH)}_3^- + \text{OH}^- \]

The function of the seed is twofold:
1. To shift the above equilibrium to the right by reaction with OH⁻ ions (a dissolution process), and
2. To provide a nucleus on which Al(OH)₃ can grow.

In today's practice, the seed is finely divided Al(OH)₃ obtained from a previous precipitation batch by classification, i.e. allowing the coarse product to settle as a product

* Mordant dyeing was described by Pliny in 80 A.D.; at that time naturally occurring alum (aluminum sulfate) was used.
and recycling the fines as a seed. The seed usually amounts to about one-third of the product — an appreciable amount.

Bayer then argued that if the mother liquor could be used to solubilize aluminum hydroxide in bauxite then such a process should be more economical than the Le Chatelier Process. The reaction of bauxite with NaOH was, however, very slow unless a high temperature was used so that it was necessary to use a reactor which was closed and able to withstand the high vapor pressure generated.

Pressure Technology

Colouring matters have been used since ancient times, but these were all naturally occurring, for example indigo, alizarine, etc., were mainly extracted from plants and insects and were imported to Europe from far countries. In the 1870s synthetic dyestuffs were produced for the first time on a commercial scale and it was in this industry that high pressure reactors were first applied. Organic intermediates which were needed to manufacture the synthetic dyestuffs were produced in heated agitated reactors that were able to withstand the reasonable pressures required for reactions such as sulfonation, nitration, reduction, etc.

After discovering the precipitation method for obtaining crystalline aluminum hydroxide, Bayer found that alumina contained in bauxite could be dissolved selectively by heating with a solution of sodium hydroxide under pressure in an autoclave to form sodium aluminate solution. A typical reactor of this period is shown in Figure 4.

Recent Trends

Bayer’s process involving both pressure leaching and controlled precipitation is used today in practically the same way as when it was discovered one hundred years ago. However, tremendous improvements in the engineering aspects have taken place and these are responsible for decreasing the cost. Furthermore, a new importance was given to the process when gallium became needed by the semiconductor industry and its recovery from process solution became desirable.

A problem of the Bayer Process, however, is the disposal of the so-called red mud — the residue of the leaching step. This is mainly iron oxide with small amounts of rutile, ilmenite, quartz, and hydrated sodium aluminum silicates. Each ton of alumina produces about one ton of red mud (on a dry basis). With increased production of metallic aluminum world-wide, the amount of red mud produced has reached tremendous proportions — about 100 million tons each year. No use for this material has yet been found and each plant has large ponds to store its waste.

Energy Economy

In the early days of the Bayer Process 1 ton of bauxite required 2.33 tons of coal for processing; today it requires only 0.15 tons\(^6\). Thus in the early days the bauxite had to be transported to regions where coal was available, whereas now alumina plants are situated near the bauxite (Fig. 5). The economy in energy is due to two factors:

1. An increased heat recovery — heat exchangers and flash tanks are now extensively used to economize energy. Flash tanks serve an additional purpose: the evaporation of solutions.

2. The use of large autoclaves. The larger the reactor the less will be the heat losses. Construction of such equipment that can be used reliably is due to improvements in engineering design and manufacturing. Along with autoclaves, precipitation tanks have increased in size correspondingly (about ten times the size of an autoclave, Table 1).

Furthermore, steam is now used for heating and agitation, and autoclaves are connected in series to permit continuous operation. This allows automation and decreasing manpower. A typical modern autoclave is shown in Figure 6.

Tube Autoclaves

The use of tube autoclaves (Fig. 7) began in Germany in
Gallium Production

Gallium was discovered in 1875 in zinc blend and its first production was from this source. In 1896 it was discovered in bauxite by W.N. Hartley and H. Ramage\textsuperscript{81}. Gallium is now produced mainly from this source although its average concentration is usually less than 0.01%. Gallium is also found in coal fly ash. Alusuisse, a major aluminum manufacturer in Switzerland, commenced industrial scale production of gallium from bauxite in 1955. Gallium follows aluminum in the Bayer Process, and is recovered from the aluminate solution either by electro-deposition on a mercury cathode, or by cementation using sodium amalgam, or by selective carbonation. After precipitation of Al(OH)\textsubscript{3}, and recirculation of the caustic solution, gallium may be enriched to a concentration of about 0.2 g/L. The present Western world production of gallium is about 50 tons per year.

Gallium (Fig. 8) is a vital material in the semiconductor industry which started in the 1960s. It is mainly used in the form of arsenide and phosphide, GaAs and GaP, respectively. Both compounds show electroluminescence, and are used in the manufacture of light-emitting diodes. Gallium arsenide in combination with (Ga, Al) As-layers is a material used in solid-state lasers, especially in telecom-
Bayer and His Medal

Bayer was born on March 4, 1847 in Bielitz in Silesia, at that time a Province of the Austrian Empire, now in Poland. The empire was composed of a vast territory encompassing the present-day Austria plus northern Italy, Bohemia and Moravia (parts of present-day Czechoslovakia), Hungary, Transylvania (part of present-day Romania), Croatia, Dalmatia and Bosnia-Herzegovina (parts of present-day Yugoslavia), and Silesia and Galicia (southern Poland). While the official language was German, many other languages were spoken, e.g. Hungarian, Romanian, Czech, Polish, Ruthenian (Ukrainian), Slovak, Slovenian, Serbo-Croatian and Italian. The multitude of nationalities and languages caused many revolts and political agitation.

Bayer went to school at his home town and at the wish of his father, who was an architect, started to study architecture. Later he switched over to science and went to Wiesbaden in Germany to study chemistry under the famous analytical chemist C. Remigius Fresenius (1818-1897). After a short period of interruption during which he worked in a steel factory in Belgium, he went to Heidelberg to continue his chemistry studies at the University of Heidelberg. There, he became assistant to Professor Robert Bunsen (1811-1899) in the period 1869 to 1871. At that time Bunsen's laboratory was visited by many chemists who became famous later on. Among those were Dimitri Mendeleev, Friedrich Bollstein, Henry Roscoe, Auer von Welsbach, Lothar Meyer, Victor Meyer, and many others. Bunsen's reputation stems from his discovery together with the physicist Gustav Kirchhoff (1824-1887) of the spectroscopic method of analysis, and the discovery of the two metals rubidium and cesium by this new tool. Bunsen is also famous for the burner known by his name and now found in every chemical laboratory.

In Heidelberg, Bayer was granted a doctorate after submitting a thesis entitled "A Contribution to the Chemistry of Indium". Indium had been discovered a few years earlier (in 1863) by the two German chemists Ferdinand Reich (1790-1882) and Hieronymus Theodor Richter (1824-1898) utilizing Bunsen's spectroscope. Afterwards, Bayer returned to his home country Austria where he was appointed a lecturer at the University of Technology at Brno in the Moravian Province of the Empire (now in Czechoslovakia). He remained there until 1873, when he left the University to establish a research laboratory in Brno. However, he later gave up this venture and in 1885 moved to St. Petersburg in Russia.

Bayer's years in Russia were most fruitful and creative. He joined the Tentelev Chemical Plant near St. Petersburg in 1887, to work on problems of production of pure aluminum hydroxide for the dyeing of cotton fabrics. There in 1888 he made the discovery that aluminum hydroxide could be precipitated from sodium aluminate solution if a seed of a freshly precipitated aluminum hydroxide was agitated vigorously in the cold solution. Bayer mentioned that the product obtained by his process was purer than that obtained by CO₂ precipitation because no phosphates or SiO₂ precipitated. Further, the aluminum hydroxide precipitated was also crystalline and could be easily filtered and washed. The process was soon adopted by the Tentelev Plant.

Bayer's second discovery was in 1892 at the Elabuga Plant on the Kama River (Tatar S.S.R.). He found that alumina contained in bauxite could be dissolved selectively by heating with a solution of sodium hydroxide under pressure in an autoclave to form sodium aluminate solution. He found also that the alkaline mother liquor obtained after the precipitation of aluminum hydroxide by the seed could be used.

While at Elabuga, he received numerous contracts from foreign countries to build alumina factories. Thus he built one in England for the British Aluminium Company in London, one in France for the Compagnie des produits chimiques et électrométallurgiques in Floges, one in Italy for the Società Italiana Elettrochimica in Bussi, one in the U.S. A. for Merimae Chemica Company in Boston, and one in Germany for Gebrüder Giulini in Ludwigshafen. He was awarded the gold medal of the Académie Par- sienne des Inventeurs industriels et expositors for his scientific and technical services.

In 1894, Bayer returned to Austria and finally settled in Rietzdorf in southern Styria in order to devote himself to scientific research. During this period he developed a method for the manufacture of synthetic cryolite which is used as an electrolyte in the aluminum industry. While in Rietzdorf, he planned to build an aluminum industry in Austria. However, he was unable to raise the necessary capital, and thus his plans failed in his home country.

Bayer was married to Alma von Witte, a niece of the Russian prime minister Count von Witte. They raised five sons and a daughter. One of his sons, Fritz Bayer, was professor of electrochemistry at the Technische Hochschule in Vienna (now known as Technische Universität Wien) until 1956 when he retired.

Bayer's home in Rietzdorf was a meeting centre for many famous industrialists among whom were Hérout and Hall. He loved music and the arts; he himself was a talented artist. He spoke six languages: German, French,
English, Russian, Italian and Slovak. He had an excellent collection of minerals which he displayed at the Chicago exhibition in 1890.

Bayer died suddenly on October 4, 1904 at the age of 57, and his widow died in 1962 at the age of 94. The foreign companies (except two) who were applying his patents stopped paying royalties after his death. It was difficult at that time to sue them, and consequently his house and laboratory had to pay for his debts.

Bayer is honoured in his native country Austria by the medal bearing his name (Figure 9) *; this is awarded every three years to a distinguished researcher in the field of aluminum. The award ceremony takes place during the International Light Metals Congress which is held in Leoben and in Vienna.

Thus, Bayer's invention to satisfy the need of the Russian textile manufacturers soon turned out to become most important for supplying the need of the growing electrolytic aluminum industry which had been discovered four years earlier by Hall and Héroult.

* Bayer's first name appears as "Karl" in his patents but as "Carl" on his medal.

Acknowledgement

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REFERENCES

One Hundred Years of Cyanidation

Fathi Habashi

Introduction

The year 1887 marked the centennial of the beginning of modern hydrometallurgy. On October 19, 1887 British Patent No. 14174 entitled Process of Obtaining Gold and Silver from Ores was issued. The discovery was made by John Steward MacArthur (Fig. 1), a chemist-metallurgist, supported and assisted by Robert and William Forrest, two brothers practicing medicine in Glasgow. The discovery was made in the basement of their clinic. The process, which became known as the cyanidation process in reference to the alkali cyanide reagent used, immediately received industrial success in New Zealand in 1889 and in South Africa in 1890. It displaced the chlorination method which had been in use. In 1903 MacArthur was awarded the first Medal of the Institution of Mining and Metallurgy in London. The cyanidation process was responsible for the doubling of gold production in the world in the two decades which followed its first application (Fig. 2).

To the modern metallurgist, it may be surprising to think of MacArthur coming across an obscure reagent such as an alkali cyanide as a leaching agent for gold ores. It is the purpose of this paper to discuss the background and the factors that led to this invention, the development of the understanding of the process, and the application of modern technology.

Background

To understand the factors that led to the discovery of the cyanidation process it is essential to know the activities of chemists at that time. The nineteenth century was the century of great discoveries that were the result of earlier activities. In particular, the discovery of the cyanidation process relied both on the attempts by alchemists to transform base metals into gold, and on the emerging chemistry of the so-called “Blue Acid” and its compounds during the eighteenth century.

Transmutation of Base Metals

Gold played a central role in metallurgy. It was used and valued by ancient peoples who formed it into a variety of objects. At one time, there was no need for any chemical or metallurgical knowledge to get gold; it occurred in nature in the native state and simple panning was enough to collect the glittering particles, and sometimes nuggets, which could readily be melted to obtain an ingot. However, since this required great patience and hard work, the alchemists thought of a faster way: transmutation of base metals into gold.

When an alchemist dipped a piece of iron into a solution of copper vitriol i.e., copper sulphate, the iron was immediately covered by a layer of metallic copper. This apparent transmutation of iron into copper is represented in modern terms by the equation:

\[ \text{Cu}^{2+} + \text{Fe} \rightarrow \text{Cu} + \text{Fe}^{2+}. \]

The major question, however, that remained unanswered was: how can the transmutation of iron or copper into gold be effected? Gold, the most noble of all metals, was insoluble in all acids and alkalies known at that time. The discovery of aqua regia by the Arab alchemist Jabir Ibn Hayyan (720-813 A.D.) and its ability to dissolve gold introduced a new technology for the extraction of gold from its ores. Aqua regia or royal water (the water that dissolves the king of metals) is a mixture of hydrochloric and nitric acids, neither of which alone can attack gold. In modern terms the action of the mixture is due to chlorine and nitrosyl chloride:

\[ 3 \text{HCl} + \text{HNO}_3 \rightarrow \text{Cl}_2 + \text{NOCl} + 2 \text{H}_2\text{O}. \]

Chlorine was discovered and isolated by the Swedish chemist Carl Wilhelm Scheele (1742-1786) in 1774 who also noted that it attacked all metals. This was first applied to the recovery of gold from its ores in 1851 by Karl Friedrich Plattner (1800-1858) of Freiberg using aqueous solutions of the gas. Although the concept of the transmutation of metals was abandoned by the chemists of the nineteenth century, metallic iron was used to displace gold from the chloride leach solution by a reaction similar to the above:

\[ \text{Au}^{3+} + \text{Fe} \rightarrow \text{Au} + \text{Fe}^{3+}. \]

The Blue Acid and its Salts

Chemists of the eighteenth century were occupied with a number of blue coloured compounds obtained by heating dried blood with potash (K₂CO₃) and then treating the aqueous extract with iron vitriol (FeSO₄). The precipitate so-obtained yielded an intense blue pigment. Its accidental discovery was made in Berlin in 1704 by J.C. Dippel and H. Diesbach and hence the pigment became known as Berlin blue. In the English literature it became known as Prussian blue to refer to Prussia, the former state of northern Germany. This new blue pigment immediately displaced the naturally occurring ultramarine blue because it was much cheaper. It was the first artificially manufactured pigment; its discovery opened up a new field of chemistry — the chemistry of cyanogen compounds.

The French chemist Pierre Joseph Macquer (1718-1784) found in 1752 that when Berlin blue was boiled with alkali, iron oxide separated out and the remaining solution when concentrated by evaporation yielded yellow crystals, now known as potassium ferrocyanide. In 1782 Scheele heated the blue pigment with dilute sulphuric acid and got an inflammable gas that dissolved in water and reacted acidic with litmus paper; he called it Berlin blue acid or simply the blue acid (Blausäure). In 1811 the French
chemist Joseph Louis Gay-Lussac (1778-1850) liquified the gas (boiling point 26°C) and determined that its composition was HCN. When Berlin blue was heated with nitric acid a red violet compound was obtained which became known as prussiate in reference to Prussia. In 1822, Leopold Gmelin (1788-1853) in Germany prepared potassium ferrocyanide by passing chlorine into a solution of potassium ferrocyanide until it no longer gave a blue colour with a ferric salt; it crystallized from solution as deep red prisms.

As we know today, blood contains the red pigment hemin which is an iron chelate containing a porphyrin structure. The building units of this structure are alkylated pyrrol nuclei. On heating with potash, potassium ferrocyanide is formed; this is extracted by water and reacted with iron salt to form ferriferrocyanide:

\[ 3[\text{Fe(CN)}_4]^{3-} + 4 \text{Fe}^{3+} \rightarrow \text{Fe}_3[\text{Fe(CN)}_4]_2 \]  

The linkages of the iron atom in hemoglobin appear to be quite analogous to the linkages in the ferrocyanide ion (Fig. 3). When numerous new compounds of the blue acid became known, Greek terminology was introduced to name them. Kyanos in Greek (κύανος) means blue, hence the modern term cyanide was used for compounds derived from the blue acid. The oxidation of ferrocyanide by chlorine yields ferricyanide:

\[ 2\text{K}[\text{Fe(CN)}_4] + \text{Cl}_2 \rightarrow 2\text{K}[\text{Fe(CN)}_3] + 2\text{KCl} \]

while oxidation by nitric acid yields the prussiates in which a cyanide group is exchanged for an NO group:

\[ \text{K}[\text{Fe(CN)}_4] + \text{NO} \rightarrow \text{K}[\text{Fe(NO)}(\text{CN})_3] + \text{KCN} \]

Surprisingly enough, the toxic action of cyanides was known by the ancient Egyptians. Compounds of hydrogen cyanide with sugars, known as glucosides, occur in nature, for example, in the leaves and seeds of the peach tree. A solution made by boiling peach leaves in water was used by the ancients Egyptians as a poison. The reaction taking place, for example, during the hydrolysis of cyanogenetic glycoside amygdalin (which is the bitter constituent of the kernels of the fruit pits of the prune family and which is present in high concentration in the bitter almonds) can be represented by:

\[ \text{C}_6\text{H}_5\text{C}(-\text{OH})\text{CN} + \text{H}_2\text{O} \rightarrow 2\text{C}_6\text{H}_5\text{O}_2\text{H} + \text{C}_3\text{H}_7\text{CHO} + \text{HCN} \]

Amygdalin Glucose Benzaldehyde cyanhydrin

Readings in Historical Metallurgy
It is remarkable that while HCN and its salts are extremely poisonous, cyanates, thiocyanates, ferro-, and ferricyanides are not.

Potassium ferrocyanide became a commercial product about 1825. It was obtained as yellow crystals by evaporating the solution obtained by leaching the product of calcining blood with potash. It found two major applications:

1. Case-hardening of iron, i.e. immersion of iron in the fused salt to produce a hard surface layer on mild steel.
2. Preparation of light-sensitive paper for reproduction. Today, when we speak of “blue prints” we are unknowingly referring to Prussian blue. It was found in 1842 that paper treated with a solution containing potassium ferrocyanide and a ferric salt (ferric ammonium citrate) becomes blue when exposed to light owing to the reduction of the ferric salt and subsequent formation of Prussian blue. Paper sensitized in this way was used for the reproduction of engineering drawings.

Production of potassium cyanide followed by a few years the introduction of potassium ferrocyanide onto the market. By about 1834 it was being produced exclusively by fusing potassium ferrocyanide with potash:

$$K_{2}[Fe(CN)_4] + K_2CO_3 \rightarrow 6 KCN + FeCO_3$$

The product was only 70% pure being contaminated by much carbonate and cyanate.

In 1835 the formation of HCN in the blast furnace became known, and in 1843, the existence of cyanogen compounds in coal gas was discovered*. Coal gas was produced in large amounts for heating and illumination purposes; it was the equivalent of today's electricity. Because of their poisonous action, it was necessary to remove cyanogen compounds; this was accomplished by absorption on a mixture of ferrous and ferric hydroxides. The repeated use of the iron hydroxides caused the cyanogen compounds to accumulate in them in a relatively concentrated form. By reaction with a ferric salt solution, Prussian blue could be formed:

$$6 HCN + Fe(OH)_3 \rightarrow H_2[Fe(CN)_4] + 2 H_2O$$
$$3 H_2[Fe(CN)_4] + 4 Fe^{3+} \rightarrow Fe_3[Fe(CN)_6] + 12 H^+.$$  

This process, developed in England, soon competed with the German process for two reasons:
1. It was cheaper because the raw material was a waste product.
2. Nitrogenous organic matter became less available because of their increased use as fertilizers.

Potassium cyanide was mainly used to prepare the baths necessary for electroplating gold and silver by E.R. and H. Elkington’s Process (1840)**. The impure product was replaced in 1876 by the white crystalline product from the Erben Meyer Process; this contained a mixture of KCN and NaCN with a total potassium cyanide equivalent of about 98%. It was obtained by fusing potassium ferrocyanide with metallic sodium:

$$K_2[Fe(CN)_4] + 2 Na \rightarrow 4 KCN + 2 NaCN + Fe.$$  

In 1900, a pure white crystalline product containing 97%-98% NaCN was produced by the Castner Process in which sodium was fed from the top of a bed of red hot charcoal through which a current of dry ammonia was passed from below:

$$2 Na + 2 C + 2 NH_3 \rightarrow 2 NaCN + 3 H_2.$$  

Later, better methods were used to prepare sodium, potassium, and calcium cyanides independent of ferrocyanide; the most important of these methods were based on the use of atmospheric nitrogen, leading eventually to the production of synthetic fertilizers. For example, Adolf Franck and Nikodem Caro in 1895-98 in Germany reacted calcium carbonate with nitrogen at high temperature to produce calcium cyanamide:

$$CaCO_3 + N_2 \rightarrow CaCN_2 + C.$$  

The cyanamide-carbon mixture was then either mixed with NaCl as a flux and melted in electric furnaces to form calcium cyanide:

$$CaCN_2 + C \rightarrow Ca(CN)_2,$$

or hydrolyzed with water to form ammonia (needed to make fertilizers):

$$CaCN_2 + 3 H_2O \rightarrow 2 NH_3 + CaCO_3.$$  

Many other processes were developed that contributed considerably to an understanding of the chemistry of cyanides, cyanogen, thiocyanates, cyanate, cyanamides, amides, nitriles, guanidines, and other related compounds.

The preparation of Prussian blue, an inorganic compound, from typically organic matter like blood occurred many years before Friedrich Wöhler (1800-1882) in Germany made the reverse operation in 1828: preparing a typically organic compound, urea, from inorganic matter, ammonium cyanate:

$$NH_2OCN \rightarrow (NH_2)_2CO.$$  

**The Cyanidation Process**

The cyanidation process as patented by MacArthur **et al.** involved two steps.
1. Dissolution of gold from ores by an alkali cyanide solution
2. Precipitation of gold from the leach solution by metallic zinc.

**Dissolution of Gold**

The dissolving action of cyanide solution on metallic gold was known as early as 1783 by Scheele who experimented with the blue gas he discovered. The solution obtained by dissolving gold in cyanide solution was used by Elkington to prepare the bath necessary for electroplating gold***. The dissolution reaction was studied by noted chemists of the time: Bagration** in Russia (1844), Elsner** in Germany (1846), and Faraday** in England (1857). It remained, however, for MacArthur in 1887 to apply this knowledge to gold ores. Faraday described the dissolution mechanism as follows: “Air voltaic circles are formed in these cases, and the gold is dissolved almost exclusively under their influence. When one piece of gold leaf was placed on the surface of a solution of cyanide of potassium, and another moistened on both sides was placed under the surface, both dissolved; but twelve minutes sufficed for the solution of the first, while more than twelve hours were required for the submerged piece. In weaker solutions, and with silver also, the same results were obtained; from sixty to a
hundred fold as much time being required for the disappearance of the submerged metal as for that which, floating, was in contact both with the air and the solvent."

When the process was applied industrially research was initiated in many universities[22]. The most important was that by Boeländer[23] in 1896 whose contribution was two fold:

1. He confirmed that oxygen was necessary for the dissolution as claimed by Elsner and Faraday and doubted by MacArthur.
2. He discovered that hydrogen peroxide was formed as an intermediate product during the dissolution of gold, and therefore proposed the following two equations to represent the dissolution reaction:

\[ 2 \text{Au} + 4 \text{NaCN} + \text{O}_2 + 2 \text{H}_2\text{O} \rightarrow 2\text{Na[Au(CN)]}_2 + \text{NaOH} + \text{H}_2\text{O} \]

Later work however, showed that the first equation was the most important one during leaching[23].

The action of cyanide solution on gold remained a mystery for a long time for three reasons:

a. It was difficult to understand why gold, the most noble of all metals, which could not be attacked by any strong acid except hot concentrated aqua regia, could be dissolved at room temperature by a very dilute solution of NaCN or KCN of 0.01% - 0.1%.

b. A strong solution of NaCN was found not to be better than a dilute solution. The question always asked was why the effect of NaCN concentration on gold always followed the pattern shown in Figure 4 whereas in the case of other metals dissolving, for example, in an acid, the metal dissolved faster the more concentrated the acid.

c. It was not obvious why oxygen was necessary even though alone it had no action whatsoever on gold.

Many theories were put forward, all of which later proved to be wrong. For example, it was thought that cyanogen gas, (CN)_2, or sodium cyanate, NaCNO, might be formed by the action of oxygen on NaCN in the so-called nascent state, and that these attacked the gold. Direct experiments, however, using cyanogen or sodium cyanate proved that they did not have the slightest dissolving action on gold. An answer to the above questions was put forward only in the last thirty years as will be discussed below.

The experimental work of G.A. Deitz and J.M. Halpern[24] in Canada in 1953 resulted in a solution to the mystery of why a dilute NaCN solution was as good as a concentrated one for the dissolution of gold and silver. In this work the effect of cyanide concentration was studied in two series of leaching tests done at different oxygen pressures (Fig. 5). An interesting observation was made: at low cyanide concentration, oxygen pressure had no effect on the rate while at high cyanide concentration where the rate of dissolution was independent of cyanide concentration, the reaction rate was dependent on oxygen pressure. In other words, there was a relation between the cyanide concentration and the oxygen pressure; the reaction rate being controlled by either one or the other but not both at the same time.

According to the stoichiometry of the process, 4 moles of cyanide are needed for each mole of oxygen present in solution. At room temperature and atmospheric pressure, 8.2 mg oxygen are present in 1 litre of water. This corresponds to \(0.27 \times 10^{-3}\) mole/l. Accordingly, the sodium cyanide concentration (molecular weight of NaCN = 49) should equal to 4 \(\times 0.27 \times 10^{-3} \times 49 = 0.05\) g/l or approximately 0.01%. This was confirmed in practice, thus explaining why a dilute cyanide solution was capable of dissolution. However, it did not explain the overall effects of cyanide concentration and oxygen pressure. Experiments done in 1947 by P.F. Thompson[25] in Australia showed clearly that the cyanidation process was actually a corrosion phenomenon in which cathodic and
anodic zones are formed on the surface of gold. He used sodium cyanide solutions, free from air, to which some gelatin was added. When the solution solidified, small spheres of gold were introduced in the jelly and their dissolution was studied by allowing a stream of oxygen to diffuse in one direction. In this way convection currents were eliminated. He observed that the gold spheres were corroded on the side away from the source of oxygen. This illustrated the formation of local cells in which parts of the gold acted as anodes and other parts acted as cathodes at which oxygen was reduced.

Combining Deitz and Halpern’s experimental data with Thompson’s demonstration of the electrochemical dissolution of gold it became possible to formulate the cyanidation process mathematically in the form of the rate equation:

$$\text{Rate} = \frac{2A \rho_{\text{CN}} \rho_{\text{O}_2}[\text{CN}][\text{O}_2]}{\delta[\rho_{\text{CN}}[\text{CN}] + 4 \rho_{\text{O}_2}[\text{O}_2]]}$$

where $[\text{CN}]$ and $[\text{O}_2]$ are the concentrations of cyanide and dissolved oxygen, respectively, $\rho_{\text{CN}}$ and $\rho_{\text{O}_2}$ are the diffusion coefficients of cyanide and dissolved oxygen; $A$ is the surface area of the metal in contact with the aqueous phase, and $\delta$ is the thickness of the boundary layer. From this equation it follows that at low cyanide concentration, the first term in the denominator may be neglected in comparison with the second, so that the equation simplifies to:

$$\text{Rate} = \frac{A \rho_{\text{CN}}[\text{CN}]}{\delta} = k_1[\text{CN}],$$

in agreement with the observation (Fig. 5) that at low cyanide concentration the rate of dissolution depends only on the cyanide concentration. In the same manner, it follows from the same equation that at high cyanide concentration, the second term in the denominator may be neglected in comparison with the first, and the equation simplifies to:

$$\text{Rate} = \frac{2A \rho_{\text{O}_2}[\text{O}_2]}{\delta} = k_2[\text{O}_2].$$

This coincides also with the experimental fact (Fig. 5) that at high cyanide concentration the rate of dissolution depends only on the oxygen concentration.

The fact that cyanide solutions are poisonous and that disposal of waste solutions necessitates extensive treatment provided an incentive to look for other solvents that are less toxic. The dissolving action of thiourea on gold was known at the same time as was the action of cyanide solution; the Russian chemist I.N. Paksin promoted its use[2]. Thiourea can dissolve gold from acid solution much faster than a cyanide solution. The process, however, was not widely adopted because the reagent is more expensive, it undergoes slight oxidation, and an efficient precipitant of gold from such solution has not yet been found. It is interesting to note that while urea is an isomer of ammonium cyanate (Wöhler’s reaction; preparation of an organic compound from an inorganic compound), thiourea is an isomer of ammonium thiocyanate from which it can be obtained by heating at a temperature slightly above its melting point (159°C):

$$\text{NH}_2\text{SCN} \rightarrow (\text{NH}_2)_2\text{C} + \text{S}.$$

### Precipitation of Gold

By analogy with the “transmutation” of iron into copper, MacArthur used zinc shavings to precipitate gold from the cyanide solution. The process became more efficient when zinc dust was introduced by C.W. Merrill in 1904 and further improved when T.B. Crowe removed air from the solution by passing it through a vacuum tank before introducing the zinc. This new technique developed in U.S.A., later became known as the Merrill-Crowe Process. However, after World War II a surplus of charcoal became available at prices that made its use cheaper than that of zinc for precipitating gold. The adsorption power of charcoal was already known, having been discovered by the German-born Russian chemist Tobias Lowitz* about 1790. Activated charcoal was being used to recover gold from the chlorine leaching process[2]. It is important, however, to realize that the two mechanisms are different. Metallic gold is precipitated on charcoal from the chloride solution by the oxidation-reduction reactions:

Oxidation: $\text{C} + 2 \text{H}_2\text{O} \rightarrow \text{CO}_2 + 4 \text{H}^+ + 4 \text{e}^-$

Reduction: $\text{Au}^{3+} + 3 \text{e}^- \rightarrow \text{Au}$

Overall reaction: $4 \text{Au}^{3+} + 3 \text{C} + 6 \text{H}_2\text{O} \rightarrow 4 \text{Au}^+ + 3 \text{CO}_2 + 12 \text{H}^+$

In cyanide solution, the gold cyanide complex is physically adsorbed on the charcoal. In contrast to the Merrill-Crowe Process, oxygen is necessary here, as it is reduced on the surface of charcoal by the reaction:

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* Tovil Egorovich Lovits (1757-1804)
Recent Trends

Scheele’s “blue gas” became an essential industrial reagent. It entered into the synthesis of dyestuffs, e.g. indigo, of synthetic resins, e.g. metacrylic acid esters (Plexiglass), of synthetic fibres, e.g. acrylic nitrile (Orlon, Buna N rubber), and of adipic acid (Nylon). It is now produced in large tonnages (Table 1), by processes such as the BMA Process (Blue gas from Methane and Ammonia):

\[ \text{CH}_4 + \text{NH}_3 \xrightarrow{300^\circ \text{C}} \text{HCN} + 3 \text{H}_2. \]

The cyanidation process has remained practically unchanged since its discovery, but tremendous engineering developments have occurred, making it very efficient\(^{26}\). The developments include the introduction of heap leaching, the application to the so-called refractory ores, the use of activated charcoal technology, and the increasing attention paid to the abatement of pollution, as now discussed.

Heap Leaching

The increased price of gold in the 1970s made possible the leaching of old tailings containing as little as 1 ppm gold thanks to the newly developed heap leaching technology that was developed for the leaching of low grade copper and uranium ores. To improve the percolation of the leach

* Uncomplexed metal ions in solution are precipitated as hydroxides from neutral or slightly acidic dilute solutions on the surface of charcoal due to the OH ions liberated, while the anions are attracted to the positively-charged charcoal surface. In this case the reaction is described as chemisorption and is irreversible.

Solution in the bed, the fine tailings were sometimes agglomerated by slurrying with water and a small amount of cement which was allowed to set. The heaps were then constructed of the hard agglomerates and leached with the alkaline cyanide solution.

Treatment of Refractory Ores

A problem in gold metallurgy was the treatment of gold locked up in pyrite crystals or in organic matter and therefore unresponsive to cyanidation. Roasting followed by leaching was an expensive proposition. A hydrometallurgical approach proved to be a successful solution for both types of ores. The Homestake Company now treats a pyrite ore — water slurry in autoclaves at high temperature and oxygen pressure at the McLaughlin gold mine in California\(^{30}\). Three horizontal autoclaves are used, each being 16 m long and 4 m diameter, operating at 160-180°C and 2000 kPa, with a retention time of 2 hours (Fig. 6). After this treatment, the ore is then suitable for cyanidation.

Gold ores containing carbonaceous material are difficult to treat, not only because part of the gold is tied up with the organic matter but also because dissolved gold is adsorbed on the carbon present in the ore and therefore reports in the tailings. This problem was solved by aqueous oxidation using chlorine\(^{31}\). Some of the gold may be solubilized by the chlorine water but the major function of the controlled chlorination is to oxidize organic matter before cyanidation. A plant at Carlin, Nevada, uses this technology. Another solution to this problem was found by using the carbon-in-leach process as described below.

Cyanidation Under Pressure

Cyanidation under high oxygen pressure has recently been used commercially at the Consolidated Murchison Mine near Gravellote in South Africa. The process has been developed by Lurgi in Germany using tube auto-
claves 1.5 km long and 5 cm inner diameter. Leaching is conducted at ambient temperature but at about 5 MPa oxygen pressure. As a result, residence time is only 15 minutes at 85% recovery. It should be noted that at high oxygen pressure, a high cyanide concentration must be used to achieve high reaction rates. This is evident by extrapolation from the dissolution curves shown in Figure 5. In practice the leach solution is 0.2%-0.5% NaCN. Although cyanide solutions are susceptible to oxidation, the short residence time renders this drawback negligible.

**Carbon-in-pulp and Carbon-in-leach**

In the early 1950s, interest in the application of activated carbon was revived when a procedure was developed for the elution of gold and silver adsorbed on carbon granules. This allowed re-use of the carbon, and the utilization of the process as a concentration-purification step[10], leading to wide-spread application of this technology.

Adsorption of gold on activated charcoal was usually carried out in columns charged with granular charcoal through which the leach solution was allowed to percolate. When the beds were saturated they were desorbed, and the depleted carbon was washed, dewatered and then activated. It was necessary that leach solution be clear to prevent blocking of the bed with suspended matter. When slurry ores are leached, filtration to prepare clear solutions becomes an expensive operation; for this reason the so-called carbon-in-pulp process was developed[10]. In this process the leaching pulp is agitated in tanks with charcoal pellets. When adsorption is complete, the pulp is screened to separate the gold-laden pellets for washing and desorption. The process has thus the advantage of eliminating an expensive filtration operation.

The carbon-in-leach process is another variation that was developed to circumvent problems faced when leaching gold ores containing carbonaceous material, i.e. low recovery because of the adsorption of the gold cyanide complex on the organic matter which then reports in the tailings. In this process the carbon pellets are added during leaching of the ore so that any gold dissolved is immediately retained by the pellets and not by the carbonaceous matter in the ore[10]. This procedure takes advantage of the aerating necessary for leaching since adsorption on activated carbon, like leaching, requires oxygen in solution. Furthermore, it has the advantage of eliminating a filtration step similar to the carbon-in-pulp process since the gold-laden pellets can be removed by sieving.

**Electrowinning**

When a purified and concentrated solution of gold cyanide solution was obtained by the activated charcoal adsorption-desorption process, an electrowinning process was developed, making use of a cathode of large surface area made of steel wool (Figs. 7 and 8). The gold-laden steel wool was then melted to get an ingot. The process proved to be economical.

**Environmental Control**

It is always necessary to bleed some of the leach solution to maintain a tolerable level of impurities since the bulk of the cyanide solution is recycled. The bleed solution, however, is poisonous and must be destroyed before disposal in the environment. Many processes have been developed, including oxidation, acidification, adsorption, as well as biological processes[16-31].

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**Summary and Conclusions**

The basic information for the cyanidation process invented one hundred years ago was already laid down by chemists of the eighteenth and nineteenth centuries. It remained for the chemist metallurgist John Stewart MacArthur to apply this knowledge to the leaching of gold ores. The process is a milestone in modern hydrometallurgy; it is used today in its original form practically unchanged. The blue gas originally prepared from Prussian blue which in turn was prepared from blood and waste organic matter is now an essential reagent in the organic chemical industry as well as in gold extraction.

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Canadian Metallurgical History
An overview of the early history of the metallurgical industry in Canada

P. Tarassoff

Introduction
Extractive metallurgy is concerned with the winning of metals from their ores and concentrates. It is therefore not unexpected that extractive metallurgy in Canada has always been closely linked with mining. The discovery and exploitation of new orebodies has provided the impetus for the building of many of our smelters and refineries — past and present — and for the development of new metal extraction processes. The purpose of this paper is to provide a brief overview of those early metallurgical developments, and to highlight the continuing expansion and diversification of the Canadian metallurgical industry.

Iron Making
Extractive metallurgy in Canada began with iron making which provided the numerous articles of cast and wrought iron needed by the early settlers. The first iron works was established in 1729 at Les Forges du Saint-Maurice, near Trois-Rivières, Québec, to exploit deposits of bog iron ore. This works continued in operation until 1883 by which time its basic technology — the charcoal blast furnace — had become obsolete. In the intervening years many other iron works sprang up in eastern Canada; Figure 1 shows the location and dates of operation of many of these early works. Most of the early iron works were short-lived, and all were supplanted by the large integrated steel companies established during the early 1900s: Algoma Steel Corporation Limited (1901); Dominion Iron and Steel Company, forerunner of Sydney Steel Company (1901); Steel Company of Canada, now Stelco Incorporated (1910); and Dominion Steel Limited, now Dofasco Incorporated (1912).

Copper Smelting and Refining
Canada’s first copper smelter, employing a Welsh reverberatory furnace and three Swansea furnaces, had a brief existence between 1848 and 1850 at Bruce Mines, Ontario. Twenty years later, Bruce Mines was the scene of one of the first hydrometallurgical operations in Canada when a plant was built to treat copper ore by the roast-leach-precipitation Henderson process developed in the United States. This process was also used in 1872-1873 at the Huntingdon Mine in the Eastern Townships of Québec. This area was the centre of a thriving copper mining industry, and several small smelting furnaces were in operation at various times between 1862 and 1907. Accounts of the history of the Eastern Townships copper industry contain reference to closures due to a drop in the price of copper, the imposition of heavy import duties by the United States, and problems with SO₂ emissions — all familiar issues today.

By the turn of the century, the copper industry had moved to the Sudbury Basin, where copper had been discovered in association with nickel, and to British Columbia. The Sudbury copper-nickel ores proved to be a real challenge to extractive metallurgists. At first the ore was shipped to smelters in the United States where unsuccessful attempts to produce copper by standard copper smelting practices led to the development of the Oxford process for separating copper from nickel in copper-nickel matte. In order to save freight costs, the Canadian Copper Company, a forerunner of the International Nickel Company, decided to build a smelter at Copper Cliff to reduce the ore to matte before shipment. It went into operation in 1888, and although rebuilt many times it is the oldest operating smelter in Canada. This was a period when many mining companies were active in the Sudbury Basin, and most built their own smelters. The location of these and later smelters is shown in Figure 2. All of the early smelters produced matte which was shipped to refineries in New Jersey, Ohio, England and Wales. Electrolytic copper was not produced in the Sudbury Basin until 1930, when the Ontario Refinery Company Limited commenced operation of a refinery at Copper Cliff.

A large copper industry had also developed in British Columbia, as is indicated in Figure 3. Five major copper smelters had been built between 1896 and 1902, all of which smelted roasted copper ore in blast furnaces. The significance of these smelters can be judged from the fact that in 1913 the Granby smelter at Grand Forks (started up in 1900 by the Granby Mining and Smelting Company) had a smelting capacity of 4100 tons of ore per day and the Trail smelter (1886) of the Canadian Smelting Works, the forerunner of Cominco, had a capacity of 3000 tons. Both surpassed the 2500 tons smelting capacity of Copper Cliff smelter in Sudbury. The much smaller Hall Mines smelter (1896) appears to have been the first Canadian smelter to produce blister copper, using a reverberatory furnace to refine copper matte. The Granby smelter, however, which from 1902 to its closure in 1919 converted matte to blister copper in Great Falls type converters, must be accorded the distinction of being the first large integrated copper smelter in Canada. British Columbia was also the scene of the first electrolytic copper refinery in Canada, which went into operation at Trail in 1916. The last of the early British Columbia smelters, at Anyox, shut down in 1935.

By this time, the copper industry was again centred in eastern Canada with the startup of the Noranda Mines Limited smelter at Noranda, Québec in 1927, and the Hudson Bay Mining & Smelting Company Limited smelter at Flin Flon, Manitoba, in 1930. In 1931, Noranda Mines in partnership with the Nichols Copper Company of New York (which later became Phelps Dodge Corporation) and British Metal Corporation brought an electrolytic copper refinery into operation at Montreal East; Canadian Copper Refiners Limited refined copper from both the Noranda and Hudson Bay smelters, and was to become one of the largest copper refineries in the world.

Nickel
As noted above, copper-nickel ore was being smelted in...
the Sudbury Basin by 1888. The earliest attempt to smelt nickel in Canada, however, took place some ten years earlier in the Eastern Townships of Québec. The Orford Nickel and Copper Company was established in 1878 to exploit a nickel deposit in Orford Township. The deposit proved to be neither rich nor large, and attempts to smelt the ore in a reverberatory furnace were unsuccessful. The venture was abandoned, but the company continued on as the Orford Copper and Sulphur Company with a copper mine near Lennoxville, Québec, and a smelter at Constable Hook, New Jersey. Ironically, its future lay in nickel. In 1886, it undertook to smelt what was ostensibly copper ore from Sudbury at its smelter in New Jersey. This led to the first recognition of nickel in the Sudbury ores, and culminated in the development of the Orford process which was put into practice at Constable Hook in 1891.

The first production in the Sudbury Basin of Bessemer matte from low-grade furnace matte, using a Manhès converter, was recorded at the short-lived Murray smelter of H.H. Vivian & Co. in 1890. The next year a Bessemer converting plant was added to the Canadian Copper Company’s Copper Cliff smelter; the Bessemer matte was shipped to the Orford Copper Company’s Constable Hook refinery, and also, after 1892, to the Canadian Copper Company’s own refinery which had been built near Cleveland, Ohio. Here, refined nickel was produced by roasting nickel sulphide after its separation from copper sulphide, and then reducing the nickel oxide.

In what eventually proved to be a twist of fate, the Canadian Copper Company evaluated the newly invented Mond nickel carbonyl refining process in 1895 and rejected it as being unsuitable to “the rigorous Canadian climate”. The earliest commercial attempt to refine nickel electrolytically in Canada was made in a plant built near Hamilton, Ontario, by the Hoepfner Refining Company in 1900. The Hoepfner process was unsuccessful, and the plant closed in 1902. This process had also been evaluated and rejected by the Canadian Copper Company.

The year 1902 was a landmark in the history of Canada’s nickel industry for it saw the merger of the Canadian Copper Company with the Orford Copper Company and several other companies concerned with the production of nickel to form the International Nickel Company, today’s Inco.

The International Nickel Company continued to ship Bessemer matte to the Orford refinery until after the First World War. With the outbreak of the war, nickel became a strategic metal and pressure was applied by the Canadian government for the establishment of a nickel refinery in Canada. In 1918, International Nickel put Canada’s first nickel refinery into operation at Port Colborne, Ontario, employing the Orford process and roast-reduction. In the same year, construction was started on Canada’s first successful electrolytic nickel refinery at Deschenes, Québec, near Ottawa, by the British American Nickel Corporation. This refinery employed the Hybinnette process and operated briefly between 1920 and 1922. Its timing was unfortunate since the nickel industry was in extreme depression. The British American Nickel Corporation, a victim of this depression, went into bankruptcy in 1924. Its assets, including the rights to the Hybinnette process, were acquired by the International Nickel Company which promptly adopted the process at its Port Colborne refinery. Interestingly, N.V. Hybinette had been employed at the Orford Copper Company’s nickel refinery some years before inventing the process which bore his name, and just a few years before this company became part of International Nickel.

During the decades that followed, the foundation was laid for Canada’s modern nickel industry with the start-up of the Falconbridge Nickel Mines Limited smelter in the Sudbury Basin in 1930, the Sherritt Gordon Mines Limited nickel refinery at Fort Saskatchewan, Alberta, in 1954, and Inco’s Thompson smelter at Thompson, Manitoba, in 1961. The Sherritt Gordon refinery employed a novel ammonia pressure leaching process, and was the world’s first commercial hydrometallurgical operation to extract nickel from nickel sulphide.

Lead Smelting

As was the case for copper and nickel, much of the early smelting and refining of Canadian lead ores was carried out in the United States. What was probably Canada’s first lead smelter was built in 1880 at the Frontenac Mine near Kingston, Ontario, but it is not certain that this smelter actually operated. About the same time, a small smelting furnace was installed at the Leadvale Mine near Truro, Nova Scotia, but its operation is also uncertain.
The first major lead smelting plants were all in British Columbia. The Vancouver smelter, located in the City of Vancouver, was started up in 1883, but operated for only a few hours before technical problems forced its closure. The Revelstoke, British Columbia, smelter commenced operation in July 1891, and operated intermittently for less than a year before shutting down because of a shortage of feed; this operation likely was Canada’s first “successful” lead smelter. Other lead smelting facilities were later established at Pilot Bay (1895) and Nelson. The Hall Mines smelter at Nelson started out in 1896 as a copper smelter. A decline in the availability and quality of copper ore, and the discovery of rich silver-lead deposits in the nearby Slocan mining district created a demand for lead smelting capacity, and resulted in a decision in 1900 to establish a custom lead smelting business. One of the two copper blast furnaces was converted to a lead blast furnace. Copper smelting ceased entirely in 1902, and until its closure in 1907, the Hall Mines smelter operated as a lead smelter.

Like the Hall Mines smelter, the Trail copper smelter of the Canadian Smelting Works also responded to the growing need for regional lead smelting capacity, and in 1899 began smelting lead ores. By 1901, it was operating three lead blast furnaces in addition to three copper blast furnaces. At first, the lead bullion was softened and sent to the United States for refining and silver recovery. In 1902, Trail became the site of the world’s first commercial production of electrolytic lead. The process had been developed and patented by the American Betts in 1901, and the Trail smelter had purchased the Canadian rights.

**Zinc Production**

Zinc was associated with many of the silver-lead deposits in British Columbia. In the beginning it was essentially valueless because the shipment of zinc ore to the nearest zinc smelters, in Kansas, was prohibitively expensive. This led to the construction of Canada’s first commercial zinc plant, employing the Belgian horizontal retort process, at Frank, Alberta; it treated zinc ores from the Slocan mining district and operated for the brief span of 1905-1906. An experimental zinc plant based on electric furnace reduction was operated intermittently between 1908 and 1913 at Nelson, British Columbia. The failure of both these ventures, for economic and technical reasons, probably dissuaded other companies from adopting carbothermic reduction processes and helped to launch Canada early on the electrolytic process route.

Although the basis for the electrolytic zinc process had been substantially worked out by the Frenchman Létrange around 1880, its successful commercial implementation dates to just before the First World War when development work was undertaken almost simultaneously by the Consolidated Mining and Smelting Company at Trail, British Columbia, and the Anaconda Copper Company at Anaconda, Montana. Commercial electrolytic zinc production began at Anaconda late in 1915, and at Trail in 1916, so that Canada had an almost “first” in this important technology.

**Gold**

Much of Canada’s early gold production came from placer deposits, from the first discovery on the Chaudière River in Québec (1823), the Fraser (1858), Cariboo (1859) and other discoveries in British Columbia, and the Klondike (1896) in the Yukon. The gold was recovered by gravity methods, sometimes augmented by amalgamation. Lode gold ore was treated in much the same way after being finely crushed in stamp mills. Two of the earliest lode gold discoveries in Canada are of particular interest because of their association with new metallurgical processes: those made in Nova Scotia in 1860, and at Deloro, Ontario, in 1868.

The fine gold in the Nova Scotia ores was not efficiently recovered in stamp mill-gravity circuits, and efforts to treat the tailings by chemical methods were unsuccessful. The development of the MacArthur-Forrest cyanide leaching process in Scotland in 1887-1888, the basis of modern cyanide leaching, and its successful application in New Zealand and South Africa soon led to its being tried in Nova Scotia. Several plants were built and operated with varying degrees of success; one of these built in 1903 at the Mic-Mac mine, near Leipsigas, is among the first technically and economically successful cyanidation plants in Canada. Other early cyanide leaching plants were operated at the Regina mine (1897) in the Lake of the Woods district of Ontario, and at the Doratha Morton mine (1898) in British Columbia. The quick adoption of cyanide leaching in Canada was timely; silver discoveries at Cobalt (1903), and gold discoveries at Porcupine (1909), Kirkland Lake (1912) and elsewhere were all to benefit.

The gold at Deloro was intimately associated with ar-
senopyrite and, as reported in the 1891 report of the Ontario Bureau of Mines, “notwithstanding that various methods were tried and large sums were wasted on new experiments, conviction slowly settled down on the minds of capitalists, miners and metallurgists alike that there was no money to be made in mining or milling the mispickel ores of Hastings”. The solution to the problem was found in 1896 in the Sulman-Teed bromo-cyanide leaching process which was especially applicable to arsenical gold ores. A plant was built by Canadian Goldfields Limited not only to recover gold but also to produce arsenic trioxide by roasting the leach residue. The arsenic was an important by-product which at the time found a ready market.

It was soon recognized that the basic technology developed at Deloro could be applied to the treatment of the arsenical silver-cobalt ores at Cobalt, Ontario. In 1906, M.J. O’Brien acquired and modernized the Canadian Goldfields plant and operated it as the Deloro Mining and Reduction Company (later Deloro Smelting and Refining) using the newer Kirkpatrick-Kirkegaard process to extract silver. In 1914, the Deloro plant was commercially producing cobalt metal for the first time in Canada, and had acquired rights to manufacture and market the alloy stillite, one of the most important applications for cobalt.

Aluminum and Magnesium

The turn of the century also saw the establishment of a Canadian metal extraction industry not associated with mining. The invention in 1886 of a viable electrolytic reduction process for aluminum by the American, Hall, and the Frenchman, Héroult, laid the foundation for the modern aluminum industry. The first commercial aluminum plant in North America was built in 1887 by the Pittsburgh Reduction Company. In 1899, this company decided to build an aluminum plant at Shawinigan Falls, Québec, to take advantage of the available hydroelectric power; the plant went into operation in 1901 as the Northern Aluminum Company, the forerunner of Alcan. When the Shawinigan plant was shut down in 1945, it had the distinction of being the oldest existing aluminum smelter in North America; by that time it had been superseded by Alcan’s smelters at Arvida (1926) and Beauharnois. The first Bayer process alumina plant in Canada was put into operation at Arvida in 1936, ending dependence on imported refined alumina.

Shawinigan was also the site of the first commercial production of magnesium in North America. The Shawinigan Electro Metals Plant operated from 1915 to 1919, producing magnesium by the fused salt electrolysis of magnesium chloride. A significant Canadian development in the extraction of magnesium was the Pidgeon process. Developed as a result of wartime exigencies, the process was put into production by Dominion Magnesium Limited in 1942, near Haley, Ontario. An interesting footnote on magnesium is that in the 1940s Cominco considered producing magnesium at Trail, using an electrolytic cell which it had jointly developed with the Mathieson Chemical Corporation.

Uranium

Hydrometallurgy in Canada has had a long association with uranium. In 1930, rich silver-uranium ore was discovered at Great Bear Lake in the Northwest Territories. Within two years Eldorado Gold Mines, later Eldorado Resources, had built a leach plant at Port Hope, Ontario, to extract radium. At that time uranium had few applications, but this changed dramatically with the discovery of uranium fission, and the resulting wartime activities. The first modern uranium acid leaching plant in North America was placed in operation at the Eldorado mine in 1952. One organization which played a key role in uranium extraction metallurgy in Canada was the Mines Branch, later CANMET. It was responsible for developing both the radium extraction process used at Port Hope in 1932, and the uranium extraction process used twenty years later at the Eldorado mine.

Froth Flotation

The froth flotation process, invented in the United States, had an enormous influence on the economics of mining and metals extraction after the turn of the century. The first flotation plant in Canada operated at Rossland, British Columbia, in 1904. Flotation was widely adopted in Canada from 1905 onwards. The flotation process meant that for the first time extractive metallurgists could deal with concentrates rather than ores, greatly simplifying metals extraction and paving the way for pyrometallurgical and hydrometallurgical processes which would not have been possible with ores.

The Role of Universities

The early historical record of the metallurgical industry in Canada contains many references to university professors. This was a time when operators were given to a great deal of experimentation on a production scale, and professors were often called in for advice. Metallurgy was then associated with chemistry and mining; departments of metallurgy were not established until much later. For example, one of the early Canadian extractive metallurgists, Dr. T. Sterry Hunt, who developed the Hunt-Douglas process for leaching oxidized copper ores in collaboration with Dr. James Douglas in 1875, was a professor of chemistry and mineralogy at both Université Laval and McGill University. Professor S.F. Kirkpatrick of the School of Mining at Queen’s University was a co-developer of the Kirkpatrick-Kirkegaard process employed at the Deloro smelter. The School of Mining was established in 1893 to provide instruction in physics, chemistry, geology, mineralogy and mining engineering. In the early years, ore dressing and extractive metallurgy received the most attention in the curriculum, and it was not unusual to find university laboratories equipped with rather large ore dressing and smelting equipment.

Of the many contributions of Canadian universities to metals extraction, one needs to be especially mentioned. This is the ammonia pressure leach process proposed by Professor Frank A. Forward of the University of British Columbia in 1948 and applied by Sherritt Gordon Mines at its Fort Saskatchewan nickel refinery. This Canadian achievement literally revolutionized hydrometallurgy.

Concluding Remarks

The early history of Canada’s metal extraction industry was characterized by a high degree of technology transfer. The metallurgists of the time became skilled at adopting and adapting processes developed elsewhere. Some of these processes were old and proven, whereas others were new and untried. There was a propensity toward experimentation on a large scale, and plants often were built before their technical or economic viability, or even the guarantee of an adequate supply of ore had been ascertained. As a result, there were many failures for every success. There were several reasons for all of this metallurgical activity. There was a strong incentive to reduce the cost of shipping ore to distant smelters in the United States and

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elsewhere, as well as to increase the return to the mine owner. Before the advent of froth flotation, the only recourse was to smelt the ores locally. More complex ores might defy treatment by the conventional processes of the day, and new processes had to be devised to separate the valuable metals or to increase recoveries. Many of the early smelters stopped short of producing refined metal and shipped matte or crude metal to refineries outside Canada. This practice also eventually gave way to domestic refineries, often with the adoption of new technology.

Within a few decades, around 1900, a strong metallurgical industry had been founded in Canada that encompassed iron and steel, copper, nickel, cobalt, lead, zinc, silver, gold and aluminum. With it, Canada's strength in metals extraction was firmly established, paving the way for the development of uniquely Canadian technology, a process that continues today.
On the early use of iron in the Arctic

Michael L. Wayman

Introduction

Archaeological and ethnographic work during the last century has shown clearly that metals have played a small but important role in prehistoric Arctic cultures. For at least the past several millennia both iron and copper have been used in the North, primarily for blades, projectile points, knives, awls, engraving tools, fishhooks, scrapers and axes. In addition they were used for joining purposes as staples and rivets. It is of great interest to enquire into the origins of these metals.

In this connection one begins with the observation that to date there has been no clear evidence for the smelting or melting of metals in the Arctic prior to the arrival of Europeans. This can be explained partly by the absence of traditional pyrotechnologies such as the high-temperature firing of ceramics and the heat treatment of bone and wood. Another factor which surely inhibited the development of smelting/melting operations is the relative absence of wood in the region. The production of charcoal for smelting requires large quantities of wood, to the extent that the desertification of a number of areas in the Old World is believed by some to have been at least partly caused by charcoal production. In the North this was obviously not possible. Thus the Arctic peoples were restricted to using native (naturally occurring) metals and metals obtained by various direct and indirect forms of contact with areas of the world where smelting was practiced. Both of these sources were important in the case of iron: we know that meteoric iron, telluric (native) iron and smelted wrought iron were all used in the Arctic. These three types of iron will be considered in turn.

Meteoritic Iron

Meteorites can be classified on the basis of their structures as stony meteorites (“chondrites” and “achondrites”), iron meteorites and the “stony irons” which contain a mixture of stony and metallic constituents (mainly “pallasites” and “mesosiderites”). In this article we are concerned only with the metallic iron meteorites. There is a long history of the use of “the metal from heaven” by many cultures in many parts of the world, including both north and South America. This is not surprising when it is realized that in 1975 some 530 identifiable iron meteorites were catalogued by the Danish metallurgist Vagn F. Buchwald and undoubtedly countless others have been made use of as objects of utility or of veneration over the past millennia.

Of major import for the inhabitants of the Arctic is the occurrence of large quantities of meteoritic material in the vicinity of Cape York, northwest Greenland. When this area was visited by a British exploratory expedition under Captain John Ross in 1818, iron-bladed tools and weapons were observed among the possessions of the region’s inhabitants, the “Polar Eskimos.” Typical of these were knives and harpoon heads made of bone or walrus ivory into which a slot had been grooved; into this slot an array of dime-sized iron flakes were wedged to create a blade and the assembly pinned by an iron rivet (two such artifacts are shown in Fig. 1). Upon enquiry, Ross was told that several blocks of iron existed in the vicinity, from which the flakes of metal had been obtained. In 1894 in the same area the Polar Eskimos showed to the American explorer Peary these blocks which turned out to be three meteorites, which were called “Womant” (3 tons), “Ahnighito” or “Tent” (30 tons) and “Dog” (0.4 tons). By that time these were no longer being used as sources of iron, presumably because of the easy access to smelted iron from trade sources. With considerable difficulty, if not adventure, Peary took the three meteorites to New York where they still reside in the American Museum of Natural History (Hayden Planetarium).

Since that time a number of other meteorite fragments have been located in the Cape York area including one weighing 20 tons which is now in the Copenhagen museum. The total mass of meteoric material recovered from the Cape York area to date totals some 58 tons. When it is realized that much of the area in which these meteorite fragments have been found is ocean, it is apparent that the original meteoroid likely had had a mass of at least several hundred tons prior to its disintegration in the earth’s atmosphere, and that other fragments likely exist on the ocean floor. It is not known when this meteorite fell occurred but the absence of impact structure in the vicinity of the fragments suggests that the meteorites may have originally landed on ice which subsequently melted away. The extent of corrosion of the fragments has been interpreted as indicating that the fall probably occurred more than 2000 years ago.

Although it is obviously possible, even likely, that other meteorites have fallen in the Arctic, and that these may have been used by Northerners, only two other meteorites have been reported from the Canadian Arctic — both in the Yukon. A meteorite named Akpahiqook discovered in 1914 on Ellesmere Island is believed to be another fragment from Cape York.

For purposes of archaeometry it is necessary to be able to identify an artifact as having been made from meteoric iron and this is normally accomplished on the bases of elemental analysis and microstructure. The chemical compositions of metallic meteorites have been well studied. These are invariable iron-nickel alloys containing between 5% and 60% nickel, however, of the 530 analyses reported in 1975 by Buchwald only 7 contained more than 20% nickel with the average being about 8%. The Cape York meteorites are typical of the average in this respect, with a nickel content of 7.8%; other elements present, also typical of iron meteorites, include 0.5% Co, 0.15% to 0.2% P, 0.02% C and 0.15% Cu.

The microstructures of iron meteorites as observed by conventional optical metallography are highly characteristic. At some point in their past histories the iron meteorites have slowly cooled from the high-temperature facencentred cubic solid solution phase; thermodynamics predicts that during slow cooling these iron-nickel alloys

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should transform to the body-centred cubic solid solution phase. This transformation is notoriously sluggish in the iron-nickel system so that equilibrium microstructures cannot be obtained experimentally at normal slow laboratory cooling rates. However, the heat treatments that the meteoroids experience as they journey through space are such as to permit the attainment of near-equilibrium microstructures which would never otherwise be observed. This can simplify the archaeo metallurgist’s task of determining the attribution of artifacts made from meteoritic iron.

The major phases which are normally present in meteoritic iron are kamacite (from the Greek word for bar), which is the meteoritic equivalent of ferrite, and taenite (from the word for band) which is nickel-rich austenite. The precipitation of kamacite in the taenite occurs along the close-packed (111) octahedral planes of the taenite; this often results in a highly characteristic crystallographically zoned structure (Fig. 2) which can be coarse enough to be visible to the naked eye after metallographic preparation, and which is responsible for the categorization of this type of meteorite as an “octahedrite”. These crystallographic bands were first observed in a meteorite early in the 19th century by William Thompson and a few years later by Aloys von Widmanstätten in one of the earliest metallographic determinations of structure. Metallogists still refer to this type of crystallographic transformation product morphology as a “Widmanstätten structure”. The kamacite laths contain less than 8% nickel while the remnant regions of untransformed taenite have much higher nickel contents, typically between 25% and 50%. The range of possible morphologies of the kamacite and taenite phases has given rise to a picturesque terminology by which the microstructure is described (e.g. “comb plessite” and “net plessite”).

Other phases present in meteorites include cohenite (Fe, Ni, Co),C, schreibersite and rhodite (Fe, Ni),P, troilite FeS and chromite FeCr₂O₄. The relative coarseness of the microstructure is reflected in its non-metallic inclusions; troilite inclusions which are millimetres or even centimetres in diameter are not uncommon.

Heating and mechanical working can create substantial changes in the microstructure, however, in many cases iron which has been heated or severely mechanically worked can still be identified as meteoritic if the microstructure is interpreted with care and thought. Often the heating and shock to which meteorites are subjected as they enter the earth’s atmosphere only affect the microstructure of a surface layer, and this is frequently also the case with mechanical working.

Each fragment of the Cape York meteorite fall is, surprisingly, a single crystal, even the 20-ton fragment in Copenhagen which has been sectioned and well studied. It is not clear whether the original meteoroid was a huge single crystal or whether a coarse-grained polycrystal broke up along its grain boundaries as a result of its collision with the earth’s atmosphere. In any case these must be among the largest single crystals known. The troilite inclusions are observed to be as much as 10 cm to 15 cm long and are elongated parallel to the <100> crystallographic directions of the original taenite. Interestingly, chromite inclusions in the troilite are concentrated at one end of each troilite inclusion, an occurrence which may be explained if the meteoroid originally solidified in a gravitational field!

There is no doubt that the Cape York meteorites were used by the prehistoric people of the area as sources of iron. The reports of the Polar Eskimos of the Thule area to John Ross and to later explorers are an indication of this but in addition there is material evidence. Several of the meteorites, including Woman, Dog and Savik I were found to be surrounded by large piles of hammerstones — in the case of Woman, some 10 000 of them. These hammerstones, rounded blocks of basalt weighing 1 kg to 10 kg each, had been brought from a distance, perhaps as much as 50 km, to be used in mechanically working the meteorites because the local country rock in the Cape York area has insufficient strength and toughness to be useful for this purpose. The hammerstones were used to laboriously pound out thin projections of metal from the main body of the meteorite and then to detach these as small flakes which were used, for example in making a blade by inserting a row of such flakes into a groove in a bone or ivory shank. Similar laborious beating techniques were used by the inhabitants of the Great Lakes area to detach useful sized pieces of native copper from large blocks in distant prehistoric times (at least as early as 3000 B.C.). It is of course likely that unrecorded small fragments from the
Cape York fall were originally used in their entirety without the necessity of detaching them from larger bodies.

Artifacts assumed to have been made from meteoritic iron have been found in a broad band sweeping down from Greenland across the eastern Canadian Arctic into Hudson's Bay. It is assumed that these originated with the Cape York meteorites, although sometimes this assumption is made without sufficient evidence. Typically, their identification as meteoritic iron is made by archaeologists on the basis that the iron contains an appreciable amount of nickel while their association with the Cape York fall is made on the rather shaky basis that this iron is known to have been used in other artifacts. However, it may well be that for some iron artifacts one or both of these assumptions is incorrect. Although it is impossible to prove definitively that a particular artifact originated from a particular meteorite, metallographic examination and quantitative elemental analysis can be used to indicate a strong probability, or where appropriate to discount such an attribution. There is a great need for metallurgical analyses of artifacts such as has been carried out by Prof. Buchwald in Copenhagen.

**Telluric Iron**

It is important to note that the presence in an iron artifact of a measurable nickel content is not sufficient evidence to prove that the iron is meteoric. An alternative source of naturally occurring metallic iron was available to the Inuit prior to direct contact with other civilizations and that iron also contains nickel.

Native iron is an unusual occurrence. The high reactivity of iron with oxygen inhibits the formation of the metal and gives it an instability when it does form, an instability which is responsible for its oxidation and corrosion tendencies. In a few areas of the world, however, native iron or naturally occurring iron-nickel alloys have been reported, the latter as awaruite in New Zealand and Europe and as josephinite on the west coast of North America. These materials are not abundant and they occur only as small grains, but on and near Disko Island, on the west coast of Greenland some 800 km south of Cape York, substantial amounts of native iron have formed. Boulders of this iron first came to the attention of Europeans in the 1850s, although a large block may have been taken to Amsterdam a century earlier. During the 1870s numerous analyses of this iron showed that it contained nickel, in amounts of the order of a few per cent. Furthermore, early reports ascribed to it a Widmanstätten structure when it was polished and etched. As a result of these observations as well as the physical appearance of the boulders, the iron was initially assumed to be meteoric. However, a series of later studies and investigations, beginning with those of Steenstrup and Lorenzen, showed that the boulders are mixtures of metallic iron with greater or lesser amounts of basalt, and that outcrops of a similar basalt containing millimetre-sized particles of iron occur at a number of locations in this part of Greenland, notably at Oulifak (Ulvafak) on the south coast of Disko Island. It thus became evident that the iron is not meteoric, but rather the boulders are fragments which have been eroded from the outcrops as a result of weathering and that the Widmanstätten structure is a normal consequence of the composition and thermal history of the material. It is now known that there are two types of the native iron, one of which had been used by the prehistoric inhabitants of the region as a material for the fabrication of tools.

Large boulders of Oulifak telluric iron can be seen at museums in Stockholm, Copenhagen and Helsinki. The largest of these, in Stockholm, weighs 29 tons.

There has been much speculation as to the genesis of these unusual deposits, but to date no single convincing explanation is generally accepted. One clue is that the metallic iron is found in association with a graphite-bearing feldspar. Clearly a strongly reducing environment is involved, perhaps related to the presence of coal beds below the basalts. One suggestion is that a block of pyrrhotite containing nickel penetrated into fissures in the country rock, through which basalt subsequently rose after having absorbed carbon from the coal beds. However, all suggestions to date are speculative and it is not even clear whether the metallic iron formed as the result of a secondary reduction and carbonization process or whether it is a primary constituent of the basalt flow.

From the perspective of a metallurgist wanting to identify archaeological artifacts made from this telluric iron as opposed to meteoric and smelted ırons, it is important to know both the compositional and the microstructural characteristics of all three types.

Many elemental analyses of the Greenland native iron have been performed since its discovery; some of the results are compiled by Boggild (1953) and by Buchwald and Møsgård (1985). The nickel content is in the range 0.05% to 4%, with one isolated and suspect data point at 6.5%. Other elements present include cobalt (0.1% to 0.4%), phosphorus (less than about 0.1%) and copper (0.05% to 0.4%). By far the most noteworthy constituent of the telluric iron, however, is carbon, which has been reported to range from less than 0.2% to as much as 10% although recent analyses suggest that the maximum is closer to 4.5%.

The telluric iron has a nickel content which is sufficiently low that it should be distinguishable from meteoritic iron by careful quantitative nickel analysis, however, carbon is also a useful distinguishing element. The carbon levels of the telluric iron are well above those of meteoritic iron and, at the upper end of the range, are comparable to those of cast iron. Because of the range of carbon contents an enormous range of properties and a variety of interesting microstructures would be expected in the telluric iron, and indeed this has been found to be the case. In fact it is the carbon content which is the basis for the classification of the telluric irons into two types. The "Type I irons contain more than 1.5% carbon and are sometimes thought of as nickel-bearing white cast iron while the Type II irons have less than 0.7% carbon (frequently much less) and can be described as malleable nickel-iron. The large blocks from Oulifak fall into the first category. The small millimetre-sized pea-shaped grains disseminated in the basalt are generally of Type II, and it is these which were used by the early inhabitants of the region.

One of the first indications that this iron was used for tools in prehistoric times was the 1879 discovery in a Greenland grave of nine pieces of basalt containing iron inclusions, together with irregular pieces of iron and knives consisting of iron flakes inset in a grooved bone, the same technology which was used for the iron-bladed knives shown in Fig. 1. A recent search by Buchwald and Møsgård (1985) through the collections of several museums yielded ten knives and ulus and one harpoon point, all dating from before 1650 A.D., which could be shown by chemical and microstructural analysis to have been fabricated from the Type II telluric iron. It appears that the iron inclusions were
extracted from the basalt by crushing, following which they were cold hammered into coin-sized flakes, then inserted into grooves in bone or walrus ivory and pinned by a rivet. The microstructures of the iron consisted of nickel-bearing ferrite grains approximately 1 mm in size, with some cementite lamellae and inclusions of limonite, cohenite, troilite and pearlite.

The Type I (cast-iron type) telluric iron is unsuitable for the fabrication of useful tools and weapons. Its microstructure is most interesting, consisting of pearlite with cementite in both massive and Widmanstätten needle form (Fig. 3). The microhardness is in the range of H, 1000. The pearlite is in many cases globular rather than displaying the lamellar morphology found commonly in steels. Not surprisingly, experiments carried out at the University of Alberta and elsewhere have shown that this material, consisting as it does of high-carbon iron grains in a matrix of basalt, cannot be forged either cold or after heating to temperatures as high as 1000°C.

Was there trade in this telluric iron in prehistoric times? Such a possibility would be indicated, for example, by the discovery far from the Disko Island region of artifacts made from this material. Of particular interest in this context would be the diffusion of such material into the Canadian Arctic as was the case with the Cape York meteoritic iron. Unfortunately, however, the artifacts examined which have been identified as having been made from the telluric iron have been found close to, and particularly north and northeast of Disko Island. To date no artifact made from this material has been found in the Canadian Arctic, or from the northern part of Greenland opposite Ellesmere Island. However, the number of analyzed artifacts is not large and it may well be that more telluric iron objects will be found, with important consequences for our understanding of the ancient trading patterns.

Smelted (Wrought) Iron

No evidence has ever been obtained which suggests that the prehistoric peoples of the Arctic carried out either smelting or melting of metals. Nevertheless, there is no doubt that among the iron artifacts which have been obtained by the excavation of archaeological sites and by ethnographic collectors, there are numerous examples of smelted wrought iron. Obviously, these were obtained as the result of some form of contact with cultures external to the area, cultures which practiced smelting technologies. However, “contact” is not as simple a concept as might appear at first thought. Presumably ever since people started to use iron, pieces of wood containing the occasional scrap of iron must have found their ways into the oceans of the world. As an illustration of this phenomenon from more recent times, an extensive compilation of Japanese junks wrecked on the west coast of the Americas, including Alaska, was published by Brooks in 1876. During the exploration voyages of the 18th and 19th centuries (A.D.), instances were reported of iron being known and its value recognized at the time of what was presumably the first direct contact between local cultures and iron-using people. Furthermore, a significant number of early iron artifacts from several cultures appear to have been manufactured by the reworking of iron nails. Certainly, the burst of exploratory activity throughout the Atlantic Ocean, which began in the 15th century, must have resulted in the launching into the ocean currents of significant quantities of wrought iron from the fittings of wrecked ships. Thus indirect contacts with smelting peoples could very well have yielded some of the iron in use in the Arctic long before the direct contacts of the past millennium.

In the eastern Arctic, these direct contacts began, it seems, with the Norse colonies founded on the southwest Greenland coast about 985 A.D. by Eric “the Red”. These colonies, which at their peak had a population of several thousand people, continued to exist for a few hundred years before all European contact was lost. It is not known to what extent iron smelting (from bog ore) was carried out by the Norse on Greenland, but they certainly had iron. In this context, it is of interest that at the Norse site at l’Anse aux Meadows, Newfoundland, which dates from the early years of the 11th century, there appears to be evidence of ironworking, evidence which is presently being investigated by Henry Unglik in the laboratories of Parks Canada in Ottawa.

Direct European contacts were renewed in the 16th century with the voyages of Froebisher (1576-8), Davis (1585-7), Lindenow and Hall (1605-7), Hudson (1610), Button (1613), Baffin and Bylot (1615), etc. It is interesting to note that the commercial exploitation of eastern Arctic waters followed closely upon the heels of the initial discoveries, with extensive multinational fishing and whaling operations off the Labrador Coast in the 16th century (in fact, Davis reported encountering whaling ships during his voyage). It is reported that in the first half of the 17th century, hundreds of whaling ships were active in Baffin Bay. Iron obtained from the whalers may have been custom-forged to order by smiths associated with the whaling fleet, a service which was also provided by the seafarer fur traders off the coast of British Columbia two centuries later.

In the western Arctic, the iron-related contact with outside civilizations began at an early date. Although conclusive evidence is lacking, it is generally accepted that wrought iron smelted in Siberia, in particular along the Amur River, found its way to Arctic North America at least as early as the first millennium A.D. and possibly much earlier. This trade in iron is believed to have involved as intermediaries the Chukchi people of the Bering Straits area and the natives of the Diomede and St. Lawrence Islands. Thus, iron was being traded and used long before the last major migration of population across the Arctic, that of the Thule culture which is believed to have spread.
from Alaska across to Northern Greenland approximately 1000 A.D. It is of interest in this context that native copper from the deposits of Alaska and the west central Canadian Arctic (including the Coppermine River area) was also being used in the Arctic by the beginning of the Christian era and some “technology transfer” undoubtedly occurred!

**Early Wrought Iron**

One of the problems associated with the unravelling of the ancient trade patterns, and the prehistoric technological development and use of iron, is that early wrought iron, made by the bloomery process, is difficult to attribute to a particular geographical location, ore or time period. Further complications arise as a result of iron’s well known tendency to corrode under many common environmental conditions, so that early artifacts and evidence of iron smelting are not often well preserved. It is generally believed that while iron smelting as an occasional, perhaps accidental, occurrence could well have taken place as early as the fourth or fifth millennium B.C., significant iron production and the beginning of the “Early Iron Age” occurred in Asia Minor about the first half of the second millennium B.C. The full Iron Age, when iron replaced bronze as the predominant material for the manufacture of tools and weapons, began some thousand years later. The origins of iron smelting are not known but many researchers believe that smelted iron could quite possibly have first appeared as a result of the inadvertent reduction of the iron oxide fluxes used in copper smelting furnaces to lower the softening temperature and viscosity of silicate copper smelting slags. This could explain the occasional discovery of large pieces of metallic iron at ancient copper smelting sites.

The process by which early iron was smelted is known as the bloomery process, its product being a bloom — a mixture of solid iron, slag and charcoal. Early iron bloom production was carried out in bowl furnaces, in which broken ore and charcoal were smelted using air supplied by bellows through one or more tuyères. The temperatures attainable in such a furnace are more than sufficient for the direct reduction smelting of iron. After the bloom was removed from the furnace and hot forged to expel as much of the slag as possible, nevertheless, the resultant wrought iron always contained a substantial amount of slag in the form of inclusions elongated by the forging. Later developments included bowl furnaces from which the slag was tapped, and then shaft furnaces, with natural and forced draft. The forced-draft shaft bloomery furnace was the final stage in its evolution; this presumably led to the blast furnace, which produced molten pig (cast) iron. Pig iron was readily converted to wrought iron by oxidation of its carbon using either the finery or the two-stage finery-chafery process in the mediaeval period and the puddling process in the 18th century.

At this point, it must be noted that although bloomery iron was widely produced and used by the beginning of the Christian Era, in China the route to iron was quite different. By 500 B.C., the Chinese were producing cast iron by the blast furnace process and their iron technology was based to a large extent on this material nearly 2000 years before the advent of cast iron production in the West.

The bloomery process, with charcoal as a fuel and a reductant, was used by many cultures in many parts of the world, even into the present century in parts of the Third World, and under such a wide range of conditions that it is generally not possible to prove provenance or date it by the examination of its microstructure, its chemical composition, or the composition of its incorporated slag phase, even with the best modern techniques. Numerous investigators have shown, for example, the futility of attempting to determine the provenance of iron artifacts by means of trace element analysis — the iron composition reflects not only the composition of the ore but also that of the flux, the fuel and possibly the furnace lining materials, as well as the characteristics and operating conditions of the furnace. This is not meant to imply that no useful information can be obtained from chemical and microstructural analysis. On the contrary, such analysis reveals much about the material. For example, it is used to characterize the material, to distinguish wrought iron from other ferrous materials such as meteoric iron, telluric iron and steel. As well, it provides information on the fabrication technology and about some aspects of the smelting technology (e.g. wrought iron containing high sulphur levels is likely to have been smelted using coke rather than charcoal). It is especially for the purposes of dating and provenancing that metallurgical analysis is often unproductive.

Evidence, both direct and indirect, supports the belief that wrought iron has been used across the Arctic from Alaska to Greenland. In the west, the evidence includes iron engraving points, carving tools, drill bits, adze and knife blades, and fish hooks from a number of cultures which span the past 2000 years, ranging from the Bering Straits across Alaska and as far east as the Mackenzie River delta. This is confirmed by the ethnographic record; for example, the Bering expedition of the early 1740s and the Cook expedition of the late 1770s found some of the inhabitants of the Aleutian Islands and Alaska to be in possession of iron.

In the central Arctic, along with much archaeological evidence for the use of native copper, a few wrought iron artifacts have been found (for example a knife blade from a Thule culture site on Victoria Island). That the inhabitants of this region were able to obtain iron by trade with Inuit groups to the west was commented on by early ethnographers, such as Vilhjalmur Stefansson.

In the eastern Arctic, numerous wrought iron artifacts have been excavated at sites dating from the past thousand years, for example in northwestern Hudson’s Bay, Cornwallis Island, Ungava, and Ellesmere Island. The earlier specimens seem to be consistent with their having been obtained from the Norse — on Ellesmere Island, for example, Peter Schledermann of the Arctic Institute of North America found segments of wrought iron chain mail! It is also of interest that Martin Frobishers, during his first expedition in 1576, found the inhabitants of Frobisher Bay to be in possession of a box of nails and a “tryvet of iron”, and further noted that the southwest Greenlanders had tools edged with iron.

The microstructure of wrought iron is characterized by stringer-like inclusions of slag in a matrix which consists of ferrite (low carbon iron) and possibly some pearlite (a mixture of ferrite and iron carbide). The slag typically contains the iron silicate fayalite (Fe₂SiO₄) as well as a glassy silicate phase considered by some workers to be close to anorthite, plus in many cases dendrites of the iron oxide wustite (FeO). A typical slag microstructure is shown in Figure 4. Other common constituents of bloomery slag include manganese oxide and the oxides of potassium, calcium and aluminum. A number of investigators have
tried without success to associate the slag composition with the ore from which it was smelted but it appears that, as is the case for artifacts, the possibility of such provenancing, using either trace elements or minor oxide constituents, is remote.

The fact that the matrix of the wrought iron varies from ferrite to pearlite reflects a heterogeneity of carbon content which is, in fact, quite characteristic of wrought iron produced by the bloomery process. Although the basic process produces an iron which is very low in carbon, recaulkuration of the bloom can occur below the tuyère level of the furnace shaft so that the carbon content can vary between virtually 0 and as much as 1%. Furthermore, the processing of the bloom by forging, an integral part of the bloomery process, and the techniques for the production of tools and weapons frequently involved the forge-welding together of pieces of iron of varying carbon content, either intentionally or inadvertently, thus adding to the heterogeneity. Subsequent heating of the iron was in general insufficient to homogenize the carbon. The phosphorus content of the bloom, which could be as high as 0.5% and heterogeneously distributed, also contributed to the variations in carbon content. The heterogeneous nature of the microstructure is reflected in microhardness values ranging from about HV100 to more than HV600. Figure 5 illustrates this heterogeneity.

As a result of all these factors, the metallurgist studying ancient iron cannot be optimistic about his or her ability to associate a wrought iron artifact with a particular time or place, and this, along with its biodegradability, has limited our understanding of the history of wrought iron in the Arctic. Nevertheless, much has been and continues to be learned.

**Wrought Iron in the Arctic**

That wrought iron was utilized prehistorically is clear from the archaeological record as revealed by the excavation of sites as well as from the ethnographic record as recorded by explorers and other observers of cultures prior to their contamination by contact with the outside world. The direct archaeological evidence exists in the form of artifacts of iron or, frequently, iron oxide in the shape of (corroded) iron artifacts. Examples include, antler, bone or walrus ivory shanks or handles with slots containing one or more flakes of rusty iron or iron oxide. In other cases, excavations have uncovered slotted handles in which the slots contain mere traces of rust which are interpreted as being the remnants of iron blades. Evidence which is even more indirect occurs in the form of slotted handles where the narrowness of the slot is such that it can be presumed to have been made for a metal blade rather than a slate blade which of necessity would have been thicker. By means of the archaeometric technique known as "use-wear-analysis", tool marks on artifacts have been studied in order to identify the manufacturing processes; in this manner drill marks on the inner surfaces of drilled holes as well as cut marks on bone harpoons and other objects have been attributed by several investigators to the use of metal tools. The least direct evidence is that carvings or engravings have been made in stone or other media which are so hard that it is assumed that a metal blade must have been utilized. It should be remembered, however, that in the cases where "metal" has been indirectly associated with artifacts, it cannot always be assumed that the metal was iron, because native copper has also been used in the Arctic for several millennia.

There are undoubtedly significant numbers of iron artifacts dating from the past millenium in museum collections in Canada and elsewhere and it would be of great interest to investigate these metallurgically. Just such an investigation has been performed by Dr. V.F. Buchwald on the Greenland collections of the Copenhagen museum. Excluding artifacts which had been subjected to (now obsolete) conservation treatments involving annealing, Buchwald was able to locate and study some 26 wrought iron artifacts, five of which dated from before 1500 A.D. These include a piece of what was possibly chain mail, as well as harpoon heads, knives and ulus. One surprising discovery made by Buchwald was that one of the artifacts studied, a harpoon head obtained in 1818 by John Ross in the Cape York area and long assumed to be meteoritic iron, is in fact wrought iron, and there is now evidence that one of the iron flakes in another of the implements obtained by Ross, and shown in Figure 1, is also wrought iron.

Although there is clearly much need for further study, the pattern which emerges from the work to date is that throughout the past few millennia iron obtained from a range of sources, meteoritic, telluric and smelted wrought
iron, all contributed to the "tool kits" of the inhabitants of the Arctic and played a role in some of their activities. As archaeological work continues in the north, it is to be expected that further evidence will accumulate to give support to this hypothesis. Here is yet another graphic demonstration of how our efforts to understand at least some aspects of the past can be aided by the metallurgical analysis of archaeological artifacts.

FURTHER READING
Highly recommended is the monograph Meteoritic iron, telluric iron and wrought iron in Greenland, by V.F. BUCHWALD and G. MOSDAL, Meddelelser om Grønland, Man and Society, No. 9, 1985, which has been drawn on heavily in the preparation of this article. In addition, see:

METEORITIC IRON

TELLURIC IRON

WROUGHT IRON
FRANKLIN, U.M., BADONE, E., GOTTARDT, R. and YORGA B., An Examination of Prehistoric Copper Technology and Copper Sources in Western Arctic and Subarctic North America, National Museum of Man, Mercury Series paper No. 01, Ottawa, 1981.
Frobisher’s gold on Kodlunarn Island — fact or fable?
Donald D. Hogarth, John Loop and W.A. Gibbins

Introduction
Background

The Frobisher voyages of 1577 and 1578 were organized by the Cathay Company for the express purpose of finding and mining gold ore. Since publication of the Baffin expedition of Charles Francis Hall[1], the location of Kodlunarn or Countess of Warwick Island, the headquarters of Frobisher’s third (1578) voyage, and focal point of the largest Arctic voyage of all time, has been known to the world. Through Hall we also know the location of the Countess of Warwick Mine on Kodlunarn Island. However, the exact locations of the other mines are not known.

Of the “ore” we know little. Pieces of the original “black stone”, once as large as “a half-penny loaf” have now disappeared altogether. Specimens collected at random from the repository of shipments in England and from loose rock on Kodlunarn Island were termed, respectively, “hornblende pyroxenite” and “amphibolite” and from a locale about 200 yards from Kodlunarn as (hornblende) “pyroxenite”[2]. However, it is uncertain whether these specimens are representative of Kodlunarn or of the other mines, although Blackadar[3] notes that, in the northern trench of Kodlunarn, biotite gneiss contains lenses and bands of amphibolite.

The “gold” itself is even less well defined. It has long been accepted that gold was virtually absent in Frobisher’s ore. The “gold” has been termed pyrite, biotite and both biotite and pyrite together. Even the word “marquesset”, which appears and reappears in contemporary descriptions, is nebulous and is defined as “a stone participating in the nature of some metal, yet in so small quantity, that the metal cannot be melted from it” (O.E.D., definition of 1616).

Many assays done in England and reported in Stefansson[4] indicated high-grade concentration of gold (greater than 60 g/t). The exact methods of the assays were not revealed but it appears from information in the British State Papers and by reference to accounts of the standard technology of the day (e.g. Sisco and Smith[5]), that the method used in England involved lead fusion, cupellation, and acid parting, much the same as in modern fire assaying.

Since the return of Frobisher’s third voyage, these high assays have generally been regarded as false. Considering the well-developed assay technology in Elizabethan times and the apparent lack of ulterior motive of the assayers for falsifying data, these suspect high assays are indeed puzzling.

Assays (“proofes”) made in the field may have been performed in a similar manner. Furnaces were set up on Kodlunarn Island and two other locations nearby, but details of their operations, the assay results and the provenance of the samples are not known. The “Booke of

Register”, reportedly listing these details, has long since disappeared.

In an attempt to solve some of these problems and provide further details on the operation of Canada’s first “gold mines”, the authors visited Kodlunarn Island in August 1983, collected specimens and made a detailed geological survey of Kodlunarn Island. Specimens returned to Ottawa were studied petrographically, analytically (XRF, DCP, AA and NAA) and mineralogically (electron microprobe). The present paper is condensed and somewhat modified from an earlier publication of Hogarth and Gibbins[6].

Geographic Information

Kodlunarn Island is on the west side of Countess of Warwick Sound, some 188 km southeast of the community of Frobisher Bay (Fig. 1). Its latitude-longitude is 62°48’ N - 65°26’ W, and its grid location, on the NTS 1:250,000 “Loks Land” sheet, is 20VLE7668. It can be reached by helicopter from Frobisher Bay or by boat, following a circuitous route around points and islands that normally takes two full days to travel. The islet rises abruptly 6 m to 7 m as a sea cliff from a gently sloping tidal flat; the summit is about 17 m above high tide. This tide is normally about 10 m. The island measures about 400 m long and 350 m across at its widest and totals 8 hectares in area.

Kodlunarn Island and adjacent Baffin Island are well within permafrost limits. Even in August, ground is frozen within 0.3 m to 1 m of the surface, which is covered with loose shingle and boulders and interrupted by the occasional protrusion of solid rock. The only continuous outcrop is found along the sea cliffs. These hostile soil and climatic conditions permit scant flora only, including rare ground-hugging Salix arctica (a willow), Betula glandulosa (a birch) and thin tufts of grass and sedges.

The north side of the island is cut by the well-documented “Ship’s Trench” while the central portion is pierced by the “Reservoir Trench” (Fig. 2). Of the numerous publications dealing with geographical features and historical incidents related to this island, probably the most accessible, and certainly the most complete, are the two volumes edited by Vilhjalmur Stefansson[7]. Further information is given by Kenyon[8].

Geology

Little has been written on the geology of this area. A summary, with accompanying geological map on 1:500,000 scale, was published by Blackadar[9]. All rocks were described as early Proterozoic (Aphebian) metamorphics. More particularly, the rocks of Kodlunarn Island were placed in unit 3, a charnockitic quartz-feldspar gneiss. No other geological report exists. The following description is based on data gathered during the present investigation.

The principal rock type at Kodlunarn Island and
TABLE 1. Typical modal analyses of black ore

<table>
<thead>
<tr>
<th>Specimen No.</th>
<th>K5A</th>
<th>K5B</th>
<th>K5C</th>
<th>K9C</th>
</tr>
</thead>
<tbody>
<tr>
<td>Hornblende</td>
<td>45</td>
<td>50</td>
<td>30</td>
<td>75</td>
</tr>
<tr>
<td>Biotite</td>
<td>20</td>
<td>10</td>
<td>-</td>
<td>+</td>
</tr>
<tr>
<td>Hypersthene</td>
<td>-10</td>
<td>-</td>
<td>-</td>
<td>+</td>
</tr>
<tr>
<td>Olivine</td>
<td>-</td>
<td>-35</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Spinel</td>
<td>-</td>
<td>30</td>
<td>-20</td>
<td>-</td>
</tr>
<tr>
<td>Plagioclase</td>
<td>30</td>
<td>30</td>
<td>-5</td>
<td>-</td>
</tr>
<tr>
<td>Opaques</td>
<td>5</td>
<td>5</td>
<td>+</td>
<td>-</td>
</tr>
<tr>
<td>Location*</td>
<td>iST</td>
<td>iST</td>
<td>1ST</td>
<td>1RT</td>
</tr>
</tbody>
</table>

* abbreviations: iST = in-situ, "Ship’s Trench"
1ST = loose, "Ship’s Trench"
1RT = loose, "Reservoir Trench"

TABLE 2. Partial x-ray fluorescent analyses of specimens from Kodlunarn Island

<table>
<thead>
<tr>
<th>Unit</th>
<th>1 (wall rock)</th>
<th>2 (with mica)</th>
<th>3 (with hypersthene)</th>
<th>3 (with spinel)</th>
<th>3 (with forsterite)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Number of specimens</td>
<td>2</td>
<td>2</td>
<td>2</td>
<td>3</td>
<td>1</td>
</tr>
<tr>
<td>SiO₂%</td>
<td>68.3 ± 2.3</td>
<td>46.1 ± 0.6</td>
<td>44.8 ± 1.0</td>
<td>39.9 ± 0.4</td>
<td>31.0</td>
</tr>
<tr>
<td>Al₂O₃%</td>
<td>15.4 ± 0.3</td>
<td>15.5 ± 0.2</td>
<td>14.9 ± 0.5</td>
<td>15.4 ± 0.8</td>
<td>21.0</td>
</tr>
<tr>
<td>FeO%</td>
<td>3.3 ± 1.1</td>
<td>12.5 ± 0.3</td>
<td>11.3 ± 0.3</td>
<td>10.9 ± 0.4</td>
<td>14.7</td>
</tr>
<tr>
<td>MgO%</td>
<td>1.4 ± 0.5</td>
<td>10.3 ± 0.2</td>
<td>13.3 ± 0.8</td>
<td>14.8 ± 0.5</td>
<td>17.8</td>
</tr>
<tr>
<td>CaO%</td>
<td>4.0 ± 0.6</td>
<td>9.3 ± 0.1</td>
<td>9.9 ± 0.9</td>
<td>11.4 ± 0.5</td>
<td>11.2</td>
</tr>
<tr>
<td>Na₂O%</td>
<td>4.2 ± 0.2</td>
<td>2.5 ± 0.1</td>
<td>1.8 ± 0.05</td>
<td>1.9 ± 0.2</td>
<td>0.5</td>
</tr>
<tr>
<td>K₂O%</td>
<td>1.1 ± 0.1</td>
<td>1.7 ± 0.1</td>
<td>1.6 ± 1.2</td>
<td>0.3 ± 0</td>
<td>0.1</td>
</tr>
<tr>
<td>Ni ppm</td>
<td>15 ± 5</td>
<td>220 ± 10</td>
<td>380 ± 10</td>
<td>400 ± 30</td>
<td>600</td>
</tr>
<tr>
<td>Cr ppm</td>
<td>20 ± 0</td>
<td>200 ± 10</td>
<td>330 ± 160</td>
<td>180 ± 20</td>
<td>290</td>
</tr>
</tbody>
</table>

TABLE 3. Some results of heating experiments on “black ores” from Kodlunarn Island

<table>
<thead>
<tr>
<th>No.</th>
<th>Microscopic appearance of unheated Mg-Mica</th>
<th>400°C</th>
<th>600°C</th>
<th>800°C</th>
<th>1000°C</th>
</tr>
</thead>
<tbody>
<tr>
<td>K5A</td>
<td>Green biotite (A₉) 20%</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>K5B</td>
<td>Green biotite (A₉) 10%</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>K5C</td>
<td>Not observed</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>K9C</td>
<td>Green biotite (A₉) 1%</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

In parentheses, compositions, mol % anneite (phlogopite-anneite series) from chart of Winchell and Winchell (10).
(measured values = K5A = 1.632; K5B = 1.624; K9C = 1.632; Second figure represents volume % of thin section.

adjacent Baffin Island is a fine-grained, grey leucocratic gneiss (unit 1, Fig. 2) characterized by oligoclase, quartz, minor brown biotite and varying amounts of microcline. This rock apparently corresponds to unit 4 — grey granitic gneiss of Blackadar. Dark green, hornblende-rich gneiss (unit 2) is less common.

The mafic rocks (unit 3 — “amphibolite”) include Frobisher’s “black ores” and are, therefore, of special significance. From various accounts, this rock, a “black stone, much lyke to a seacole in colour”, seems to represent the type sought, and almost exclusively mined, in the last two Frobisher expeditions. At Kodlunarn, the rock is present as two principal layers or lenses trending north-northwest and dipping steeply west, and is concordant with the local gneisses (Fig. 2). One of these, traced across the island, is included in the two Frobisher trenches. Specimens were first collected from a shallow hole excavated in the floor of the Research Trench in 1975, then in situ in the Ship’s Trench in 1983, and finally from remnants of stockpiles near the trenches themselves. As distinct from nearby fragments, rocks in the stockpiles were very dark and partly covered with blackened lichen, believed to indicate great age (cf. ref. 2). Some specimens on the stockpiles closely resembled the rocks in situ, but those selected for study were macroscopically different. Modal analyses of typical ‘black rocks’ are given in Table 1.

Chemical Composition

X-ray fluorescent analyses (data summarized in Table 2) show that all “black ores” are rich in aluminum (14 to 21% Al₂O₃) and the “black ores” specially rich in magnesium (10 to 18% MgO) and iron (10 to 15% FeO).

An abundance of lithophile elements (e.g. 2 to 3% Na₂O, 1 to 2% K₂O, 9 to 12% CaO) and a low content of mantle-type elements (100 to 300 ppm Cr, 100 to 400 ppm Ni) suggests that the “black ores” were crustally derived. However, specimen K5C’s enigmatic, being quite different from the other rocks; it is comparatively poor in alkalics and silica but richer in nickel. These and other differences suggest that this rock, one of the specimens collected loose from the Ship’s Trench area, may have been derived from a locality separate from other spinel-bearing amphibolites and perhaps from all the other “black ores”.

Ore Mineralogy

Seven polished thin sections were examined with the ore microscope. From these, five polished thin sections representing the three “black ore” types were selected for microprobe work at Carleton by Peter Jones, analyst. A brief summary of oxide and sulphide mineralogy follows:

The main black oxide is magnetite with subordinate ilmenite, the latter either as exsolved lamellae in magnetite or as isolated grains.

The green oxide is pleonaste (65% Mg-spinel, 35% Fe-spinel) in specimen K5C but hercynite (45% Mg-spinel, 55% Fe-spinel) in specimens K5B and K9D.

Sulphides are mainly present as small rounded grains. The main sulphide would appear to be pyrrhotite, with lesser amounts of chalcopyrite and pyrite. The pyrite is
partly oxidized to hematite. A very small amount of millerite is present in specimen KSC closely associated with calcite.

Needless to say, with the very minor amount of precious metals determined by chemical analysis (Table 4), no native gold, silver or platinum was detected in any specimen.

The Frobisher Minerals

What mineral first attracted the attention of the “venturers”? According to George Best (in Stefansson 8), back in England one of the sailors’ wives cast into the fire a piece of coal-black rock of the first voyage, removed it, quenched it in vinegar, and found that it “glistened with a bright Marquess: of golde”. Another report (an unpublished manuscript) has “Robert Garrand” (Robert Garrett), the discoverer of the black rock, tossing a piece into the ship’s fire “to prove it would burne”. The last report seems the most likely but, common to both accounts, the rock was subjected to ignition.

In order to investigate the appearance of the black rock after heating, 10 specimens were roasted at various temperatures in a muffle furnace. As can be seen (Table 3) a golden mica is readily produced. This yellowing was first observed in one specimen at 400°C, and at 800°C is more or less complete for all specimens containing phlogopite-biotite. 800°C also marks the temperature of greatest contrast of golden yellow against a pitch-black background. By 1000°C magnetite has partly oxidized to hematite, and hornblende has taken on a slightly brownish cast. Pyrite has long since oxidized to hematite at these high temperatures.

We may conclude that Best’s “marquesset of golde” was phlogopite-biotite, oxidized after heating to moderate or high temperatures, and whose true identity as a mica was missed because of its minute size (average grain length 1.5 mm). However, it is possible that the “marquesset” was bronzite-hypersthene, also yellowed after roasting.

We have few clues to the identity of the so-called gold in the unheated rock. Best notes that the ore of Kodlunarn Island “in the washing held golde plainly to be seen”9. Roy and Tilley believe the “gold” to have been biotite10, but the mica encountered in this investigation was greenish black and can hardly be considered a candidate. On the other hand, pyrite was noted in five of the eight specimens of “black ore” collected at Kodlunarn. At the bottom of the Ship’s Trench a network of thin pyrite veinlets was observed in situ. Pyrite would, therefore, seem a more likely mineral. However, the weathered brassy to bronzy biotite, described by Roy and Tilley, while not observed in the present investigation, presents a possible alternative.

Precious Metal Analyses

Eight “black ores” from Kodlunarn Island were tested for Ag and Au by direct current plasma (DCP) at the University of Ottawa.

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**TABLE 4. Analyses of precious metals in samples of “black ore” from Kodlunarn Island**

<table>
<thead>
<tr>
<th>No.</th>
<th>Location (1)</th>
<th>Analyses in PPB (2)</th>
<th>Ag</th>
<th>Au</th>
<th>Pt</th>
</tr>
</thead>
<tbody>
<tr>
<td>K5A</td>
<td>in-situ, bottom S.T.</td>
<td>&lt;50</td>
<td>8</td>
<td>-</td>
<td></td>
</tr>
<tr>
<td>K5B</td>
<td>in-situ, bottom S.T.</td>
<td>&lt;50</td>
<td>6</td>
<td>5</td>
<td></td>
</tr>
<tr>
<td>K5C</td>
<td>loose, bottom S.T.</td>
<td>&lt;50</td>
<td>2</td>
<td>&lt;5</td>
<td></td>
</tr>
<tr>
<td>K5B</td>
<td>loose, top S.T.</td>
<td>&lt;50</td>
<td>7</td>
<td>5</td>
<td></td>
</tr>
<tr>
<td>K5C</td>
<td>loose, bottom R.T.</td>
<td>&lt;50</td>
<td>8</td>
<td>5</td>
<td></td>
</tr>
<tr>
<td>K5E</td>
<td>loose, top R.T.</td>
<td>&lt;50</td>
<td>11</td>
<td>-</td>
<td></td>
</tr>
<tr>
<td>Crusal abundance (3)</td>
<td>75</td>
<td>3.5</td>
<td>46</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Ultrasonic rocks (4)</td>
<td>80</td>
<td>11.4</td>
<td>65</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

(1) Abbreviations: S.T. = “Ship’s Trench”; R.T. = “Reservoir Trench”.
(2) Ag, Au by DCP, J. Loop, Univ. of Ottawa, analyst; Pt by NAA, E. Hoffman, NAS, Hamilton, analyst.
(3) Li and Y10 quoted by Boyle (reference 11).
(4) Boyle (reference 11, 12).
Silver
A crushed and ground, 100 mg-sample was dissolved in hot HNO₃, and the solution was analyzed by DCP.

Gold and Platinum
Five to ten-gm samples were made from crushed and ground specimens. A measured amount of silver, in solution, was added to the ore-flux mixture and a silver bead containing all the gold and platinum in alloy was obtained by standard Pb-fusion fire-assay methods. The bead was then dissolved in aqua regia and the DCP readings were taken at the University of Ottawa. Data are summarized in Table 4.

Six of the samples were then analyzed by neutron activation analysis (NAA). Nickel-sulphide extractions were made from 50 g samples and NAA determinations were made for platinum.

Gold contents approximate the crustal average and reached Boyle's ultrabasic rock average in one specimen only (K9E). Eight specimens averaged 6 ppb.

Silver approached crustal abundance in one specimen only (K8A). This rock, strangely, was one of the few specimens showing no sulphide at all, microscopically or macroscopically. All other assays were below the detection limit (50 ppb).

Platinum is also low and gives no support for a mantle-derived rock. NAA data suggest the "black ores" contain approximately 1/10 crustal abundance Pt.

Conclusions
Rock was quarried from two sites on Kodlunarn Island, the Ship's Trench and the Reservoir Trench, along a band containing micaceous amphibolite. High-grade material was stockpiled nearby, possibly together with rock brought from other locales. The Ship's Trench was at tidewater and may have been used as a drydock in the third voyage, after the finish of mining operations at this site. Likewise, the Reservoir Trench may also have had an auxiliary use, here as a source of fresh water in the later stages of the third voyage. However, Best[1] infers that a closely guarded "spring" supplied at least some of the drinking water requirement.

Of the three types of "black rock" found loose on Kodlunarn, only one can be correlated with rock in situ. A second type perhaps belongs here also, but no evidence whatsoever could be found linking specimen K5C (Table I) with any rock in place. It is thought that the last type was derived from a locale bordering the Countess of Warwick Sound.

The golden mineral originally washed from crushed rock is thought to have been pyrite or chalcopyrite which occur irregularly in small amounts through the "black ore". On the other hand, "gold" produced in the fire probably represented a green mica that oxidized to golden yellow long before red heat. In the latter case, it is assumed that the rock from Little Hall's Island (the original discovery, not seen) reacted on firing similarly to the Kodlunarn rocks.

Gold and silver contents in our specimens were very low, mostly below crustal abundance. This is not at all surprising, considering the specimens represent rocks of basic to ultrabasic compositions — probably crustally derived as metamorphic rock.

Platinum affords a special interest. The metal was unknown in Froebisher's day (it was first characterized by Sir William Watson, an English physicist, in 1750) and the early assayers could have confused the high Pt in ultrabasics with Au. During fire assaying, P would, like Au, be partitioned into the lead fraction and, ultimately, into the bullion bead. This could explain the high assays reported in the State Papers. Unfortunately, this theory could not be substantiated in our analyses. We must dismiss Froebisher's gold as an interesting but perplexing myth.

Acknowledgment
The authors would like to thank R. Hartree of the University of Ottawa for chemical analyses and P. Jones of Carleton University for microprobe work. The study was made possible by a Department of Indian and Northern Affairs contract in favour of one of us (D.D.H.).

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Technical investigations of iron-working remains from the French Jesuit mission of Sainte-Marie among the Hurons (1639-1649)

Sandra K. Zacharias, James R. Hunter and Ursula M. Franklin

Introduction

Archaeometallurgical remains can be a valuable source of information about past technological activities and, by inference, the societies that undertook them. Material traces of former metal-producing or metal-working centres can include raw materials, products, or metallurgical installations such as mines, furnaces or forges. Of these remains, the most common is slag, often left undisturbed since its original production.

Physical and chemical analyses of archaeological slag and metal can often reveal the processes and materials involved in their formation. Mineral composition and softening temperature of a slag, for instance, can indicate the conditions of the metallurgical operation, such as temperature and atmosphere and, thus, help identify metallurgical processes which took place at a site.

The present report illustrates the value of such analysis. Metal-working remains from Sainte-Marie among the Hurons, near Midland, Ontario, on the south side of Georgian Bay have been analyzed in order to determine the techniques and processes used by the 17th-century blacksmith who worked at the mission headquarters from 1642 to 1649. Samples were selected from a very large collection of blacksmith refuse which was excavated from the site by archaeologist Kenneth Kidd in 1941-42.

Historical Background

Sainte-Marie among the Hurons was the first European community, in what is now the province of Ontario. It was a Jesuit community made up primarily of French Jesuit priests, lay brothers, donnés, engaged young boys, who were engaged in apostolic work among the Huron Indians. The founding of Sainte-Marie goes back to 1632 when the French missionaries and traders returned to New France to re-establish their former colony, which, for three years, had been controlled by the English-based “Company of Adventurers to Canada”. Two years after their return, the Jesuits (led by Father Jean de Brébeuf) reopened missionary work among the Hurons of the upper country, a large area west of the Ottawa river in what is now south-central Ontario.

By 1638 eleven priests were working among the twelve or fourteen thousand people who made up the Huron confederacy. That same year Father Brébeuf was replaced as Superior of the Huron mission by Father Jérôme Lalemant, an able administrator who was sent to assume control of the developing mission. One of the many new policies developed by Lalemant was the idea of a permanent Jesuit residence located in the centre of the Huron homeland.

All the religious needs of the Huron could easily be administered from this central base rather than by Europeans dispersed throughout the Huron country. Furthermore, the central residence could act as an example for the Hurons of what a Christian community was like. Sainte-Marie was designed to become agriculturally self-sufficient and eventually self-sustaining. Construction began in the summer of 1639, and by 1641 fourteen priests were working on the mission and some twenty workmen, donnés and lay brothers were providing support services.

The Blacksmith

In 1642 the blacksmith, Louis Gaubert, arrived at Sainte-Marie. Louis was born in Champenois, France in 1608. He arrived in Canada in 1633 on board the flotilla of ships commanded by Samuel de Champlain who was returning to assume the role of Governor and representative of the King in New France. Louis was one of seven workmen who were sent to repair the buildings originally erected in 1625-26 by Father Charles Lalemant at Notre Dame des Anges, near Quebec City. He was employed as a blacksmith. Father Paul le Jeune, Superior of the Canadian missions, spoke well about his abilities in his trade. Two years after arriving in Canada, Louis decided to enter the Jesuit Order. He was sent back to France in 1635 where he entered the novitiate at Paris on December 8. The following spring he returned to Canada and was sent to Notre Dame des Anges. Sometime in 1637 or 1638, he pronounced his vows and became a Brother in the Society of Jesus. He resided at Notre Dame des Anges until the summer of 1642 when he was sent up to the Huron country where he became a blacksmith at the residence of Sainte-Marie. He worked as a blacksmith and labourer until the Huron mission was abandoned in June of 1650. Whether he returned to France after the collapse of the Huron mission is not known. Louis was listed as a joiner-carpenter at the College of Quebec in 1659 and died at Quebec on July 22, 1679.

The Forge

During the 1941-42 excavations of Sainte-Marie, Kenneth Kidd discovered a forge, or at least the foundation for a flue, twenty-five feet south of the central residence building and ten feet north of the bastion. The flue “consisted of a rectangular base, open on the north side. Carefully cut limestone slabs were laid in mortar with larger slabs forming a rough footing in two courses.” Around the forge foundation large quantities of metal tools, metal scraps, slag ash and charcoal were recovered. In addition, almost a thousand pieces of slag or slag-like material were distributed throughout the site. Based on a preliminary examination using a low-power microscope, the specimens were categorized and representative samples chosen for analysis. Four slag samples from the forge area and four samples from the other areas of the site were selected. The sample numbers refer to the excavation unit provenience level.

Analytical Techniques

For the slag, specific gravity was calculated and approximate melting point was determined by heating small cubes of slag in a furnace in air by increments of 25°C to the softening point and then to the fluidity temperature. Mineral phases were identified by microscopy of polished sections and by X-ray diffraction analysis (XRD) using the Debye-Sherrer powder method with Fe Kα radiation.

An iron fragment was examined by metallography. The surface of the specimen was prepared by grinding, polishing and etching with 2% Nital solution.

Wood charcoal remains were identified by Stephen G. Monkton. Material was examined microscopically and identifications were made with the assistance of both a wood identification key and comparative material made available by the Royal Ontario Museum. Each fragment was broken in half in order to expose a transverse section of cell tissue.

The Samples

Slag

Slag samples were categorized into three types based on morphology and surface features.

(1) Concave-convex round slag:

(Sample Units S14.85Y' .7; S3.85V1; S3.770'.10)
round shape approximately 9 cm in diameter; heterogeneous layered structure; light grey to nearly black in colour with small and large pores throughout; some glassy regions and surface charcoal inclusions; very little surface rust and relatively light weight suggest low iron content.

Sample S14.85Y' .7 (Fig. 1) was analyzed further. Specific gravity of a piece of this slag is 2.7. This is very light compared to a typical iron smelting slag (usually the iron silicate, fayalite. S.G. ~ 4.1 to 4.4), and reflects both the porosity and the low iron content of the sample. The softening point is about 950°C and a very viscous fluid state is reached between 1150°C and 1200°C.

An examination of the microstructure shows that this slag has a layered structure. Several phases are present and some phase segregation has occurred due to melting point and density differences. In the lowest layer (convex side) pale dendrites and darker laths are present within a glassy matrix (Fig. 2). Characteristically, the iron silicate, fayalite, forms these laths or needles when it recrystallizes above 1150°C. The dendritic structure of the other phase, probably iron oxide, indicates that it solidified at a higher temperature than the fayalite, and that this slag reached a liquid state.

In the upper layers of the slag the pore size increases, the dendrites decrease in size and number, the laths increase in size and irregular orthorhombic grains (characteristic of fayalite) appear in the glassy matrix. The last indicate a slow cooling. Near the upper surface, the slag is very glassy and the pores are larger still. On the outer rim of the slag mottled grey ash and vitrified areas are present.

XRD analyses made of powdered samples from the rim and from the interior of the slag help identify the phases observed in the microstructure. In the surface sample, the major crystalline phase is wollastonite (β-CaSiO₃). In the interior sample three crystalline phases are present: two major phases, fayalite (Fe₂SiO₄), and a spinel similar to hercynite (Fe₃Al₂O₆), and a minor phase, anorthite (CaAl₂Si₂O₈). Anorthite and wollastonite both result from the vitrification of calcium-rich charcoal fuel ash. Wollastonite generally forms below 1100°C, suggesting that the outer rim of the slag remained below this temperature.
In summary, the concave-convex type slag is a heterogeneous vitrified mixture of fuel ash (mainly calcium oxide) and iron forging by-products: hammer scale (iron oxides) and bits of slag from the wrought iron (iron silicates). The layered structure suggests that this slag was not formed all at one time, and that it was subjected to repeated heating and cooling — as would be expected for iron smelting slag formed over time in a blacksmith's shop.

(2) Irregular chunks of heterogeneous, glassy slag:
- (Sample Units S4.61S’’.7; S2.620’’.4; S5.75H’’.3K5-3; S3.770’’.10) irregular shapes; very corroded, rust coloured; some glassy areas and inclusions of charcoal and very rusted iron; interior has large and small pores throughout and very heterogeneous structure, colour ranges from grey to dark brown; relatively heavy weight suggests higher iron content than type (1).

These samples were considered too heterogeneous to give meaningful XRD analysis. Microscopic examination suggests, however, that they have a composition similar to the concave-convex round slags but a much coarser and unlayered structure. Localized vitrification indicates that some portions of the slag reached a fluid state, at about 1100°C. For the most part, however, these are poorly coalesced masses of corroded iron, ash, charcoal and slag such as would be found in the area surrounding a smelting forge.

Wood charcoal fragments attached to sample S4.61S’’.7 (Fig. 3) were identified by Stephen Monckton. According to his analysis, “this material was unusual in that it was extremely hard and had an appearance similar to graphite. In those fragments large enough to analyse (approximately 3 mm to 4 mm in width), cell structure was fairly well preserved. Two genera were identified: Acer (maple) and Betula (birch).”

(3) Light weight glassy grey slag:
- (Sample Units S4.61S’’.7; S2.59K’’.7) small irregular shaped chunks of glassy slag; very light weight with large pores throughout; almost translucent pale grey colour with some regions of surface rust; very heterogeneous interior structure.

This vitrified material was considered too amorphous and heterogeneous to give meaningful analysis, but it is almost certainly impure vitrified fuel ash.

Iron

Six irregular-shaped chunks of corroded iron (Sample Unit S1.630’’.1) were examined. From these, one specimen of iron (Fig. 1) was polished and etched for metallographic examination.

The microstructure reveals a substantial layer of carbon steel beneath the corroded surface of this specimen. Figure 4 shows the ferrite and pearlite structure near the surface of the sample. In Figure 5 the region of lower carbon content beneath the steel layer can be seen; equiaxed ferrite grains with elongated slag inclusions indicate that this metal was heated after it was worked sufficiently to deform the slag inclusions.

For such a carburized layer to form, this iron fragment had to remain for a prolonged period (at least several hours) in a carbon-rich area, such as buried in a charcoal fire, at a temperature over 900°C. This piece of carbon steel could have been an accidental product lost in the smelting hearth or formed during the burning of Sainte-Marie. Two hand-wrought iron nails from Sainte-Marie (of uncertain provenience) examined by Franklin have a similar steel microstructure. However, a skilled smith could have intentionally produced steel by carburizing wrought iron in the smelting hearth at Sainte-Marie. This technique was known in medieval Europe.
Summary
The slags examined are by-products of iron smithing rather than smelting, as evidenced by their mineral composition, heterogeneous structure, relatively low iron content and characteristic round concave-convex shape. The slag was repeatedly heated to 1100°C to 1200°C or higher; to achieve these temperatures a forced air draught, probably produced by bellows, was necessary. The amount of smithing slag provides a rough estimate of the scale of iron working. The quantity present at Sainte-Marie suggests small scale production of tools and other items for local use.

The local hardwoods used for fuel provide excellent charcoal (heat retaining, long burning) for smithing. Their use indicates that the smith was knowledgeable about wood types and adaptable enough to recognize and use the best in this new land.

The iron fragment analyzed had a much higher carbon content than expected. This may, however, be steel which was accidently produced — most likely during the fire at Sainte-Marie.

These remains suggest that a typical medieval-Renaissance European blacksmithing technique was used at Sainte-Marie, similar to that illustrated in Figure 6. The smith’s skill is evident in his use of high quality fuel and his control of heat. Whether or not he intentionally produced steel is unclear. An examination of some of the iron tools made at Sainte-Marie may help solve this problem.

Acknowledgments
The technical studies were carried out as part of a general research project in Archaeometallurgy at the Department of Metallurgy and Materials Science, University of Toronto. The authors gratefully acknowledge the sponsorship of NSERC for our research. We also thank Steve Monckton for his work in identifying the carbon remains embedded in the slag, and Bill Byrick, manager of Sainte-Marie among the Hurons, for permission and encouragement to investigate some of the fascinating details concerning life in Ontario’s earliest permanent community.

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Technological changes at
Les Forges du Saint-Maurice, Québec, 1729-1883

André Bérubé

Introduction
Since the disappearance of Les Forges du Saint-Maurice at the end of the last century, many historians have taken an interest in the lengthy past of this enterprise. They considered the political, socio-economic, administrative and anecdotal aspects of the industry, but barely dealt with the elements of technology. The purpose of this article is to try to fill this gap.

Les Forges du Saint-Maurice was an integrated industrial concern, specializing in the production of semi-finished wrought iron and finished cast iron, that operated almost without interruption between 1734 and 1883. Les Forges first came into being in 1729, when Montréal merchant François Poulin de Francheville submitted to the King of France a request for the right to exploit the bog iron ore deposits on his seigneury of Saint-Maurice. This request, agreed to in 1730, was one of a long series of attempts, previously unsuccessful, to establish a local iron industry in the colony. Several such proposals were held up by a lack of capital from promoters or a too rigid application of the mercantilist policy by the metropolis. In 1730, however, the French minister Maurepas favoured a more liberal policy which would encourage private enterprise. The benefits of an iron industry to the naval construction industry was a factor that caught the attention of administrators.

The choice of the site for Les Forges was based on the following factors:
• the abundance of surface deposits of bog iron ore in the Saint-Maurice region;
• the presence of a vast supply of hardwood and conifers needed in large quantities by this industry;
• the proximity of sandstone and limestone deposits, indispensable materials for the construction and operation of the blast furnace;
• the presence of a water stream, with sufficient flow to provide the hydraulic power needed for the operation of the machinery.

The Period 1729-1735
With all the necessary legal documents in hand, Francheville began his project. He was immediately faced with the problem of technological know-how which did not

FIGURE 1. Map showing the location of Les Forges du Saint-Maurice in Québec.
exist in the colony. So he turned toward New England, and sent a few of his workers there as observers during the summer of 1732 and 1733. The American firms at that time had a reputation for iron production by economical methods. Francheville chose these methods, as his limited

FIGURE 2. The St. Maurice blast furnace complex. In the centre of the photograph, taken around 1870, a cluster of wooden buildings — moulding and model sheds, water-wheel shed — surrounds the blast furnace itself and forms the blast furnace complex. Copyright: Lawrence McDougall, Montréal.

FIGURE 3. The remnants of the blast furnace at the turn of the century. Due to the scarcity of stone building materials in the Trois-Rivières area, the blast furnace was rapidly dismantled to the tuyere level. Copyright: Québec Ministry of Cultural Affairs.

FIGURE 4. Aerial view of the first blast furnace complex after archaeological excavation.
1. The furnace stack. Note the circular shape of the crucible.
2. Space for charcoal sheds.
3. The moulding sheds. The overlapping remnants of these sheds suggest that the moulding branch of the industry was geared to changing markets.
4. Pit for moulding cannons.
5. Support for bellows.
The Period 1735-1846

The failure of the first enterprise convinced colonial authorities to undertake serious feasibility studies before starting the industry again on a more solid footing. The results of the investigation carried out by a young ironmaster from the Champagne region, Pierre-François Olivier de Vézin, confirmed the potential of the industry, but the conclusion was that the technical procedures chosen by Francheville were unprofitable. In 1735, a new company was formed, with one of the participants being Vézin, which proposed to take over the exploitation rights granted to Francheville. The company planned to set up an integrated complex using the indirect reduction process. In this process, cast iron was first produced from iron ore in a blast furnace, then converted in a forge into wrought iron by decarburization; i.e. getting rid of the carbon it contained. The forge was composed of one or more "fineries". A finery was a hearth with a tall chimney where cast iron was melted, then oxidized in front of a tuyere to reduce its carbon content from 3% to 4% to about 0.05%. The application of this technology required heavy investment, however it provided for the production of iron in large quantities. The construction of a blast furnace with a daily capacity of 2.5 tons of cast iron began in 1736, and the furnace was in regular operation by the summer of 1738. The construction of a forge equipped with fineries began in 1736, and a second forge similar to the first was added in 1739.

The breaking-in of technical procedures — especially the adaptation to changing seasons and the rigors of winter — took some ten years before the industry reached a viable rate of production. From that moment on, and until 1846, work went on almost without any technical development. However, if we examine expansion opportunities in regard to markets we note some changes. When the project was initially introduced in 1735, it was planned that most of the sale would be semi-finished wrought iron from the forge, while cast iron that came directly from the blast furnace would provide extra revenue. This prediction proved to be incorrect; the naval construction industry that was to buy all the semi-finished wrought iron never developed and the only customers for this type of product were small craftsmen. Meanwhile, the demand for finished cast iron products — military, agricultural and light industrial equipment, and domestic articles — continually increased, so that by 1800, only one-third of the revenue came from the sale of wrought iron and two-thirds from cast iron products. The only technical changes made in this period, therefore, involved moulding techniques.
The changes consisted of cast iron moulding through the use of a cupola furnace and the construction of a drying oven to bake the mould cores. Until the introduction of the cupola furnace, the cast iron was taken directly from the blast furnace forehearth. The production of wrought iron in the finery was not subject to the pressure of demand and so remained unchanged. The puddling furnace, technically more advanced than the finery, was never introduced at Saint-Maurice.

The legal ownership conditions of Les Forges also greatly restricted technical progress. Les Forges reverted to the government after bankruptcy of the company in 1741. In 1767, the British colonial authorities handed over management to a private company through exploitation leases. An important clause of this lease was that no improvements made were the responsibility of the tenant. This perhaps explained the technical tardiness of certain operators, whose main interest was maximum short-term profit. The entrepreneurs were obviously not inclined to invest in infrastructures that did not belong to them.

The Period 1846-1883

This period covers the last years of operation of Les Forges. It is a time of reorganization, involving some technological updating and a conversion of production to meet the requirements of a new market. After Les Forges were sold by the government to private business in 1846, the introduction of innovations in technical procedures, mainly for the modernization of the blast furnace complex, could begin. These innovations included:
- the introduction of the hot-blast stove, a recent Scottish invention, that provided savings of up to 25% in fuel and an increase in the amount of cast iron produced;
- a metal air compressor to replace the wooden bellows;
- the use of two water-cooled Scotch tuyeres, instead of one ordinary tuyere;
- the replacement of the water wheel by a turbine for powering the air compressor;
- renovation of the blast furnace, mainly an increase of its dimensions, both in height and width, doubling its daily production up to five tons;
- the mechanization of loading operations;
- the introduction of cupola furnaces, cranes, drying ovens, cold ovens and gradual cooling ovens for the moulding of railway wagon wheels;
- the use of bricks rather than sandstone as refractory material inside the blast furnace;
- in regard to the preparation of charcoal, the carbonization of loose wood at the cutting site in the forest was replaced by the use of brick ovens built near the blast furnace.

These changes, introduced between 1846 and 1883, indicate an effort to adapt to new demands, resulting from the expansion of the railway construction industry. During the 1850s, the demand for cast iron for agricultural and domestic purposes paralleled the new demand for industrial material. Managers at Les Forges reacted by renovating certain elements of the technical procedures in order to meet both demands. Technical innovations introduced were thus aimed at increasing the volume of cast iron from moulding. There were no changes in the wrought iron production methods, as this product only represented an insignificant percentage of sales; it finally disappeared in the late 1860s.

During the last twenty years of operation (1863-1883), iron castings production was abandoned. Cast iron, then the only product of Les Forges, was sent to the large foundries in Montreal that specialized in the production of railway equipment. An increase in the demand for cast iron near the end of 1880 resulted in the construction of a second blast furnace in 1881. This new charcoal-fueled blast furnace was of a modern design; it was made of refractory bricks encircled with metallic plates, and was equipped with a circular crucible, three water-cooled Scotch tuyeres, a hot-blast stove and an air compressor run by a turbine assisted by a steam engine.

Conclusion

By the middle of the 19th century, charcoal needed for the production of cast iron was being replaced by coke throughout the Western World. This change, first achieved in England during the 18th century, was taking place in France, the United States and Canada, but not in Quebec because of the absence of coal deposits. After the 1860s, iron production based on charcoal owed its survival to the demand of the railway industry for cast iron with special properties. Les Forges succeeded in remaining temporarily active through innovation borrowed from the coke production process. If it had not been for the financial problems that caused the closing of the firm in 1883, Les Forges du Saint-Maurice could perhaps have survived until 1911, as did the rest of the Quebec charcoal-based industry, with the help of government subsidies.

NOTES

1. More than 200 titles, books or articles dealing with the history of Les Forges du Saint-Maurice have been listed and analyzed by Louise Trotter in a publication entitled Les Forges du Saint-Maurice: Their Historiography, available from Parks Canada in the History and Archaeology series.


4. Between 1865 and 1872, the John McDougall and Company foundry in Montréal signed two contracts with John McDougall and Sons, the company operating Les Forges, stipulating that the Montréal foundry would buy all the cast iron production of Les Forges less 150 long tons per year kept by John McDougall and Sons. These two contracts are kept in the Archives judiciaires de Montréal under the seal of notary W.F. Lightall, n° 3297 and n° 5293.

5. Charcoal iron is used for the moulding of railway wagon wheels. These tempered cast iron wheels have a surprising resistance to wear by friction. A pair of wheels made at the Radnor Forges with this technique were used over a distance of 150,000 miles and exhibited at the 1862 International Exhibition. Bull. Recherches historiques, Vol. 24, (9), pp. 267, 1918.

6. SHALLENBERG, R.H. and AULT, D.A., have shown how in certain American states the charcoal iron industry was able to effectively compete with the coke iron industry until the end of the 19th century through access to high-quality deposits of iron ore and the use of technical innovations borrowed from the coke iron industry. "Raw Material Supply and Technological Change in the American Charcoal Iron Industry", Technology and Culture, Vol. 18 (3), pp. 436-466, 1977.
Ironworking in Upper Canada: Charles Hayes and the Marmora Works

Rita Michael

The Marmora Ironworks was not the earliest ironworks in Ontario, but it was distinguished from other early works in a number of ways. It was established on an ambitious scale as a self-sufficient and, for the time, sophisticated community. In effect it was an iron plantation on the scale and scope of those at Hopewell, Pennsylvania and Saugus, Massachusetts. It was, besides, the only one of the few early ironworks in Upper Canada which operated almost to the end of the nineteenth century, albeit in fits and starts, and which used magnetic ores rather than bog iron ores (limonite). Furthermore it is the only works which has produced archaeological interest over a number of years.

The ironworks is on the east bank of the Crowe River near Highway 7, on Lots 7 and 8, Concession 4, Marmora Township, Hastings County, Ontario. It is the property of the Village of Marmora, and the extant remains encompass about 2.5 acres of the original 2000 or so which made up the original grant.

The mineral riches of the Canadian Shield had been known since the days of the Jesuit missionary-explorers. A consortium headed by Alexander Henry and the Duke of Gloucester planned to exploit the copper near Sault Ste. Marie during the last part of the eighteenth century, but nothing came of the venture. This attempt may have grown out of Lord Dorchester's suggestion that private enterprise be encouraged in order that the iron ore which “abounds” in Canada might be exploited. John Graves Simcoe, the first Lieutenant Governor of Upper Canada was also anxious to see ironworking develop. Simcoe was eager to encourage many enterprises, among which “the development of iron mines.” Governor Gore had been approached by John Mason for help with his ironworks in Norfolk County. Three Americans had applied for permission and support to erect an ironworks on the Gananoque River at the falls to become known as Furnace Falls. They were granted 1000 acres of land in fee simple to encourage their endeavor. Sir Peregrine Maitland, only a few months after taking up his duties as Lieutenant Governor in 1818, intended to visit the “bed of iron ore on the Crow River.” Maitland was to provide a good deal of support, both moral and substantive, to Charles Hayes when he came to Canada in 1820.

The first ironworking in the province seems to have taken place near Chippawa using local bog ores in the last years of the eighteenth century but little is known about the operation. A few years later, about 1800, Sunderland, Sherwood and Jones, all Americans, set up a furnace on the Gananoque River in Lansdowne Township, Leeds County. They exploited the local bog ores and used the river as their power source. However, Lord Selkirk tells us in a note in his 1804 journal, that the works “is mismanaged.” The quality of the bar iron was “not esteemed.” The works closed down a year or two later with Jones obtaining possession of the lands. Jones emerged from what probably was a power struggle for ownership because Selkirk also notes that the “three men...do not agree amongst themselves (sic)”

A few years later an Englishman named John Mason obtained land in Norfolk County. He set up a furnace on Potter's Creek at its entrance into Lake Erie. The area supplied bog ore and everything seemed propitious for a successful enterprise. However Mason had trouble getting both skilled labour and materials at a reasonable rate. He seems especially to have had trouble with labour for he complains to Robert Gourlay in a letter dated 1817 about many problems but especially about labour. “They, meaning iron men, “are the very worst sort of men to manage, colliers not excepted. Not one in a hundred of them but will take every advantage of his master in power.” Mason had asked the government to pay the passage of “five or six families” from England but was turned down. He was unable to find anyone “capable of working the furnace.” Mason died soon after leaving the works to his wife and son who sold it to Joseph Van Norman about 1820. Van Norman was to make a success of the works using innovative technology and good management. When the bog ore ran out in the vicinity it became uneconomical to operate the works and he abandoned it. He later bought the Marmora works, investing a large sum of money, but he was unsuccessful.

The ores were different from those used at Norfolk and required a different technology which he never mastered.

Charles Hayes must have had connections in England which put him in touch with the financial possibilities of ironworking in Upper Canada. He had an agent in England who had worked for him at last throughout the years he was at Marmora, for he tells us in a letter that he blamed his financial situation on his “Agent in London.” But it seems that Hayes over-extended himself by building too much too soon.

Hayes arrived in Upper Canada in the fall of 1820. He had been in communication with Maitland's secretary, Major Hillier, at various times prior to his arrival. He writes to Hillier in the spring to say he was deferring his arrival until “the question of timber duties is settled.” Upon his arrival he went to York to petition the governor for land on which to establish his works. He was granted 1200 acres with the provision that he would erect his works and provide accommodation for his labourers. He was also to reserve another 1200 acres in the area for fuel supplies. He would receive patent to the lands upon having fulfilled his obligation to the Crown by surveying several townships and reserving a percentage for woodlots and for settlers. He cut a road 12 miles long during the winter of 1820-21 from between Lots 13 and 15 at the Sydney-Rawdon Township lines north to the site of the ironworks. The transportation problem was one which would cause him the most trouble and which he did not solve during his tenure as manager.

An ironworks was required to satisfy a number of criteria in order to be an economically viable enterprise: 1) a nearby ore supply; 2) abundant water for power; 3) abundant fuel supply; 4) abundant limestone for flux; 5) a

hill into which the furnace could be built so that a charging bridge could be attached to it from above; 6) near markets with cheap transportation; 7) sufficient capital to carry the project for at least three years; and 8) and, a supply of sufficiently skilled labour. All but the last three criteria seem to have been met at Marmora.

The works were located about 36 miles from Belleville, the nearest town on Lake Ontario. The Crowe River is not navigable below the falls at Marmora because of a series of rapids. The Trent River is the nearest suitable waterway for transport to the Lake, but it also has a series of rapids which needed to be by-passed. Water transport was the preferred route to markets. Land transport was hazardous, time consuming and thus expensive. Hayes saw the possibilities of opening the Trent-Rice Lake route and informed Maitland that it could be made profitable through a toll system. The governor supported the proposition but it was not begun immediately. Transportation, i.e. cheap transportation remained one of the fundamental obstacles to the lack of success of the works. Hayes must have had difficulty finding people to take his products to the Kingston markets because he “offered an extra 100 dollars to the transporter who takes the iron the furthest by water”.[16]

The furnaces had been put in blast in March, 1822[17]. By summer of that same year the Kingston Chronicle carried an advertisement for “mill irons” and “other heavy castings” at the price of 5 dollars per 100 lbs, cash[18]. Thomas Wiltaker was named as agent for Hayes. The following year, Hayes had expanded his product line to include: single and double stoves, dog iron, sleigh shoes, cauldrons and sugar kettles; potash kettles and coolers; pots and bake ovens; cart and wagon boxes, and fencing mill irons[19]. What sort of profit he made if indeed he made anything at this early period is not known. A profit of £2672 is shown for the year of 1825 in the Manahan and Ridley Report of 1837[20], but this may have been inflated to impress the government with the works’ potential. Marmora was not the only ironworks selling its products through the Kingston market. Les Forges du St. Maurice had an agent for its wares in the city and Samuel Shaw was also selling iron products among an assortment of wares[21].

Hayes had started out with enough capital to erect a large number of buildings on his site. Besides the two furnaces, he had a foundry with two trip hammers and four forge fires, carpenter shop, charcoal house, blacksmith, grist mill, saw mill, tannery, bark mill, stone store, a dozen dwelling houses, sheds and barns, adding up to a substantial community. The return on his capital investment must have been slow and the needs of the works required a constant infusion of funds to meet wages and expenses. In an effort to find more cash he offered for sale several lots suitable for dry goods or other tradesmen enterprises. Applicants had to submit a certificate of good conduct[22]. By October 1824, Hayes was forced to turn over all his real and personal assets in trust for the benefit of his creditors to Peter McGill, A. Manahan and Robert Hayes. He was to continue as manager of the works until his return to England to try and secure further funding, but he was unsuccessful and was never able to redeem his ironworks[23].

The origin of the labour pool at Marmora is not known but it is likely that it was made up of recent immigrants, probably from Ireland and Britain. Hayes seems to have had the same sort of problems with his workers as did John Mason. In a letter to Hillier he writes that he was delayed by “Wicked combinations among my work people — whom I have no power to control”[24]. Some of his workers were unskilled and inefficient, and he was “much plagued by them”[25]. Drunkenness was part of the problem because Hayes tried to prevent licensed taverns from operating in the vicinity of the works[26]. The environment was a harsh one and the men perhaps not known to each other nor used to the process of ironworking itself sought solace in alcohol. Unlike Les Forges du St. Maurice and other similar 'plantations' where a labour force was maintained through subsequent generations of workers and their descendents, Marmora was in the process of building a new labour force as well as a business[27]. Besides coping with the cold and isolation of a Canadian winter, there were the black flies and mosquitoes of summer. Unfortunately, we know little about the relationships between owner and labour at Marmora, only that the worker was treated in a paternal manner. He was lodged on the site, provided with accommodation by the company, and must from necessity purchase his food and clothing from the company store. He may have had a garden, but that would take time and effort to achieve. Alcohol abuse continued to be a problem through the Marmora years. Joseph Gander an itinerant social minister, visited various people in the area of the works helping tend the sick and those trying to cope with alcohol abuse[28].

The Hayes years at Marmora amounted to little more than four, but they were years of incredible activity, carving out a new world. They were years of great financial loss and eventual despair. A renewed interest in the ironworks in the last few years has produced a better understanding of its history. Newly located maps indicate its physical layout and archival re search is gradually fitting the Marmora puzzle together. With the addition of archaeology as a resource for producing further evidence it is now possible to learn more about the technology used at this early period and through metallurgical analyses determine how successful that technology was.

In response to possible destruction in 1978, the regional archaeologist for the Ministry of Culture and Recreation, Phillip Wright, conducted testing on the site. The archival record was meagre and mis-interpreted. Testing was carried out in the north end in that area occupied by a large lumber mill owned by the Pearce Company at the turn of the century[29]. The buildings from this enterprise had obliterated most of the ironworks except, it appears, the foundation of the furnace house and the tailraces which serviced it and the forge. When the Pearce Company purchased the site in 1883 it did not destroy the furnace immediately. When it was destroyed is not known but a house on Main St. has a stone from one of the furnaces set into its foundation. The stone is stamped “Farnley Ironworks Wortley Leeds”. The owner of the house thinks it (the house) is about fifty years old. No doubt other stones were carried away and used for similar purposes.

The site is a difficult one; it is compressed between the 30-foot limestone cliff on the east and the Crowe River on the west. The road which services the site is now much higher than originally and consequently the furnace house area is swampy because water seeps from the cliff and has no outlet as the tailraces are blocked.

Slag, iron and soil samples have been taken for analysis. The slag sample appears to be from a cold-blast furnace and it contains, besides the limestone, small pieces of unburnt charcoal. The ores used at Marmora contained
a high sulphur content and required special knowledge to be worked successfully. Charcoal may not have been the best fuel for this purpose, but the iron from Marmora was considered superior in every way, so perhaps the iron founder was able to maintain the correct recipe at this early period.

The ironworks underwent a number of re-building periods as owners changed with regularity after the Hayes years. However, no superstructures remain and archaeology will uncover the foundations only. This will be useful nevertheless as dimensions will be obtained and possibly fragments of discarded iron products, which can be studied. The problems of the furnace bricks must be sorted out — their place of origin, cost and date imported. It would be useful to learn more about the workers, about their origins and lifestyle at Marmora.

REFERENCES
8. INNIS, MARY QUAYLE, “The Industrial Development of Ontario, 1783-1820”. In Historical Essays on Upper Canada, p. 140.
10. MASON, JOHN, see note 5.
11. Ibid.
21. Kingston Chronicle. August 13, 1819 advertises St. Maurice Stoves; October 24, 1823. Besides Shaw, a John Watkins was also acting as agent for stoves, but whose the ad does not say.

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Nineteenth-century Nova Scotia iron works

Christopher Andree

Introduction

Years before iron and steel works were established at Sydney, several thriving although short-lived iron works operated in Nova Scotia. These furnaces reflected the general nineteenth-century North American experience with iron works. At the beginning of the century, small furnaces struggled to supply local needs. Charcoal was the principal fuel and water powered most furnace equipment. By the end of the century, a few huge primary iron and steel works dominated production for Nova Scotia and the rest of Canada.

Two sites achieved prominence in the nineteenth-century Nova Scotia iron industry: Londonderry and Ferrona (Fig. 1). Londonderry, the larger and more lasting of the two sites, is today a small village on the south flank of the Cobequid Mountains, about 30 km west of Truro. Prior to the establishment of an iron works in 1849, no community existed. At the peak of operation in the 1880s, however, over 2600 people were located there. The second site, Ferrona, today is a mere crossroads on the map, about 10 km south of New Glasgow. Following the construction of a blast furnace there in 1892, a small village grew up around the furnace. But once the furnace was blown out in 1904, the population gradually drifted away.

Early History

The history of Nova Scotia’s iron industry began in 1604 when Samuel de Champlain became the first European to identify iron ore in the future province[2]. Despite this early acknowledgment, iron was not smelted commercially in Nova Scotia for another two hundred years. Although primitive smelters of the period produced more iron than could be used locally, early settlers still depended on imports of iron from Britain and the United States.

By the late eighteenth century, entrepreneurs began to make the first tentative attempts at smelting ore in Nova Scotia. The earliest effort to smelt iron occurred at Nictaux in 1789, through the labour of some New Englanders. Apparently the expense associated with operating this furnace led to its abandonment after a short time[2]. About the same time construction started on a mine and furnace beside the Sissiboo River on St. Mary’s Bay. One or two of the original promoters soon lost interest in the project and it lapsed before actual operation could begin[3]. A third attempt to mine and manufacture ore was carried out beside the Moose River near Clementsport in 1826. Although a well-designed furnace and foundry were constructed, the site was never very successful. Like earlier operations, the furnace had been built in advance of the colony’s needs for iron. The furnace ran for a few years and was then abandoned. In 1862, 1872, 1873 and 1885, the works were again in blast, but with little commercial success[4].

The most innovative early smelting efforts were under-

taken at New Glasgow by the General Mining Association. In 1823, King George IV granted to his brother the Duke of York all reserved mines and minerals in Nova Scotia for 60 years. In turn, the Duke transferred these rights to one of his large creditors, the jewellers Rundell, Bridge and Rundell. This group then formed the General Mining Association in 1826 to undertake the exploitation of its property.

The Association was attracted to Nova Scotia primarily because of the coal deposits, but it also perceived iron smelting as a possible market for its coal. In 1829, a coke blast furnace was constructed near New Glasgow and about 45 tonnes of iron produced; this was the first use of coke for metallurgical purposes in Canada. The iron turned out to be a poorer quality than expected and labour problems prevented the experiment of smelting iron from continuing[5]. The General Mining Association experiment at New Glasgow marked the end of the pioneer period of iron manufacture and the technological problems that plagued early operations. Subsequent iron works overcame these problems and prospered from an increased demand for iron.

Londonderry

Twenty years after the efforts at New Glasgow, the first sustained mining and smelting operation in Nova Scotia commenced at Londonderry. Detailed surveys of the district were conducted by William Dawson in 1845-46 and 1849, and by Abraham Gesner in 1845. Favourable reports by these men led to the formation of the Acadian Iron Mining Association, a British company organized about 1848, to develop mines and foundries at Londonderry[6]. Because iron ore had not had not been reserved for the Crown in the deeding of the land, the property was exempt from the control of the General Mining Association.

Unlike earlier smelting enterprises in Nova Scotia, the works at Londonderry had a large and ready market for its iron in Sheffield, England. In the eighteenth century, Sheffield, had developed a world reputation for the excellence of its steel. Much of its pig iron was imported from Sweden and Russia. The 1830s and 1840s saw an expanded demand for Sheffield steel which required new sources of pig iron. The high-quality ore at Londonderry seemed to be a potential source. How and why an unknown and isolated corner of the British Empire attracted the interest of Sheffield steel manufacturers is something of a mystery. The only clue may be that one of the British promoters of Londonderry, Charles Archibald, had come from Truro and could have been familiar with reports of iron deposits in the vicinity of Londonderry. His brother married the daughter of Richard Smith, a prominent figure in the General Mining Association, providing perhaps an introduction into the British iron and steel world.

The Acadian Iron Mining Association began mining its property in 1849 or 1850. Five Catalan forges were constructed on the west bank of the Great Village River. These forges were never intended to be permanent, but

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could be built quickly as a stop-gap measure until a proper blast furnace could be erected. The company completed a charcoal blast furnace in 1855. Unlike the monolithic tapered stone furnaces that were still being built in North America, the blast furnace of the Acadian Iron Mining Association was built to modern British design. The furnace stood 10.7 metres high and was constructed of firebrick cased in iron. The furnace had a yield of approximately 7.1 tonnes of iron per day, but operated only 6 to 7 months of the year. Water power provided the blast for the furnace\(^5\). Charcoal for the furnace was burned in heaps in the woods by farmers who contracted with the company to supply fuel.

Limestone was quarried locally, probably at Lorneville. In addition, large deposits of ankerite, a limestone with small quantities of iron ore, were mined from a site adjacent to the blast furnace. Ankerite increased the iron content of the furnace charge, boosting furnace capacity\(^6\).

The association with steel manufacturers in Sheffield provided a guaranteed market for the furnace output of Londonderry. All pig iron was shipped in an unrefined form to be manufactured into steel in England. The crucible method, a commonly used technique of manufacturing steel at Sheffield, required a high-quality pig iron such as that found at Londonderry. By the early 1860s, however, the crucible steel method for steel production was displaced by the Bessemer process, which was cheaper and could use lower-grade pig iron. Consequently, the British market for Londonderry iron collapsed in the mid-1860s. Fortunately for the company, increasing demand in Canada for railway car wheels and crucible steel that could be manufactured at Londonderry more than compensated for the loss of Sheffield markets\(^7\). With these markets established, the company continued as a profitable concern throughout the 1860s.

In the early 1870s, the company came to the attention of Hugh Allan and George Stephen of Montreal. These men represented a consortium promoting the construction of the Canadian Pacific Railway. They saw that the granting of a railway charter would create a large demand for steel rail which could not be met by any existing Canadian plant. Consequently, they undertook a search for a suitable facility for the manufacture of rails\(^8\). At the time, Londonderry, the largest and best equipped iron works in Canada (Fig. 2), had a low production capacity, but the area seemed to possess significant reserves of high-quality iron ore. An independent study by the Geological Survey of Canada in 1872 indicated a reserve of 4,789,000 tonnes or sufficient for 53 years at the contemplated rate of production\(^9\).

In 1874, a group of British and Canadian investors, including Stephen, purchased the company and renamed it the Steel Company of Canada (no relationship to the modern company of the same name). Unfortunately, the Canadian Pacific Railway scheme did not go ahead as planned in 1873, thus destroying the possibility of large Canadian markets. Nevertheless, the promoters proceeded with their plans to create one of the largest iron and steel complexes in North America.

The sponsors of the Canadian Pacific created a plant at Londonderry capable of producing 27,000 to 45,000 tonnes of steel per annum\(^10\). Two coke furnaces with a total capacity of 1,200 tonnes per week were constructed; the first being blown-in in 1877, the second in 1879\(^11\). Coal for coke in the blast furnaces was obtained from collieries around New Glasgow and from a colliery which the company purchased at Springhill\(^12\).

Essential to the Steel Company's expansion plans was the introduction of a new steel-making method known as the Siemens process. Developed by Charles William Siemens during the 1860s, it was a technique for converting iron ore directly into steel. In 1873, Siemens himself commented on the feasibility of using this process at Londonderry, and later became president of the Steel Company\(^13\).

Despite Siemens' association with 'the works, his process for manufacturing steel failed. A few tonnes of steel were produced, but the furnaces closed after only six months. Precise reasons for this failure were never given, but one report stated that the process had not worked as designed because the steel required "further processing"\(^14\). In connection with a steel works he established in Wales, where another failure of his process occurred, an observer wrote:

"As a man of science, Dr. Siemens had a great reputation. . . . as a practical iron master, Dr. Siemens was a failure"\(^15\).

The incorporation of the Steel Company of Canada marked the zenith for the fortunes of iron mining and smelting at Londonderry. For the rest of the century, the
operation moved from one bankruptcy to another and the property gradually deteriorated. Londonderry was an inherently poor location to pursue an iron smelting operation, because transportation was expensive and could not rely on nearby water facilities. The company was dependent upon expensive rail transport to serve distant markets in Ontario and Quebec. As well, the projections of ore reserves provided by the Geological Survey of Canada proved greatly overstated. By the 1890s, Londonderry was running out of inexpensive ore and had to import ore from Torbrook, over 200 km away. Finally, the company suffered from several expensive mistakes, including failure of the Siemens process and the unusable quality of coal from a colliery the company purchased at Springhill.

In 1891, one of the two blast furnaces was shut down. Five years later, mining ceased and the second blast furnace was closed. In 1899, the company, then known as the Londonderry Iron Company, went into liquidation. Until the foundry burned in 1902, it continued to produce cast iron pipes.

The property was then purchased by T.J. Drummond of Montreal, with the intention of rebuilding the pipe foundry and blast furnace and reopening the mines (Fig. 3). By the end of 1903, the mines and furnaces had been cleaned and re-built and in January of 1904, one blast furnace was once again blown-in (Fig. 4). Despite a few promising years of operation, however, a financial depression in 1907-08 hastened the company's demise. Mining ceased and the furnace shut down for the last time in March of 1908.

In June 1908, the Canada Iron Company came into existence as a re-organization of Drummond’s mines, furnaces and foundries in Ontario, Quebec and Nova Scotia. The pipe foundry at Londonderry continued to operate until September 1914. Despite hopes of the people there that the foundry might remain open, the plant had been in difficulty since 1913 and after it closed, all operations were moved to Trois-Rivières, Québec. Thus ended the iron industry at Londonderry.

The initial success of the iron works at Londonderry arose from the presence of high-quality ore and a specialized market in England. As long as the demand for quality iron remained, the site prospered. Decline began in the early 1870s with the ill-fated expansion of the works to manufacture pig iron and castings that had no particular competitive advantage over other furnaces more conveniently located to serve markets in Ontario and Quebec. Consequently, costs of raw materials and transportation became critical. Without any benefit to its location, the works gradually lost business to other, better-sited works.

**Ferrona**

Although the furnace at Ferrona was in blast for only 12 years, it was more successful than Londonderry because it contributed to the rise of the Nova Scotia Steel and Coal Company, which eventually established a major iron and steel works at Sydney. This furnace had its humble beginnings as a small foundry established in 1872 in New Glasgow. Ten years later, the proprietors began to manufacture steel from imported pig iron and scrap steel. Soon after, the company decided to establish a local supply of pig iron for their foundry by exploiting iron deposits of the East River area near Bridgeville. Consequently, a subsidiary company, the New Glasgow Iron, Coal & Railway Company, was incorporated in 1888 for the purpose of developing these properties. In 1895, the railway company amalgamated with the steel company to form the Nova Scotia Steel & Coal Company.

Developmental work on the company's properties began in 1889. A blast furnace, ore washing plant and coke oven were constructed at Ferrona and a railway, completed in 1891, was extended to the iron deposits near Bridgeville. Limestone was quarried at Springville and shipped to Ferrona. Pig iron smelted at Ferrona was then shipped to New Glasgow for conversion into steel. The steel plant at New Glasgow used the Siemens-Martin open hearth process, a somewhat different and more successful process than that tried unsuccessfully at Londonderry twenty years before.

Whether due to insufficient prospecting or to poor judgment by management, the ore supplies at East River were soon found to be totally unsatisfactory for the needs of the Ferrona blast furnace. The company sponsored exploration at various other iron deposits in Nova Scotia and for a while even imported ore from Spain. In all cases, supplies of ore proved inadequate. The turning point for the fortunes of the Nova Scotia Steel & Coal Company occurred in 1893 with the purchase of iron deposits at Bell Island, Newfoundland. This vast deposit insured a virtually unlimited supply of cheap ore and permitted the company to embark on a major expansion program to
meet demands for iron and steel created by an improvement in the Canadian market.

Proven reserves of the Bell Island mines encouraged the Nova Scotia Steel Company to embark on a large expansion of its iron and steel works. A search for a suitable site determined that Sydney in Cape Breton was the most appropriate location. Large coal deposits and a fine, deep-water harbour made the location especially attractive. In 1900, the Nova Scotia Steel Company purchased the General Mining Association's collieries at North Sydney and began erecting blast furnaces and a steel plant. The blast furnace at Ferrona closed in 1904 and the following year the new steel works opened at North Sydney.27

Closure of the blast furnace at Ferrona marked the end of iron smelting in Pictou County. The new facilities opened at North Sydney provided a major, world-class iron and steel company, but its history is beyond the scope of this paper.

Other Nineteenth-Century Sites

One of the most ill-fated iron works in Nova Scotia was located at Bridgeville and constructed soon after the nearby furnace at Ferrona was blown-in. In 1892, the Pictou Charcoal Iron Company blew-in a charcoal blast furnace in Bridgeville. At this late date, when most companies in North America had switched to coke as a blast-furnace fuel, this company adopted the antiquated method of manufacturing pig iron with charcoal. Until the technology permitted the production of cheap, high-quality steel, charcoal iron had to be used in high-grade castings and wrought iron, especially for the manufacture of railway car wheels. Charcoal iron was more expensive to produce than coke iron, but sold for a premium price. By the 1890s, however, steel metallurgy had caught up with the last remaining advantages of charcoal iron. Open-hearth steel furnaces and other developments around this time completely destroyed any remaining markets for charcoal iron. Against this background, the Pictou Charcoal Iron Company's furnace was technologically obsolete from the moment it opened. Moreover, the choice of Bridgeville as the site of a charcoal furnace seemed unusual because of its proximity to a large coal field, although it was also near sources of ore, flux and large stands of hardwood.

A small furnace with an annual capacity of approximately 4500 tonnes of pig iron was constructed, although in fact it never produced more than 1500 tonnes in any year. After only two years in operation, the furnace closed in 1895. For a few years after, mining was continued and the ore sold to the Nova Scotia Steel Company to smelt at Ferrona. In 1899, even mining ceased.28

The last site in Nova Scotia to be associated with the domestic iron and steel industry was Nictaux-Torbook. As mentioned, some of the earliest smelting operations in Nova Scotia were located in this area. Later the mining of ore, rather than smelting, became the area’s important contribution to the iron industry of the province.

In 1891, the Londonderry Iron Company developed the iron ore potential of Torbrook as a source of supply for its blast furnace at Londonderry by incorporating the Torbrook Mining Company to open a mine near Torbrook.29 After operations began in 1892, ore was shipped to Londonderry via the Dominion Atlantic Railway. Mining continued successfully until 1896, when the furnace at Londonderry was blown-out, terminating the primary destination for Torbrook ore. Without any market for its ore, the mine closed in July 1896. In 1905, following the re-opening of the Londonderry furnace, mining activity resumed at Torbrook. Three years later, closure of the blast furnace for a second time eliminated the main market for Nictaux ore. The company, having invested considerable funds to develop the property, decided to try instead to establish export markets for the ore.30

An ore shipping port was constructed at Port Wade, 88 km away on the Bay of Fundy, where a dock and ore storage bin were constructed in 191C. Several thousand tonnes of ore were exported to Pennsylvania, Scotland and England between 1910 and 1913. When the volume of sales did not justify continuing operation of this facility, the works were closed completely in 1913.31 This was the last commercial iron mining operation in Nova Scotia.

Conclusions

The pre-1900 iron works of Nova Scotia represented traditional nineteenth-century uses of iron resources. The industry was characterized by numerous small operators extracting local ore, each with a small production capability. Although these conditions matched the needs of the period, they were unsuited to the increased demands for iron and steel products by the turn of the century. As well, locations of nineteenth-century plants were inappropriate for twentieth-century requirements. Londonderry began by mining ore literally within metres of the blast furnace. Charcoal was provided from the forests surrounding the works. By the end of its history, iron ore came 220 km from Nictaux and coke 100 km from New Glasgow. Thus neither fuel nor ore, the original attractions of the site, played any role in the operation of the furnace in its final years. The works at Ferrona lacked the necessary ore supplies to increase production. Nictaux ore deposits proved to be too small to supply the modern needs of an iron and steel works. Consequently, all of the nineteenth-century iron works disappeared to be replaced by two large iron and steel works at Sydney depending on coal deposits nearby, ore imported from Newfoundland and good transport connections.

Footnotes

9. HOW, Mineralogy, p. 89; and Halifax Herald, August 15, 1903.
13. Ibid., p. 75; and Nova Scotia Department of Mines Annual Report (1875).
15. Nova Scotia, Mining, Report (1874); Donald, Canadian Iron and Steel, p. 59; Steel Company, Reports Upon the Property, pp. 56-57; and Harrington, Iron Ores of Canada, p. 259.
25. GOUDGE, Limestones of Canada, p. 97.
27. CANTLEY, Iron and Steel, pp. 328-329.
31. Ibid., p. 54; and Maritime Mining Record (Stellarton, N.S.), March 27, 1907.
Discovery and technological change:  
the origins of steelmaking at Sydney, Nova Scotia

Kris Inwood

Introduction
In 1879, when the Dominion government initiated its controversial program of industrial promotion, iron smelting in Canada was limited to one company producing coke iron on the mainland of Nova Scotia and three small charcoal blast furnaces in Quebec. There was no steel production. By the eve of the First World War, iron and steel were being produced in very large quantities, the companies active in 1879 had disappeared and new centres of integrated iron and rolled steel production at Hamilton, Sydney and Sault Ste. Marie had come to resemble the industry as we know it today.

The most dramatic change was that at Sydney, Nova Scotia, where in the early 1900s, two steelmaking companies established works. The Sydney developments have been explained as the result of resource discovery and technical change. W.J. Donald, in his authoritative account of the Canadian iron and steel industry, observed:1

“The most rapid growth of the Canadian iron and steel industry has occurred since 1897. . . . The remarkable growth has been caused largely by the new conditions of the industry. The discovery and use of Newfoundland ores, together with the availability of Cape Breton coal, has made the Nova Scotia industry not only possible but profitable. . . . It is a safe conclusion that the Canadian industry would not have been an important one, had it not been for the discovery of the Michipicoten and especially the Wabana (Newfoundland) ores.”

Donald also argues that metallurgical progress had been necessary to render the ores valuable.

“Newfoundland ores . . . contain a considerable amount of phosphorus, and were made available for steel-making by recent inventions.”

The purpose of this paper is to re-examine Donald’s story. It will be argued that the origin of steel-making at Sydney is somewhat more complicated than Donald has indicated. The difficulty with Newfoundland ore extended beyond its phosphorus content and a resolution required more than the basic open hearth furnace identified by Donald. Moreover metallurgical change was only one of several developments necessary to induce industrial investment at Sydney. Surprisingly, however, the discovery of iron ore in the late nineteenth century does not appear to be part of the explanation.

The “Discovery” of Iron Ore
A belated discovery of iron ore in Newfoundland would not be entirely surprising in that knowledge of Newfoundland resources appears to have been limited as late as the mid-nineteenth century. D. Thompson, the historian of

surveying in Canada, points to “a curious paradox. Newfoundland, one of the first parts of North America to be discovered by Old World navigators, became almost the last part of the New World to be systematically surveyed”2. In spite of this paradox, iron ore at Bell Island, Newfoundland, had been known at least as early as the 1570s, during which decade ore samples were sent to England. Detailed plans for mining and smelting the Bell Island ore, which 300 years later provided the basis for industrial expansion at Sydney, were first formulated during the early 1600s. Although nothing of a sustained character came of that effort the remains of an abandoned iron mine were still known to the residents of Bell Island two centuries later and were mentioned in a book first published in 1818 that remains an important source of information on early Newfoundland3.

Within twenty years of the appearance of the book the Newfoundland legislature commissioned a geologist, J.B. Jukes, to survey the colony’s mineral resources4. Jukes failed to visit the Bell Island iron mine, did not mention it in his report, and probably never learned of it. The immediate reaction to Jukes’ report was mixed although contemporaries softened their criticism by noting the immense difficulty encountered by an inexperienced geologist making the first systematic survey of a large island during two summers’ fieldwork.

Jukes likely would have encountered the ore except for a curious combination of circumstances. The geologist was unable to stop for lengthy discussions with local inhabitants and appears not to have read the book published twenty years earlier. Moreover Bell Island lay in Conception Bay, which was then the centre of Newfoundland population and commerce. In sense the iron ore lay under the very noses of the geologist and colonial officials. A false confidence may have been reinforced by the plausible but apparently misleading belief in a resemblance between the strata underlying Bell Island and those exposed on the adjacent mainland. Idiosyncratic influences were also at work. The ore was situated on the seaward end of Bell Island, which was inspected by Jukes late one afternoon. Not untypical Newfoundland weather had set in. Fog impaired visibility and a heavy sea cut short the geological inspection by reducing Jukes, in his own words, to “a state of body very unfavourable to the appreciation of the picturesque, that of incipient seasickness”.

The failure of the sea-sick Jukes to mention or even to recognize the Bell Island ore was an influential omission. Subsequent geologists and public officials used Jukes’ report as the benchmark it was intended to be. When Newfoundland established a permanent geological survey on the British and Canadian model in 1864, colonial authorities may have tended to defeat their own ends by relying upon professional geologists who took their cue from Jukes. The next thirty years of geological investigation failed to locate the Bell Island iron ore. Valueless iron ore was discovered at several sites but not on Bell Island, which the Geological Survey of Newfoundland never examined closely.

The ore was not completely forgotten, however. Private merchants appear to have taken an interest independently of the efforts of public officials. The old iron mine was examined in the early 1870s and again in the late 1880s by which time efforts were underway to sell the mine to British interests. A Nova Scotia iron company also appears to have examined the mine in 1888. Neither the British ore markets nor the Nova Scotia company took a serious interest in the ore although the latter was desperately short of workable reserves and the British industry was fast increasing its dependence on imported ore.  

The Sydney Steelmakers

The Geological Survey of Newfoundland finally learned of the Bell Island iron ore late in 1892, just as commercial interest was becoming more serious. The first step to commercial development of the ore was taken by a Concepcion Bay native working on a street railway in Boston. The man told his employers about the ore. The street railway organizers happened to be in the midst of amalgamating several Cape Breton coal mines into the Dominion Coal Company (DCC). This group, centering on F.S. Pearson and H.M. Whitney of New England and B.F. Pearson of Halifax, was sufficiently interested to ask for an appraisal of the ore and to obtain an option in 1893. Somewhat surprisingly in retrospect, the option was allowed to lapse, permitting a rival group, the Nova Scotia Steel and Coal Co. (NSSC) to acquire the ore for use in a Pictou County blast furnace established by the NSSC in 1892. There was some discussion of a joint DCC-NSSC enterprise but the plan was not implemented.

It is far from clear why the DCC did not proceed in 1893 with an ironworks to complement its coal mines in Cape Breton. One local source indicates that the DCC was at that time preoccupied with putting the coal merger in working order. Another local source claims that the DCC had received an unfavourable report on the ore. Both stories may be correct, although the latter is especially intriguing. The silicious and mildly phosphoric Bell Island ore posed a perplexing metallurgical problem, particularly if used together with the sulphuric Cape Breton coals. The technical expertise of the DCC principals, which was not widely respected, may not have been adequate to the challenge.

The NSSC was certainly better prepared than the DCC to use the Bell Island ores. The former had the incentive of owning a blast furnace, which, once constructed, was unexpectedly found to be without an adequate local ore supply. The ensuing crisis had forced the NSSC to rely on mainland Nova Scotia ore with a chemical composition similar to that of the Bell Island ore. The result may have been that the NSSC was partially prepared to meet the metallurgical challenge of using Bell Island ore.

Although the NSSC purchased the Bell Island reserves in 1895 for use in its Pictou County furnace, a pooling of resources between DCC and NSSC was a promising idea. In 1893 the NSSC lacked coal and ore but had metallurgical experience and impeccable Tory credentials that might prove useful as long as Conservatives formed the government in Ottawa. The DCC principals lacked experience in coal and iron but possessed a number of coal mines, first option on the Bell Island ore and a higher profile on American security markets. Together the two groups might have reduced the unit costs of overhead expenses. The scheme for joint action foundered, however, because the NSSC principals were unwilling to commit themselves to the very large scale of enterprise advocated by the DCC principals. The NSSC acknowledged that a smaller scale of production would increase substantially the cost of rolling but nevertheless preferred a smaller steelworks not requiring entry into the competitive heavy rail trade.

A renewed attempt at NSSC-DCC cooperation was made in the late 1890s but the two groups continued to differ on the wisdom of undertaking large-scale production. By this point, however, the DCC had organized the coal business as well as it ever would; the international iron markets had recovered from an extended depression, the Liberal party had taken office in Ottawa and, once in office, the Liberals quickly showed a willingness to suppress the free trade sentiment on which they had campaigned. The favourable turn of circumstance prompted the Liberal-aligned DCC to proceed with plans for a steelworks without equal participation by the NSSC. Thus the DCC in late 1898 purchased from the NSSC the greater part of the Bell Island ore and began construction of a fully integrated steelworks at Sydney that was incorporated as the Dominion Iron and Steel Company (DISC). Production began in 1901. The Liberal Minister of Finance, under whose aegis DISC was organized, played an active role in selecting directors for the new corporation. Carefully chosen Liberals were given a majority of seats on the Board and it was agreed to deny representation to the NSSC.

Although excluded from participation in DISC even in a secondary capacity the NSSC principals could not be stopped from establishing their own steelworks at Sydney. The NSSC used the profits from resale of the greater part of its Bell Island reserves to purchase in 1899 a large Sydney coal company and the only major Cape Breton mine that had not entered into the DCC merger. With this new coal source the NSSC was able to move its furnaces from Pictou County to Cape Breton where, as a result, two independently owned steelworks faced each other across Sydney harbour by 1905. The two companies relied upon immediately adjacent sources of iron ore and coal, and produced complementary goods. In the next fifteen years repeated attempts to merge DISC and NSSC failed because the smaller NSSC was a decidedly reluctant bride. A marriage of convenience was finally arranged in 1921 giving life to a succession of corporate offspring including, most recently, SYSCO.

The first decade of the twentieth century was a good one for iron mining at Bell Island and the Sydney steelworks that it supported. By the eve of the First World War, Sydney’s share of Canadian ferrous metal production was already declining and Bell Island ore shipments to all destinations had reached a level that remained substantially unsurpassed until the 1950s. Bell Island was recognized to be one of the largest and lowest cost sources of iron ore in the world. This advantage, however, was seriously undermined by the quality of the ore. A relatively high content of silicon and phosphorus increased steel production costs and obstructed efforts to produce a quality of steel comparable to that made from Spanish and Lake Superior ore. The peculiar quality of the ore helps to explain why it became commercially valuable in the final decades of the nineteenth century.

The Metallurgical Challenge

The metallurgical challenge of producing steel from Bell Island ores at Sydney arose from three elements: silicon,
sulphur and phosphorus. Cape Breton coal was sulphuric while the Bell Island ore was siliceous and mildly phosphoric. Each of the impurities would have been independently troublesome but in combination the dilemma was aggravated considerably. The easiest way to lower silicon, for example, was to reduce furnace temperature and flux with added limestone, both of which increased consumption of the sulphuric coke. The mere presence of a siliceous slag induced absorption of sulphur by the metal. A similar dilemma defined the silicon-phosphorus combination. A high phosphorus pig was tolerable if its silicon content was low; a low-phosphorus, high-silicon combination was also acceptable. For example, acid Bessemer converters and acid open-hearth furnaces required a low phosphorus, high silicon pig. Even within the acid technology, however, open-hearth furnaces found silicon difficult to control and converters released too much energy if the silicon level was excessive. Basic Bessemer, on the other hand, permitted, indeed it required, a highly phosphoric pig. Unfortunately economical production of a consistent steel required that pig charged to the basic converter and open-hearth furnace not be heavily siliceous.

The fundamental reason for this state of affairs lay with the nature of the oxidation processes in acid and basic technology respectively. An acid lining oxidized silicon, which was thereby eliminated; in the process energy was released to the steelmaking bath. However, there was no easy way to eliminate the phosphorus. A basic lining, by contrast, oxidized phosphorus, thereby eliminating it and releasing energy, but left the silicon to be removed with difficulty into the slag.

The difficulty of eliminating these impurities helps to explain why the Bell Island ores had not been commercially valuable before the 1890s. Cast iron, the use of which dominated ferrous metal markets before 1850, required a metal with minimal phosphorus. Even after the turn of the century the Nova Scotia furnaces could not run foundry iron with more than 50% Bell Island ore in the mixture. In the third quarter of the nineteenth century the use of ferrous metal for cast iron was overtaken by its use as wrought iron, which in turn was displaced by acid Bessemer steel. The siliceous nature of the Bell Island ores, however, would have produced a costly wrought iron and the phosphorus level was sufficient to render it useless in the acid Bessemer converter. Bessemer technology had a major impact in bringing siliceous ores into use, but the valorization was limited to low phosphorus ores such as those in Northern Spain, Elba, Cumberland and French North Africa.

In the fourth quarter of the nineteenth century basic metallurgy emerged to permit use of high phosphorus ores, but at first it was necessary to have very high phosphorus and very low silicon levels. The Minette ores of western Europe provide the most dramatic example of ores suddenly valorized by the introduction of basic Bessemer technology in 1879. Only in the late 1880s did the basic open-hearth furnace become a useful steelmaking process. Production costs for open-hearth furnaces were falling relative to Bessemer costs but the relative decline was slowest for basic open hearth. Basic steel of both varieties remained more costly to produce than their acid equivalents without a fully offsetting quality advantage. Hence ores and pig fit only for basic metallurgy, especially in the open-hearth furnace, remained less valuable than acid ores and could be exploited only in conditions of low-cost ore transportation and extraction.

With the commercial development of the basic open-hearth furnace in the late 1880s, the phosphorus problem was substantially reduced. Acid Bessemer would accept very low phosphorus pig, basic Bessemer very high phosphorus pig and basic open hearth some of the intermediate range. The solution was not yet complete in that basic open-hearth furnaces still produced a high-cost steel, the phosphorus gap had been narrowed rather than eliminated and the silicon problem remained. Low silicon pig could be converted as could high silicon pig if the phosphorus was low, but moderate levels of silicon, and silicon in combination with phosphorus, continued to pose a problem. As late as 1912, it was costly to reduce silicon levels and the theoretical understanding of silicon in steel was considered inadequate. The problem might have been less bothersome if silicon had not been undesirable in finished steel. For several decades steel rails called for especially low silicon. Even if the silicon was left in the metal, costs were increased because the productivity of virtually all inputs to the blast furnace and steel furnace was adversely affected. The silicon-phosphorus combination plagued Cleveland (England) ironmasters trying to turn to steel, probably delayed successful development of Lapland ores and prompted contemporary assessments of the Bell Island ore as being low grade although, at last, usable.

Not surprisingly the idea of smelting Bell Island ore with Cape Breton coal met with some skepticism. DISC officials and promoters were called upon publicly to assuage concern about sulphur in the coal and silicon and phosphorus in the ore. Public disclaimers notwithstanding, DISC encountered considerable difficulty in using its resource combination. Any new steel company could expect an initial period of learning to use its resources to best advantage. The DISC experience, however, as described by M.R. Campbell, went beyond the normal learning process.

Campbell describes “serious operating difficulties” during the first 25 years of DISC production. Successive steelmaking techniques were introduced, experimented with, and then rejected or modified considerably. The succession of techniques served to keep the steelworks in operation although the product was frequently low quality and costly. Eventually, in the mid-1920s, the accumulation of metallurgical experience at Sydney led to the innovation of a process for slag removal that became standard practice throughout North America and reduced to manageable proportions the cost and quality dilemma of smelting with Cape Breton coal and Bell Island ores. Campbell observes that the slag removal process, “as now carried out in the Sydney open-hearth furnaces, is the evolution of fifty years of trial with every known method of making basic steel. It had been developed into the most flexible and effective means of solving what is undoubtedly one of the most difficult problems in the whole industry.”

Important as is the slag removal process, our interest focusses on the earlier techniques, which kept the company in operations during its first 25 years. Three kinds of steelmaking innovations were involved. One was the principle of a tilting open-hearth furnace first employed in the United States during 1889 and installed at Sydney in 1901 to facilitate removal of the large quantities of slag. Another type of tilting furnace was developed in Britain during the late 1890s and tried at Sydney in 1920. A second innovation involved the principle of duplexing, or oxidizing the silicon and phosphorus in successive furnace
TABLE 1. The share of Newfoundland ore in blast furnace charge of the Nova Scotia Steel and Coal Company

<table>
<thead>
<tr>
<th>Year</th>
<th>Share</th>
</tr>
</thead>
<tbody>
<tr>
<td>1896</td>
<td>19%</td>
</tr>
<tr>
<td>7</td>
<td>n.a.</td>
</tr>
<tr>
<td>8</td>
<td>37%</td>
</tr>
<tr>
<td>9</td>
<td>57%</td>
</tr>
<tr>
<td>1900</td>
<td>61%</td>
</tr>
<tr>
<td>1</td>
<td>64%</td>
</tr>
<tr>
<td>2</td>
<td>70%</td>
</tr>
<tr>
<td>3</td>
<td>85%</td>
</tr>
<tr>
<td>4</td>
<td>89%</td>
</tr>
<tr>
<td>5</td>
<td>84%</td>
</tr>
<tr>
<td>6</td>
<td>n.a.</td>
</tr>
<tr>
<td>7</td>
<td>99%</td>
</tr>
<tr>
<td>8</td>
<td>94%</td>
</tr>
<tr>
<td>9</td>
<td>92%</td>
</tr>
<tr>
<td>1910</td>
<td>100%</td>
</tr>
<tr>
<td>1</td>
<td>100%</td>
</tr>
<tr>
<td>2</td>
<td>n.a.</td>
</tr>
<tr>
<td>3</td>
<td>100%</td>
</tr>
</tbody>
</table>

Source: PAC R6 87/18
n.a. not available.

operations A double Bessemer duplex was first tried during the 1880s but was not successful until the following decade. The most popular type of duplex, acid Bessemer and basic open hearth, was introduced at Sydney in 1906 but soon abandoned in favour of other variations. One alternative, the Bertrand-Thiel, was a double open hearth first used in Austria in 1894 but not yet widely employed when installed at Sydney in 1913. The third innovation was a very large (500 ton) pig mixer that did not convert to steel but would homogenize iron of uneven quality (such as the Wabana ore produced) and permit a certain amount of refining as well. Silicon and sulphur could be removed in the pig mixer, which was first developed c. 1890 and extensively employed on the northeast coast of England from which region DISC obtained the necessary plans in 1908-09.

The pig mixer, varieties of duplexing and the tilting open-hearth furnace were all developed outside the Sydney-Wabana axis and had become available by 1895. Their use at Sydney was both a cause and a result of the industrial expansion at Sydney. DISC engineers made use of the techniques first developed elsewhere to remedy deficiencies in the local resource base. The transfer of new technology to a young industry was, in this case, part of an attempt to protect the reputations of politicians and financiers who had supported the establishment of DISC and to generate a return on the not inconsiderable resources committed by investors. The resources would not have been committed in the first place, however, had not the general advance of metallurgical knowledge and technique created the possibility of using low-quality resources. Certainly in 1880 and probably also in 1890 it would not have been possible to secure a favourable (and credible) appraisal of the prospects for steelmaking at Sydney. By 1900, however, scientific metallurgy had more to offer and it was possible to organize the Dominion Iron and Steel Company. Indeed, the attempt succeeded although the company remained a decidedly marginal operation during its first twenty-five years.

Less is known about the technical steps taken by the NSSC, which was the first company to smelt with Bell Island ore and the second to locate at Sydney. We do know, however, that the NSSC did not rely simply on the Bell Island ore (Table 1). A substantial part of the ore smelted by the NSSC was secured from the mainland of Nova Scotia where the company's first blast furnace was situated and from foreign sources after the furnace had been moved to Sydney. In Pictou County the dilemma of resource quality could be minimized by admixtures of local manganiferous ore and the use of local coal, which was less sulphuric than in Cape Breton (although it had a higher ash content). Moreover a substantial portion of the NSSC steel was forged rather than rolled, and could be refined slightly in the shaping process. The NSSC relied on tilting and mixing furnaces, as did the DISC, and in 1912 added a fluid compression hydraulic press to improve ingot quality. Although the NSSC quality control dilemma was undoubtedly eased by its small scale of production and reputed technical competence, it too appears to have relied on techniques that only became available in the late nineteenth century. Indeed, the distinct techniques identifiable by an historian eighty years later probably represent only a fraction of the necessary knowledge and skills. In spite of remaining mysteries, the scientific understanding of iron in relation to other elements improved substantially in the late nineteenth century as did methods of instrumentation and precision control. These changes undoubtedly assisted the Sydney steelmakers even within the framework of existing techniques.
Conclusion

Apart from the metallurgical advances a number of other factors had a role in the appearance of steelmaking at Sydney. For example, the growth of iron demand in the face of a limited ore supply in the north Atlantic region created a favourable environment for the development of peripheral sources such as Bell Island in the late nineteenth century. The Bell Island ore was offered for sale, unsuccessfully, in the import dependent British market before being offered to Canadian companies. The Canadian economy was expanding quickly, however, and with it grew a domestic consumption of ferrous metal. The Canadian market in, say, 1870 simply would not have been large enough to absorb Canadian ferrous metal production at its 1910 level. The visible hand of public policy, which became progressively more generous to the iron and steel industry between 1870 and 1900, may also have been a factor.

There are two further influences of a technological nature deserving special mention because of their potential impact on iron ore in several regions during the late nineteenth century. One involves the transportation sector and the other concerns the preparation and carbonization of bituminous coal. In the transportation sector the replacement of steampower for sail, and metal for wood, was part of a revolution in ship design that lowered ocean freight rates dramatically during the late nineteenth century. Because long-distance freight rates fell relative to transportation costs on the short haul, remotely situated mines such as at Bell Island were especially favoured as were the Sydney steelmakers who relied on ore shipments from Spain, Cuba, Sweden and even Lake Superior. The simultaneous and equally strong decline in overland freight rates by rail provided the DISC and NSSC with access, albeit tenuously, to the central Canadian market for metal.

Changes in the transportation sector have been extensively examined by historians and are somewhat better understood than developments in the technique of coal washing and retort coking. Both techniques appeared first in Europe as part of an attempt to prepare low quality coals for metallurgical purposes. The United States and Britain, with their ample supplies of high quality coal, were slower to adopt these techniques. By the final decades of the nineteenth century, however, a combination of threatened depletion of the better coals, a growing market for the by-products obtainable in retort coking and improvements in the washing and coking techniques created a favourable environment for the diffusion of these techniques to the United States and Britain. This implied the emergence in these countries of independent engineering consultants able to assist small companies to install appropriately designed equipment. One of these companies helped the NSSC in 1892 to install a washing and coking plant that was by North American standards quite modern. The new equipment permitted a reduction in the sulphur and ash content of Nova Scotia coals and an increase in coke density to support a larger blast furnace burden. These advantages, which had provided the impetus to develop the techniques in Europe, now permitted Nova Scotia coal to be used for metallurgical purposes.

Technological change in the coal and transportation sectors, we have argued, was necessary for the emergence of steel-making at Sydney and helps to explain its timing. Domestic and international metal market growth and favourable movements in Canadian government policy are also part of the story. One of the more important and certainly the most complicated precondition for steelmaking at Sydney was the advance on a number of fronts of metallurgical science and technique. This combination of influences sufficed to attract investment in steelmaking at Sydney during the first decade of this century, but the effect was not strong enough to overcome the inherent liability of low quality resources and a remote location. As a result steelmaking at Sydney remained a somewhat marginal enterprise until the 1920s, and even that meagre prosperity was quickly extinguished by the Great Depression. Perhaps the most surprising aspect of the origin of steelmaking at Sydney is the alleged discovery of iron ore on Bell Island, Newfoundland. Professional geologists and colonial officials were certainly surprised to learn of the ore in 1892, but other people knew of the ore's existence for some time. The fundamental change leading to the mining of ore on Bell Island was not an autonomous resource discovery but the combination of economic and technological changes outlined above.

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The Woodstock iron works
Carleton County, New Brunswick

R.R. Potter

The Years Prior to 1848

New Brunswick became a separate province in 1784, shortly after the influx of a large number of Loyalists from the United States. Many of these people settled along the Saint John River, including the area of excellent farmland near the present town of Woodstock.

There were indications that some of these early settlers recognized iron formations in the vicinity of Woodstock about 1820, however the nature of the deposits was not known until 1836 when Dr. C.T. Jackson, a geologist commissioned by the State of Maine, investigated the mineral potential of the area. Jackson described the “ore” as compacted hematite yielding 44 per cent pure metallic iron, and suggested that sizable tonnages may be present. It was about this time that an extensive limonite deposit was found in north-central Maine, near Mt. Katahdin. Other iron deposits were found in the Nictaux area of Nova Scotia, and a few years later at Londonderry (Fig. 1).

In 1837, Richard Ketchum of Jacksonville, on whose land iron ores had been found, petitioned the provincial government to incorporate a company. After much study, the York and Carleton Mining Company Bill was passed on March 15, 1847. Several other petitions were passed permitting the company to construct roads, obtain woodland, etc. The first blast furnace was erected in 1848 on the west bank of the Saint John River near the mouth of Lanes Creek.

The Iron Deposits

The area west of Woodstock is underlain by steeply folded and faulted calcareous and siliceous slates of Silurian age. Detailed mapping by Hamilton-Smith indicates that the ferruginous and manganiferous beds are restricted to the Smyrna Mills Formation of Upper Llandovery (Silurian) age (Fig. 2). This unit has been recognized within an area of about 150,000 km² in eastern Maine and western New Brunswick (Fig. 1). Associated manganiferous iron deposits have been described by Miller, Pavlides and Sidwell. Hamilton-Smith estimates the true stratigraphic thickness of the manganiferous and ferruginous strata in the Woodstock area as about 15 m, however apparent widths are much greater due to steeply plunging complex folds and faults.

Detailed exploration by Stratmat Ltd. has resulted in the delineation of six deposits. Tonnage to 152 m is based on limited drilling and extrapolation from gravity surveys.

The manganiferous iron strata of the Woodstock area are dense, fine-grained, cherty and argillaceous rocks which vary in colour from dark brown to grey green. Weathered surfaces are heavily stained with manganese. Hematite, manganiferous hematite, braunite, quartz, chlorite, apatite, calcite and magnetite are the most abundant minerals present, although bementite and manganiferous carbonate have been identified in similar strata in Maine. Some narrow sections contain in excess of 35% to 45% combined iron-manganese, however most of the associated strata are much lower grade. The over-all chemical composition of a 93.4 m drill intersection provides an indication of the chemical composition of the strata.

Blast Furnace Operation

The first blast furnace was erected in 1848 on a terrace on the west bank of the Saint John River about 1500 metres south-southwest of the bridge crossing Lanes Creek. About 40 men were employed quarrying the ore from the Iron Ore and Moody Hill deposits at Jacksonville. Ten to twelve pairs of horses with wagons and sleds transported the ore to the top of the second river terrace, which was at the same elevation as the top of the blast furnace.

According to Mr. Norris Best, manager of the Woodstock Charcoal Iron Company in 1865, each charge contained 612 kg (1350 pounds) of ore, 32 kg (70 pounds) of limestone and 0.73 m³ (20 bushels) of charcoal. The limestone was obtained on the Becaguimec River, 11 km to the northeast. Charcoal was produced in six rectangular burners. Four beehive burners were constructed near the blast furnace. About 43,500 m³ (12,000 cords) of wood were required annually to sustain a monthly production of 2700 m³ (75,000 bushels).

The following is a complete description of the blast furnaces which were in production in 1874.


*See article by Andrews, this volume.

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Stourbridge bricks. There were two high-pressure engines, with cylinders sixteen inches in diameter and four feet stroke, of twenty-six nominal horsepower; two blowing cylinders, six feet by five, with an air receiver, twenty-eight feet by four feet ten inches. Steam was generated in five boilers, twenty-eight feet in length by three feet three inches in diameter, and was maintained by the waste gas from the head of the furnace. The fuel used was chiefly hardwood charcoal, from maple, birch and beech, yielding at the kilns, forty-five bushels of charcoal to the cord of wood. The cost of the wood at the kilns was about $2 or $2.50 per cord. There were ten charcoal kilns, with an average capacity of seventy-five cords of wood, and a production of 2,800 to 3,200 bushels of coal. The quantity of ore used was, on an average, three tons to the ton of pig, and the cost at the furnace $1.20 per ton. One hundred and twenty-six bushels of charcoal were required per ton, at a cost of seven cents per bushel, and the cost of pig produced was $20 to $22 per ton. Much delay and expense has occurred by the frequent stoppage of the furnace for repairs, which were deemed necessary every four or five months, keeping the furnace idle about two months in the year."

Between 25 and 50 men were employed in the operation of the first blast furnace. The annual capacity was 2150 tonnes (?) of pig iron, enough to plate two frigates[19]. It is doubtful if production was sustained continuously for any more than a few years during the entire life of the operation.

**Quality of Iron Produced**

Much has also been written about the superior quality of the iron produced at the Woodstock furnaces. According to Hind[20] it had a silver grey colour, a high density and made excellent steel. He provided the following table:

<table>
<thead>
<tr>
<th>Resistance in tons/sq.in.</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Yorkshire Iron</td>
<td>24.50</td>
</tr>
<tr>
<td>Derbyshire</td>
<td>20.25</td>
</tr>
<tr>
<td>Shropshire</td>
<td>22.50</td>
</tr>
<tr>
<td>Staffordshire</td>
<td>20.00</td>
</tr>
<tr>
<td>Woodstock</td>
<td>24.80</td>
</tr>
</tbody>
</table>

The reason for the remarkable properties of the Woodstock iron was the subject of considerable discussion and debate. Bailey quotes a report of the Maine Scientific Society which stated that[21]: "The English Admiralty instituted experiments at Shoeburyness, England in order to test the resistance which iron plates would offer to the heavy ordnance of Sir William Armstrong." In that trial, every plate was shattered, except a triple plate made of Woodstock iron. "This plate was indented, but not pierced, and immediately attracted considerable attention." Several other tests were attempted with considerable success. According to the manager, Mr. Best, this was due to the smelting process employed, which permitted a higher density and a greater ability to absorb carbon. Despite the phosphorus content, which should impart a high degree of brittleness, it is difficult to reconcile the favourable results of tests and confirm the high tensile strength of the Woodstock iron.

The pig iron produced between 1843-1884 was hauled in sleds to the Houlton Road Terminus of the St. Andrews Railroad, 9 miles southwest of Upper Woodstock. It was then transported to St. Andrews, transferred to schooners for shipment to Saint John, and thence to England. When
water conditions were suitable, it was shipped down the Saint John River.

On March 2, 1880, Norris Best was declared insolvent, and on November 9 of the same year a group of men from the United States and Yorkshire formed the Woodstock Iron Company. About this time, bog iron ore from Maugerville, near Fredericton, was mixed with the Woodstock ores. Production ceased in 1884.

About 73,000 tons of iron-manganese ore was mined and smelted between 1848 and 1884. Competition from newly developed, better quality deposits in the United States, high transportation costs and the increasing scarcity of fuel forced the Woodstock operation to close.

Recent Work

When production ceased at Woodstock, there was little interest in the deposits until 1943, when Noranda Mines Limited investigated the possibility of making separate iron-manganese concentrates by flotation methods. As there is no domestic production of manganese, and because of its use in steel production, a review of the province’s manganese occurrences was made by the New Brunswick Resources Development Board.

In 1953 the area was staked by Stratmat Ltd. and both exploration and metallurgical tests were carried out concurrently. An electric-reduction process to produce metallic iron and ferromanganese was developed by Dr. Marven Ucyc of that company. Although tests indicated it was technically feasible, further development was prevented by high energy costs.

In the early 1970s, John Wark of Mandate Refining Ltd. initiated work on the co-roasting of pyrite tailings from the province’s base metal ores and iron-manganese from Woodstock. Soluble iron and manganese sulphates were produced and the base metal content of the tailings recovered by electrolysis. The process was evaluated by Ethyl Corporation. However, no further work was done.

More recent research by the New Brunswick Research and Productivity Council to test the application of their sulphation roast-leach technology to the Woodstock ores gave encouraging preliminary results.

Any future development of the Woodstock deposits or similar deposits in Maine will depend mainly on the development of a new technically and economically acceptable process to produce marketable products. It is equally important to identify areas where stratigraphic and structural conditions are such that production can be achieved by low-cost open-pit methods.

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Edward Haycock and his Gatineau Mine

D.D. Hogarth

Edward Haycock, Engineer and Public Contractor

Edward Haycock was born in Shrewsbury, England, June 14, 1813. His father was John Haycock, a timber merchant and soap manufacturer, residing at Swan Hill Court, Shrewsbury, and his mother Esther was the daughter of William Hilton of Hempstead, Hertfordshire. Edward’s paternal grandfather, John Hiram Haycock, was the first of a line of architects that “changed the face of Shrewsbury” and “provided some of the best examples of typical Victorian architecture in Shropshire.” Nothing is known of Edward’s early life or education, although it appears that he made no attempt to follow this artistic trend.

In 1833, Edward immigrated to Canada. The following year, he married Mary Butler, daughter of Johnson Butler, a Lieutenant-Colonel in the Lincoln Militia during the War of 1812. Mrs. Haycock, as co-heiress of Johnson Butler’s estate, was assigned 600 choice acres of farmland in Brooke Township near present-day Sarnia. This alone would have guaranteed a tidy income, but the Haycocks soon disposed of this land, the last lots being sold in May 1840. Instead, Haycock turned his hand to engineering and, for a number of years, was employed by the Great Western Railway first in Woodstock and later in Hamilton. He then moved to Port Hope, tried farming, worked in the engineering office of the Port Hope, Lindsay and Beaverton Railway Company, and later constructed a portion of this railway.

During his stay in Port Hope, Haycock met his future partners Ralph Jones, a contractor, and Thomas Curtis Clarke, chief engineer with the P.H.L. and B. Railway Company. The alliance was styled Jones, Haycock and Company and their most successful venture was the construction of the Government Buildings in Ottawa. This work was the crowning achievement of Haycock’s career and, no doubt, his proudest moment was at 11:30 a.m., April 2, 1860, when his daughter Mary Emily, then a little girl of 4, with a special silver trowel ceremoniously laid the first stone.

Of special interest in this project were the quarrying operations, which introduced Haycock to the art and difficulties of mining as well as to the potential returns of mines. Sandstone, for facing, was drawn from four quarries, 10 to 12 miles from the Departmental Buildings. Haycock seems to have been in charge of these operations, which in the early stages involved an average labour force of 86 men and 43 teams. However, by 1861, quarrying had declined in importance and the contractors were able to concentrate whole-heartedly on the construction site.

After a lengthy delay, due to exhaustion of Government funds, the East and West Blocks became available for occupancy in the autumn of 1865, but by the time of final payments to Jones, Haycock and Company in December 1868, the original tender of $278,810 had escalated to $1,036,423.

Jones, Haycock and Co. also tendered the construction of the proposed Governor-General’s residence on the recently purchased property at Majors Hills. Their bid was accepted and a contract was drawn up; however, the plans were dormant until April 1864, when the Government rented Rideau Hall and Majors Hill became a park. Jones, Haycock and Co. had a minor contract in the alterations of Rideau Hall in 1866. The Government’s account with the partnership was closed in December 1868.

Discovery of Iron Ore

Hull township was laid out by the surveyor Theodore David in 1801 and 1802, and the southwestern part of Templeton township by S.Z. Watson in 1805. Land near the Haycock Mine was, however, not settled until much later, the first lot being deeded in 1860. Iron ore was said to have been discovered along the Hull-Templeton line in 1865 by George Austin, at that time a land surveyor with the firm of Austin and Thistle, real-estate agents in Ottawa. Unfortunately, no details are available for this discovery, but it is known that Austin shared his information with others, including Sir William Logan, director of the Geological Survey of Canada.

Logan set out in mid-August 1866 and spent three days in his examination. However, that at time roads were mere tracks and travel, by horse and buggy, painfully slow. It took a full half-day from Ottawa center town to the Darby farm, near the discovery site. Logan proceeded to the newly constructed Wright’s Bridge, crossed the Gatineau River, travelled northward along the Cantley road, branched right along what is now Rue Ste. Elizabeth, passed Morris Foley’s farm, and then headed eastward a mile and a half to the Darby homestead. Room and board was obtained at the Darby’s and one of the sons (probably Hugh Darby) acted as guide. Logan examined occurrences near the main deposit, in detail, and two deposits near the north end of Elizabeth Darby’s lot, briefly. His description in the Geological Survey’s Report of Progress for 1863-1866 is clinically detailed, but hardly one to incite a mining stampede. In spite of this, some geologists, including Henry Venner of the Geological Survey, did visit the deposits in the succeeding 6 years, but they were certainly not enthusiastic about the prospects. In fact, the Ottawa Citizen critically referred to them as “scientific but unspeculative gentlemen.”

Curiously, the first activity seems to have appeared not here at all, but 4 miles to the northeast. In 1867, a half lot (100 acres), with indications of iron ore, was acquired by Thomas Watson of Montreal. Watson was the principal shareholder of the Canada Iron Mining and Manufacturing Company, at the time developer of the Forsyth iron mine on the west side of the Gatineau River. The original deed was made out to the Canada Iron Mining and Manufacturing Company in October 1867, and the property was sold to Gardner Church of Templeton in May 1872.

Close by was the half lot of John Grier. Grier, a lumber merchant in Cumberland, had been deeded his 100 acres....


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in August 1871, but turned over his land and mineral rights to Dr. James Alexander Grant three months later. Grant, the well-known Ottawa physician, had a keen interest in the progress of the Ottawa Valley as well as geology (he had already published several geological articles in technical journals and was elected a Fellow of the Geological Society of London in 1860). Grier and Grant both knew of iron ore near Rainville Creek, but did not attempt to develop the vein. Instead, Grant sold his property to George R. Godson of Tenbury, England and Ottawa for $20,000. This was a phenomenally high sum in those days, especially for a mere 100 acres of land, mostly unsuitable for farming and with the sole mineral potential based on a single small iron deposit. The terms were $10,000 down and $10,000 to be paid as soon as Godson “shall have bona fide sold the first ton of ore out of said lot of land.” However, it appears that he made little attempt to develop his ore. He moved to England, where he died a wealthy man in 1909 and the property passed into the hands of his trustees.

It was in the fall of 1872 that Edward Haycock and one of his sons, out hunting for partridge, first visited the area that was to be called the Haycock Mine or Haycock

Location. They were attracted to the occurrences which had been stripped free of overburden by their predecessors. The Haycocks were immediately impressed and sent a sample to Dr. Otto Wuth in Pittsburgh. The report, received early in December, was most encouraging. Samples, said to be representative, yielded 64% iron and “the ore, if uniform, would produce a Bessemer pig of unsurpassed quality.” However, this was preaching to the converted. Edward Haycock had already paid the Darby family $8000 for mining and mineral rights to 200 acres in Templeton township and 100 acres in adjoining Hull. He also acquired timber rights on an adjoining 100-acre lot and right-of-way to the south from Farrell McGovern and others. Locations are shown in Figure 2.

Almost simultaneously, Dr. James Grant acquired mineral rights to the east from Farrell McGovern, the original grantee. Dr. Grant paid McGovern $1000 for mineral and mining rights to 100 acres and right-of-way through 200 acres to the south “to make road or tramway for the purpose of carrying away ore from the mines.” Edward Haycock and Grant had made up their minds to begin operations. The stage was set for a rather feverish activity which was to last, almost without interruption, for the next two years.

The Haycock Enterprise, 1872 and 1873

Even before a rough appraisal had been made, Edward Haycock seemed determined to develop his mine and set up a steel industry in the Ottawa area. On December 30, 1872, he wrote to his friend the Prime Minister, Sir John A. Macdonald, informing him of these intentions.

By 1873, Haycock had completed his purchase of the property. As the winter was comparatively snow-free, the
The railroad was 3-foot gauge. Ore cars were horse-drawn and it was necessary to provide a solid horse track beside the rails. The rails themselves were birch and maple, 4 inches broad and 3 inches wide. They were fastened to the timbers below with tamarack pins.

Engineering and construction feats were impressive, especially when one considers that the railroad was constructed privately with rather primitive and limited equipment. The grade itself was not steep (the steepest grade was 120 feet in 500 yards at "Paddon's Hill", 2 1/4 miles north of the Gatineau terminus), but in order to avoid a sharp rise in the northern part, the line veered east, thereby easing the grade, but running into swampy land (Fig. 2). At one point, the ground was too soft for horses and the track was laid entirely by manual labour. Along the entire 6-mile run, some 4000 feet of trestle were necessary, with 4-inch planking to support the rails and horse track. The rather smooth downhill grade allowed loaded cars to descend from mine to river mainly by gravity, with most of the energy required to draw the empty cars back to the mine.

A 10-acre lot was purchased as a storage and shipping depot at the southern terminus of the railway. Wharves were constructed in the summer and fall of 1873. Thirty ore cars were purchased.

A 20-hp sawmill was acquired from a western firm, laboriously drawn through the bush and set up at the mine site. Timber for the railroad, bunkhouse, etc. was originally taken from the right-of-way, but also from the timber lot in Templeton township, where hemlock, beech, maple and elm were said to abound on the high land and tamarack, spruce, hemlock and cedar in the swamps. The reserve was, however, insufficient for future needs, because in October 1875 Edward Haycock picked up an additional square mile of timber land to the east, at public auction, for the bargain price of $92. Long after the iron ore had proved valueless, the timber resources would be considered a precious asset.

The bunkhouse, sawmill, office building and storage sheds were erected in a clearing above the Steel Ore Pit. Nearby, a year-round spring provided water. The little community became known as Hematite. Railroad spurs branched to the main pit, ore and waste dumps.

In the beginning, this was exclusively a family enterprise. Edward Haycock was the proprietor and master planner. The four sons, Samuel Hatt, John Hilton, Edward Burton and Richard Henry, each lent assistance in one way or another. Samuel was an experienced engineer and was put in charge of the railroad. John Haycock, who acquired an early interest in minerals, was active in prospecting. Edward B. and Richard, who had studied law, were particularly valuable in legal matters. The sole "outsider" was Donald B. MacDonald, a former colleague of Edward, Sr., on the Intercolonial Railway and a recent employee at the Forsyth Iron Mine who, in the summer of 1873, was made mine foreman.

In mid-November 1872, apatite was discovered on the lot of John Thibert, Jr., 1 1/2 miles northwest of the main occurrence of iron ore. It was said that Edward Haycock himself was the discoverer. What is definitely known is that mining rights were acquired by John Haycock, who worked the property for phosphate from December 1872. The new phosphate industry captivated the interests of John and Edward B. Haycock and soon put an end to their active participation in the Haycock Iron Mine.
Edward Chapman’s Reports

Edward Haycock next took the logical step of having the property appraised by a mining man of high repute. For this purpose, he secured the services of Professor Edward John Chapman, who had held the Chair of Mineralogy and Geology at University College, Toronto, for 20 years(2). Chapman was then 52 and near the peak of a distinguished career. He had published numerous articles in scientific journals and written a book on the Geology of Canada, already in its second edition. He also was a well-known mining consultant. Among other attributes, he had examined the iron deposits near Madoc and Marmora, which were similar in many respects to those of the Haycock Mine.

Chapman arrived at the Haycock Location, as he called it, early in April 1873, by his own admission a poor time for examination because “snow covered the greater portion of the ground”. The report notes that “4 or 5 distinct beds of good width” occur in the northern hill on the property and another on the middle hill. Chapman calculated a reserve of 21,000 tons of high-grade ore in the main vein to a depth of 100 feet.

During the summer, Chapman again visited the property (but this time under better circumstances) and issued a supplementary report dated November 22, 1873(3). All exposed veins were said to “widen rapidly” below the surface and closely spaced veins were thought to “run together at depth and form a workable deposit”. Although these features may have been observed in exceptional cases, it is inconceivable that they should be the general rule and be applicable to the calculation of ore reserves. These generalizations could be attributed to an inexperienced speculator, but would hardly be expected from the country’s foremost economic geologist and mining consultant.

In the second report, the number of “distinct beds” on the north ridge had increased to 13 and the minimum ore reserves, in the central part of the property alone, were put at $6,300,000 tons containing 64% iron, to a depth of 200 feet. These figures were based on an aggregate thickness of 70 feet and an “average length” of 35,000 feet, neither of which was supported by factual data. The “established” reserve would furnish “a daily output of 100 tons of ore or 60 tons of metal during a period of a century and half”. Chapman regarded the estimate as conservative.

As a yardstick of comparison, today’s Marmaron Mine, near Marmora, Ontario, regarded as a medium-sized iron deposit, initially contained 20,000,000 tons of 37% iron, or about twice the iron attributed to the Haycock deposit.

The Haycock ores were indeed high in iron (average 60% to 68%), but also high in titania (most analyses showed 2% to 4% titanium dioxide). This content of titania was definitely one to contend with. In the 1870s, many steelworkers would have found the Haycock ore unacceptable. Yet Chapman blithely claimed that titania “does not, in any way, affect the practical value of the mineral”.

In view of the large reserve of seemingly high-quality ore, Chapman recommended that blast furnaces be put up at the Gatineau storage ground.

On reading the report, Haycock, already convinced of the value of his ore, was elated. Surely this report would provide the necessary data to convince investors. Some geologists, however, probably viewed this report with scepticism. Henry Vennor and Bernard Harrington of the Geological Survey of Canada were factual, but less than enthusiastic, in their descriptions. Dr. Alfred Selwyn, then director of the Geological Survey, never one to mince words, went so far as to say that Chapman’s report was entirely misleading(4). Needless to say, no officers of the Geological Survey are mentioned as guests attending the official opening of the mine under company ownership, October 26, 1875. Probably they were not invited and if they were they didn’t accept.

Notwithstanding these criticisms, Haycock knew otherwise and felt it his duty to convince the public. On the suggestion of Samuel and Richard Haycock, Dr. James Grant, along with the Hon. James Skead and Joseph Currier, signed affidavits supporting Chapman’s report. Sir John A. Macdonald himself telegraphed his confidence in the report. Their names were prominently affixed to the Prospectus of the Ottawa Iron and Steel Manufacturing Company, which became manager of the Haycock Mine and is the subject of the next section.

The Ottawa Iron and Steel Manufacturing Company

Just when Haycock began to seriously consider forming a joint stock company is uncertain, but rather firm plans were announced by The Citizen in August, 1873. The paper refers to an early Prospectus, which the writer has not seen. Details are, however, given in the Prospect of 1874 and 1875, copies of which are available in the National Library and Public Archives of Canada, respectively. The charter of the Ottawa Iron and Steel Manufacturing Company (Ltd.) was passed as a private bill in Quebec, which was given final assent on January 28, 1874 and amended on February 23, 1875.

Up to 1875, Edward Haycock had spent over $100,000 of private funds, the savings of a lifetime and a snug fortune in those days. Naturally, he looked for some return on his money and, for this, due provision was made in the company’s charter. The price to be paid the vendor for purchase of the property was $250,000, half in cash and the balance in stock. However, under the terms of the charter, the company was unable to commence work until $200,000 in stock had been subscribed and, in order to comply with this clause, Haycock felt obliged to forfeit $36,000 cash in favour of stock, so that operations could begin by January 1st, 1876. The sale of common stock had been slack, there was little money on hand and therefore cash payments to Haycock were deferred to March 1878.

An attraction for prospective shareholders was Haycock’s guarantee of a minimum dividend of 10% per annum on paid-up capital for 3 years from the allotment of shares (January 1875). As security, Haycock declared a personal liability on the guarantee. The three dividends were, in truth, paid up, the last on December 30, 1877.

Officers of the company from 1874 to 1882 included such well-known figures as James Skead, R.S. Cassels and W.R. Thistle.

The Hon. James Skead (1816-1884) was a lumber baron, an active industrialist and a champion of agriculture in the Ottawa district. He sat for the Rideau Riding in the Legislative Assembly from 1862 until confederation, when he was appointed Senator.

Richard Scougall Cassels (1822-1896) was the manager of the Quebec and later the Ottawa Branch of the Bank of Upper Canada. From 1869 to 1875, he was
The furnaces were officially opened on September 12, 1875. For a while, the furnaces produced 4- or 5-tons of metal a day and it appeared that the company would move ahead.

A big problem was what to do with the metal. Cunningham favoured boiler plate, and tests and feasibility studies were made. Other products tested included merchant bar iron, munitions, razor blades and files. Later, Haycock unsuccessfully applied to Sir John A. Macdonald to supply rolling stock to the C.P.R.

However, this was all wishful thinking. Haycock had over-extended himself and, for the last year, the mine had been operating on a pitifully thin margin. Creditors made their demands. Suddenly, yet quite unavoidably, Haycock found himself out of funds completely. The venture collapsed before the new company could be given a fighting chance. On the approach of winter 1875, the mine was abandoned, never to be reopened.

After 1875, the buildings and railway rotted in disrepair. A map of 1877 shows the lower part of the Haycock railway as a road. The camp burned during the winter of 1878-79.

At various intervals, attempts were made to re-activate the mine. W.H. Fuller was stationed in England in 1876 and 1877 to organize a new company and negotiate a loan. Captain Pill, a mining engineer, and L.G. Bennet, a mineral importer, representing English financiers, sailed to Canada and examined the property. Their report was optimistic, but failed to attract investors back home.

Another flurry took place in 1880-1884. Haycock himself visited London in the spring of 1880. Sir John A. Macdonald, in England in 1881, was interviewed by Frederick Lake, who attempted to put the company on the London market. Three mining experts were sent to the Haycock Mine in 1883 and 1884. "They examined the property minutely and the main occurrence was probed by an early prototype of the diamond drill. John Birkinbine of Philadelphia and Richard Howson of Middlesboro (England) heartily endorsed Chapman's report."

Howson went so far as to recommend doubling the area of the proposed furnace and storage site on the Gatineau and replacing the narrow railway with standard gauge. The third expert was not specifically named, but apparently lacked the exuberance of his predecessors. The over-all conclusion was that although the quality of the ore was good, the quantity was insufficient to justify a commercial venture.

In the meantime, creditors had closed in. First the Baskerville Brothers, provision merchants in Ottawa, demanded seizure of the land, buildings and equipment by the Sheriff of Ottawa County. This took place in 1881 and again in 1882. The mine was actually advertised for sale, but, due to legal opposition, this action did not proceed further. The Ontario Bank foreclosed their mortgage of 1875. Finally, the mine was seized a third time, advertised for sale and, in March 1884, acquired by Alexander Simpson on behalf of the Ontario Bank. From here the mine was purchased by John Payne, a chartered accountant in Toronto, who acted as a bankruptcy agent. It was not until October 1907, when the last of the outstanding debts were liquidated by Richard Haycock, that the mine finally passed out of the hands of the Haycock family, out from obligation. Strangely, a search of legal documents failed to show that the charter was ever surrendered or that Edward Haycock officially declared himself insolvent.
Epilogue

It is true that the Ottawa Iron and Steel Manufacturing Company was founded at a very poor time. The mid 1870s were depression years. The railway boom of the late 60s and early 70s had collapsed and iron and steel were, by 1875, in surplus. Had the company been organized a decade earlier, it is likely that investment would have been much livelier. One might speculate that operations would have been larger, somewhat more ore would have been located and the mineral resources better defined, but in the end would be the inevitable realization that large quantities of iron ore did not exist and a larger number of investors would have been impoverished.

As it was, the principals in the operation were all affected, to some extent, by their association with the company. The Haycock sons, Edward Burton and John Hilton, in the long run probably benefited from the experience. It was the beginning of careers in mining that first took the brothers to the Ottawa Valley phosphate deposits. Their most profitable enterprise was the North Star Mine near Glen Almond, which produced phosphate for the Haycocks from 1879 and was sold to the Dominion Phosphate Company in 1882. Curiously, the sole Ottawa director of the latter company was the well-known engineer Thomas C. Keefer, who also owned an iron property adjacent to the Haycock Mine.

Later, John and Edward Haycock, Jr. worked mica veins in the same district. Their most successful venture was the Nellie and Blanche Mine near Cantley, which produced phosphate and mica from 1879 and, along with other lots, was sold in 1892 to the Lac Girard Mica Mining System for $38,000. In 1888-1890, Edward worked silver veins in the Temagami-Cobalt district, precursor to the Cobalt rush of 1904. He went on to develop alluvial and bedrock gold deposits on the Rivière du Loup, Quebec, and became an early gold prospector in the Rainy River district of northwestern Ontario.

Samuel Haycock appears to have been closest to Edward, Sr. Until he moved from Ottawa he was joint owner of the Haycock residence at 9 Rear Street in Ottawa, and later Edward Haycock seems to have been a frequent visitor at Samuel’s house at Cardinal, Ontario. Samuel was instrumental in organizing the Cantley Road Company in 1878 and surveyed the road (the present Chemin St. Antoine), which was to pass within half a mile of the Haycock lots. Probably he had a mind to aid his father’s ailing iron mine. On the other hand, it was through Edward Haycock’s acquaintance with Sir John A. Macdonald that Samuel was able to embark on an engineering career in the construction of the St. Lawrence Canals, which occupied the last 30 years of his life. No doubt his engineering experience in the Intercolonial Railway, Departmental Buildings and the Haycock Mine were invaluable in this work.

While Edward Chapman, William Thistle and Richard Cassels achieved merits that outshine the tarnish of the Haycock Mine, the same cannot be said for the Hon. James Skead. James Skead’s empire toppled in the middle seventies during the depression that swept Canada. The failure of the Ottawa Iron and Steel Manufacturing Company was just another factor that contributed to his own demise. He died insolvent.

Edward Haycock himself was the person most affected by the failure of the Ottawa Iron and Steel Manufacturing Company and he took personal responsibility for many of the debts. In spite of the advice of his friends, he stubbornly refused to admit the gross mistake in calculation of ore reserves and optimistically looked forward to a resumption of mining activities. This, of course, was to be his personal ruin. “In his old age he emerged from the struggle bereft of everything save the honour and credit of his good name.” He died insolvent on May 19, 1894. In his will, the only personal item specified was a gold pocket watch, at once a symbol of his greatest triumph and deepest failure. It had been given in 1862 by his partners in the construction of the Departmental Buildings, Ralph Jones and Thomas Clarke. Attached to its chain was a hand-forged iron ring, the first metal produced from the Haycock Mine.

Acknowledgments

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REFERENCES

6. Route located from H.G. Venmore, Map (1 in. to 4 mi.) showing...position of the most important mines in Ottawa County, Geol. Surv. Canada, Report of Progress for 1876-1877; legal papers; personal observations and other sources.
The Grantham iron works

David J. McDougall

Introduction

The production of pig iron, wrought iron bars and castings in Quebec during the 18th and 19th centuries has been so closely identified by historians with Les Forges du St. Maurice on the St. Maurice River a few miles north of Trois Rivières (Three Rivers), that one is sometimes left with the impression that it was the only iron works that ever operated in the province. There were however, about twelve others, one of which was so completely re-built that it should probably be considered as a separate entity. All but two were constructed after the midpoint of the 1800s, coinciding with the growing Canadian demand for "charcoal pig" for railway car wheels and castings for industry, and about half of them were technically, if not economically, successful. The misconception about Les Forges du St. Maurice may have come about either because of its long history of somewhat less than continuous production over a century and a half, or a tendency of Quebec historians to concentrate on events in the French-Canadian milieu, but whatever the reason, the other Quebec iron works have been generally ignored. All but one of the Quebec blast furnaces could be described as "pre-industrial" with capacities ranging between three and ten tons per day, the exception being the Radnor furnace built in 1892 with a maximum production of about 35 tons per day and an alleged capacity of 50 tons per day.

Until a railway network was completed in the 1880s, the location of all the Quebec iron works except the Grantham Iron works was dictated by close proximity to adequate ore reserves because horse transport was of little use beyond a ten-mile radius. The other essentials of forest resources (mainly soft woods) for charcoal, limestone for flux and water power, were generally available except within a few miles on either side of the St. Lawrence River. Among the dozen iron works established in Quebec between 1732 and 1880, seven used bog iron ore in charcoal-fired furnaces. Five of these, including Les Forges du St. Maurice, were located on or near the St. Maurice River which enters the St. Lawrence River from the north, and two, the St. Francis Forges and the Grantham Iron works were on or near the St. Francis River which enters the St. Lawrence from the south. In a general way deposits of bog iron ore in the St. Lawrence lowlands of Quebec were contained within the limits of the post-glacial Champlain Sea, with many clustered along the flanks of the St. Narcisse recessional moraine which is crossed by the St. Maurice and other near-by streams, and the Drummondville recessional moraine which is crossed by the St. Francis River.

Discovery of Bog Iron in Grantham

Bog iron was discovered near Drummondville in Grantham township sometime between 1815 and 1830 and was reported to occur there "in great plenty" by Lieut. F.E. Baddely of the Royal Engineers who had been engaged in reconnaissance surveys for the Surveyor-General of Lower Canada (Baddely 1831). Thirty years later the "Geology of Canada" (Logan 1863) listed some fifty bog iron occurrences, but the Grantham deposits had been either forgotten or ignored because none were on or near the St. Francis River. About four years later, Sir William Logan, the Director of the Geological Survey examined specimens of bog ore which had been discovered in 1867 on a tributary of the St. Francis called the Rivière aux Vaches in the seigneury of DeGuire (Gazette de Sorel 1868). The development of the Rivière aux Vaches deposits was described in a Geological Survey report in 1874, (Harrington 1874) but there was no mention of the iron ore in Grantham township by the Geological Survey until after 1880.

St. Francis Forges

In April 1869, production began from the St. Francis Forges at St. Pie de Guire on the Rivière aux Vaches about fifteen miles northwest of Drummondville by "La Compagnie des Mines de la Rivière St. François", headed by a Montreal merchant named Narcisse Birs dit Desmarteau. This iron works was purchased in 1874 by John McDougall of Montreal, the owner of two Montreal manufacturing companies: the Caledonia Foundry and Machine Works which specialized in manufacturing mill machinery and steam engines, and the John McDougall and Company which had been formed in 1864 to manufacture railway car wheels from charcoal pig iron produced at Les Forges du St. Maurice. Robert McDougall, a co-owner with his seven brothers of Les Forges du St. Maurice and the Forges l’Islet, was engaged by the new owner to manage the St. Francis Forges and shortly thereafter was joined by his brother, David McDougall. Both were the sons of John McDougall of Three Rivers (Trois Rivières), who had purchased Les Forges du St. Maurice and the Forges l’Islet in 1863. There is no known kinship between the John McDougall of Montreal and the John McDougall of Three Rivers but the similarity of names has caused considerable confusion among historians of Quebec’s iron industry.

Grantham Iron Works

Beginning

In the fall of 1879, Robert McDougall and John McDougall, the proprietor of John McDougall and Company, made an inspection of a piece of property in Drummondville on the west side of the rapids upstream from Lord’s Falls to determine its suitability as the site for a new iron works. At that time, the town of Drummondville was a small community which had a number of advantages as the site for an iron works. A description in the Gazetteer of British North America (Lovell 1881) which appears to have been prepared in 1879 because it does not mention the iron works, stated that it was:

"The chief town of the Co. Drummond, Quebec on the St. Francis river and on the South Eastern railway, 68 miles from Montreal. The district court of the county is held here. It has two churches (Church of England

and Roman Catholic), a telegraph office, 2 hotels, 11 stores, 2 spool factories, 2 saw mills, 1 grist mill and a carding mill. Pop. 750."

There was abundant water power and untouched nearby resources of bog iron ore and wood for charcoal, but probably the most important consideration was that the South Eastern Railway provided access to both Montreal and Quebec City via either Grand Trunk Railway from where the two lines intersected at Acton, or via the Richelieu and Ontario Navigation Company steamers from Sorel. The Grand Trunk Railway was the only satisfactory way to ship heavy goods by rail because it had exclusive use of the only bridge to Montreal Island from the southern side of the St. Lawrence River. The South Eastern’s main line went to Longueuil on the south shore of the St. Lawrence opposite Montreal where a ferry had to be taken to reach Hochelaga (then a separate community adjacent to Montreal). However, during the very cold winters of 1879-80, and 1880-81, the railway train went directly onto the island of Montreal via an ice bridge over the St. Lawrence. In December of 1879 Robert McDougall wrote his principals in Montreal that he had made a proposal to the owners of the lot to rent part of it and enclosed a copy of the reply from W.J. Watts, the son of the owner, and outlined the conditions for a renewable ten-year lease, along with a sketch map of the property and his comments on the legal and financial aspects of the lease (R. McDougall Letter Book).

The winter of 1879-80 was extremely cold but despite the weather the routine operations of the St. Francis Forges and preparations for the establishment of the new iron works at Drummondville proceeded more or less normally. In January 1880, the St. Francis furnace was operating with few problems and as late as March 22, pig iron was being shipped directly via the South Eastern Railway "for as long as the ice bridge lasted." Wood was being cut at several locations and carted to the St. Francis kilns, and bog ore mined and washed during the previous summer was being brought in by carters from Grantham, Wickham and Simpson townships near Drummondville and from St. Guillaume in Upton township near St. Pie de Guire. Bog ore shipments were also coming by rail to Drummondville from Acton, twenty miles away and from Gentilly, over a rail distance of about 120 miles. The possibility of bringing bog ore 60 miles from Farnham was being considered for the following year (R. McDougall Letter Book). (These shipments appear to have been the first long-distance railway transport of iron ore in Canada).

The establishment of pig iron manufacturing in Drummondville required an entrée into local politics and what seems to have been an amicable inter-family relationship between the McDougalls of Three Rivers and the wealthy and politically active Watts family of Drummondville apparently helped in obtaining a ten-year exemption from municipal taxes. Construction of the furnace foundations began on March 22, 1880, but the events of the next fortnight were chaotic. Within the week the building which had been acquired as an office in Drummondville burned down and immediately thereafter the mill which was to supply the lumber for construction also burned. The other nearby mills had contracts for all the lumber they could produce and arrangements had to be made to purchase lumber from mills which were remote from the site. A few days later, the ice in the St. Francis broke up and went out abruptly and the resulting ice jams and flooding, in addition to damaging flour and saw mills along the river, caused some anxiety for pig iron from the St. Francis Forges which had been moved in January to the river schooner landing on the St. Francis.

![Figure 1: Robert McDougall, manager of Les Forges du St. Maurice (1863-1874), Forges St. Francis (1874-1880) and Grantham Iron Works (1880-1885). Notman Photographic Archives, McCord Museum, McGill University.](image)

**Design and Construction of Blast Furnace**

The design and construction of the new blast furnace at Drummondville appears to have been based on Robert McDougall’s experience in managing several smaller blast furnaces over the previous decade and a half plus a knowledge of the charcoal iron technology practiced at that time in New York and New England. The building housing the blast furnace and the casting floor had ground-plan dimensions of 88 feet by 40 feet and was shed-like over the casting floor and rose to three and a half stories around the blast furnace. The lower part of the furnace was built of locally quarried stone with a brick stack at a total cost of $1,725.00 and was larger than any previous blast furnace built in Quebec for bog iron ore. The bricks used in construction were tapered and beveled, fourteen to sixteen inches in their greatest dimensions, and had to be ordered from Montreal because suitable local brick-yard facilities were not available. An enclosed loading ramp on wooden pilings led from the top of the river bluff behind the furnace to a four-foot diameter loading door covered with a hinged cast iron cover which was opened for charging with a counter-balanced chain block. At the boshes the velocity and direction of the air blast could be adjusted by the use of inter-changeable nozzles on the tuyères. The compressors provided air pressures up to “two air inch”
(about one pound per square inch) and later experience showed that satisfactory smelting could be done with as little as “one half air inch” (about one quarter pound per square inch). A short “wing dam” directed water from the rapids into a canal where a turbine-type “water wheel”, capable of supplying 150 horsepower, provided the power for the air compressors. A few years later, a longer wing dam was built about half way across the river from the eastern bank to direct the flow toward the town’s industries on the opposite side and just prior to 1900 the remaining gap was closed by a wooden dam built by the town of Drummondville to improve the performance of its small hydro-electric power plant located in the Grantham Iron Works power house. (R. McDougall Letter Book, Obalski 1889, L. Millar, pers. comm.).

The building housing the blast furnace, the “prooving” shed where the pig iron was cast and the slag drawn off, a machine shop, the power house and air compressor building were on the low ground near the river. A road (rue des Forges) went up the bluff from a road along the river and across the property to the Melbourne road (rue Heriot). Above the bluff, the office, weigh scales, workmen's houses and the company store and bakery were on the north side of rue des Forges facing the company's stables on the other. Stock piles of iron ore, charcoal and limestone were located near the ramp to the furnace and the wood yard and the battery of bee-hive charcoal kilns were in the southwestern corner of the property near the Melbourne road. (R. McDougall Letter Book, L. Millar, pers. comm.).

Construction of the blast furnace continued through the summer of 1880 and it was “blown-in” during the fall. By December it had been worked up to full capacity and just before Christmas the manager reported:

“The furnace is doing remarkably well — our cast at noon yesterday (Tuesday) was 5/4 tons* and at midnight the same quantity forming 7 1/2 tons for the 24 hours. This was obtained from fifty charges of ore of 650 lbs each or 14 1/2 tons in all showing a yield from the ore of over 50 per cent. These results are particularly satisfactory to me for it shows that under favourable circumstances an ore can be made to give out all its iron when properly treated in a well constructed furnace. This has been my aim and study for years but the other furnaces with which I have practiced have denied me this result and I am greatly pleased at having at last arrived at what may be called true results in blast furnace practice — or in other words — of being able to extract from the ore as much iron in the furnace as in the laboratory by chemical action.”

The furnace produced only 80 tons, 1500 pounds of pig iron in December but by January production had settled down to a steady rate of about five tons per day and the new cast-in mark of “Grantham” was in use, which had been chosen for the very practical reason that it had fewer letters than “Drummondville”. Iron was being shipped to Montreal on the South Eastern Railway at about the same rate it was being produced, with both production and shipments averaging about 142 tons per month during the first four months of 1881. In the early winter, sleighs brought in 3510 long tons of iron ore which had been mined during the summer of 1880 in Grantham, Wickham and Simpson townships and an additional 525 long tons of
ore were brought into Drummondville by rail. In December 1880 and January and February 1881, another 3020 long tons of ore were brought in by carter at a cost at the blast furnace of $1 to $2 per ton. During the period of December to March, something in excess of 7250 cords of wood (some data is missing in the Letter Book) were cut at a cost delivered to the kilns of about $1 per cord. However, erratic behaviour of the furnace at the St. Francis Forges had become a matter of concern to the manager and on January 20, 1881, the first mention appeared in correspondence of the possibility of replacing the furnace with a second one at Drummondville. After some initial uncertainty as to whether there was sufficient water power and ore available to support two furnaces at the same location, at the end of April 1881, the foundations for the new furnace were being cleared and a moulding machine to make bricks was in use at the site. Robert McDougall's reports show that the St. Francis Forges were still in production in mid-March 1881, after which it was dismantled and its rectangular kilns re-erected at Drummondville during the summer.

The second furnace was located about 90 feet south of the one built the previous year in a building of the same size but was constructed entirely of brick (Harrington 1883) with a lower stack and somewhat larger dimensions at the throat and boshes. The blast for both furnaces was modified for both hot and cold blast by the installation of hot air stoves using waste gases from the furnaces and a steam power plant was built between the furnaces for emergency power for the air compressors.

<table>
<thead>
<tr>
<th>Grantham No. 1 (Built 1880-81)</th>
<th>Grantham No. 2 (Built 1881)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Height</td>
<td>35 feet</td>
</tr>
<tr>
<td>Width of throat</td>
<td>5 feet</td>
</tr>
<tr>
<td>Width of boshes</td>
<td>9 feet</td>
</tr>
<tr>
<td>Width of hearth</td>
<td>3 feet 4 inches</td>
</tr>
<tr>
<td>No. of tuyeres</td>
<td>5</td>
</tr>
<tr>
<td>Temp. of blast</td>
<td>Cold or hot (400-450 degrees)</td>
</tr>
<tr>
<td>Air Pressure</td>
<td>Unknown</td>
</tr>
<tr>
<td>No. blowing cylinders</td>
<td></td>
</tr>
<tr>
<td>Power source</td>
<td>Water turbine and steam</td>
</tr>
<tr>
<td>Daily productive capacity</td>
<td>7 tons cold blast, 10 tons hot blast</td>
</tr>
</tbody>
</table>

The maximum productive capacity of the two Grantham furnaces can be estimated at about 6000 tons of pig iron per year but from 1880 until mid-1889 the total production was reported as 30,000 tons averaging 3000 to 4000 tons per year from about 10,000 tons of ore per year (Obalski 1889). This was the iron works period of greatest activity and it was apparently at this time that it was reported to be using 20,000 cords of wood, employing 250 to 300 men and disbursing about $80,000 per year for wages, wood and ore (St. Amant 1932).

Despite changes in the Quebec and Canadian blast furnace capacity, technical advances in the metallurgy of iron and steel and the introduction of protective tariffs and government subsidies, the Grantham production remained fairly constant until the early 1890s. Between 1876 and 1883, competition in Quebec had nearly disappeared with the closure of the Moisié Iron Works, Les Forges l’Ilet and Les Forges du St. Maurice and, although Robert McDougall's brother Alexander McDougall (the manager of Les Forges du St. Maurice from 1876 until mid-1880) and a partner Louis Dusseauilt had reopened the Grondin iron works under the name Forges Shawini- gan in mid-1880, it closed in February 1881. After 1883, the Radnor Forges was being operated by Robert McDou- gall's cousin, George McDougall, but its production from then until 1890 averaged little more than 750 tons per year, about half its rate of production during the previous decade. In 1880, the Conservative government of Sir John A. MacDonald imposed an import duty on foreign iron of $2 per ton which was raised to $4 per ton in 1887. Incentives for iron making were further increased on July 1st, 1883 by a government subsidy of $1.50 per ton of pig iron made from Canadian ore and over the next three decades a number of large, mainly coke-fired blast furnaces were built in Nova Scotia and Ontario with a corresponding dramatic increase in Canada's production of iron and steel. However, during the same period, Quebec's share of Canada's primary iron production dropped from about ninety per cent in 1875 to about twenty-two per cent in 1887, about one and a half per cent in 1907 and nil in 1912, reflecting the fact that by the late 1880s, iron could be produced in coke-fired furnaces which had properties equivalent to the more expensive "charcoal pig".

Following the construction of the second Grantham blast furnace, several events took place which had both positive and negative effects on the operations of the iron works. Two days after Christmas 1885, Robert McDougall died at the age of 52 after having been in ill-health for about two years (L. Millar, pers. comm., McDougall family records). He was succeeded as manager of the Grantham Iron Works by his brother, George McDougall, who had been superintendent of Les Forges du St. Maurice from about mid-August 1880 until about March 1883.

Sometime between 1882 and 1885, a half-mile spur line of wooden rails capped with strap iron was built along the rue des Forges from the South Eastern Railway to the Grantham Iron Works. In 1887, the Canadian Pacific Railway absorbed the bankrupt South Eastern and the same year completed the Lachine railway bridge which for the first time gave the railway and its subsidiaries access to the island of Montreal from the south shore of the St. Lawrence. In 1888, using these facilities, the Grantham Iron Works began shipping bog iron ore from St. Lazare in Vaudreuil Seigneurie about twenty miles west of Montreal and a rail distance of about 130 miles from Drummondville. Shipments from these ore deposits continued the following year and bog ore was also obtained from Grantham, Wickham and Bulstrode townships in Drummond county. At about the same time, an unsuccessful attempt was made to use a mixture of bog ore and magnetite as furnace feed (Ells 1888), the magnetite having probably come from a small deposit near Sherbrooke, Quebec. From 1890 until the early 1900s bog iron from St. Wenceslas in Nicolet Seigneurie was shipped into Drummondville over the newly completed Drummond County railway (after 1899 part of the Intercolonial and still later the Canadian National).

On July 1, 1889, the government subsidy was reduced from $1.05 to $1.00 per ton, the iron works' exemption from municipal taxes was coming to an end and during that year only one furnace was in continuous operation. At that time the future of the iron works must have been in considerable doubt because David McDougall left the John McDougall and Company organization and moved to Montreal to work for the "Gazette" newspaper after having been company store manager (and sometimes in charge of the furnaces) at Les Forges du St. Maurice, the St. Francis Forges and the Grantham Iron Works for more than twenty-five years (McDougall family records).

By 1892 the Radnor Forges had been enlarged and
revitalized as the Canada Iron Furnace Company under the ownership of George E. Drummond, his brother Thomas J. Drummond and their associates, to the point that it dominated the production of “charcoal iron” in Quebec for about the next twenty years. The same year John McDougall of John McDougall and Company died at the age 66 and his estate was managed from then until 1899 by his partner and brother-in-law, Robert Cowans whose primary interests by that time were the Cumberland Railway and Coal Company and the Sprinlight coal mine in Nova Scotia. Pig iron production at the Grantham Iron Works averaged close to 3000 tons per year in 1890 and 1891 but from 1892 onward, it seldom exceeded 2500 tons and frequently fell far below that figure. There seemed to have been no attempt to take advantage of the Federal government’s subsidy on pig iron which was raised from $1 to $2 per ton on July 1, 1892, and further increased on April 29, 1897 to $3 per ton for pig iron made from Canadian ore and $2 per ton for pig iron made from foreign ore. By then it is probable that John McDougall and Company had lost much of its market for railway car wheels and pig iron to the iron manufacturing and processing companies controlled by the Drummond brothers.

From 1898 to 1906, part of the ore used by the Grantham Iron Works came from large bog iron deposits at Nicolet near the St. Lawrence river, which was shipped to Drummondville over the Intercolonial railway system. In 1902, 60 tons of magnetite from a small deposit near Richmond, Quebec and in 1903, 200 tons of “hard ore”, (apparently magnetite) from near Eastman, Quebec which had been found by prospecting in the autumn of 1902, were shipped to Drummondville. Much of the prospecting activity can probably be ascribed to George McDougall who seems to have been responsible for attempts to use other kinds of ore than bog ore at both Les Forges du St. Maurice and the Grantham Iron Works. However, the Eastman ore caused problems with the furnace which brought a burst of profanity from the otherwise taciturn manager of the iron works which was still recalled with awe by some of his relatives 70 years later (L. Millar, pers. comm.). There are no records of later shipments from these deposits and ore from the United States (possibly magnetite) was only used in sufficient amounts to collect “foreign ore” subsidy payments in 1910 when about 93 per cent of the pig iron was made from Quebec ores. During the following and final part of the year of production, when “foreign ore” payments had been suspended, about 42 per cent of the ore came from outside the province.

By about 1900, wood for charcoal began to be scarce in the vicinity of Drummondville and coke was tried with unsatisfactory results. To overcome the fuel problem, new kilns and a number of buildings were erected at a place called “The Kilns” about seven miles up the St. Francis River from Drummondville. These kilns operated during the winter months and the charcoal was hauled to the iron works in large charcoal sleighs over an “iced road” (L. Millar, pers. comm.).

On July 1, 1903, the government subsidies were reduced to $2.70 for pig iron made from Canadian ore and $1.80 for pig iron made from foreign ore with the intention that they cease in 1906-07 at $1.05 and $0.70 respectively. Although they were reinstated at slightly higher rates after 1907, they were again progressively reduced to $0.90 and $0.40 and ceased on the last day of 1910. However, despite the falling subsidies, when John McDougall’s son, Edgar McDougall, took over the management of the Montreal car wheel foundry and its subsidiary, the Grantham Iron Works, from 1900 to 1907, at least one Grantham blast furnace was operated on a twelve-month-a-year basis in contrast to the much lower production between 1892 and 1899 when his uncle, Robert Cowans was in charge of the “State John McDougall and Company”.

Final Shutdown

The end of an era which had begun in 1863 with the purchase of Les Forges du St. Maurice and the Forges l’Islet by John McDougall of Trois Rivières came on November 6, 1906 with the death of his son George McDougall who, in one fashion or another had been associated with the management of Les Forges du St. Maurice, the Forges l’Islet, the Radnor Forges and the Grantham Iron Works for at least 35 years. However, by then the Grantham furnaces were reported to be the smallest charcoal blast furnaces in active operation anywhere in the world with a daily production of only 3.5 tons of pig iron from each furnace (Sweeiser 1909). During 1907, steps were initiated to bring several iron manufacturing and processing companies under one management and in June 1908, the Canada Iron Corporation was formed to consolidate the Canada Iron and Foundry Company in Ontario; the Canada Iron Furnace Company and John McDougall and Company in Quebec; and the Annapolis Iron Company and the Londonderry Iron and Mining Company in Nova Scotia. The executives of the new corporation were: Thomas J. Drummond, president, George E. Drummond, vice-president Edgar McDougall, H.E. Dyment, H. Cockshutt, and George Gudewill, directors; plus a committee of directors in London, England. However, despite an impressive looking list of foundries, blast furnaces and iron mines extending from the head of the Great Lakes to the Maritimes, the corporation had a number of weaknesses and beginning in 1908, neither the Grantham Iron Works nor the Radnor Forges operated for more than six or seven months a year. The corporation’s final financial illness was the end of the Federal government’s subsidies and a depression in the North American iron and steel industry. The Radnor furnace was shut down for the last time after 41 days in blast in 1910 and the last operating Grantham blast furnace was blown out after operating for 106 days in 1911. No more primary pig iron was produced in the Province of Quebec until production began from the electric furnaces of Quebec Iron and Titanium at Sorel half a century later (by chance, the initial production came

*Nearly a decade earlier, a very similar merger was planned with the incorporation of the International Car Wheel Company under the laws of New Jersey in 1899. This company was to include car wheel foundries in Buffalo, New York City, Philadelphia, Pittsburgh and two in Boston in the U.S.A., one in St. Thomas, Ontario and two in Montreal, including the John McDougall and Company. The Weston Furnace Co., Manistique, Michigan, manufacturers of charcoal pig iron was also part of the merger and it can be assumed that the John McDougall and Company’s New Jerseyvestment, the Grantham Iron Works may also have been included. Nine of the directors of the International Car Wheel Company were Americans, including R.H. Griffin of Buffalo who had earlier been involved with the Drummonds in the formation of the Canada Iron Furnace Company at Radnor. The four Canadians included T.J. Drummond, Edgar McDougall and Robert Cowans. To add to the international aspect, there was one director from France and one from Belgium. (Iron Trade Review, 1899). Nothing more is known about this company and it can be assumed that the merger scheme collapsed.
from a small deposit of ilmenite found and mapped by the writer.

The Canada Iron Corporation's annual report for the year ending May 31, 1912 explained the shutdowns as the result of dumping of American pig iron on the Canadian market, but despite some optimism for the future, the corporation went into voluntary liquidation in August 1913. The trustees for the liquidation — Edgar McDougall, who had been vice-president and general manager since March 1911, and the Montreal Trust Company — disposed of the corporation's blast furnaces and iron mines, and in March 1915, the foundries and machine shops were re-organized as the Canada Iron Foundries Limited (the present day Cannon) with Edgar McDougall as president. By 1915-16 the Grantham furnaces had been dismantled and in 1917, the Southern Canada Power Company built a hydro-electric power plant at Lord's Falls and the water backed up by its dam inundated the low ground where the blast furnaces had stood. The remainder of the site on higher ground is now occupied by several textile factory buildings and the only obvious evidence of the existence of the Grantham Iron Works is the street called rue des Forges and the railway spur which now serves the textile plant. Two less visible reminders, known mainly to the Drummondville city engineering department, are the glassy slag which forms the base course of several streets in the older part of the town and the blockages from deposits of iron oxide which occur from time-to-time in the city water lines from the iron saturated waters of the district.

Conclusions

The Grantham Iron Works could be described as a direct descendant of Canada's oldest iron works because the builder and first manager of the Grantham Iron Works had not only learned his skills as an "Iron Master" at Les Forges du St. Maurice but except for a greater size and some up-to-date innovations, the Grantham blast furnaces were little different from the blast furnace built at the older iron works nearly a century and a half before.

The explanation of why two archaic "pre-industrial" style furnaces were built at Drummondville as late as the 1880s is not known, but a number can be suggested. The first is that although Robert McDougall appears to have had some technical training (probably at the Three Rivers Mechanics Institute), he lacked the engineering background to design and build a more up-to-date blast furnace. That, however, would not have precluded the design of a furnace by a consultant, so that there may have been other factors. An additional possibility is that a taller furnace would have caused the soft-wood charcoal to crush under the weight of the ore. Some indication that this may have been a reason is that the metal-jacketed, water-cooled Canada Iron Furnace at Radnor used soft-wood charcoal and was only 40 feet high while the 59-feet tall Deseronto furnace used hard-wood charcoal. A third possibility is that the owner, John McDougall of Montreal and his partner Robert Cowans may have been conservative enough that they would not consider financing a type of furnace which they probably considered to be experimental. This view may have been reinforced by the failure of the "new" furnace at Les Forges du St. Maurice which was a small, "modern" furnace.

The special requirements by the manufacturers of railway car wheels for "charcoal iron" permitted the continued operation of the Grantham Iron Works until the early 1900s. By then, however, coke-fired blast furnaces could produce equally satisfactory pig iron at a lower cost and at that point the only valid reason for the continued operation of small charcoal-fired furnaces had virtually disappeared. Problems with supplies of bog iron ore, charcoal and the financial difficulties of its last owners, the Canada Iron Corporation, undoubtedly contributed to the final shutdown, but the real reason can be summarized in one word: obsolescence.

Although the type of furnace was obsolete, for at least two-thirds of its period of operation, efforts were made to improve the efficiency of the iron works. It appears to have been the first in Canada to have used long distance rail transport, the furnaces were larger than any previously built in Quebec and, in contrast to older Quebec charcoal-fired blast furnaces, apparently were equipped to recirculate the flue gases to provide auxiliary steam power. In contrast, although nearby blast furnaces in northern New England and New York had the technology to use magnetite ore, a number of the attempts to use magnetite in the Grantham furnaces caused problems. To what extent this was due to the design of the furnace or lack of skill on the part of the second manager (George McDougall) is not known.

Among a number of unknown aspects of the history of the Grantham Iron Works are the reasons for the greatly reduced production during the 1890s, at a time when Canada's over-all production for charcoal pig iron was rising. In the early 1900s production again reached levels not much different from the 1880s, so that the problem was not a technical one. It seems probable that it was either a loss of markets for railway car wheels by John McDougall and Company or disinterest on the part of the administra-

tor of the estate of John McDougall of Montreal.

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2. L. Millar, pers. comm. — Leslie Millar, one of Robert McDougall's grandsons, was born in Drummondville in 1890 but left there at an early age to work in Montreal. However, as a youngster he and his older brothers were very familiar with the day-to-day operations of the iron works under the management of their great uncle George McDougall. After his retirement from business, Leslie Millar's keen interest in the local history of the lower St. Francis River resulted in his sorting and assembling information from his family's papers and those of several other families. The author is deeply indebted to him for documents, photographs, sketch maps and numerous long and lively communications. Many of the glass plate negatives from which the photographs were made are now in the Nottam Archives of the McCord Museum, Montreal. Mr. Millar continued to be actively engaged in his various historical interests until a short time before his death in 1984 at the age of ninety-three.

3. McDougall family records — Genealogical and other documentary and published information assembled by the author from many sources. Included among them are numerous pieces of documentary information provided by the Historic Sites group of Parks Canada, in particular by A. Bérubé and M. Bédard.

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An historical sketch of the Canadian steel industry

W.M. Williams

Introduction

The Canadian steel industry is now in many ways a mature one, having achieved many technical successes during its hundred-year history. It is interesting to look back over this period, to learn something of the histories of the different producers and of the people and forces that shaped their development. The following article provides a thumbnail sketch of the Canadian steel industry as it exists today. It is intended for the general reader, not for the specialist who will want to consult the several books and theses written on the development of the industry during the last hundred years.

The famous Les Forges du Saint-Maurice, an important iron producer for a century and a half, provides a natural historical starting point for a look at the four major players — Dosco, Algoma, Stelco and Dofasco — who have provided most of the hundreds of millions of tonnes of steel, in every form from horseshoes to tinplate, needed in the twentieth century.

The smaller companies which have sprung up in the last 20 or 30 years are dealt with briefly — but only because their story is so recent. Not so long ago the mini-mill, based on scrap rather than on the blast furnace, was thought to be specially suited to the economies of the smaller developing countries of the Third World. However, recent history has made it evident that the small plant, specializing in a few basic products (e.g. reinforcing bar) or taking advantage of special circumstances, can successfully compete with the large blast furnace/primary steel producers.

Les Forges du Saint-Maurice

Les Forges du Saint-Maurice did not produce steel, but no historical survey of the Canadian steel industry could omit mention of this important iron producer, established in 1738, with a subsidy from the French government, just north of the town of Trois-Rivières, Quebec. This was Canada’s first iron works and the only major industrial venture that spanned all three periods of Canadian political history: the French regime, the British colonial period and the first years of Confederation. Its importance, not only to Lower Canada but also to the more western regions of the country which came under federal jurisdiction during the nineteenth century, was great: it provided farmers, foresters and other consumers with necessary cast and wrought ironware and tools as well as supplying blacksmiths with bar and plate. Les Forges were in active business for 145 years, closing in 1883 when local iron ore was exhausted and local supplies of wood (for charcoal) had run out. Today, the site — a very attractive one on the banks of the Saint-Maurice River — is in the charge of Parks Canada who are gradually restoring the buildings and arranging displays.

The plant was a large one, with more than 1000 people on the payroll at one time. There was a blast furnace, still to be seen, which was fuelled with charcoal and operated with a cold blast. The production, about 5 tons/day of high quality iron, was sand cast into pigs, was as customary at the time, and used for two purposes: (a) for remelting and casting into stoves, kettles, kitchen ware, etc.; and (b) for further refining in charcoal furnace to reduce the carbon and thus produce wrought iron (the forerunner of low carbon steel) which could be hammered into bars and plate for the manufacture of tools, nails and general hardware.

Product quality at Les Forges du Saint-Maurice was very high as the following extract from a letter in the Windsor Castle archives testifies:

(From Sir Jeffrey Amherst to Lord Ligonier, October 21, 1761)

“I sometime since desir’d Governor Burton to send me some Iron from Trois Rivieres that I might have it prov’d here; he send me two pieces, which the broadest and shortest was hammer’d twice, the other once.

The Principal Blacksmith of New York, who made the Experiment of the goodness and properties of the Iron, reports the largest bar extremely hard, tho’ mallable and easily wrought for several purposes of hard Iron; but what makes it most valuable, it will make as good Steel as any that goes under the Name of German Steel.

From the above report, I have order’d the remains of the Bars to be packed up, and to be sent by the first occasion to your Lordship from Falmouth, to which place the packet will take them. If these Blacksmiths are not deceiv’d, it may hereafter turn out to great advantage, perhaps that the goodness of the Iron in His Majesty’s Forge at Canada should be known, and I therefore send it to your Lordship to make that Use of it You judge proper.”

As for cast ironware, (particularly wood stoves), much survives to the present day, mostly in museums, where the castings may be admired for their decorative designs as well as for their obvious durability and historical interest. Many of these items have the distinctive ‘FSTM’ letters cast into them, making identification easy and adding considerably to their value. In a sense, Les Forges du Saint-Maurice still exist as part of the corporate body of present-day Cancon Limited, (formerly Canada Iron Foundries, and before that called Canada Iron Corporation), which took over what remained of the technical and financial property of Les Forges in the early years of this century.

Dominion Steel and Coal Company

The closing of Les Forges du Saint-Maurice in 1883 was due not only to the gradual exhaustion of local ore, as well, advancing technology had made the plant obsolete. The use of charcoal in the blast furnace had been on the decline (in favour of coke) for a hundred years, and the idea of a hot blast had been tried successfully in the early part of the 19th century. Furthermore, the manufacture of wrought iron was a dying industry by 1883 — Bessemer had introduced low carbon steel 25 years earlier and the new material was rapidly supplanting wrought iron. Steel plants were springing up all over the world.

Canada's first integrated steel works was formed from the amalgamation of the Nova Scotia Steel Company (founded in 1882) of New Glasgow, a small company operating an open hearth furnace, and the Nova Scotia Forge Company of Trenton, Nova Scotia. They were soon joined by the New Glasgow Coal, Iron and Railway Company which had secured large supplies of local hematite; before long a new company, Dominion Iron and Steel Company, emerged with a new steel plant at Sydney, Nova Scotia. It had a bright future: with local supplies of ore and coal, and with easy access to the whole Eastern seaboard as well as the inland waterways of Canada, it was obviously well placed to supply steel goods, particularly rails, to a continent in the throes of development.

Steel production expanded quickly. Between 1901 and 1911, steel production (mainly from the Sydney plant, which had far outdistanced the New Glasgow works) jumped from 29,000 to 882,000 tons/year. The war years contributed to the well being of the company but the post-war years were difficult. Demand for rails dropped steeply and in 1920 the Nova Scotia companies were taken over by British and American interests and merged with the Halifax shipyards to become the British Empire Steel Corporation.

Following the 1929 stock market crash the company was again re-organized under the name Dominion Steel and Coal Corporation (DOSCO) but another war, World War II, had to come before real profitability returned. In the early 1960s the company was sold, some of its components going to the new Sidbec steel company established by the Quebec provincial government. The central steel-making unit at Sydney, however, was taken over by the Province of Nova Scotia and re-named Sydney Steel Company (SYSCO).

Although the company has produced many millions of tons of steel during the last hundred years, particularly rails which are exported to all parts of the world, its future, like that of many older steel plants in the Western world, has been uncertain for some time. With a strong local need for jobs, it may well continue, with subsidies, for a limited time. However, it is difficult to see how this old steel plant is going to survive into the 21st century.

Stelco Incorporated

Canada’s biggest steel company, Stelco, was the impressive result of heavy ‘wheeling and dealing’ in 1910. The most celebrated participant was a 30-year old Maritimer, Max Aitken, who was already well known in and beyond Montreal financial circles as the man who, in a time of mergers and financial deals of all kinds, had already put together Canada Cement and the Canadian Car and Foundry Company.

The story, reduced to its simplest elements, began with the offer by its managing director, William McMaster, of a profitable finishing plant, Montreal Rolling Mills, to Max Aitken for $4,200,000. It took Aitken only two days to decide to buy what turned out to be a bargain. At about the same time, several Ontario companies, including a primary producer — Hamilton Steel and Iron Company — were negotiating a merger of their interests. Aitken called everyone together for a meeting at the Windsor Hotel, Montreal. No decisions were made but there was enough agreement among the parties for Aitken to adjourn the negotiations to give him time to raise money in London.

On his return, the participants went into seclusion in the Waldorf Astoria Hotel in New York, cutting themselves off from all outside activities until their negotiations were finished one way or another. The result, ten days later, was The Steel Company of Canada Limited, created by letters patent on June 8, 1910 and presided over by Charles Wilcox of Hamilton Steel and Iron.

Aitken had no interest in the day-to-day management of the new company. “When a building is finished”, he said, “I do not stay in it but move on to build another”. He soon moved to England, eventually becoming Lord Beaverbrook, a great friend of Winston Churchill and a major newspaper publisher. His finest moment was no doubt his stint as the successful, resignation prone, ruthless, (British) Minister of Aircraft Production during World War II.
The new Company had a difficult start, largely caused by personal differences and resignations, particularly of salesmen who had been fighting each other for business for years but now suddenly had to work together in common cause. It was a somewhat desperate management that decided to organize its first sales convention in Hamilton in 1911 in an attempt to get people to work together harmoniously. Fortunately, the new company had a few years to organize production and sales before World War I erupted.

The outbreak of war in 1914 was a near disaster for the new company. Agricultural and railroad orders dried up; lay-offs and drastic cost cutting were the orders of the day. But with the intensification of the war, and the realization that it was not “going to be over by Christmas”, came a change in company fortunes. Munitions production was the driving force, a force that demanded more employees, more open hearths, more production and, most importantly, the development of a greater variety of skills and improved technology. The company bought an interest in American iron and coal mines, a sheet mill was built in 1917 and by-product coke ovens in 1918. Steel capacity approached 500,000 tons in 1918 when the war came to an end.

The 1920s saw the introduction of the automobile to the mass market, canning became more and more important, and what today are lumped together as consumer goods (washing machines, refrigerators, stoves etc.) began to be needed in large quantities. These trends toward the recognizable steel ‘mix’ of today, were accompanied by changes in plant practice: from steam to electric power, from coal to oil and gas, from horse and manpower to overhead electric cranes. Testing and research laboratories were built and the first university-trained engineers found their niches in production, quality control and research.

The long and severe depression of the 1930s, which affected all the steel companies, ended only a few years before the outbreak of World War II. After producing at only a small fraction of its capacity in the early 1930s, production gradually climbed back to an all-time high by 1937. A profit of four million dollars surpassed the previous record of 1929 and the Steel Company of Canada was in a good position to contribute in an important way to the Canadian War effort.

After France collapsed in 1940, Canada was Britain’s most important ally. Steel was needed, particularly for the hundreds of ships that were built in Canadian yards. A 110-inch wide plate mill was brought into production early in 1941 and worked continuously on three shifts. Wire (for wire rope) and pipe were other items needed in large tonnages. Steel for anti-submarine nets, spring steel and armour plate were only a few of the items produced as part of a remarkable technical contribution to eventual victory in 1945.

The 1950s was a period of expansion and quiet revolution. In that decade, the company’s steel capacity increased from 1 to 3 million tons. The challenge of the newly invented oxygen process acquired by its neighbouring competitor, Dofasco, spurred the Steel Company to improvements in open hearth technology, notably a reduction in refining times through auxiliary oxygen injection. Automation, quality control, pollution control had arrived or begun to arrive, and the modern world of job-classifications, personnel management, management-union relations, all partly stimulated by a major strike in 1946, began to take shape.

Other changes belong to the more recent past. Notable among these were the setting up of Canada’s largest steel research laboratory in the early 1960s, the large investment in the new Lake Erie steel plant during the steel boom of the late seventies, with 100% reliance on modern continuous casting technology, and the shutting down of most of the old Quebec operations in and around Montreal. Stelco, like most other steel companies, is ‘rationalizing’.

**Dofasco Incorporated**

Canada’s second largest steelmaker (its present annual steelmaking capacity is 4.5 million tonnes) began in 1912 as a steel foundry, Dominion Steel Limited, located on Hamilton Bay in Ontario. The founder of the business was an American, Clifton W. Sherman, who was soon joined by his brother Frank A. Sherman. It was a substantial business for the period, employing more than a hundred workers and capable of producing about 80 tons of steel castings/day from a single open hearth furnace. At first, the main market was the railroad industry, a large user of steel castings, but less than two years after start-up World War I had broken out and munitions production became increasingly important. By the end of the war in 1918, the company had 11 open hearth furnaces and a total capacity of 850 tons/day. Its payroll had expanded to 258 workers.

Not unexpectedly, the company ran into trouble soon after the war ended when steel demand dropped and market patterns changed. In 1921, the company’s credit at the bank ran out and it was on the point of bankruptcy but salvation came, just in time, in the shape of cast steel undercarriages for railroad cars. Welded undercarriages were the standard in Canada at the time but, fortunately, the company had bought the Canadian rights to a method of casting them. Disaster was averted and the company survived the difficult period of re-adjustment to the peace-time economy of the 1920s. The steel casting division is still important to Dofasco; today it employs almost 900 people and produces a wide range of castings for the transportation, mining and electrical industries.

By 1928, the company had commissioned Canada’s first universal plate mill and had thus begun to develop its expertise in metal rolling, skills which are central to its corporate interest today. In the mid-1930s, the company (renamed Dominion Foundries and Steel Limited) installed a 20-inch cold rolling mill which in turn led to an expansion into the production of tin plate and the other flat-rolled materials basic to most consumer products, and such metallurgically sophisticated items as grain-oriented silicon steel.

By the time World War II broke out, Dofasco was poised to play an important part in the war effort. Like other steel producers, the company was called upon to make new products as well as to increase output, e.g. armour plate became a specialty. By the end of the 1940s it had become obvious that if Dofasco were to expand even further, a blast furnace was essential; it no longer made sense to work with iron and steel ingot brought in from outside the company. In 1951 with the commissioning of its first blast furnace, (it now has four) Dofasco became Canada’s fourth fully integrated steelmaker.

Its finest hour, however, was still to come. By the early 1950s Frank A. Sherman had succeeded his brother Clifton as president and the bold (some people probably
called it foolhardy) decision was made to abandon open hearth steelmaking in favour of acquiring exclusive Cana-
dian licensing rights to the new oxygen steelmaking
process successfully developed in Austria only a few years
earlier. In 1954, Dofasco tapped the first hearth from North
America's first basic oxygen furnace. This was followed in
1955 by the installation of the first continuous galvanizing
line in Canada and, in 1956 by the adoption of oil injection
in the blast furnace, an innovation that led to record
productivity and low coke rates. By 1960, the company
had three blast furnaces in operation with associated coke
oven batteries and by-product plant, a hot strip mill, two
pickle lines, two galvanizing lines and two 56-inch mills.

Of all the major Canadian steel companies Dofasco has
weathered the recent economic squalls most successfully,
a consequence of keeping up-to-date technologically and
having the right 'mix' of steel products. There is also no
doubt that good management has played an important
part in this success story.

The Other Steel Companies

The history of the Canadian steel industry is very much the
history of the four integrated steelmakers, all owners of
that venerable and gargantuan piece of technical machin-
ery, the blast furnace, which has its origins in the small
bellows-operated furnaces of several millenia ago. Even
this brief history of steelmaking in Canada, however,
cannot omit mention of the smaller companies which have
sprung up in the last 20 or 30 years, e.g. Lasco, Ipsco and,
more recently, Ivaco, to mention only the more prominent
enterprises that have concentrated exclusively on electric
furnace steelmaking largely based on steel scrap. Few
people in 1960 could have guessed how successful these
non-blast furnace companies would become. Today in
Quebec, for example, a province which has not had a blast
furnace for more than a century, there are companies
making electric furnace steel from scrap as well as a
company (Sidbec) that makes electric furnace steel directly
following the reduction (Midrex process) of iron ore using
natural gas. A recent development is the new plant of QIT
Fer-et-Titane Incorporated to make steel via the K-OBM
process from the high quality pig iron which results from
the smelting of ilmenite, a titaniferous iron ore that also
yields titanium dioxide (for paint pigment).

It is too soon to say how important these smaller
companies will become in the over-all pattern of Canadian
steel production. The history of Canadian steelmaking,
and the steelmaking business, is still unfolding, bringing
about changes in commercial and geographic emphasis in
interesting and sometimes unexpected ways. What is
unchanging are the commercial, scientific and technical
challenges which have been so successfully met during the
last 100 years and which will continue to face Canadian
steelmakers into the next century.

Conclusion

There has been a tendency in recent years for the ferrous
metallurgist and his financial masters to look with envy at
other industry sectors and to wonder whether his own
industry has stagnated, has been by-passed or is headed
for slow extinction. These notes may help to place the
present Canadian industry in historical perspective, allow-
ing us to stand back and see how the separate components
make up the whole, to see what is good, well-managed
and technically enterprising, to see where the new trends
lie and to consider how strong and lasting they might be.

On the reverse side of the coin they allow us to look
dispassionately at the industry, its methods and products,
its geography and financial structure, so that we may be
properly informed about the technical decisions which
must continuously be made to keep it strong and profit-
able.

Steel itself, whether made from blast furnace iron or via
some other metallurgical route, will remain the staple
engineering material for as far into the future as anyone
can forecast. With its three great advantages: (1) its
cheapness, (2) its integrity at high temperatures, even in
unalloyed form, and (3) its unrivalled mechanical proper-
ties, particularly its outstanding stiffness, steel will with-
stand for a long time the intensifying competition from
polymers, ceramics and the other common metal alloys.

Acknowledgments

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made a number of valuable comments. His help is gladly
acknowledged.

FURTHER READING

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Trottier and is available from Parks Canada.

2. Algoma Steel Corporation: Steel at the Sault: Sir James Dunn and the
Algoma Steel Corporation. 1966-1956. by D.L. McDowell and
presented as a doctoral thesis at Carleton University in 1978, provides
a detailed history of the company.

3. Stelco Incorporated: The Elements Combined, a very readable
account of the company's origins and development to 1960 has been
provided by the distinguished historian William Kibbourn (Clarke,
Irwin). The wood engravings by Rosemary Kibbourn are noteworthy.

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of the company.
Early Nova Scotia gold mining

John Hartlen

The 19th century was an “age of gold” and Nova Scotia experienced its share of early gold rushes. Gold mining activity in the province was focussed into three “rushes” which occurred, approximately, in the 1860s, 1890s and 1930s; a total recovery of 1.1 million oz. Au has been recorded. The purpose of this note is simply to chronicle the technical development of gold mining and processing in the province and to comment briefly on the recently renewed interest in this precious metal.

The first public discovery of gold in Nova Scotia was made by a farmer in 1860 at Old Tangier or Mooseland, and this first sparked the province’s first gold rush. The mining and milling methods of the 1860s were ingenious. Initially, arrastras, crude stone-hewn grinding mills, drawn by men, mules or horses, were widely used. Nova Scotia’s first conventional stamp mill was installed at Mooseland by 1862; it was comprised of four wooden stamps and is believed to have been imported from Scotland. One of the largest waterwheels of the 1860s, reportedly 16 feet in diameter and 8 feet wide with a 50-horsepower rating, was used in the Renfrew Gold District to drive gold stamps. Steam power was also employed, and many of the engines were of native construction, as was the unit at Waverley’s Lake Major Company, which had a piston 1 foot in diameter by 12 feet long and was connected to a 16-ft-diameter flywheel.

Nova Scotia miners were also quick to try new extraction methods. In the 1860s, a Bavarian running a mill near Halifax first applied quicksilver to crushed gold ores. Plattner’s Process for the separation of gold using chlorine was tried at Tangier in 1863. Quartz kilns were marked on maps of the province in the 1860s; however, by the end of the same decade, ore roasting was abandoned at Tangier when the cost of wood rose to $4 a cord.

In September 1865, there were 34 gold mills in Nova Scotia, 22 driven by steam and 12 by waterpower, servicing 95 mines where 836 hands were employed. The first gold rush peaked in 1867 with the production of 26,000 oz Au. Thereafter, gold production declined steadily as the known shallow gold reserves were depleted.

Nova Scotia’s second gold rush from 1895 to 1903 was the most productive period in its gold mining history, with about 240,000 ounces of gold having been recovered and new technology introduced. Activity peaked in 1897 when 35 localities were being worked by about 4000 persons. One of these early operations, the Lakeview Gold Mining Property at Waverley, Nova Scotia, is shown in the 1896 photo presented in Figure 1. The mill is the foreground, with the actual mine perched on the ridge above it. This and similar mills flourished for a period, but by 1907 the province’s second gold rush had ended and only about 500 persons were still employed in the gold processing field.

In 1891, portable electric power machinery was first employed at a Nova Scotia gold mine. The use of steam-powered drills and compressed air machinery was established in the same decade. Stamps were still in common use for crushing, as Figure 2 clearly shows from an 1890s view of the Lakeview stamping mill. A Bostonian named H.S. Badger experimented with cyanide gold milling in the 1890s shortly after the process was patented. From 1900 to 1902, Wilber Libbey, President of The Mining Society of Nova Scotia, evaluated both cyanidation and chlorination gold milling processes at his Brookfield Mine. In 1903, a member of the Geological Survey of Canada reported that a cyanide plant was about to be installed at Brookfield and that a cyanide plant at Leipsigade had already been running “successfully” for about two years.

As the near-surface gold deposits were rapidly becoming depleted, interest shifted to deeper underground mining methods. The focal point of the deeper-mining lobby became “On Gold Measures of Nova Scotia and Deep Mining”, a technical paper given by GSC geologist Eugene Farbault at the Annual Meeting of the Canadian Mining Institute in Montreal in March 1899. By the following year, Farbault’s “pay zone theory” was the main topic of conversation of provincial mining men. In April 1903, as a result of Farbault’s theory, the Nova Scotia...
legislature passed a *Deep Mining Act* enabling a subsidy to be given to aid the cost of sinking shafts deeper than 500 feet. However, the anticipated benefits of deep (for the period) mining were not realized, and this sparked a controversy involving both miners and politicians that remained intense even after the then current phase of gold mining and milling activity had died down.

As an aside, the 1890s also marked the start of several mining-metallurgical organizations. In 1892, the Nova Scotia Gold Miner’s Association joined coal and iron miners to form The Mining Society of Nova Scotia. In 1896, the Federated Canadian Mining Institute was formed, two years later becoming the Canadian Mining Institute, which was the forerunner of The Canadian Institute of Mining and Metallurgy created by Charter in 1920.

Nova Scotia’s third gold rush lasted from 1932 to 1942 and resulted in the recovery of an additional 158 000 ounces of gold. Activity was stimulated both by the *Canada Domestic Fuels Act* of 1927 that promised cheaper coal-fired energy and enabled lower-grade ore deposits to be mined at locations with no access to waterpower and especially by a two-third increase from $20.97 to $34 an ounce in the U.S.A. price for gold effective January 31, 1934. Gold production for this phase of activity peaked in 1939 when 30 000 oz. Au were reported; production was abruptly curtailed in 1941 because of wartime labour regulations.

Will Nova Scotia ever experience another period of active gold mining and milling or did the earlier workers recover all the gold there was? Certainly the prospectors still believe there is a future for gold in the Maritimes. By 1979, gold prices had risen in uninflated dollars above even the record prices of the early 1900s and 1930s, and Nova Scotia’s fourth gold prospecting rush was on. It was reported by the Mineral Resources Division of the Nova Scotia Department of Mines and Energy in December 1980 that numerous individuals and small companies had lately staked ground at former mining camps. With such activity and the application of modern exploration, mining and milling technologies, Nova Scotia may again become a significant gold producer.
Metallurgical developments at Deloro, Ontario: 1868-1919

Roy T. Bowles

This paper focusses on technological developments which occurred in the village of Deloro, Ontario between 1868 and 1919. These developments illustrate the manner in which improved metallurgical processes linked Ontario mineral resources to industrial expansion.

Overview of Deloro

The village of Deloro is in Hastings county, about 5 miles from the village of Marmora, the site of the old Marmora iron works and the more recent Marmornaton iron mine. In this area, the Canadian Shield meets the Great Lakes lowlands. The first settlement in the vicinity occurred in the 1820s and was associated with the Marmora iron works. Settlement grew around agriculture.

Gold was discovered in Hastings county in 1868. Deloro — the name derives from gold — developed as a concentrated settlement in proximity to several gold mines and the related processing activities. By 1901, substantial smelting works were successfully producing gold and arsenic from local ores. The village was developing as a company town within the framework of benevolent paternalism. For reasons associated with markets and with local ore supplies, the Deloro mills were idle for a period beginning in 1902.

Developments in later periods connected Deloro to world events. In 1903, silver was discovered at Cobalt, Ontario. By 1907, M.J. O’Brien, who owned mines at Cobalt, had purchased the Deloro works and was using them to process Cobalt-area ores.

The Deloro Smelting and Refining Company (in its early years called the Deloro Mining and Reduction Company) operated as a subsidiary of M.J. O’Brien, Ltd. until 1961, when the plant was closed. The smelter produced silver, cobalt, arsenic and small quantities of other metals. The metal manufacturing department produced stellite, some other alloys and stellite products.

This paper focusses on the period up to the end of World War I, but a brief overview of later periods will be useful. Prosperity at Deloro depended on the demand for and alternative supplies of strategic alloy metals. Early in the O’Brien period, there was experimentation with cobalt metals and alloys. By the beginning of World War I, Deloro was producing stellite, an alloy which retains its hardness at high temperatures. Demand for stellite cutting tools in the munitions plants provided the basis for rapid growth at Deloro. General prosperity (but with some slack periods) characterized the 1920s. Activity declined through the 1930s, but by 1938 contracts to supply stellite to the British aircraft industry brought renewal and expansion. Through World War II and the Korean War, the Deloro Smelter was one link in the strategic materials programs of the Canadian and American governments and as part of these programs processed concentrates from African mines.

Following World War II, the metals department engaged in precision casting of alloy products, including components for jet engines.

The closure of the Deloro Smelting and Refining Company appears to have resulted from the movement of the metals manufacturing to Belleville, Ontario (where it continues, but now under different ownership) and the decline in the price of cobalt partially as a result of its production as a by-product by large smelters, including Inco. The village of Deloro continues to exist as an organized municipality of 250 residents. The only economic activities are two nursing homes and a general store.

Beginning late in the nineteenth century and continuing through the first three decades of the twentieth century, there was a period of rapid economic expansion in the Canadian economy. This expansion was characterized by a growth of prairie wheat production, a growth of manufacturing, and the rapid development of the new staple industries of minerals, pulp and paper, and hydro-electric power. This expansion was associated in time and technological processes with what some historians have called “the new industrialism.” To the iron and steel of the “old industrialism” were added: (1) new materials in the form of aluminum alloys and alloy steels; (2) new sources of energy in the form of electric power and petroleum; and (3) new processes such as the reduction of complex ores and the recombination of metals to form alloys.

Development of Gold and Arsenic Technology

Two types of technological development important in the new industrialism were improved processes for the reduction of complex ores and the production of alloy metals.

* The relative importance of the Deloro Smelting and Refining Company in the Ontario mineral industry in relation to the Cobalt-area mines is indicated by a number of things. Under the Ontario Metal Refining Bounty Act which was in force between 1907 and 1917, Deloro received $49,000, Coniagas received $67,000 and four other smelters combined received $11,000.

By 1925, all other smelters had closed and “the Deloro Smelting and Refining Company remained as the only smelter in Canada capable of treating Cobalt, Ontario ores.”

In the early 1930s Deloro Smelting and Refining Company... and Union Minière du Haut Katanga established an agreement to control the marketing of their products and to maintain the cobalt price.” By 1936 this Cobalt association included six members (one being an association of German producers) who among them controlled almost 90 per cent of world cobalt production. During World War II, when Canadian cobalt became strategically important, Deloro Smelting and Refining Company was appointed agent to buy cobalt ores from Cobalt, Ontario by the Metals Reserve Company of the United States government and the Canadian Crown Company, War Supplies Limited. In 1951, to stimulate production considered essential because of the Korean War, C.D. Howe, Minister of Trade and Commerce, announced that the government would purchase cobalt concentrates and appointed Deloro Smelting and Refining Company as authorized agent.


Readings in Historical Metallurgy
Developments in both of these areas occurred at Deloro, and occurred there at times which gave Deloro a significant place in the Ontario mineral industry.

Developments in the processing of ores occurred in relation to Hastings county gold mining. Hastings county gold, which had been discovered in 1868, occurred in complex mispickel ores in combination with arsenic. Between 1870 and 1900, various companies operating at Deloro and other central Hastings locations established mills and attempted to develop smelting processes which would extract the gold. Many of these attempts were technical and financial failures.

The First Report of the Ontario Bureau of Mines reported activities up to 1891:

“It is now more than a quarter of a century since gold was discovered in the county of Hastings. For several years mines were worked at various points in the townships of Marmora and Madoc, and a number of mines gave good promise to their owners. Several mills for treating the ores were built, and excitement at times was raised to a fever heat as new discoveries were made. But in almost every case it was found that when a shaft had been sunk to the water-line the ore ceased to be free milling, and no process then known made the economic treatment of it possible. The mispickel, although still rich in gold as shown by assays, was too refractory to yield the precious metals in paying quantities; and notwithstanding that various methods were tried and large sums were wasted on new experiments, conviction slowly settled down on the minds of capitalists, miners and metallurgists alike that there was no money to be made in mining or milling the mispickel ores of Hastings.”

The report provides a very optimistic description of a new mill in the village of Marmora which was applying the Walker-Carter process developed in the United States. The Fourth Report returned to pessimism, stating that the hundreds of thousands of dollars spent on elaborate reduction works only resulted in failure.

A major turning point in development at Deloro appears to have occurred in 1896. Canadian Goldfields Limited (based in London, England) acquired properties at Deloro. It also obtained rights to use “the bromo-cyanide process for the treatment of gold ores — sometimes called the Sulman-Teed process, for the names of the inventors — which had a special aptitude for extracting gold from arsenical and other refractory ores by means of solution…” In 1898 and 1899, this process was used for the extraction of gold, but apparently fully integrated operations were not yet established, because the residue was being stored to be treated later “for recovery of arsenic, itself a valuable by-product.”

Peter Kirkegaard, the superintendent, and S. B. Wright, a metallurgist, described the process in the following terms:

“The concentrating plant consists of three hydraulic classifiers, and five smooth belt vanners for the 10 old stamps; and three Willfrey tables, one classifying cone, and one Bartlett table for the 10 new stamps… The concentrates were transported in cars to the leaching plant, where they are treated by the bromo-cyanide (Sulman-Teed) process. This consists of 1) extraction of gold by leaching the finely ground ore with a solution of potassium cyanide to which is added a small quantity of a solution of cyanogen-bromide — a haloid salt of cyanogen; (2) precipitation of the gold from this bromo-cyanide solution by means of metallic zinc; (3) removal and smelting of the zinc-gold slimes, thus obtaining the pure gold… The concentrates, after the extraction of the gold, are sent to the arsenic works, where they are calcined for the arsenical contents. The crude arsenic resulting from the roasting is refined and produces white arsenic… analyses of which show 99.6 to 100 per cent pure arsenious oxide (As₂O₃).”

A 1902 article on “Arsenic in Ontario” indicated that arsenic was becoming central to economic activities at Deloro. Concerning arsenic production in Canada, it stated: “The whole output was white arsenic from the Marmora district in Hastings County, Ontario.” Production had increased from 57 tons in 1899 to 300 tons in 1900 and 700 tons in 1901.

The Arsenic Period

In the development of techniques to produce Hastings county gold, it had been necessary to deal with arsenic. Whether accidentally or intentionally, the production of arsenic at Deloro became an important economic activity. (Arsenic was marketed as a herbicide, insecticide, and for industrial purposes such as the manufacture of glass.) There was a brief period of prosperity at Deloro, based on the production of gold and arsenic from local ores. Prosperity ended, apparently for two reasons. In March 1902, a mine shaft at the Canacian Goldfields was flooded, preventing access to a major orebody, and there were difficulties in centralizing operations of the many small properties in the area which might have provided a supply of ore. The facilities were then idle or operating at only partial capacity for a few years.

The technological challenge which confronted those who sought the gold of Hastings county was the separation of various metals bound together in complex ores. This challenge had been solved by the personnel of Canadian Goldfields Limited. After careful reading of the reports, this author concludes that the challenge was met by bringing together at Deloro machinery and processes developed elsewhere and combining them to accomplish the difficult task of treating local ores. Machines are identified by brand name or referred to in contexts which imply that informed members of the mining industry will recognize them. Some processes are treated as generally familiar. Others are identified by registered rights. Some local modifications in machines or processes are noted, but no reference seems to indicate an “invention”. These qualifications do not minimize the accomplishment. Entrepreneurial and engineering achievement is measured by the extent to which the solution of problems makes an enterprise financially viable.

The Link with Cobalt-Area Ores

The technological capacity developed in the nineteenth century to treat Hastings county ores was significant in the early twentieth century mining boom based on Cobalt-area silver.

The developing relationship between the Cobalt Camp and Deloro can be traced through the reports of the Bureau of Mines. In the fall of 1903, silver deposits were discovered at Cobalt. By 1904, efforts had been made to resume production at Deloro, using Cobalt-area ores. By 1905, the O’Brien mine (owned by M.J. O’Brien) was in
production. By 1906, several mines were producing. Output had increased substantially, with 5335 tons of ore being shipped.

The report for 1906 provides several indications which make a link with Deloro not surprising. The difficulties of smelting the ores are noted:

“There is no doubt that the metallurgical processes involved in the complete separation of the four valuable ingredients of the Cobalt ores, namely silver, cobalt, nickel and arsenic, present a good many difficulties, but there is no occasion to doubt that the difficulties will yield to skill and ingenuity, and that ere long these refractory ores will be satisfactorily treated within the boundaries of the province.”

The ores shipped contained 160 tons of nickel, 321 tons of cobalt, 1440 tons of arsenic and over 5 million ounces of silver. Mine owners, however, typically received no payment for cobalt, nickel and arsenic. The principal purchaser of cobalt ores was the American Smelting and Refining Company of New Jersey, but there is mention of various refineries under construction in Ontario.

One quotation indicates a significant link between the technological capacity developed by Canadian Goldfields and the metallurgical challenge presented by the Cobalt-area ores:

“At Deloro in Hastings county, the gold-arsenic extraction plant formerly operated by Canadian Goldfields has been acquired by the Deloro Mining and Reduction Company, Limited, which is now refitting it with machinery required for the Kirkpatrick-Kirkegaard process, with a view to treating ore from the Cobalt Camp.”

Also significant is the mention of S.F. Kirkpatrick, a professor at the Kingston School of Mines, and the reference to a specific process developed by him and Kirkegaard, the engineer who had directed the successful development of processes for treating Hastings county ores.

The relationship between the high arsenic content of Cobalt-area ores and arsenic refining capability at Deloro appears to have been recognized. The report stated the expectation that the new Deloro firm would “shortly be treating the product of the... Cobalt Camp for the extraction of arsenic and other constituents of value.” The importance of the arsenic capabilities in existence at Deloro is also reflected by the fact that in 1907 the Deloro smelter was the only purchaser paying for the arsenic content of Cobalt ores.

A special article on the Deloro Mining and Reduction Company was printed in the November 15, 1907 issue of The Canaanian Mining Journal. Brief quotations reinforce the interpretation that this company was using technology developed to conquer Hastings arsenic to confront the complex ores from Cobalt.

“A smelting establishment, as interesting as it is unique, has been completed at Deloro, Ontario. The cause of its erection was the crying need of a Canadian metallurgical plant that would treat the arsenical ores of the Cobalt district. Such a plant, of course, does exist at Copper Cliff. But the increasing output of Cobalt camp, the complexity of the ores, and a growing sense of the futility of shipping ores to American smelters, pointed indisputably to the necessity of building a new plant... After a year or more of preparation the plant of Canadian Goldfields has been remodelled, large additions have been made and the new, complete and modern metallurgical establishment operated by the Deloro Mining & Reduction Company has commenced treating ore from Cobalt... The mill practice has been evolved, of course, with reference to the heavy, slimes arsenical ores of Deloro. Experience has proved already that it is well adapted to Cobalt ores.”

FIGURE 1. Ledyards Gold Mine, near Madoc, Ontario (Public Archives of Canada).

Technological Challenge, the Manufacturing Condition and the War

Before more specifically examining technological developments and economic growth at Deloro, some general points need to be made concerning technological challenges, the development of a manufacturing condition in Ontario and the association in time with the development of specialized productive capacity at Deloro.

The ores of the Cobalt Camp were complex; that is, they contained a number of minerals bound together in such a way that the extraction of each metal required sophisticated metallurgical processing. Two technological challenges faced those wishing to profit by processing Cobalt ores. The first was the separation of silver, the most valuable metal and the metal for which there was clear demand. The second was the separation and refining of the other metals, which in this case were cobalt, nickel and arsenic. The properties of cobalt and nickel and the potential uses of these metals were not well developed when Cobalt mines began production. A third technological challenge which emerged was the development of uses for cobalt which would make it a valuable commodity.

Many smelters developed the capacity for extracting silver, and a few became efficient in the extraction and refining of cobalt. The features of the Deloro Mining and Reduction Company which seem to have been important in its early and continued success were: (1) the existing and continuous capacity to use the arsenic content of ores to produce commercial by-products; (2) the capacity to extract silver at an early date; (3) the development of capabilities for the separation and refining of other metals, particularly cobalt; and (4) the development of uses for cobalt metals. Focus in the discussions which follow will be
on developments in the refining and use of cobalt.

Through the late nineteenth and early twentieth centuries there was a focussed concern with the development of a manufacturing condition in Ontario. This concern permeates discussions of the treatment of Cobalt-area ores. In 1907, the Bureau of Mines was optimistic “that ere long these refractory ores will be satisfactorily treated within the boundaries of the province.” The report for 1916 discussed at length and with great pleasure the developments which had permitted most of the processing of Cobalt-area minerals to become located in the Province of Ontario. Concern with the manufacturing condition, as it applied to minerals, was reflected in government policy:

“With a view to encouraging the production of refined nickel, refined cobalt metal, oxides and salts, the Ontario Government in 1907 passed the “Metal Refining Bounty Act” under which a bounty of six cents a pound was payable to smelters on the cobalt content of their finished products.”

The Act was in force until 1917. In that year, the Bureau of Mines claimed:

“It may fairly be said that the Act was successful in achieving the object aimed at, namely, the establishing of a refining industry in Ontario for the treatment of Cobalt silver ores not only for their silver contents, but also for the cobalt and nickel.”

While perhaps claiming too much for the specific effects of the bounty, it reflects the continued ideological commitment to the development of a manufacturing condition in the mineral industry.

Two essential components of the technological infrastructure for the manufacturing condition were energy and transportation. There is frequent mention of the importance of hydroelectric power in the processing of Cobalt-area ores. Hydroelectric power lines arrived at Deloro from the Trent River in 1909. A railroad spur line connected Deloro to the Central Ontario Railroad in 1913.

By way of overview, developments at Deloro were responding to the drive to establish an Ontario manufacturing condition as reflected at the levels of ideology, government policy and technological infrastructure.

World War I created a demand for increased industrial output and thereby for materials which would increase manufacturing efficiency. Expanding uses of cobalt metal and cobalt alloys helped meet this demand in such a way that producers had ready markets. At the beginning of World War I, the Deloro Mining and Reduction Company had an established physical plant with good transportation connections and energy supply. It also had an experienced labour force which could serve as the core of an expanded work crew. As will become clear in the following discussion, personnel there had a developed orientation toward and capability for product development. Because of the confluence in time of the integrated capabilities of the Deloro Mining and Reduction Company and the outbreak of World War I, Deloro was visibly connected to the force of world events.

The Development of Cobalt Metals and Stellite

Developments in cobalt technology can best be presented in a simple chronology. In 1907, the Deloro smelter was producing silver, but “had not made any attempt to separate the cobalt and nickel, which are retained in residues.” In 1909, both the Deloro and Coniagas smelters were preparing to manufacture cobalt oxide. However, the Bureau of Mines noted that the market for cobalt oxide was likely to remain depressed unless new uses were developed, because one year of production would meet several years’ supply. The price had dropped from $2.50 per pound in 1903 to 80¢ per pound in 1909. In 1910, the smelters at Deloro and Thorold were producing cobalt oxide for the first time, but market conditions were much the same and the price had dropped to 75¢ per pound. There was cause for optimism in the fact that Elwood Haynes had produced stellite, an alloy which included cobalt. The market remained depressed through 1911, and the Bureau did not detect any important developments with stellite. In 1912, stellite was being developed for cutting tools, and German steelmakers were making large purchases of cobalt.

Important experimentation was underway during this period. In 1913, Professor H.T. Kalmus of Queen’s University published Preparation of Metallic Cobalt by Reduction of the Oxide, in which different experiments are reported in technical detail. The following quotation relates these efforts to Deloro:

“Nearly 1000 lbs of commercial black cobalt oxide have been given to this laboratory for these experiments and the parts to follow, by the Deloro Mining and Reduction Co. Deloro, Ontario, to whom we take this opportunity of expressing our thanks. We particularly thank Prof. S.F. Kirkpatrick in this connection and as well for many valuable suggestions during the process of these researches.” (Kirkpatrick was a member of the Board of Deloro Mining and Reduction.)

Given the importance of stellite in the expansion of cobalt markets and the importance of Deloro as a manufacturer of stellite, some rather specific documentation is appropriate.

Haynes himself reported on his work in 1910: “An alloy of 75 per cent cobalt and 25 per cent chromium, to which small quantities of other metals are added, is not only sufficiently hard for good edge tools, but is quite tough and can be bent much beyond its elastic limit without cracking; resembling in this respect the alloy steels, but generally speaking, it is much harder.” Haynes thought his new alloy particularly appropriate for pocket knives, cutlery and laboratory tools because it did not tarnish.

According to S.B. Wright, general manager of Deloro, Haynes had begun attempts in 1897 to produce an alloy which would resist oxidizing and maintain a cutting edge. In 1907, he acquired patents on both a nickel-chromium and a cobalt-chromium alloy. He continued experiments with the addition of other metals. Regarding the cobalt-chromium alloy: “By introducing from 8% to 20% of tungsten, ternary alloys were made and their hardness was so great that they would scratch quartz.” Lathe tools made of these alloys were then tested in the machine shop, and the remarkable results led rapidly to the adoption of stellite tools in many shops. Patents covering the ternary alloys were granted in 1913.

It is clear that Haynes spent some time at Deloro, and local tradition has it that some of his inventions took place there. However, the specific dates of his presence and the nature of his work at the site have not been documented. Extracts from speeches presented years later by two company officials are the best that can be done at present:
As the production of cobalt oxide exceeded consumption, the directors (per T. Southworth — vice-president) invited a Mr. E. Heynes to visit Deloro. He experimented with cobalt, chromium and tungsten. This discovery became "Stellite", which proved to be the best cutting tool known for lathe work.

"Haynes came to Canada again to arrange for supplies. A deal was made and Deloro received exclusive rights to market stellite in the British Empire, France, Switzerland and other countries in return for a supply of cobalt metal.”

The information presented above seems to firmly establish three important points about the technological and market situation of the Deloro Mining and Reduction Company at the beginning of World War I. Stellite had been patented. Deloro had manufacturing and marketing rights to stellite. Deloro had a market for the cobalt metal in Haynes’ U.S. manufacturing enterprises which no doubt also participated in war manufacturing. The results of experimentation by Haynes and Kalmus were soon applied in production. The Bureau of Mines reported that in 1914 Deloro produced 226,079 pounds of metallic cobalt and that cobalt was being used in cutting materials. The nature of this technological breakthrough and the apparent relationship between the Kalmus research and applications are indicated in a statement made by a company official some years later:

"Until 1912 no cobalt metal was produced commercially. The production from Deloro was cobalt oxide... Dr. H. Kalmus of Queen's University was appointed by the Ontario Government to investigate this problem and in 1914 at the beginning of World War I, cobalt metal was made commercially at Deloro. Until then the small quantity of metal made on a laboratory scale was far more expensive than gold.”

In 1915, the Bureau of Mines noted that cobalt was becoming a metal of importance in electroplating and steel making. Deloro was reported to be manufacturing stellite and experimenting with cobalt electroplating. Continued technological development at Deloro is reflected by the fact that it was the first smelter to produce metallic nickel from Cobalt-area ores. Three smelters received bounty payments. Deloro receiving more than the other two combined. A detailed report on the Deloro Mining and Reduction Company and the processes it employed was produced.

In 1916, the name of the Deloro Mining and Reduction Company was changed to the Deloro Smelting and Refining Company. Although the motivations of company officials are not known, the change in name is consistent with changes that had taken place in the mining industry of the province and in operations in Deloro. Concentrators and smelters at Cobalt and Gogwanda had for some years been doing most of the reduction of ores. While Deloro continued to receive some cobalt-silver ore in lump form, it also received "jig and table concentrate, and ore residues such as those produced by the Nipissing Mining Company..." Increased concentration at Cobalt, together with availability of electrical power and the transportation costs of other inputs to the smelting and refining process, makes the location remote from the mineral supply understandable.

In 1917, Deloro was manufacturing refined silver, refined arsenic, cobalt oxide, metallic cobalt, nickel oxide and stellite. Output of stellite had been greatly increased. Facilities were expanded by the construction of new offices, a new arsenic bag house, a metals manufacturing building and a warehouse. The new plant expansions were designed to accommodate expanded metal manufacturing and new experimental work with metals. The plant employed 400 men, the largest number mentioned during any war year. By 1918, Deloro had increased production of stellite to 21,000 pounds a month to meet the demands of the munitions factories, had developed a new stellite alloy called "festeelite" and had added a new extension to the metals manufacturing building.

In 1918, Sydney B. Wright, the general manager at Deloro, published an article in The Canadian Mining Institute Bulletin describing "the Smelting and Refining of Cobalt Silver Ore." The following year he reported on "The Production and Uses of Stellite." Both articles focussed on the Deloro operations. An extended quotation will provide significant insight into the role of cobalt metal technology in war manufacturing and technological developments more generally characteristic of the new industrialism:

"The variety of uses to which Stellite is being put is increasing steadily. It now enters largely into the manufacture of many makes of automobiles. In connection with the production of 6-inch and 75-mm shells in Canada production was accelerated to a surprising degree by the use of Stellite tools. As an example of this it may be permissible to submit the following figures furnished by a Canadian shop: on 6-inch shells, average depth of cut 1/2 inch, the average speed with high-speed steel tools was 45 feet, the average speed with Stellite tools was 60 feet. Using Stellite tools, the average speed was 55 to 65 feet, the average speed 3/2 to 1/2 of an inch, and the production per hour 7 1/2 shells; with Stellite tools the production increased to 23 shells per machine per hour. Stellite has also been employed with success at mines in the Porcupine district in facing off pump liners, which high-speed steel had failed to do effectively. The use of Stellite in the form of welded tools is also developing rapidly. The Stellite tips are usually welded to carbon steel shanks, and various tools are made with the aid of the acetylene torch, by building up Stellite tips or edges on steel bosses or shanks.

"During the past year, much time and money has been devoted to experimentation along the lines originally contemplated, namely, the application of Stellite to the manufacture of cutlery, surgical instruments, etc. The following articles have been made to a greater or less extent: pocket knife blades, table knife blades, surgical instruments, spatulas, dental instruments, evaporating dishes, annealing dishes, ornamental work, valves, plumbing fixtures, pens and combs.

"Recently, Mr. Haynes was asked to furnish a special grade of Stellite for use in boring the cylinders of the Liberty motor now used in aeroplanes. This he succeeded in doing..."
Microcosm, Technology and the World

How is the notion of microcosm useful in organizing thoughts about the technological processes which occurred at Deloro? Deloro was a special locale with regard to the development of technology. The mineral resources located in the area — gold and arsenic — could only be exploited with the technology which could reduce complex ores. The motivation to exploit these resources brought the techniques and personnel together in that spot. The discovery of complex silver-cobalt ores at Cobalt at a time when the Deloro technology was available, but under-utilized, provided the opportunity for a connection between Deloro and this new resource supply. The association in time between events at Deloro and world events is very significant. At the beginning of World War I, in 1914, Deloro had the facilities for production and it achieved the technological breakthrough of producing metallic cobalt on a commercial scale. Deloro was a significant site in the development of the technological capacity to transform Ontario’s mineral products into commodities required for the new industrial technologies of the world.

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The first cyanide leaching plants in Canada

J.E. Dutrizac and J.B. Sunstrum

Introduction

Until relatively recent times, virtually all gold was recovered by gravity and/or amalgamation methods. Although such techniques are very effective on coarse gold, they become increasingly less useful for finer grains especially when the gold is partly enclosed in rock matrix. Throughout the nineteenth century a number of bonanza gold fields were found in the United States, South Africa, Australia and Canada. In every case the abundant supply of gold nuggets and coarse alluvial gold soon gave way to hard rock mining with its inevitable losses of fine gold in the stamp mill tailings. Continuing efforts to develop new technologies capable of dissolving gold from finely ground ore were eventually successful first with chlorination (the Plattner process) and subsequently with cyanidation.

Even before the nineteenth century, however, European chemists such as Scheele and Berthollet were aware that gold could be readily dissolved in an excess of free cyanide\(^{1}\). As early as 1840, a patent was issued to the Elkington brothers for the dissolution of pure gold in cyanide solution and its subsequent plating onto various metals, with or without an applied electrical current. In 1846, Elsner showed that oxygen was also required to solubilize gold or silver in cyanide media\(^{11}\). Despite the fact that much work was done on the reaction of cyanide solutions with gold, a viable leaching process for gold ores did not quickly emerge although cyanide solutions were used to enhance amalgamation from the 1860s onward. Major problems were associated with the use of concentrated cyanide solutions; the costs of cyanide were excessive and it was difficult to recover gold from the concentrated media.

These difficulties were finally overcome in 1887 – 1888 by MacArthur and the two Forrest brothers who patented the use of dilute alkaline cyanide solutions and the subsequent recovery of all the dissolved gold by cementation on zinc turnings. The success of the MacArthur-Forrest process was immediate and soon cyanide plants were operating in New Zealand (1889) and at two locations in South Africa (1890). The process was applied using either vat leaching or agitated tank methods; feed was commonly stamp mill-gravity circuit tailings although raw ores and concentrates were leached from the inception of the technique. During the decade 1896 – 1906, the world’s production of gold doubled, and this is largely attributable to the arrival of cyanide leaching technology\(^{15}\).

Early Canadian Cyanide Leaching Plants

Gold has been produced in Canada since 1824, with major output coming from the Caribou district of British Columbia in 1857, Nova Scotia in 1860, the Klondyke in 1895, the Forcupine camp in 1909 and Kirkland Lake in 1912\(^{20}\). As was the case elsewhere in the world, most of the gold was recovered by gravity or amalgamation methods, which often left significant gold values in the tailings. Gold mining in Nova Scotia was at its peak\(^{20}\) in the 1890s and efforts were being made to recover this fine gold, including the use of Plattner’s chlorination process which had been first tried in Nova Scotia in 1863. As a general observation, however, chlorination processes suffered in their lack of selectivity toward the precious metal and, hence, they tend to be expensive.

The Brookfield Mine, Nova Scotia

Knowledge of the cyanide process and its great achievements in South Africa in the early 1890s spread rapidly throughout the world. The first Canadian application of cyanide leaching seems to have taken place in 1893 – 1894 at the Brookfield Mine, North Brookfield, Nova Scotia, but this was far from a resounding success\(^{20}\). The intent was to treat ~ 6000 tons of stamp mill tailings using a “novel” cyanide leach process advocated by one Professor Kendall of New York. Despite rising costs, little gold if any was recovered, and there were suggestions of fraud — an offense not entirely unknown in early gold mining camps. The mine’s owner, having concluded “that Kendall and his fellow swindlers knew nothing about their business”, shut down the cyanide circuit and developed the mine using conventional gravity methods, later coupled with the Plattner chlorination process\(^{20}\). Thus, the country’s first cyanide endeavour was a technical and economic failure.

The Sultana Mine, Ontario

The Sultana Mine and mill (Fig. 1), was located on Sultana Island in the Lake of the Woods of northwestern Ontario about 12 km from Rat Portage. The deposit consisted of wide quartz veins containing mostly free gold but also some gold-bearing sulphides. Mine development commenced in 1892, and the processing mill was completed by January 1893\(^{26}\). The ore was crushed and then milled in two batteries of five stamps each. Each stamp weighed ~ 385 kg, and the stamp mill readily reduced the ore to ~ 40 mesh. Ground ore was processed at a rate of 20 to 30 t/day in a series of Frue vanners to recover the coarse gold and then on amalgamation plates to recover the finer gold values. It was stated\(^{26}\) that a cyanide plant was operated in the summer of 1893, but that cyanide leaching proved to be of no value “as nearly all the gold in the ore is saved on the plates”. Details of the cyanide plant are not available. In any event, by 1897, the cyanide plant had been replaced by a chlorination facility to recover refractory gold associated with the sulphides in the mill tailings\(^{26}\). Thus, the cyanide leaching plant at the Sultana Mine probably was not technically successful because of the refractory nature of the gold present in the amalgamation plant tailings. It is interesting, though, that this early attempt to employ cyanide leaching technology occurred within six years of the MacArthur-Forrest discovery and at about the same time as the unsuccessful cyanide tests at the Brookfield Mine in Nova Scotia.

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The Regina Mine, Ontario

The Regina gold mine was located on the shores of Regina Bay on the Lake of the Woods about 50 km south of Hawk Lake station on the main CPR rail line. Mining operations began in the spring of 1895 and the processing mill was completed in September of that year (Fig. 2). The mill consisted of a jaw crusher and two batteries of five stamps each. Each stamp weighed 400 kg and the stamping circuit was capable of reducing 24 t ore/day to ~45 mesh. Gold initially was recovered by gravity concentration and amalgamation. A fire destroyed much of the mining operation in the winter of 1895, and a new mine-mill complex was constructed by February 1896.

The newly completed mill contained a MacArthur-Forrest cyanide circuit for the treatment of both the gravity concentrates and the gravity circuit tailings. The cyanide plant was housed in a separate building which was 12 m by 15 m in extent and was capable of treating up to 30 t of feed daily. As was common practice in the early days of cyanidation, vat leaching techniques were employed to extract the gold. Ten wooden vats 3 m in diameter and 2 m high were used for the leaching of 20 t/day of gravity circuit tailings. The drained tailings were loaded into a vat and were flooded first with 0.1% cyanide solution and subsequently with 0.2% cyanide solution; the total leaching time was 6 days. About 0.5 kg lime was added for each ton of tailings processed to maintain the required alkalinity. At the end of the leaching period, the tailings were drained and washed; gold was recovered from the pregnant solution with zinc turnings. The washed tailings were simply discharged into the nearby Lake of the Woods. Four ten-ton vats were also available for the treatment of the 0.3 t/day of gravity circuit concentrates. These richer grade concentrates were leached directly in 0.2% cyanide solution. Although leaching times as long as six weeks were employed for the concentrates, gold recoveries did not achieve the anticipated 90% levels.

It was noted that the cyanide leaching plant at the Regina Mine was put into operation on the first of January 1897.

The Deloro Mine, Ontario

While cyanide leaching technology was being pursued in the Lake of the Woods district of northwestern Ontario, interesting developments were also taking place in the Deloro mining camp, near Marmora in southern Ontario. Gold had been produced in the Deloro area for a number of years before 1890, but depletion of the high-grade free-milling ores had resulted in a significant decline in activity. In the mid-1890s, however, interest was refocused on the fine-grained gold commonly associated with arsenopyrite. Initially, roasting of the ore followed by chlorination and ferrous sulphate precipitation of the dissolved gold was used, and this practice yielded ~98% gold recovery. Several instances of arsenic poisoning from the roaster gases were noted, however, and efforts were made to find processes capable of recovering the gold without roasting. The Deloro Mine of Canadian Goldfields Limited was developed during the autumn of 1896 and a cyanide leaching facility was completed by March 1897 (Fig. 3). This plant did not employ the conventional MacArthur-Forrest process, but selected the bromo-cyanide or Sulman-Teed process instead. In this technique, a bromocyanogen salt was added in small amounts to the conventional alkaline cyanide solution. It was claimed that the bromocyanogen compound greatly accelerated the leaching process, thereby shortening the retention time in the plant. At Deloro, the ore was crushed in Blake jaw crushers and then was fed to one of three pulverizing machines. The ground ore was screened and subsequently passed to the leaching section. The bromo-cyanide process was also carried out in vats flooded with lixiviant. At Deloro, leaching was done using four wooden vats each of 50 tons capacity; pregnant solution and wash water also were stored in wooden vats prior to gold recovery on zinc turnings. With the use of the bromo-cyanide process, leach retention times were only 24 h to 36 h as opposed to the many days retention times commonly used for the MacArthur-Forrest cyanide leaching process in that period.

It was noted that the Deloro bromo-cyanide process commenced operation in March 1897; some three months after the startup of the leach plant at the Regina Mine in northwestern Ontario.

The Mikado Mine, Ontario

The Mikado Mine was located on the shore of Shoal Lake which drains into the Lake of the Woods; the mine was about 50 km from Rat Portage on the CPR rail line. Mining operations commenced in August 1896, and most of the mining infrastructure was in place by the end of that year. Ore grades exceeded 3 oz/t, and the ore was "unusual" in that up to one-third of the values were associated with

FIGURE 1. Sketch of the Sultana gold mill where cyanide leaching was tried unsuccessfully in 1893.

FIGURE 2. Photograph of the Regina gold mine and stamp mill in 1895; the lake steamboat used to supply the mine also is shown. Canada's first successful cyanide leaching plant was operated at this site in 1897.
sulphides. A twenty-stamp mill consisting of four five-stamp batteries was constructed to pulverize the ore prior to primary gold recovery on amalgamation plates. No gravity concentration units were installed as the idea was to treat all the product from the amalgamation plates by the bromo-cyanide process.

The bromo-cyanide process, however, never was operated at the Mikado Mine. Instead, a conventional MacArthur-Forrest cyanide leaching facility was installed to treat the fine gold and the gold associated with sulphides in the amalgamation tailings. At the Mikado Mine, the tailings were processed by the “double treatment” method whereby the tailings were allowed to settle, in the presence of cyanide solution, in one series of tanks to eliminate the slimes. The settled tailings were then transferred to a second series of tanks where most of the leaching occurred. The wooden leaching tanks were 4 m in diameter by 2.5 m high and were contained in a separate leaching building (Fig. 4). A five-day leaching cycle was employed for the 50 t/day of tailings processed. The operation was claimed to be highly efficient, and continued in use, with minor modifications, until at least 1902.

The cyanide leaching plant at the Mikado Mine was constructed between September 17 and October 24, 1898. Apparently, it was “put into operation about a month later”; i.e. about December 1898. Consequently, the start-up of the leach plant at the Mikado Mine virtually coincided with that at the Doratha Morton Mine in British Columbia. Of course, both facilities came on stream more than one year after the cyanide leaching plant at the Regina Mine.

The Doratha Morton Mine and Cyanide Plant, British Columbia

The Doratha Morton mine was developed in a quartz ledge situated at an elevation 2600 ft and about 11/4 miles from the coast of Fanny Bay. Fanny Bay itself is a branch of Philips Arm which is located on the mainland 120 miles northwest of Vancouver. Rich gold ores were locally encountered and these were mined with compressed air drills, supplied with air from a compressor located at the mill site some 11/4 miles distant. The ore was loaded into the buckets of a wire rope tramway and conveyed to the cyanide plant which was located at the mill site on the shore of Fanny Bay.

At the mill site, the ore was passed successively through a jaw crusher and a stamp mill. The ground ore was then passed to one of three settling tanks, 20 ft in diameter by 71/2 ft high, which were intended to eliminate slimes from the ore. From the settling tanks the ground ore was transported by car to one of six cyanide leaching vats, also 20 ft in diameter by 71/2 ft high. In the vats, the ore was flooded with dilute cyanide solution (2 to 3 lb/t), and was held in contact with the lixiviant for 6 to 8 days. At the end of the leaching cycle, the pregnant solution was drained into three solution holding tanks, 20 ft in diameter by 10 ft high. The washed residue was discharged into Fanny Bay. Gold was recovered from the pregnant solution by passage through zinc turnings, and the zinc consumption was ~ 0.5 lb/t of ore.

Figure 5 presents a general view of the Doratha Morton cyanide plant in 1898. At the top is located the stamp mill and the power plant with its stack. Immediately below is the bank of three settling tanks which received the ground ore from the stamp mills. Below the settling tanks are the cyanide leaching vats which are obviously of wood stave construction. At the water's edge are the three slightly higher solution storage tanks, and presumably the zinc boxes for gold recovery.

Although the Doratha Morton cyanide plant operated for less than a year (between December 1898 and November 1899), it successfully processed 9700 tons of ore and recovered 4400 oz of gold and 10200 oz of silver. Depletion of high grade ore was cited as the reason for the
closure of the mine and leach plant, but the advent of the Boer War might also have been a contributing factor as the management of the company was apparently British South African (pers. comm., C. R. Harris, 1985). There was general satisfaction with the operation of the cyanide vat leaching circuit, and this satisfaction was to be reflected in the steady growth of cyanide leaching practice in the early years of the 20th century.

**Nova Scotia Operations**

In the spring of 1901, J.R. Stuyvesant constructed a cyanide leaching plant in the Caribou mining district of Nova Scotia, and entrusted its operation to one H.S. "Cyanide" Badger. The plant had hardly begun operations when the decision was made to move the facility to Isaac's Harbour. Nova Scotia to treat stamp mill tailings from the adjacent Richardson Gold Mining Co. [12, 13]. In the autumn of 1901, expanded cyanide leaching facilities were completed at Isaac's Harbour and these were housed "in an enormous building 407 feet long and varying in width from 30 to 70 feet". Figure 6 shows a general view of the Richardson Gold Mining Co. in 1902. The central building with the smoke stacks is the stamp mill-gravity circuit which supplied feed to the cyanide plant. Although not identified on the original photo, the building on the left is clearly the "enormous building 407 feet long" that housed the cyanide vat leaching facility. The mill and the cyanide plant were described [13] as being connected by "a trestle 50 ft high and 100 ft long", and this is evident in the centre of the photograph. Figure 7 illustrates the cyanide leach plant and connecting trestle in a bit more detail although little can be seen of the actual leaching vats, etc. Unfortunately, photos of the actual cyanide leaching facilities do not seem to exist, and would have been difficult to obtain given the lighting conditions existing inside the wooden buildings.

At the Stuyvesant leaching installation, the roughly-deslimed tailings were vat-leached in circular wooden tanks 25 ft in diameter by 5 ft high, and having flat perforated bottoms connected to a piping system. Air tubes were also fitted into the tank bottoms to provide adequate oxygen for the cyanide leach. The deslimed tailings were loaded into the vats which were then flooded with alkaline 0.25% KCN solution. After the solution had been in contact with the tailings for ~ 12 h, the valve in the bottom of the vat was opened and the pregnant cyanide solution was discharged. A water wash cycle eliminated entrained solution, and the leached tailings were then discharged through an iron door in the side of the vat. The entire loading-leaching-discharge cycle required about 36 to 40 hours. The pregnant solution was passed through "boxes" of zinc turnings to recover gold; the spent solution was replenished with KCN and then recycled to one of the 10 leaching vats. Capacity of the plant was stated to be 150 ton/day; during the period July to September of 1901, some 4145 tons of tailings were processed to yield $5,754 worth of gold. Operating costs were about 60 cents/ton of tailings. The J.R. Stuyvesant Cyanide Plant closed by 1903 for economic reasons. A similar cyanide leaching plant with a 100 ton/day capacity was constructed in 1901 by S. Smith of the Waverly gold camp. After treating about 2000 tons of tailings, the plant was closed as insufficient gold was being recovered to pay the operating costs [14].

Following the closure of the J.R. Stuyvesant Cyanide Plant at Isaac's Harbour, "Cyanide" Badger developed a similar leaching circuit in 1903 at the Mic-Mac Mining Co. in the Leipsigate, Nova Scotia mining camp [15]. Figure 8 shows a building and connecting trestle at the Mic-Mac Mine in 1904. Again, the building is not identified on the original photograph, but this likely is the cyanide leaching building which received its stamp mill tailings in the foreground [16]. The operation at the Mic-Mac mine seemed to have been both technically and economically successful, and operated for several years until reserves were exhausted. Stamp mill circuit tailings were deslimed to increase permeability, and were then charged into one of four wooden leaching vats 16 ft in diameter by 5 ft high. Each vat held 50 tons of tailings. The tailings were flooded with 0.25% cyanide solution which was allowed to cover the tailings completely to a depth of ~ 3 inches. Following a 12-h retention period, the pregnant solution was drained for gold recovery; total batch time was about 30 h. A second, and virtually identical, cyanide vat leaching plant

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**FIGURE 5.** A general view of the Doratha Morton cyanide plant, Fanny Bay, Philips Arm, British Columbia in 1898 (Ref. 10).

**FIGURE 6.** General view of the Richardson Gold Mining Co., Isaac's Harbour, Nova Scotia in 1902. The long building on the left is likely the J.R. Stuyvesant Cyanide Plant. (Public Archives of Canada photo PA-40008.)
with a capacity of 75 ton/day was constructed in 1904 by “Cyanide” Badger at the Brookfield Mine, and this also appears to have been a technical and economic success.

Conclusions

Within six years of the patenting of the cyanide leaching process by MacArthur and the Forrest brothers, this technology was being tried in Canada. Initial efforts to use the process at the Sultana Mine in northwestern Ontario and at the Brookfield Mine in Nova Scotia were unsuccessful for technical and economic reasons. By contrast, cyanide leaching was successfully implemented at the Regina Mine, Lake of the Woods, Ontario, in January 1897. This operation may prove to have been the earliest successful Canadian application of the cyanide leaching process. Shortly after the start-up of the Regina Mine leaching plant, cyanide leaching using the bromo-cyanide process commenced (March 1897) on the arsenopyrite ores of the Deloro Mine in southern Ontario. These successful enterprises were followed by the cyanide leach plants at the Mikado Mine in northwestern Ontario and the Doratha Morton Mine in British Columbia, both of which commenced operation in December 1898. As was the case for all the early cyanide leaching operations, vat leaching methods were employed. It is interesting that this technology recently has been “rediscovered” for the leaching of low-grade gold ores.

The cyanide leaching process of MacArthur and Forrest offered a ready means of recovering low concentrations of fine gold from a variety of ores. The benefits of this technology were seen immediately by the metallurgical industry, and within a few years of the discovery in 1887, the process was being used in countries throughout the world. So rapidly was the process accepted that by 1897, cyanide leaching and cyanide leaching apparatus had become a part of the curriculum of many mining metallurgical schools such as the School of Practical Science at the University of Toronto. Significantly, this rapid dissemination of technology occurred in an era of relatively poor communications and without the “benefit” of structured technology transfer mechanisms. Perhaps good technology is its own best salesman.

Finally, it must again be emphasized that the “first” use of any technology is very difficult to discern, and this is especially true of cyanide leaching processes which were often attempted at small, isolated gold mines. Accordingly, it is not impossible that an earlier application of cyanide leaching in Canada may yet be revealed.

REFERENCES

The first copper smelter in Canada

Elsie Kossatz and P.J. Mackey

The small town of Bruce Mines lies a few miles east of Sault Ste. Marie, Ontario on the north shore of Lake Huron. This was the site of the first copper mine. This copper smelter used a reverberatory furnace, based on the Welsh process, and operated in the period of 1849 to 1850, when the plant was destroyed by fire. During the period of 1850 to about 1869, crushed and sorted ore was shipped to England and later to the United States. In 1869, a roasting plant, based on the Henderson Process, was constructed at Bruce Mines and was operated until about 1872. This short article describes the early development of this historic operation.

Although the rich copper ore found in the vicinity of Bruce Mines was known to the Indians, its discovery in or about 1847 was credited to Mr. Keating, who named the area Bruce in honour of Lord Elgin (James Bruce), Canada’s Governor General. In 1848, the property was acquired by the Montreal Mining Company for 40,000 pounds sterling and the company immediately sent a small party to investigate the deposit. The party sank a shaft on a lode within a hundred yards of the lake and also reported two adjacent lodes farther inland, which were left undeveloped until the turn of the century. Initially, the foundation of the headframe and hoist was built above the frost line and when the machinery was set in motion it collapsed. Eventually, the shaft was to reach a depth of 300 feet.

By 1849, the property was well established and the small settlement consisted of 77 miners, 65 labourers, 4 boys, 11 tradesmen, including blacksmiths and carpenters, 2 mining captains, 1 engineer, 2 clerks and 1 superintendent. The community, including the workers’ families, numbered about 250.

The president of the Montreal Mining Company, The Honourable James Ferrier, realized very early that a smelter would be the key to the success of the venture and, therefore, brought out from Wales a copper refiner and three furnacemen. The smelter was built and blown in, but unfortunately, in 1850, it was destroyed by fire. Although short-lived, this was the first copper smelting operation attempted in Canada. The furnace was not rebuilt, but the richer portions of the smelter slag were shipped to England for refining.

Although the smelter was no longer in operation, ore was processed on the property for the next few years by a crude crushing method using Cornish rolls, followed by break staff jigs and concave boulders. Evidence is available that the loss of metal values was extensive, between 40% and 60% per cent. Evidently, the crushing was not carried far enough and gave results so different from expectations that all officers of the company resigned or were dismissed. In 1852, with a new manager, A.B. Borron, a bonus system based on ore grade, tonnage and copper market price was introduced for the operators. In spite of management difficulties and technical setbacks, the company was able to declare a small dividend in 1853 and a larger one in 1854.

In 1854, a lease was acquired from the Montreal Mining Company by Mr. Sampson Vivian to mine the western portion of the property on the payment of a royalty of 5% for dressed ore. This lease led eventually to the expansion of the property and to a take over of the operation by the lessee. The new enterprise was named the West Canada Mining Company. After the formation of the new company, which combined all operations, the capital expenditure was so heavy that a dividend was not declared until 1861. On the Wellington and Huron Copper Bay sections of the deposit, an ore dressing floor was erected. A large proportion of the ore mined in the winter was stored underground in the stopes until it could be treated on the open floor during the summer months. In 1862, a bush fire destroyed the village, but the villagers had faith in their community and it was soon rebuilt. The company’s contribution of 300 pounds sterling for fire relief was directed toward a public school fund, ample evidence of a belief in the future.

Ore reserves were expanded, frequently by chance rather than by careful prospecting. For example, while searching for his strayed cattle, a teamster, George Clark, found a copper lode where fire had burned away the underbrush. Mr. Clark was awarded a keg of flour for his find. A shaft was sunk at the location, which was later called the Fire Lode. This lode and others were developed by the West Canada Mining Company in 1864. The Bruce Mines area produced, from its inception in 1848 until 1857, 3239 tons of upgraded ore with an assay of 18% copper, a total of 583 tons of copper. The value of the ore ranged from 136 to 140 pounds sterling per ton.

From 1861 to 1890, the price of cooper was good, and the mine employed 350 men, 200 of them working underground. The miners contracted with the company to drive a level for 120 dollars each. They provided the powder, caps, fuses and candles, and sharpened their own steel. The company did the mucking and the hoisting. A man’s average wage was 40 dollars a month, 4 dollars being equal to a pound sterling. In the seven years between 1862 and 1868, the company shipped 4880 tons of copper in ore form to England.

The heavy cost of dressing ore, the loss of copper during concentration and the high cost of shipping led Mr. John Taylor to recommend, in 1869, that a smelter again be erected on the property. As a result, Mr. Charles de Bussy was sent to carry out tests, based on the Henderson Process, for roasting, leaching and cement copper production. According to this process, the ore was first roasted to drive off most of the sulphur, then the calcine was crushed, mixed with salt and recalciated. Then the product was leached with hot, dilute sulphuric acid and the liquor was drained into vats containing scrap iron to produce cement copper, probably assaying about 70% Cu. The extraction appeared to be excellent, and a new smelter was erected at a cost of $200,000. However, the costs of salt, scrap iron and coal were so high that the plant was not profitable and operations were discontinued in 1873. An 1871 sketch of the plant with its tall smokestacks is shown in Figure 1.

During 1868, some efforts were made to extend the ore
reserves. A shaft was sunk on the north lode about one mile north of the Bruce section, but work was stopped because the men objected to walking so far to work. Labour relations have always been a problem in the industry, apparently. The shaft head gear was subsequently destroyed by fire. After 1872, no development was carried on. Presumably the costly experiment of the Henderson Process disheartened the owners and orders were given to take away everything of value. The end came in 1875 when the "big cave" occurred; a stope approximately 200 feet high and 12 feet wide caved in and the mine was closed from 1876 until 1898. In 1898, the property was obtained by an English group, who had the mine dewatered. They erected a costly concentrator, capable of handling 400 tons a day, but they failed to ensure adequate underground development to feed the plant. Once more, a costly fire occurred that completely destroyed the new construction, and the mine was again abandoned.

The pioneers who started the mine and the copper smelters at Bruce Mines were faced with extraordinary problems. Their contact with the outside world and needed supplies had to come mainly by water. Fire bricks for the furnace, built in 1849, cost 25 cent a piece, a very high price considering that men laboured for $10 per week. The railway had not yet arrived, and only a crude tote road connected the isolated communities of the North Shore with each other. The winters were harsh and operations in such a climate were so difficult that construction had to be done mostly in the summer months. Fire was an ever-present hazard because all buildings were of wooden construction and because the heating systems were primitive and dangerous.


Two smelting operations were tried in an effort to make a product which could be profitably marketed in spite of the high transportation costs inherent in such a pioneering operation. Both efforts failed. The successful operation of the smelters at Bruce Mines could have changed the financial picture for the investors and, thus, might have hastened the pace of industrial development on the North Shore.

REFERENCES
The historical development of copper smelting in British Columbia

S.A. Bradford

The twenty years before World War I was the Golden Age of copper smelting in British Columbia. In this short time, the smelters developed from small, primitive units to large, efficient operations. The first copper claims had been staked on Howe Sound and Knight Inlet in 1864, and there was extensive copper mining in the interior by the mid-1880s, but before 1895, all ore was shipped out of the country for smelting, converting and refining.

The first interest in smelter construction was undoubtedly inspired by a Provincial bonus and Dominion land grants offered to smelter builders. The early smelters made use of small blast furnaces which were designed to smelt any kind of ore, but usually too little ore was available. The first smelter seems to have been the McRae furnace on the Spillimacheen River (1883), followed by Vancouver (1889), Revelstoke (1891), and Golden (1891). They each smelted a few tons of galena and then quickly closed down permanently for lack of ore. Only smelters that actually produced some copper matte are described in the following sections.

The Nelson Smelter

The Hall Mines smelter at Nelson was British Columbia’s first successful copper smelter. Designed and supervised by Paul Johnson, its 160 tpd blast furnace was blown in January 1896 and a 275 tpd furnace was added in September. This was probably the largest copper blast furnace in the world at the time, since it measured 44 in. by 12 ft. at the tuyères. The furnaces’ most distinguishing features were their movable hearths, which were mounted on jack screws and carried on a truck running on rails under the furnace. A duplicate hearth was kept on the rails behind the furnace so that replacement could be made in minutes, or a hearth could be replaced by a crucible for lead smelting.

The refinery included two hand-operated calcining furnaces 16 ft. by 50 ft., and two reverberatory smelting furnaces 13 ft. by 17 ft., although in later years, the converting was discontinued and the matte was shipped to Grand Forks. No. 1 blast furnace was taken down in 1907 and the larger furnace then operated only occasionally. The equipment was dismantled and sold in 1908 and the buildings were destroyed by a fire in 1911.

The Trail Smelter

The history of the Trail smelter begins with “Fritz” Augustus Heinzé, a young mining engineer completely unhampered by scruples. He had built a custom smelter in Montana in competition with the “Copper Kings,” which brought him to the notice of the Rossland mine owners. They invited him to erect a smelter in British Columbia, with the LeRoi Mine contracting to supply 75,000 tons of ore. Heinzé brought his smelter superintendent with him and they chose Trail as the site for their new smelter. He also built a narrow-gauge railroad to Rossland and purchased the Miner, the only newspaper in the district. The paper immediately began a campaign against the Canadian Pacific Railway, arousing the public on the issue of the inadequacy of service. The result was that Heinzé was able to obtain a land grant from the Dominion and right-of-way to extend his little rail line to Victoria! At this point the Canadian Pacific woke up to the danger that faced them and bought him out, railroad, smelter, and all, for $1.2 million. Heinzé took his fortune back to Butte and began the infamous Montana Copper War.

The first furnace at Trail was fired in February 1896, and by the end of July, five furnaces were operating. The blast furnace, a 40 in. circular water-jacketed furnace with 6 tuyères, could handle 45 to 55 tons of ore per day. Four reverberatories, 14 ft. by 22 ft., had a capacity of 40 tpd each, but were found to be unsatisfactory and were later replaced by blast furnaces. In 1897 a new 200 ton blast furnace was installed and the smaller one was converted into a lead furnace. The new furnace was designed by Paul Johnson and was the same size as his big one at Nelson.

The Trail smelting practice at this time was to heap roast the ore, smelt it in the blast furnace to a low-grade matte which was crushed, roasted and smelted again. The resulting high-grade matte was shipped to the United States for converting.

After Canadian Pacific took over the Trail operation, the British Columbia Minister of Mines reported that “the old ‘Heinzé smelter’, although it can scarcely be called up-to-date, has been remodelled so far as it is possible to remodel an old smelting plant without tearing it completely down.” At that time it had been operating for less than four years.

In 1901, the plant had three copper blast furnaces, three lead blast furnaces, a softening furnace, two O’Hara roasters, and six Bruckner roasters. By 1908 there were five copper blast furnaces operating with a combined capacity of 1600 tons. This included the largest furnace in Canada, with a cross-section of 42 in. by 20 ft.

In 1913, the five copper furnaces were all enlarged to give a total rated capacity of over 3000 tpd. The largest furnace was 50 in. by 35 ft. with 28 tuyères on each side and an eight-foot ore column.

The Van Anda Smelter

Located on Van Anda Bay on the east coast of Texada Island, this smelter was directed by Thomas Kiddie for the Van Anda Copper and Gold Co. Before the smelter was built, ore from the Van Anda mine had been shipped to England for smelting. The first furnace was a 42 in. cupola blown in July 1899, smelting about 50 tpd. The chalcopyrite and bornite ore was first roasted in open heaps where the crushed ore was spread out in beds with cordwood piled on top. Figure 1 shows the Van Anda smelter as it looked in 1899.

In 1900, a 75 tpd furnace was erected. The smelter operated periodically for the next two years but finally

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closed permanently in 1903 when the two larger smelters on Vancouver Island were built.

**The Grand Forks Smelter**

The first smelter in the Boundary district was constructed by Granby Mining, Smelting and Power Co. about 3/4 mile above the town of Grand Forks on the north fork of the Kettle River. Two 200 tpd furnaces were built by A.B.W. Hodge, with the first blown in August 1900 and the second in October. They smelted a very uniform, low-grade (about 1.5% Cu) chalcopyrite ore which contained only enough sulphur to make a matte, eliminating the need for roasting or a second smelting. The Granby methods became famous for low costs and large tonnage (at one time the Grand Forks smelter was the largest non-ferrous smelter in the British Empire).

In 1902, the company installed furnaces No. 3 and 4 and also three converters which produced 99.95% blister copper; this was sent to a refinery in New York. Furnaces No. 5 and 6, similar to the others, were installed in 1904. The following year, two 300 tpd furnaces were added; these produced a higher-grade matte running 50% to 60% copper. All the furnaces were enlarged to this size in 1906. In 1909, all the furnaces were lengthened to 260 in., giving a smelter capacity of 4000 to 5000 tpd. Three new electrically operated Peirce-Smith converters were also installed.

A record year for Granby was in 1910, the year in which they first smelted over a million tons of ore. The interior of the blast furnace building in 1912 is shown in Figure 2. The smelter finally closed permanently in 1919 due to a prolonged coal strike and falling copper prices, but by then Granby had smelted over 13 million tons of ore at Grand Forks.

**The Greenwood Smelter**

The British Columbia Copper Co. (of New York) was responsible for the construction of a smelter at Anaconda, about ½ mile below Greenwood on Boundary Creek. Designed and constructed by Paul Johnson, the furnace was blown in in February 1901. Although designed to smelt 250 tpd it smelted an average of 360 tpd. This was a water-jacketed stack furnace 42 in. by 150 in. at the tuyères. The charge consisted of ore with about 2.5% copper plus coke to about 12% of the ore weight. Most Boundary ores were self-fluxing, containing calcite which breaks up the ore speedily when heated in the blast furnace, so the ore only had to be crushed to 4 in. to 6 in. for charging.

A second furnace was added about 1902 and a bessemerizing plant installed in 1904. In 1907 the smelter was completely reconstructed with three new furnaces having a capacity of 1700 tpd. In 1910 the two end furnaces were enlarged to 30 ft. with a 12 ft. ore column, giving the smelter a total capacity of 2200 tpd for an operation employing 125 men.

The converter building had two converter stands and five extra converter shells, since linings lasted only three to four charges. Capacity of the converter was 40 to 50 tpd of 35% copper matte, producing a million pounds of blister copper per month.

The downfall of the company and the smelter was the “Big Blast” at the Mother Lode Mine in 1912 which was claimed to be the largest in the history of mining. 50,000 lb. of blasting powder was packed into the mine and detonated all at one time. The intention was to break up the rebody to allow easier mining, but the actual result was to dilute the ore badly with waste rock. The smelter continued to operate through World War I but closed down in November 1918 because of low copper prices, shortage of fluxing ore, and the low ore grade.

**The Crofton Smelter**

This smelter was built by Northwestern Smelting and Refining Co. in 1901 at Crofton on Osborne Bay on the east coast of Vancouver Island. It was originally constructed to treat ores from the Lenora and other mines, with a narrow gauge railroad connecting the mines to the smelter. The equipment consisted of a 450-ton water-jacketed furnace, a 65-ton cupola, and a Bessemer converter. There was also a Garrettson furnace, built in Vancouver, which was designed to smelt and convert in practically one operation, but this proved unsatisfactory and had to be changed to permit ordinary smelting practice. The plant was completed in 1902 but lay idle for lack of ore. It was acquired in 1905 by the Britannia Copper Syndicate who used it to smelt ore from its mines on Prince of Wales Island. The converter plant had one stand and four shells with a capacity of 15 tpd of copper from 45% matte. The smelter closed about 1908 and was dismantled before 1913.
The Boundary Falls Smelter

The Standard Pyritic Smelting Co. erected a smelter three miles below Greenwood in 1901 but never used it. It was taken over by the Montreal and Boston Copper Co. and operated with occasional interruptions until 1905 when it was sold to the Dominion Copper Co. who operated it intermittently until 1908. The smelter had three water-jacketed blast furnaces with a nominal capacity of 300 tpd each. In 1907, one furnace was replaced with a 600-ton furnace with 25 tuyères on each side, originally equipped with a Girous top for heating the blast. Preheating raised the air temperature to about 100°C but this was soon found to be unsatisfactory and was discontinued. Dominion had financial difficulties and was reorganized as New Dominion Copper Co. in 1909. They rebuilt the power plant which had burned in 1908 but never got the smelter operating again. It was all dismantled before 1913.

The Ladysmith Smelter

A smelter, built by the Tyee Copper Co. at Oyster Harbour on the east coast of Vancouver Island about 58 miles north of Victoria, was blown in in December 1902. The 150-ton water-jacketed blast furnace was designed and directed by Thomas Kiddie. Later, a second furnace was added. The ores were mainly chalcopyrite with some bornite running about 4.4% copper, although small lots of gold ores were also smelted. A typical matte would run about 40% to 43% copper with 26 oz. of gold and 1.3 oz. silver, and this was shipped to the smelter at Tacoma for further treatment. The slag was granulated and sluiced down to the tidal flats.

At Ladysmith, Kiddie tested and patented a hot-blast system consisting of a thin-walled steel pipe that ran through the dust chamber and connected to the tuyères. He calculated that about one-third of the coke was saved with this arrangement. A special forehearth was invented by W.J. Watson who succeeded Kiddie as superintendent. His design allowed continuous flow of the matte and slag rather than tapping at intervals. A settler situated in front of the furnace was 4 ft. by 12 ft. with a 2 ft. square matte compartment to separate the products.

When the smelter closed in 1911, the No. 1 furnace was 42 in. by 120 in. (275 tpd) with a bosh, and No. 2 was 48 in. by 160 in. (400 tpd), not boshed. From then on, the owners made annual announcements that the smelter was about to reopen. Finally it did run again for about a year, from 1917 to 1918, but closed when the war ended.

The Kamloops Smelter

A small smelter was erected in 1904 by Kamloops Mines, Ltd., an English corporation, to smelt ores of the Iron Mask Mine. Coke was brought from the Crownest Pass and hauled from the CPR siding by horse and wagon 3½ miles up a 1600-foot elevation. The water-jacketed blast furnace was 36 in. in diameter with an eight-foot shaft and bell-hopper feed. The blast was preheated by passing it through a pipe that completely enclosed the downcomer for the waste gases from the furnace. The furnace was in blast for only a few months, smelting about 2600 tons of ore that ran around 5% copper, and was then dismantled.

The Anderson Oil Smelter

The Dominion Oil Smelting Co. of Vancouver set up an experimental smelter in the old Van Anda smelter building, based on patents of James J. Anderson. The furnace was essentially a reverberatory with a shaft-furnace stack superimposed at the end. It was fired by four injectors squirting vapurized oil so that the flame and combustion products passed up through the ore column and heated it to near the melting point. The matte and slag separated by gravity and were tapped off separately.

The impetus for the experiment was the fact that California fuel oil was selling at British Columbia ports, delivered in bulk, for 75 cents a barrel. Thomas Kiddie supervised a test in July 1911 using Boundary ore and copper slag left from Van Anda operations. About 94 tons of ore was smelted in the tests. Kiddie estimated fuel savings of 30 to 35 cents per ton of ore and labour savings of 9 cents, but copper prices started a decline and no further experiment seems to have been made.

The Anyox Smelter

The second smelter constructed by Granby Consolidated was built at Observatory Inlet, 110 miles north of Prince Rupert, just before World War I. The 2000 tpd smelter had three water-jacketed blast furnaces 50 in. by 360 in. with ore columns of 12 ft. The height from the tapping floor to the charging floor was 26 ft. It was blown in March 1914. The plant had three Great Falls type basic converters, 12 ft. in diameter and 17 ft. high. The Anyox smelter continued operation until 1935 when it was purchased by Cominco and closed. Blister copper was not produced again in British Columbia until 1978 when Afton Mines started up their top-blown rotary converter at their plant near Kamloops.

The Second Smelter at Golden

A lead smelter near Golden was in blast briefly in 1891, but a second smelter was built at Golden by the Labourer’s Co-operative Gold, Silver, and Copper Mining Co. in 1904. The furnace was described as a rectangular, water jacketed cupola 35 in. by 78 in., rated at 65 tpd. Electric power was generated from a waterfall on Hospital Creek about two miles above the works. The smelter operated for just one evening with chalcopyrite ore packed down from claims on Canyon and MacLean Creeks. The local opinion was that it was part of a stock promoting scheme, but Labourer’s Co-operative continued to develop mining claims in the area for several years. The property was eventually sold for taxes and the machinery scrapped in 1937.

The Changes

The twenty years before World War I saw tremendous improvement in copper smelting, going from little 50 tpd cupolas to the 600 to 800 tpd blast furnaces of 1914. Recovery of copper in the smelting operation was about 50% in the early furnaces; by 1914 it had risen to about 80%. In 1896, Heinze was charging about $11 per ton for smelting and freight. In 1900 the typical smelting cost was around $6, and in 1910 it was around $2.

Attitudes toward working conditions were changing, too. In 1913 the Ontario legislature passed a law limiting miners to eight hours per day underground, leading the Canadian Mining Journal to deplore that the progressive mining industry was being “experimented upon”.

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Thomas Kiddie, in an address in 1910, predicted that “the time may not be far distant when other metals, such as zinc, which heretofore have been looked upon as an impurity and a detrimen, will become sources of revenue.”12 And the time was certainly not far off; electrolytic refined copper and zinc were first produced at Trail in 1916.

In all, twelve copper smelters were constructed and actually made matte in British Columbia before World War I. While a few small ones failed quickly and disappeared, the major smelters developed into world leaders in matte and metal production.

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"Somewhere in Eastern Canada..."

a history of Canadian Copper Refiners

J.H. Schloen

One of the most important phases in Canada's relatively rapid development of its metal mining and refining industries was launched in 1928 when the directors of Noranda Mines Limited decided to establish an electrolytic copper refinery "somewhere in Eastern Canada."

Noranda's president, J.Y. Murdoch, had already evolved the company's "mine to market" policy and was determined to prove the economic feasibility of refining Noranda copper in Canada. The time appeared to be ripe as buoyant business conditions continued into 1929. In addition, proven ore reserves at the Horne Mine in northwest Quebec had virtually tripled and the price of copper had reached a healthy 15 cents per pound. Letters Patent incorporating the company bearing the date February 6, 1929 had been granted by the Secretary of State in accordance with an application that specified a capital structure of $2 million divided into 20,000 shares with a value of $100 each.

Noranda assumed 70% of the original investment in the proposed refinery, while the Nicholas Copper Company of New York (which later became Phelps Dodge Corporation) and British Metal Corporation Ltd. of London, England, each were responsible for 15%. Nicholas Copper Company agreed to supply the design and to supervise the construction of the refinery and British Metal Corporation would act as sales agent and market the refinery's products.

Montreal East was chosen as the site of the new refinery after an extensive survey of the entire eastern part of Canada. The location offered many advantages, including its proximity to the Montreal harbour.

Agreements had already been completed with Hudson Bay Mining & Smelting Co. Ltd. to refine their entire production of copper for a period of 10 years and with Noranda Mines Limited for its production for a period of not less than 15 years. Noranda also agreed to unconditionally guarantee bonds to be issued by Canadian Copper Refiners Limited as to principal, interest and sinking-fund payments.

The land for the refinery was purchased from the Town of Montreal East. The preliminary financing for the project was in place and agreements for copper refining had been signed. The time to begin construction was at hand and Canada was just entering the worst depression in its short history. Surveying the swampy site in the spring of 1930, a CCR director summed it up with: "It's a real gamble. We're taking a hell of a chance."

Construction began in April with a design similar to Nicholas Copper's El Paso refinery in Texas. It was to have a rated capacity of 75,000 short tons per year. Throughout the 1930s, the refinery throughput never reached capacity, due to the depression. Many experts gave it a life expectancy of 10 years at most.

The original refinery consisted of 24 commercial sections and two stripper sections, a power house with three motor generator sets, a casting house with a central furnace and boiler bay, a slime treatment plant and a liberator system to decopperize foul electrolyte. There was a reverberatory furnace for anodes with a daily capacity of 200 tons and a reverberatory furnace for wide bars with a daily capacity of 300 tons.

Originally, CCR did not have a silver refinery (parting plant), a selenium plant, tellurium plant, acid recovery, copper sulphate plant or coke and billet casting.

During the summer of 1930, construction proceeded and by late March of 1931 most units of the new refinery had been completed and the first refining cycle in the tank house was started. The first wire bars were poured on May 5th and Canadian Copper Refiners was "in business". The price of copper had slid to about six cents per pound, but if Noranda and its minority partners were worried, their concern was not apparent.

With Nicholas Copper Company operating the plant under contract, most of the senior supervisors were citizens of the United States enticed from the established copper refining industry of that country. This situation continued throughout the 1930s until Canadians gained sufficient knowledge and experience. A large number of workers employed in the construction of the refinery became the first employees of the new operation.

H.S. McKnight was employed by Nicholas Copper Company to be in charge of the design and construction of the refinery. After the construction was completed in 1931, Mr. McKnight was appointed manager of the new refinery and remained in this position until 1953, when the interest of Phelps Dodge Refining Corporation (formerly Nicholas Copper Company) was purchased by Noranda.

During the early 1930s, there was little market for the refined copper. More than 60% of the incoming material was unrefined copper from Noranda's Horne smelter. The Noranda anodes would be refined primarily to recover the contained gold, which could be sold to the Canadian Mint for $20.67 per ounce. The silver would also be recovered while the refined copper would be stockpiled. During one


FIGURE 1. Installation of electrorefining cells in the tank house, November 13, 1930.
period, the stockpile of copper exceeded 23,000 tons.

With the refinery in full operation, although not at its rated capacity, the directors were already planning improvements and expansion. A parting plant was constructed and an extension added in 1932. In 1933, new purification cells were added and the selenium plant was completed the following year. Two sections were added to the tank house in 1936, increasing the plant’s rated capacity to 81,000 tons per year. Nicholas Copper Company and the British Metal Corporation participated only in the original $2 million investment. Improvements and expansions were the sole responsibility of Noranda.

Before the war clouds began to darken in 1939, the management of CCR had decided to further expand the plant, and the construction of a tank house addition to house eight commercial sections and two stripper sections was in progress. The new units raised the capacity to 112,000 tons per year.

When Canada entered World War II in September 1939, Canadian Copper Refiners was ready as the second largest copper refinery in Canada to supply this country and its allies with many of the requirements necessary to the successful war effort. The policy of processing raw materials to finished product was successful.

As Canadian Copper Refiners’ first decade came to a close, the people at the Montreal East plant were directing all efforts to meet the challenge posed by World War II. A bright future lay ahead. New production and processes were under development. Noranda would purchase the interests held by its two minority partners in 1953 when John H. Schloen, who came to CCR after graduating from McGill University in 1931, was appointed manager.

Expansion continued until Canadian Copper Refiners became, in 1973, one of the largest and most technically advanced copper refineries in the world, with an annual capacity of 480,000 tons.

Today, CCR brand electrolytic copper is widely used by rod mills, by sheet and strip manufacturers and by alloy makers in many countries where primary copper of high quality is required for manufacturing.
Early Canadian lead smelters

J.E. Dutrizac and J.B. Sunstrum

Introduction

Lead was among the first metals known to man, and there are indications that lead was produced by the reductive smelting of galena in the mountains surrounding the Black Sea as early as 3000 B.C.11. The early metallurgists soon discovered that silver could be recovered from lead bullion by cupellation techniques, and that the lighthearted product of the cupellation process could be reduced to give silver-free lead metal. By the Graeco-Roman era, lead-silver ores were being extensively exploited for both their silver and lead contents. Although silver was always of prime concern, a number of industrial uses existed for lead that included piping and sheathing. Subsequently, lead metallurgical practice developed steadily in different parts of the world, and was a relatively mature technology by the time of the European colonization of North America.

In the United States, lead was produced on a small scale as early as 1720, and by 1800, the extensive Pb-Zn deposits of Missouri-Kansas and the upper Mississippi Valley were being developed.12 Many of the early American lead ores were low in silver, and seemed to have been worked primarily for their lead contents. Major uses of lead at the time included bullets, pipe and sheathing. During the period 1860-1890, the silver-rich galena ores of the western United States were exploited, and a number of lead smelters were operated. Although all the smelters produced lead metal, the recuperation of the associated silver and gold values was generally of prime concern.

In Canada, lead deposits are abundant, and galena-rich outcrops frequently attracted the attention of the early explorers and prospectors. In fact, the first record of a Canadian ore deposit seems to have been the observation of an argentiferous galena vein near Lake Temiscaming on 1741.13 The presence of silver-bearing lead deposits naturally attracted the attention of the mining community, and many outcrops were worked throughout the country. Because of the problems associated with transporting lead concentrates to existing American or overseas smelters, attention soon shifted to the establishment of local lead smelting facilities. The purpose of this article is to examine those early smelting efforts and to try to identify the first "successful" Canadian lead smelter.

Lake Temiscamingue mine was developed in 1890, and a concentrating mill was erected. The lack of suitable transportation facilities, however, soon led to the closure of this operation.6,7 By contrast, good rail transportation existed along the lower Ottawa Valley, and as a result several small mines operated successfully in the 1890s on Calumet Island where silver-bearing lead-zinc veins were abundant.8 In both Nova Scotia and Quebec plans for lead smelters were formulated, but it appears that these plans were never implemented. All the lead concentrates were exported.

A somewhat similar situation prevailed in Ontario. Galena veins were first exploited at the Kingdom mine at Galetta, in the Ottawa Valley, in 1884-1885.9 Concentrates produced at this operation were shipped by rail to the United States. Somewhat later, galena veins were exploited at Bannockburn,10 and small amounts of hand-cobbled ore were exported. It seems that a Bleiberg lead furnace was installed at Bannockburn in the mid 1890s, but the size of the furnace and the details of the installation are not known. It also appears that the furnace was never used.

Early British Columbia Lead Smelters

Lead outcrops were discovered in the Kootenay Mountains in the 1820s, and these outcrops were extensively examined after 1870 as prospectors seeking gold and silver moved steadily northward from the United States. The arrival of the Canadian Pacific Railway greatly facilitated the exploitation of these deposits, and by the early 1880s several small mining ventures were underway.11 At first, concentrates or high grade ores simply were transported by rail or river steamer to existing smelters in the northwestern United States. The high cost of this practice was obvious to the mine operators, and quite naturally there was considerable interest in establishing local smelting-refining facilities.

Although there is evidence that a lead blast furnace was constructed in the East Kootenays as early as 1883, it seems that this smelter never operated. In fact, the first operational lead smelter in British Columbia, and indeed in Canada, seems to have been built in the City of Vancouver in the latter half of 1888.12 The smelter was housed in a stone and masonry building 20 m by 13 m in size, and was adjacent to a wharf and support facilities. The blast furnace was a rectangular water-jacketed wrought iron unit about 1.5 m by 1.0 m at the tuyere line. Seven tuyeres were used for blowing, with compressed air provided by a blower driven by a steam engine. The brick-lined shaft of the furnace was 3 m high, and was open at the top to allow manual feeding. A self-raising metal hood fitted over the top opening and connected to a dust chamber and 20 m stack. The furnace was brought on line in February of 1889, and some lead bullion was poured. After only a few hours of operation, however, the metal hood connecting the top of the shaft to the brick-lined dust chamber overheated, and the furnace had to be shut down. The self-raising metal hood was replaced with a rigid brick flue, and the furnace was restarted. After about two hours of operation the slag in the furnace froze, and

Lead Mining Operations in Eastern Canada

In the early 1880s, many galena prospects were developed in Nova Scotia, Quebec and Ontario. In most instances, however, only small amounts of hand-picked ore were obtained for assay or for metallurgical testing in the United States or Europe. In Nova Scotia, systematic lead production began in 1897-1898 at the Cheticamp Gold Company's operation in the Northern Inverness district. The ore was milled and treated on Williff tables to yield a concentrate containing ~50% Pb together with 37 oz Ag/t and 0.4 oz Au/t.13 In Quebec, the

once again the unit was shut down. It was felt that the operating difficulties were caused by the high sulphur and CaO contents of the ore. Unfortunately, roasting facilities had not been provided at the Vancouver smelter, and the parent company in England seemed unwilling to invest further in this enterprise. Thus, the Vancouver smelter was shut down permanently, and was totally dismantled in 1899. There does not appear to be any photographic record of this operation, and the original site is now beneath the city of Vancouver. The failure of this pioneering enterprise likely can be attributed to a poor understanding of the direct smelting of lead sulphide ores and to the limited financial commitment of its backers.

Although initial smelting efforts at the Vancouver smelter were not technologically successful, interest in the establishment of lead smelting facilities in southern British Columbia continued. Thus in 1889, the Kootenay Smelting and Trading Syndicate began construction of a lead smelter on the banks of the Columbia River near Revelstoke[44], and a general view of this facility, taken from a 1903 newspaper article, is shown in Figure 1. The main smelting building was 55 m by 11 m in size and was adjacent to assay facilities and a steam boiler plant. Apparently the lesson of the Vancouver smelter had been learned, and provision was made to roast the ore prior to smelting. A single hand-rabbled reverberatory furnace was installed that was adequate to treat the total smelter feed. Although slow and expensive to operate, the hand-rabbled furnace yielded a product quite suitable for the lead blast furnace. This latter unit consisted of a rectangular water jacketed furnace about 2 m by 1 m at the tuyere line. Such furnaces were common in lead smelters at the time although, even then, efforts were being made to expand the hearth area to obtain greater productivity.

The Revelstoke smelter commenced operations on July 20, 1891 and successfully operated for several weeks using both charcoal and coke as fuel/reductant. Coke, which was imported from the United States, seems to have been the preferred feed, but locally-produced charcoal was much cheaper. Provision had been made for a variety of fluxes; as a result, fluid, free-flowing slags were produced. Lead bullion was tapped at regular intervals, and the whole operation ran smoothly and efficiently. Little is known of the eventual fate of the bullion, but it was likely sent to the United States for refining and silver-gold recovery.

It seems that the Revelstoke operation was the first successful lead smelter in Canada. Despite this, however, it operated for less than a year. Although numerous lead mines were being developed in British Columbia, the movement of lead concentrates in the Kootenay district was not easy in the early 1890s. As a result, the smelter soon exhausted the immediately available ore supply, and was shut down permanently in 1892.

Following the closure of the Revelstoke smelter, other lead smelters were constructed in the Kootenays. A lead smelter was erected in 1891 at Golden[45], but apparently never operated for want of adequate ore supplies. In 1895, the Kootenay Mining and Smelting Company completed its custom lead smelter at Pilot Bay on Kootenay Lake, and a general view of this operation is shown in Figure 2. The operations consisted of a concentrator, a roaster with four 20 m by 5 m reverberatory roasting furnaces, and finally the smelter building which contained a single 100 t/day water-jacketed blast furnace. Although a refinery was originally planned for the smelter, silver does not seem to have been recovered at the Pilot Bay plant[47]. In the first year of operation, over 3200 tons of lead bullion were produced, and lead concentrates contin-
used to be treated throughout 1896. The plant was sold in 1897, and smelting thereafter was discontinued although the concentrator apparently continued in use for a few more years.

The Hall Mines copper smelter at Nelson came on line in 1896, and at the time likely possessed the world's largest copper blast furnace with a daily capacity of 275 t. An overview of this substantial operation is shown in Figure 3. In the spring of 1898, an effort was made to process lead feeds in this blast furnace on an experimental basis
18. Ores containing about 40% Pb were calcined and then fed to the furnace together with appropriate fluxes. A quantity of Pb-Ag-Au bullion was collected, and the "experiment" was deemed to be a success. Thereafter, the treatment of lead feeds became an increasingly important part of the operation of the Hall Mines smelter. For example, in 1901 some 22,000 tons of lead concentrates were successfully processed, and by 1902 the facility had become essentially a custom lead smelter
19. Figure 4 shows the lower part of the lead blast furnace together with a number of lead ingots. The furnace measured 1.1 m by 3.6 m at the tuyere line, and is relatively "modern" in appearance. The Hall Mines copper-lead smelter continued operations until 1908 when the facility was permanently closed.

Lastly, mention must be made of the lead smelter built at Trail that was to become the basis of Cominco's current lead processing operations. In the early 1890s, copper-gold ores were discovered near Rossland, and initially these were shipped to the United States for processing. The shipping was done by river steamer along the Columbia River, and the "port" was Trail Creek Landing, about 10 km from Rossland. To overcome the obvious inefficiencies of this lengthy transportation system, copper smelting commenced at Trail in 1896 and persisted until 1929. Smelting of silver-rich lead concentrates commenced in 1899
20. Lead concentrates or rich ores were desulfurized in Bruckner roasters and then were smelted in a 1 m circular water-jacketed blast furnace which originally had been used for copper production. The furnace had six tuyeres and could treat ~50 t/day of roasted charge
21. Initially, the lead bullion was exported to the United States for further processing, but this practice ended in 1902 with the opening of the world's first Betts process electrolytic lead refinery
22. Unlike the earlier lead smelters which had operated in British Columbia, the Trail smelter had a substantial and continuous supply of lead ores and concentrates. This assured ore supply seems to have been one of the deciding factors in the success of this operation which was to evolve into the large Cominco lead smelter which operates to this day.

In retrospect, the supply of ore is seen as being especially important. All too often in the 19th century, smelters or processing plants were optimistically erected before adequate ore reserves were established. This seems to have been the case with many of the early British Columbia lead smelters which were often technically successful, but were soon forced to close for want of adequate smelter feed.

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The founding of the Canadian zinc industry

J.E. Dutrizac

Introduction

Canada is currently the world’s largest supplier of zinc concentrates and is also a major producer of zinc metal. This dominant position in the world’s zinc market is a fairly recent circumstance and, in fact, the country’s zinc industry was founded only in the early part of the twentieth century. By contrast, the American zinc industry was well established by the mid-nineteenth century, largely based on technology originally imported from the still older European practice. Initial attempts to create a viable Canadian zinc smelter were based on the traditional practice and were unsuccessful; success came only with new electrolytic technology, much of which was developed domestically.

The purpose of this paper is to outline the early attempts to produce metallic zinc in Canada and to trace the gradual development of these first efforts into a viable zinc industry based on modern science and technology.

The Canadian Metal Company, Frank, Alberta

The small coal mining town of Frank, Alberta, is usually remembered in Canadian history for the Frank Slide. There in the early hours of April 29, 1903, part of Turtle Mountain crashed into the valley destroying most of the mining town and claiming over sixty-five lives. In metallurgical history, however, Frank should be equally remembered as the site of Canada’s first commercial zinc plant where in 1905-1906 a serious attempt was made by the Canadian Metal Company to smelt the zinc ores of the Slocan, British Columbia, mining camp, using the Belgian retort process.

A modern smelter was erected on the site (Fig. 1) to take advantage of the plentiful coal supplies in the area. In the Belgian horizontal retort process, dead-roasted ore is mixed with coke or coal and is then charged into a fire-clay retort. The retorts are externally heated and, at high temperatures, zinc metal is evolved in vapour form and is subsequently condensed as a liquid in clay condensers fitted onto the retorts. The works at Frank consisted of a roasting section, a distillation department, a pottery to make both retorts and condensers, as well as the usual laboratory and office buildings. The roasting was accomplished in five Merton hearth furnaces fired by coal. Ore was delivered to and was removed from the roasters by screw conveyors. The cooled calcine was mixed with coke breeze and was then charged into fire-clay retorts heated in one of five furnaces of fairly conventional design. The furnaces were fired using the abundant local coal. Each furnace held 240 retorts, each about \(7\frac{1}{2} \times 11 \times 50\) inches long; the open ends of the retorts were fitted with air cooled clay condensers to collect liquid metallic zinc. Each retort held about 300 lb of charge which would consist of calcine together with coke breeze. The recovery was about 80% of the zinc in the charge, and this value dropped sharply as the zinc concentrate’s grade fell. For this reason, the smelting of feeds assaying less than about 40% zinc was considered uneconomical. Retorts were fragile and had to be carefully made from clay imported from the St. Louis (U.S.A.) area; in addition to being brittle, the retorts were subject to slag attack by lime or lead compounds in the charge.

The Frank smelter operated for only a short period before being shut down permanently. Although a number of design problems were encountered, it appears that the operations ceased for three principal reasons. The main factor was the high cost and the general lack of skilled metallurgical labour. Horizontal retort operation was an art, and the company failed to attract enough qualified European or American furnacemen to run the retorts. The second factor was the supply of zinc concentrates. Although zinc was plentiful in the area, concentrate grades were frequently low with high lead values; furthermore, individual mines were small so that a continuous feed supply was always a problem for all the early base metal smelters (copper, lead or zinc) which flourished in the area. The last factor cited for the demise of the Frank works was the poor quality of local coals, but it is more likely that high coal consumption was an equal factor.

As was the case for many Canadian “first” smelting enterprises, the Frank zinc operation was economically unsuccessful. It was, however, the first commercial zinc plant to operate in Canada and it deserves recognition for this.

The Canadian Zinc Company, Nelson, British Columbia

With the failure of the Frank zinc smelter, the numerous mines in southeastern British Columbia were again faced with severe marketing problems for their zinc ores and concentrates. Shipping into the United States was generally unattractive because of steep tariffs and high rail costs. There was a continuing need for a zinc smelter in the Kootenay area to service the numerous small mining operations thriving there. Within this general framework, F.I. Snyder of Chicago approached the British Columbia government for support in building an experimental zinc plant based on electric furnace smelting. Accordingly, in 1908 the Canadian Zinc Company erected a 2000-lb electric smelting unit at Nelson, British Columbia, to process local concentrates. The zinc concentrates being produced in that era, before flotation was well developed, were often rich in lead and silver. Electric smelting offered the potential of making zinc and a lead-silver bullion simultaneously. The concept was similar to the Imperial Smelting Process, except that the process heat was supplied electrically. The concentrates were roasted in a 14-ft diameter McDougall roasting furnace and were then cooled and mixed with coke; only enough carbon had to be added to serve as reductant, because the process heat was provided by direct electrical heating within the charge. Lead bullion was tapped from the furnace and zinc was collected in a single large condenser. Severe technical problems were encountered with the condensation of the

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reactivated. Some success was noted in condensing zinc, but consistent condenser operation was never achieved. Power costs remained high, partly because of the need to recycle large quantities of blue powder. The early metallurgists were simply unable to quench the zinc vapours rapidly enough to prevent the back reaction with the CO reaction product to give blue powder (ZnO).

Although some progress had been made, government enquiries indicated that extensive work would be needed to make the 2000-lb unit consistently operational; larger-scale operation was considered most unlikely. A close scrutiny of existing Scandinavian electric smelting operations indicated that these were unprofitable in spite of their lower power costs and very high grade feeds. Accordingly, the electric furnace operations at Nelson were terminated in 1913 and were never resumed even during the war years when the demand for zinc was great\textsuperscript{4,5}.

The Consolidated Mining and Smelting Company, Trail, British Columbia

In the early 1900s the horizontal retort process represented the accepted technology for producing zinc; this conventional technology had not, however, proved successful in the Canadian setting. About 1880, the Frenchman Létrange had demonstrated that zinc could be produced by electrolysis of aqueous zinc sulphate-sulphuric acid solutions, and the process offered exciting possibilities for treating lead-bearing zinc concentrates to give high purity metal. The process had been tried at Cockle Creek, Australia, in the latter part of the nineteenth century, and it had been a dismal failure. In spite of the Cockle Creek experiment, however, the potential advantages of the electrolytic method continued to interest extractive metallurgists.

In the early part of this century Consolidated Mining and Smelting Corporation, which already operated a lead smelter and refinery at Trail, British Columbia, began to examine the electrolytic zinc process. The company had access to low-cost hydroelectric energy and this was certainly a factor in their selection of the electrolytic route. Test work on the production of electrolytic zinc began at Trail in 1912; initial efforts were very encouraging and led to the erection of the Trail Experimental Zinc Plant in 1913. This unit had a capacity of about 1000 lb Zn/day and operated until 1915 when, under wartime pressures, a 35-ton/day commercial unit was constructed. Figure 2 shows the leaching circuit (under construction) for the commercial unit, and Figure 3 indicates the construction of the tank room as of December 1915.

The lead-rich zinc concentrates were dead-roasted in a 25-ft diameter Wedge roaster at 700°C, and the calcine was subjected to neutral leaching in wooden Pachuca tanks. The sands from the neutral leach classifier and the thickener underflows were subjected to an "acid leach" using most of the return electrolyte. The neutral leach-acid leach circuit then employed was more akin to a two-stage neutral leach than to the hot acid leach processes currently employed in the industry. The leach residues contained most of the lead and silver values in the feed and these were recycled to the adjacent lead smelter. The zinc leach solution was purified with zinc dust and was then subjected to electrolysis. The original tank room (Fig. 3) contained 448 cells, each of which was 27 in. wide by 36 in. deep by 83 in. long; two cells constituted a tank as is evident in the photograph. Lead anodes and aluminum cathodes were used and this general practice continues to the present.
Glue was added to obtain smooth deposits, and the cathode deposits were pulled and stripped manually. Metallic zinc was melted in a reverberatory furnace and was cast into 50-lb bars assaying over 99.9% Zn, a very high purity at that time.

The commercial zinc plant was an economical and technical success. Economic success was assured by the high zinc prices brought about as a result of the war in Europe. Technical success was achieved by a combination of factors including the inherent advantages of the electrolytic process, the proximity of a lead smelter to process the lead residues and, especially, the presence at the smelting site of skilled and devoted professional and technical labour. The initial 35 ton/day zinc plant was soon expanded to about 75 ton Zn/day, and this expansion both proved the viability of the electrolytic process in the Canadian economic climate and firmly established Consolidated Mining and Smelting Company in the zinc business.

Other Electrolytic Zinc Operations

The wartime demand for zinc drove the price even higher, and this naturally encouraged other efforts to make the metal. These new operations tended to be electrolytic because the Frank experiment had shown the importance of highly skilled labour for the horizontal retort process, and such labour was simply not available in 1914-1918. One of the new plants was that of the Weedon Mining Company at Welland, Ontario. This operation was established to process zinc ores from Montauban Township, county of Portneuf, Quebec, and was eastern Canada's first venture into zinc extractive metallurgy. This plant was located at Welland, Ontario, to take advantage of inexpensive hydroelectric power and of the excellent transportation facilities available. It utilized an electrolytic technique based on the Watts Process and was able to produce 6000-lb Zn/day from 50 cells. The Watts Process was based on electrolysis from a neutral solution of zinc sulphate. The acid generated was neutralized by containers of calcine suspended in the cells themselves. In essence, the leaching and electrolysis were carried out in a single vessel. The Watts Process was difficult to operate because of problems with the charging and discharging of the calcine containers. Solution purification was an obvious problem. The Welland plant ran for about a year and produced only a negligible quantity of metal. Its closure marked the suspension of zinc production in eastern Canada for nearly half a century.

The last of the early zinc ventures worthy of note was that of the Standard Silver Lead Mining Company at Silverton, British Columbia. This operation was really an experimental unit designed to test the French (Mr.) electrolytic process. In this process the roasted concentrates were leached with sodium bisulphate solution to form a sodium sulphate-zinc sulphate pregnant solution. This was purified using cementation techniques and zinc was then electrowon between a lead anode and a zinc cathode. During electrolysis the sodium sulphate was converted to bisulphate for recycle to the leaching circuit. The operation was similar to an acid leach except for the presence of sodium ion which reportedly increased the conductivity of the electrolyte and may also have helped eliminate iron as sodium jarosite in the leaching circuit. An important feature of the French process was the addition of manganese salts to the solution. These eventually deposited as MnO₂ on the lead anodes, thereby lowering the oxygen overvoltage. The importance of the MnO₂ layer on the anodes is now generally recognized by the industry. The plant at Silverton ran briefly but no really significant output was recorded. Efforts were later made to construct a plant based on the French process at Nelson, but these efforts never bore fruit. It was unlikely that the Kootenays could have supported two zinc plants and, by this time, the Trail zinc smelter was firmly established.

Thus, although various zinc processes were examined during the early development years, the sulphuric acid-based electrolysis route, combined with the emerging flotation science, proved to be the successful technology. The electrolytic route formed the basis for future expansion of the industry, first at Flin Flon, Manitoba, and later at Valleyfield, Quebec, and Timmins, Ontario, in eastern Canada.

Acknowledgments

The assistance of R.J.C. MacDonald with the assembly of the early information is recognized. Ms. Sandra Rogers of Cominco Limited provided useful references on that Company's early history.

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The Hall-Héroult Process in Canada
F. McGravie, W.W. Robertson and S.T. Solinski

Historical
During this, the Centennial year of the discovery of the Hall-Héroult process, it seems appropriate to review some of the developments which have taken place in Canada and to explain why it has become an important world centre for the production of primary aluminum.

For many years after it was recognized that the element aluminum existed, scientists were frustrated in their efforts to isolate it. In the early 1800s H.C. Oersted and Frederick Woehler prepared small quantities sufficient to measure some of the physical properties. A quasi-commercial process involving reduction with metallic sodium was developed by the French scientist H. St. Claire Deville in 1855 but the cost was very high ($30.00 per kg). It was not until 1886 that Charles Martin Hall in the United States and P.L.T. Héroult in France, more or less simultaneously but completely independently, discovered the process which is essentially that in use today.

Some of the early pioneers in the industry quickly recognized the potential of the hydro-electric resources in Canada as economic energy suppliers for aluminum production. The first commercial production in Canada was in Shawinigan, Quebec in 1901, only 15 years after the process was discovered. Subsequently operating plants were established at nine different locations in Canada (Quebec and British Columbia) (Table 1). Operations continue today in all locations except one (La Tuque). The table shows present plant capacity which for the older plants was reached in an evolutionary manner including replacement of obsolete capacity or upgrading and modernization of existing designs.


Table 1. Chronology of Canadian smelter projects

<table>
<thead>
<tr>
<th>Year</th>
<th>Location</th>
<th>1986 Capacity (tonnes per year)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1901</td>
<td>Shawinigan, Quebec</td>
<td>84 000</td>
</tr>
<tr>
<td>1925</td>
<td>Arvida, Quebec</td>
<td>432 000</td>
</tr>
<tr>
<td>1942</td>
<td>La Tuque, Quebec</td>
<td>0 NIL</td>
</tr>
<tr>
<td>1943</td>
<td>Beauharnois, Quebec</td>
<td>47 000</td>
</tr>
<tr>
<td>1943</td>
<td>Isle Maligne, Quebec</td>
<td>73 000</td>
</tr>
<tr>
<td>1954</td>
<td>Kitimat, British Columbia</td>
<td>268 000</td>
</tr>
<tr>
<td>1955</td>
<td>Baie Comeau, Quebec (1)</td>
<td>271 000</td>
</tr>
<tr>
<td>1976</td>
<td>Grande Baie, Quebec</td>
<td>171 000</td>
</tr>
<tr>
<td>1986</td>
<td>Becancour, Quebec (2)</td>
<td>230 000</td>
</tr>
<tr>
<td></td>
<td><strong>Total Canadian Capacity</strong></td>
<td><strong>1576 000</strong></td>
</tr>
</tbody>
</table>

(1) Reynolds Metal Company
(2) Aluminiere de Becancour Inc.
(All others Alcan Smelters and Chemicals Ltd.)

Table 2. Raw materials consumption per tonne aluminum

<table>
<thead>
<tr>
<th>Material</th>
<th>Consumption (t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Alumina</td>
<td>1.92</td>
</tr>
<tr>
<td>Carbona Electrode</td>
<td>0.52</td>
</tr>
<tr>
<td>D.C. Power</td>
<td>13 000 kWh</td>
</tr>
<tr>
<td>Anode Baking Energy</td>
<td>2 GJ</td>
</tr>
</tbody>
</table>

Hall-Héroult Process Fundamentals
The process involves the fused salt electrolysis of the oxide in a reduction cell illustrated schematically in Figure 1. The essential elements include a carbon anode, a molten cryolite electrolyte in which alumina is soluble, a pool of liquid aluminum and a carbon-lined container which holds the electrolyte and the metal. Appropriate electrical connections are made to the anode assembly and to the carbon lining. As electrolysis takes place, the oxide is decomposed into its elements. Oxygen is deposited at the carbon anode and reacts to form a mixture of carbon monoxide and carbon dioxide. Molten aluminum with a density higher than the electrolyte collects in the bottom of the container. The inter-polar distance is only about five centimetres. The D.C. power not only causes the electrolysis but also provides the heat required to maintain the electrolyte at a temperature of 960°C to 970°C. Alumina must be added periodically to replenish that which has decomposed and likewise the consumable carbon anode must be renewed. Table 2 shows the quantities of raw material consumed per tonne of metal.

In view of the high energy requirement of the process, areas with low-cost power are logical sites for commercial plants. To give a better appreciation of the amount of energy required it is useful to express it in other terms. A modern thermal plant would consume seven tonnes of coal or 23 barrels of fuel oil to generate the electricity required to produce one tonne of aluminum. In one captive thermal station burning low grade coal, the weight of fly ash associated with the coal burned to generate process power exceeds the weight of metal produced in the adjacent smelter.

While the basic process is essentially the same as that discovered by Hall and Héroult, there have been many improvements in process technology and plant design.

Evolution of Hall-Héroult Technology
Cell Amperage Rating
Figure 2 shows how the amperage rating of the reduction cells has evolved with time. Factors which determined the rate of this increase include improvements in cell design and in the technology for efficient generation of high amperage D.C. power. For example the development of

![FIGURE 1. Schematic diagram—Hall-Héroult cell.](image-url)
solid state silicon diode rectification was an important milestone. In addition to better rectification efficiency, higher voltage ratings were possible so that more cells could be included in a single circuit. At normal electrolysis efficiencies, one kiloampere produces about seven kilograms per day. Thus the daily production per cell has increased from about 35 kilograms to in excess of 2 tonnes. The annual production from a series of cells (referred to in the industry as a “potline”) has increased from a few thousand tonnes to in excess of 100,000 tonnes.

Another important factor which permitted significant increase in amperage rating was an improved understanding of the electromagnetic and hydrodynamic forces which exist in a reduction cell. The vertical flow of current through the electrolyte and metal layers is at right angles to a horizontal magnetic field surrounding the bus bars which conduct the current between cells. These forces cause instability in the metal pad. The forces which are proportional to the amperage squared, become very significant at levels of 100 kA and above. Calculations applying fundamental physical principles, greatly facilitated by modern computers for what is a rather complex three-dimensional system, have provided designs in which these forces are minimized.

Anode Technology

The early cells used circular carbon anodes 10 cm to 15 cm in diameter. Subsequently larger anodes with an area of about 0.5 m² and 20 cm high became the standard. The manufacturing process involves compressing a heated mixture of coal tar pitch and sized petroleum coke. The green carbon is then baked at about 1100°C. Before installation in the cell the baked carbon body is fitted with a metallic conductor for attachment to the cell busbars.

This is a relatively expensive process and resulted in development work to adapt the Soderberg self-baking anode, widely used in the electrothermal industry, to the Hall-Héroult process. The first commercial design was the “Horizontal Stud” (Fig. 3a). An unbaked mixture of petroleum coke and coal tar pitch is added to a rectangular compartment. Electrical contact is made through steel studs planted laterally while the paste is still fluid. Process heat bakes the anode mass resulting in a relatively good electrical connection between the carbon and the steel studs. As the anode is consumed, the studs in the lower row are removed and the electrical connection made to a higher row. Later a “Vertical Stud” design (Fig. 3b) with current entry from the top of the anode was found to be more satisfactory for higher amperage designs.

The Soderberg anode achieved the purpose of decreasing electrode costs but was not without problems. The coal tar fume emitted during the baking in situ was difficult to contain and gave serious problems with working conditions. Likewise the baking temperature of the anode was restricted to the cell operating temperature resulting in a more reactive anode. Anode servicing work was in general physically demanding although this was later mechanized.

In recent times with increased concern for both external and internal environmental conditions, the Soderberg process has fallen from favour. Most recent plants use the prebaked anode design (Fig. 3c). Modern techniques permit the manufacture of larger anodes which require less labour in anode servicing and the operation has been mechanized. The higher capital costs of anode manufacturing facilities are at least partially offset by lower costs for potrooms and environmental equipment.

The economic life of existing Soderberg smelters has been extended by retro-fitting improved scrubbing equipment, mechanizing difficult operations and the compulsory use of personal respiratory protection equipment.

Electrolyte Chemistry

The unique contribution of Hall and Héroult was to find an electrolyte which would satisfactorily dissolve alumina and remain stable during electrolysis. Considerable research has been carried out to find an alternate but without success. However, the application of better knowledge of electrolyte chemistry has resulted in improved performance. The compound cryolite, a 3:1 molar ratio of sodium
fluoride to aluminum fluoride, is a solvent for alumina forming a simple eutectic system. Typically 5% calcium fluoride is added which lowers the melting point from 1006°C to 960°C to 970°C. Some operators add 2% to 3% lithium fluoride which also lowers the melting point enhancing efficiencies and decreasing fluoride losses.

It has been found that the Faraday efficiency (ratio of theoretical to actual ampere hours per unit of production) is significantly improved if the mole fraction of aluminum fluoride is increased beyond that in pure cryolite to an excess of 5% to 10%. A consequence of this change is a significant increase in the loss of fluoride compounds to the cell gases, increasing costs and causing environmental problems; these were subsequently alleviated by the development of the dry scrubbing process (described below).

Environmental Control Technology

The early prebaked cells were not enclosed and fluoride lost from the cells was emitted into the working area and the external atmosphere. With the Soderberg design it was found necessary to enclose the cells and evacuate the enclosures for disposal to atmosphere. It was quickly recognized that this effluent must be treated. This posed a number of difficult technical problems. The concentration of contaminants was low as relatively large quantities of excess air were drawn through the enclosure. The principal contaminants were particulate and gaseous fluoride compounds, finely divided alumina and coal tar pitch fume. A number of processes were developed using various designs of spray towers, wet electrostatics, etc. In general good efficiencies were obtained on gaseous fluoride permitting the recovery and recycle of certain compounds. However, efficiency on particulate matter was not good and the units were difficult to maintain.

A significant technical breakthrough was achieved with the development of a process called “dry scrubbing” (Fig. 4). Treatment of cell gas is achieved by injecting into the dirty gas stream the finely divided alumina before it is sent to the cell for electrolysis. The gas stream is then passed through bag collectors which remove particulate matter with very high efficiency. The gaseous fluoride reacts with the alumina and is thus returned almost immediately to the cell, reducing the amount of make-up required to maintain the desired composition. Recovery efficiencies in excess of 99.5% are obtained for both solid and particulate matter. The process comes very close to the environmentalists goal of “total containment”. An important side benefit is that it permits the use of electrolytes containing higher percentages of aluminum fluoride with resultant improvements in Faraday efficiency.

Alumina Feeding

The replacement of decomposed alumina can be done batchwise. Periodically (usually 4 to 6 times per day) a quantity of alumina is placed on the frozen crust surrounding the electrode. The alumina is stirred into the electrolyte by breaking the crust with a self-propelled pneumatic breaking machine. This method gives rise to variations in alumina concentration during the cycle. When alumina concentration falls below a certain critical level an “anode effect” occurs. Normal electrolysis ceases and instead the electrolyte is decomposed. The resistance of a cell increases by a factor of about ten. Termination of the anode effect is accomplished by increasing alumina concentration and removing an adhering gas film from the anode surface.

The latest technique is to feed alumina frequently in shots of about two kilograms by a small pneumatic breaker installed on each cell and controlled by the process control computer. The result is more precise control of electrolyte composition, better control of anode effect frequency, less operating labour and improved efficiencies. Figure 5 compares the alumina concentration time relationship for the two systems.

Process Control

In the early days the operation of aluminum cells was considered to be more of an art than a science. The key process decision maker was usually the general foreman. His job was not only to supervise the work force but also to make day-to-day decisions on target levels for major independent variables. His preparation for this latter activity was years of practical experience as a cell operator. Many of these men performed very well at a time when knowledge of the process was insufficient to provide them with logical decision-making rules.

A major change occurred with the introduction of the first process control computers in the mid-1960s. By this time the body of knowledge on the process had increased substantially. It was appropriate to use the introduction of the computer, with the mysticism surrounding it, to change the process decision-making approach. The computer performs certain on-line functions such as control of cell
resistance and termination of anode effects much more efficiently than could be done manually. In addition it provides a very useful data logging facility. Consequently the process decision-making role passed from the general foremen to process control technicians with good process knowledge and access to pertinent historical information provided by the computer. In contrast to the situation in a flow process where many variables are monitored by the computer, essentially only one variable, cell resistance, is sensed as input. It is quite amazing how much useful decision-making information can be obtained using this one input plus of course the computer's capability as an accurate time piece.

Lining Life
During normal operation, the carbon cell lining absorbs up to its original weight of electrolyte material. A gradual deterioration occurs and periodically it must be shut down and replaced. Improved design, better materials of construction and better installation practices have resulted in an increase in lining life from under 1000 days to a range of 2000 to 3000 days.

Mechanization
In order to cope with the environmental problems inherent in the Soderberg design, equipment has been developed to reduce physical effort and improve the working atmosphere. This required considerable ingenuity as the machines had to operate in restricted space and perform operations requiring “robotic” technology. While productivity within the operating area is increased, the savings are offset somewhat by the need for additional maintenance personnel.

Labour Productivity
The unit commonly used in the industry is “tonnes per man year” where the denominator includes all personnel within the plant fence (operating, service, clerical and supervisory personnel including senior management). In older plants a typical number would be 100 to 150 tonnes per man year, while modern plants of optimum size achieve 300 tonnes per man year.

Process Economics
The improvements which have been realized in both technology and equipment have had a positive effect on process economics. The impact of higher unit labour and major raw material costs has been at least partially offset. It has been possible to cope with the need to meet more rigid standards for external and internal environment without major adverse economic effects. One cannot however ignore the importance of low-cost energy. Only in special situations can a plant based on fossil fuel-generated electricity be viable.

What of the Future?
It is clear that the Hall-Héroult process has reached a stage of considerable maturity. In the near term, one can only anticipate evolutionary improvements in the technology. By many criteria the process compares unfavourably with more elegant flow processes requiring minimum labour with sophisticated automated process control systems. Despite major research projects in many parts of the world no such process for aluminum has been found nor is any apparent on the near horizon. Therefore, the process discovered by the American and French scientists a century ago will continue to be the commercial method of aluminum production for some time to come. However, a modern plant with good internal and external environment, highly mechanized using modern process control principles is a far cry from the “smoke stack” image of earlier smelters.

In view of the energy situation, Canada will be one of the few areas in the world where economic factors will favour continued operation of existing plants and possibly modest expansion.
The early history of electrorefining in Canada

D.J. MacKinnon

Introduction
The Canadian Copper Company, formed in 1885, began mining the nickel-copper ores of Sudbury in 1887 and producing nickel-copper matte from its smelter in 1888. Although it was quickly appreciated that the separation of nickel and copper (i.e., refining) was critical to the production of a marketable nickel product, refined nickel was not produced in Canada until about 1918. From the beginning of the industry, there was considerable political agitation to refine nickel in the Province of Ontario. The companies, however, claimed that the existing processes, viz. the sodium sulphide “tops and bottoms” process operated by the Orford Copper Company of New Jersey, U.S.A., and the carbonyl process developed by the Mond Company at Clydach, Wales, were too costly to operate in Canada.

Nickel was first refined in Ontario about 1918 when the International Nickel Company opened a nickel refinery at Port Colborne, Ontario. Initially, however, the refinery at Port Colborne employed the Orford sodium sulphide process, and the first electrorefined nickel in Canada was produced in 1920 by the British America Nickel Corporation whose mine and smelter were located in Sudbury, Ontario, but whose electrorefinery was located at Deschênes, Quebec, near Ottawa, Ontario.

The purpose of this article is to outline the early history of nickel electrorefining and to trace the significant developments which led to the first successful commercial electrolytic nickel refinery in Canada. The development of an electrolytic process for refining nickel is due largely to the efforts of four individuals, viz. Carl Hoepfner, Hans A. Frasch, D.H. Browne and ultimately N.V. Hybinette, and the contribution of each is discussed separately below.

Early Nickel Electrorefining Processes

Hoepfner’s Electrolytic Method
In 1890, the Canadian Copper Company began experiments to develop a process for producing refined nickel and copper. The primary objective was to become independent of the Orford Copper Company, the main purchaser of their nickel-copper mattes. Jules Garnier, who had built nickel refineries in France to treat the New Caledonian laterite ores, was hired and was given a free hand. He built a Bessemer plant at Copper Cliff, Ontario, to refine the matte from iron and a plant at Cleveland, Ohio, to refine the metal and also to treat nickel oxide to produce pure nickel shot. He worked on the project for two years but without success.

The Canadian Copper Company then turned to Dr. Carl Hoepfner to develop his electrolytic method for separating nickel from copper. Hoepfner’s patent, “Improvements in the electrolytic production of nickel and other metals,” claims to produce nickel and cobalt by electrolyzing neutral or slightly acidic solutions of the chloride salts in a diaphragm cell. The metals are deposited either on vertical disk cathodes rotating between the anodes or on vibrating vertical plate cathodes. Chlorine is evolved on insoluble anodes separated from the cathodes by nitrat ed cotton or linen diaphragms which in some cases may be reinforced with asbestos.

The application of the above process to the electrowinning of zinc from zinc chloride solution was demonstrated on a commercial scale in Germany and England in the mid-1890s. Hoepfner also built a plant at Cleveland and worked on his process for separating nickel and copper for about a year, but without commercial success.

The Hoepfner Refining Company
In spite of the doubts cast upon his process by the Canadian Copper Company, Carl Hoepfner was hired by a group of Hamilton, Ontario, businessmen to operate his process in that city. The Hoepfner Refining Company, Ltd. was incorporated under the Ontario Companies Act on July 24, 1899 with its head office at Hamilton, and with a capital of $600,000 consisting of 6000 shares at $100 per share. Hoepfner’s services, together with the rights to his process, with improvements, were secured by the promoters who intended to establish a nickel refinery at Hamilton. This was the first electrolytic nickel process to be tried in Canada, and in 1900 a large plant was built to treat materials from the Sudbury operations. Hans A. Frasch was hired as the manager to work under Hoepfner’s supervision.

The same group of Hamilton businessmen also formed the Nickel Copper Company of Ontario, Ltd. in 1900. The arrangement was that this company would operate the mines and furnish matte which the Hoepfner Refining Company would process to the finished metal. The company bought the Whistle Mine in the Sudbury Basin and did considerable testing and development on the property, but it never achieved commercial operation.

In the meantime, the Hoepfner Refinery was hampered by unfortunate construction delays and the arrangement of the plant that involved expense and delay. In the end, the process could not be made a commercial success.

The Frasch Process
Hoepfner died suddenly in Colorado and the company then attempted to operate a new process developed by his assistant, Frasch. In this process, the Ni-Cu matte is made the anode of an electrolytic cell in which the electrolyte is a solution of an alkali salt such as NaCl and the cathodes are either carbon or copper. During electrolysis in the diaphragm cell, sodium hydroxide is formed at the cathode, and a solution of the corresponding chloride salts of the contained metals is formed at the anode. The sulphur contained in the matte reports in elemental form and remains as a residue or slime.

The metallic chloride solution thus obtained in the anode compartment is withdrawn and subsequently electrolyzed in a series of cells all having matte anodes and copper cathodes. The granulated matte is placed on a layer of car-
bonaceous material and together they form anode section at the bottom of the cell. The nickel-copper chloride solution passes through the body of the anode, through a horizontal diaphragm (usually a layer of sand), past the cathodes, and overflows to the anode of the next cell in series. The least electropositive metal in the solution; i.e. copper, is deposited on the cathodes. By careful control of the cell potential, the cupric chloride displaces nickel from the granulated matte anodes until eventually the solution becomes depleted in copper and more concentrated in the other metals, notably nickel.

To remove the last traces of copper, the electrolyte, in its passage through the last cell in the series, is not permitted to contact the cathode but circulates only through the anode, below the horizontal diaphragm. Only enough electrolyte passes through the diaphragm that the contained copper is completely removed upon contacting the cathode. The solution derived from the cathode section of this last cell contains only Ni, Fe and Co. The Fe is removed from solution by oxidation-precipitation, and the Ni is separated from Co by electrolysis or by chemical means. The Frasch electrolytic apparatus is shown schematically in Figure 1.

Although Frasch’s process was favourably reported in the technical press[12], it also proved to be a commercial failure. The process is overly complex and likely was difficult to regulate with the control devices available at the turn of the century. Neither the Hoepfner Refining Company nor the Nickel Copper Company of Ontario ever operated commercially and both incurred heavy financial losses. In 1905, the Hoepfner Refining Company suffered severe fire damage[13] that ended all hopes of further development.

Browne’s Electrolytic Process
From 1897 to 1898, the Canadian Copper Company, following its unsuccessful attempts to commercialize Hoepfner’s process, investigated the Mond carbonyl process[13] on which it had an option. It was decided, however, that this process was not suitable to the Canadian climate and environment. From 1898 to 1902, the company carried on a long and costly series of experiments in Cleveland under the direction of D.H. Browne. Browne patented an electrolytic process[14] which he claimed was better than previous technologies because it prevented the sulphur in the Ni-Cu matte from interfering with the dissolution reaction.

In Browne’s process, anodes of nickel-copper matte or nickel-copper metal and cathodes of pure copper are suspended in a solution of nickel-copper chlorides. This electrolyte is continuously regenerated by passing it through a tower filled with Ni-Cu Bessemer matte in contact with a salt solution and Cl₂ gas generated at the carbon anodes used in the nickel electrowinning cells. Copper is deposited initially, and when the electrolyte becomes enriched in nickel and the copper nearly all deposited on the cathodes, the remaining copper and iron are precipitated from the electrolyte with H₂S gas. The solution, now consisting of nickel chloride, is electrolyzed between carbon anodes contained in water-sealed compartments and pure nickel sheet cathodes. The Cl₂ gas evolved at the anodes is conveyed via the water seals to the regeneration tower as mentioned above. A schematic drawing of Browne’s apparatus for separating copper and nickel is shown in Figure 2.

Browne’s process ran satisfactorily for a short time and on a small scale in Cleveland, Ohio, in early 1902. In April of 1902, the International Nickel Company was formed by combining the refining works of Orford and Warton, and the mining properties of the Canadian Copper Company, the Anglo-American Iron Company, and the Vermillion Mining Company in the Sudbury district with the properties of the Nickel Corporation and the Société Minière Caledonnière in New Caledonia. The new company concluded that Browne’s electrolytic process could not compete with the Orford process and consequently it was abandoned.

Hybinette’s Role
N.V. Hybinette made significant contributions to the development of methods for separating copper and nickel, and his electrolytic process forms the basis of modern-day nickel electrowinning practice. In 1895, Hybinette improved the Orford Na₂S-based nickel refining process,
and in 1897 he was given complete charge of the nickel department of the Orford Copper Company. In this capacity, he was able to continue to develop his own ideas on the electrolytic nickel refining process.

By 1900, Hybinette had concluded that the Orford process had reached its limit of development. The process had many drawbacks including a large number of processing steps that made it totally uneconomical to operate on a small scale. He concluded that the solution to the problem of separating copper and nickel was to cast the matte into an anode, electrolyze the anode, remove the copper from the solution, and obtain nickel metal directly at the cathode. In 1904, while he was employed as a metallurgical engineer at the International Nickel Company, a plant was built at the Orford works to test this electrolytic process. Prior to start-up, however, a dispute over patent ownership resulted in his departure from the company[15]. Between 1906 and 1909, he further developed his process and used it successfully to treat the complex ores of the North America Lead Company at Fredericktown, Missouri. That plant, however, shut down rather suddenly and Hybinette returned to Norway where he built a nickel refinery at Kristiansand in 1910. By 1914, the plant was producing 1800 tons of electrolytic nickel per year[15].

The electrolytic cell for the separation of copper and nickel by the Hybinette process is shown in Figure 3[10]. The electrolysis cell, A, is a wooden tank with or without a lead lining. It has an overflow, G, which keeps the solution level at I and it contains a filtering-bag consisting of a wooden frame, K, and two thicknesses of cotton cloth, D, separated by the wooden frame. C is the cathode plate, usually a nickel sheet, with a wooden rib, E, which prevents the cathode from warping and making contact with the bag. B represents the anodes which are usually nickel-copper alloys. F is the inflow tube which is regulated in such a way that the solution in the bag is maintained at level H, about 1 inch above level I.

The weakly acidified pure nickel sulphate solution enters the cathode bag via the inflow tube F. Nickel is deposited on the cathode and the depleted electrolyte passes through the bag, D. At the anode, nickel, copper, and iron dissolve while the precious metals fall to the bottom of the cell as a slime. The electrolyte overflows the cell at G and goes to purification to remove copper and iron. Copper is cemented from solution using nickel-copper scrap anodes and the iron is removed by hydrolytic precipitation.

The purified solution is returned to the cathode bag. The operation is continuous until a sufficiently thick layer of nickel has deposited on the cathode and/or the anode has dissolved to the point where it must be replaced.

Canada's First Successful Nickel Electrorefinery

The British America Nickel Corporation (Banco)

About 1916, the International Nickel Company concluded that developments in the nickel industry had reached a point where it had become advantageous to have refining facilities in Canada[17]. A site was chosen at Port Colborne, Ontario close to an abundant supply of power from Niagara Falls, and construction began on a refinery with a planned capacity of 15 million pounds of nickel per year.

At about the same time, because of the unparalleled demand for nickel created by the First World War, a new company, the British America Nickel Corporation, also known as Banco, was formed to produce nickel in Canada. Banco, in which the British Government had a large interest, acquired deposits in Sudbury and began to construct a smelter at Nickelton, Ontario. A refinery also was constructed on the C.P.R. line at Deschênes, Quebec, near Ottawa, where cheap electric power was available, and other advantages, such as an excellent site, water and sufficient labour existed. The refinery employed the Hybinette electrolytic process to which the company had secured the rights for North America. The three plants (the...
mine and smelter near Sudbury and the refinery near Ottawa) were completed early in 1920. The head offices were located at Ottawa[19].

A view of the refinery at Deschènes, Quebec is shown in Figure 4. It covered an area of 170 000 sq. ft. and consisted of three buildings, each of about 5000 sq. ft., and a 250-ft stack[19]. The electrorefinery had a capacity of 15 000 000 lb of nickel per annum that could be increased to 20 to 24 million pounds at a comparatively small cost. During operation, the matte from the smelter was passed through two wedge roasters, each with eight hearths, and then to the leaching department where part of the copper was leached with sulphuric acid. This copper was subsequently plated out in electrolytic tanks. The copper cathodes were melted and cast into 81-lb ingots; about 55 tons of copper was produced for each 100 tons of nickel.

The leached matte was smelted in especially designed electric arc furnaces using 24-in. circular carbon electrodes, and nickel-copper metal anodes, weighing 200 pounds, were cast in steel moulds. These anodes then went to the nickel tank-house which covered three acres. Electrolytic nickel was produced and large amounts of nickel sulphate and nickel ammonium salts also were produced for sale to the nickel plating industry. The anode slimes were recovered and concentrated prior to refining in the precious metals department to yield platinum, palladium, iridium, rhodium and some gold and silver.

BANCO came into production in 1920 with an exceedingly efficient technical organization and a refining process which was cheaper than the Orford process used by the International Nickel Company. A further advantage of the electrolytic process was that it recovered the precious metals in the feed. In the following year, unfortunately, the demand for nickel declined to such a low point that the company was forced to close down. The British Government pulled out of the operation and the company was forced to seek markets in direct competition with the more established nickel producers.

The company restarted operations in 1922 under Norwegian and Canadian control. In spite of a price war, BANCO expanded production in 1923; by the early summer of 1924, it was producing ~ 400 tons of nickel per month and was materially aided by the value of the precious metals by-product. By July 1924, however, the company was forced into liquidation[20] and ultimately was taken over by the International Nickel Company. The refinery and smelter were dismantled and the useful equipment was moved. The municipal council of Deschènes tried to keep the stack and the principal buildings hoping to attract other industries. They also considered using the stack and large furnaces for burning the region's refuse, but Prime Minister Mackenzie King opposed this plan and gave the order "Tear it to the ground"[19]. As a result, the refinery was razed and today only the concrete foundations remain.

In securing the assets of BANCO, the International Nickel Company also secured the North American rights to the only nickel electrorefining process that had proved to be an economic success. The eventual use of a modified Hybinette process by the International Nickel Company, instead of its own Orford process, attested to the efficiency of the method. Thus, the International Nickel Company gained control, not only of the sole major Canadian nickel property remaining outside its orbit, but also over an efficient nickel refining process. This enabled the company to strengthen its position by lowering its own processing costs.

In 1928, a new nickel producer, the Falconbridge Nickel Company was incorporated in Canada to mine the nickel deposits in Falconbridge Township in the Sudbury area. Hybinette's Norwegian nickel refinery was acquired by Falconbridge Nikkelverk Aktieselsk, a wholly owned Norwegian subsidiary of Falconbridge Nickel[21] as a result, both of Canada's major nickel producers came to use the Hybinette process which had been extensively developed at the Deschènes plant in the early 1920s.

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Nickel smelting at Copper Cliff: 
the second fifty years

J.W. Matousek

Introduction
Since the late 1800s, copper-nickel ores have been mined and smelted in the Sudbury area. The Canadian Copper Company, the Copper Cliff ancestor of Inco Limited, commenced operations with a blast furnace smelter in 1888. Coal-fired reverberatory furnaces were added in 1911 and multiearth roasters in 1914. The blast and reverberatory furnaces smelted, respectively, copper-nickel ores and ore fines until 1930, when a milling process and plant began operating to concentrate and separate the minerals of these two metals. Two reasons for this delay were: (1) the availability of high grade ores suitable for direct smelting; and (2) the difficulties in making the copper-nickel separation.

The Mond Nickel Company, however, practiced flotation at its Coniston Smelter site from 1914 to 1930. A bulk concentrate containing six per cent copper and five per cent nickel was recovered from ore of 1.6 per cent each of copper and nickel. Gravity (table) and magnetic concentrates were also produced.

Inco Limited (then The International Nickel Company) began laboratory-scale flotation tests on its ores in 1918. This work led, ultimately (in 1930), to the construction of the 7500 tonne-per-day Copper Cliff Mill, designed to recover separate copper and nickel concentrates. Its capacity was expanded to 11 000 tpd in 1936, to 18 000 in 1940, and to 27 000 in 1942. Since 1971 it has operated to process only the bulk flotation products from other mills to yield individual nickel and copper concentrates.

Additional highlights in the history of the facilities for milling the Sudbury ores of Inco Limited are given below.

Figure 1 reviews the progression of smelting and refining at Copper Cliff. The "new" smelter of 1930 was commissioned with 30 multiearth roasters, five reverberatory furnaces, and eight Peirce-Smith converters. At the peak level of production, in terms of concentrate tonnage smelted, the main smelter contained 42 roasters, seven nickel and one copper reverberatory furnaces, one copper flash furnace and 19 converters.

<table>
<thead>
<tr>
<th>Mill Site</th>
<th>Operating Dates</th>
<th>Ore, tpd</th>
<th>Concentrate(s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Creighton</td>
<td>1951-1971</td>
<td>11000</td>
<td>Bulk</td>
</tr>
<tr>
<td>Levack</td>
<td>1959-1976</td>
<td>5500</td>
<td>Copper and Nickel</td>
</tr>
<tr>
<td>Frood-Stobie</td>
<td>1967-</td>
<td>22000</td>
<td>Bulk</td>
</tr>
<tr>
<td>Clarabelle</td>
<td>1971-</td>
<td>32000</td>
<td>Bulk</td>
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Ores and Concentrates
The ores mined in the Sudbury region have nickel-to-copper ratios of approximately one-to-one with the assay of each metal being typically in the 1.0 to 1.5 per cent range. In reviewing the production data for the Copper Cliff Mill it becomes apparent that the decade following start-up was one of learning the arts of copper-nickel separation. From 1930 to 1940 the nickel concentrates assayed 4.0 to 5.5 per cent nickel with the nickel-to-copper ratios shown in Figure 2. In the early 1940s this ratio was stabilized at values approaching four, where it remained through the mid-1970s with the continuously increasing grades shown in Figure 3. Combining these data with iron, sulphur, and silica assays of the concentrates and the analyses of the important sulphide minerals of the ore, given in Table 1, permits the variation diagram of Figure 4 to be calculated.

This latter figure shows that increases in nickel concentrate grades have been achieved by pyrrhotite and gangue rejection. The implications of this for roasting and smelting are discussed below.

TABLE 1. Mineral analyses

<table>
<thead>
<tr>
<th>Chemical Analysis — %</th>
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<tr>
<td></td>
</tr>
<tr>
<td>Pentlandite (Fe,Ni)₆S₈</td>
</tr>
<tr>
<td>Chalcopyrite (CuFeS₂)</td>
</tr>
<tr>
<td>Pyrrhotite Fe₇S₈</td>
</tr>
</tbody>
</table>

Readings in Historical Metallurgy
Roasting and Smelting

Nickel concentrate filter cake, pre-mixed with sand, is partially roasted prior to smelting. At Copper Cliff, fuel is burned in the multiearth roasters to supplement the heat liberated by the oxidizing reactions. While pyrrhotite levels in the concentrate have dropped, fuel ratios have shown an increase from roughly 100 to 450 MJ per tonne of dry roaster feed, clearly illustrating the values of iron and sulphur as fuels.

The reverberatory furnaces were fired with pulverized coal until 1973, and with Bunker C oil since. Energy requirements for smelting have shown a trend similar to those for roasting, although less pronounced, with fuel ratios rising from 3500 to 4500 MJ per tonne of dry roaster feed for the period being reviewed.

Matte Grades and Slag Losses

Figure 5 plots the furnace matte grades as a function of the
concentrate nickel and copper assays. The nickel matte grade follows a predictable trend. An analysis of the copper assays, however, shows that the nickel-to-copper ratio has decreased from approximately 3.8:1 in the concentrate to 2.5:1 in the matte. This occurs because of a copper transfer with the slags from the converters of the copper smelting circuit (returned to the nickel reverberatory furnaces for cleaning) where the charges are over-blown to promote the slagging of nickel.

The relationship between copper and nickel matte assays and the corresponding values in slag are shown in Figure 6.

**Summary**

Since 1930, environmental considerations and technological improvements have provided the driving force for raising the grades of nickel concentrates smelted at Copper Cliff. This has been achieved by rejecting pyrrhotite and waste rock. However, the advantages offered by the improved concentrates have not been realized without cost. Nickel and copper losses in the mill tailings have increased in typical grade-recovery relationships, although continuous research efforts have held these to acceptable levels. The content of the valuable metals in discard slag has also risen, but this loss has been compensated for by the lower slag volumes that result from pyrrhotite removal from the smelther feed. Similarly, costs associated with higher incremental fuel requirements have been offset by the reduced tonnages that have had to be smelted.

Most importantly, the higher grade concentrates have permitted reductions to be made in sulphur emissions, these having been cut in half since the 1960s decade. A portion of the pyrrhotite removed has been processed at the Iron Ore Recovery Plant where the contained sulphur is fixed as acid. The balance of the rejected pyrrhotite has gone to stockpiles or to tailings.

The above essentially describes conditions that existed to the end of 1976. In 1977, milling practices were altered to produce nickel concentrate with a nickel-to-copper ratio of 5:1 with nickel assays in the nine to eleven per cent range. Subsequently, laboratory and pilot-plant testing has concentrated on investigating the feasibility of raising the grade to 15 per cent nickel and in defining the extra costs, in metal losses, and the benefits, in reduced sulphur emissions, that would result.

1983 marked the 100th anniversary of the discovery of the nickel-copper ore deposits in the Sudbury region and of the incorporation of the community of Sudbury. The third, fifty-year period of production was entered on a strong note of uncertainty. It is becoming increasingly clear, however, that a break with the past is being made in the practices of milling, smelting, and refining by Inco Limited at its Copper Cliff operations.
Falconbridge Ltd. – 60 years in a century
Gerald A. Crawford

1880s – Unmasking the Sudbury Basin

Falconbridge Ltd. turned 60 in 1988 but its roots go back more than a century to the discovery and development of the mineral resources of the Sudbury Basin. Nickel was first identified in the area in 1856 but sustained exploration awaited the early 1880s and the advance of the CPR westward from Sudbury.

Rock was laid bare in the right-of-way and by forest fires and logging. Reddish oxidized outcropping or “gossan” were exposed, heralding the presence of nickel as the shape of this astounding geological formation gradually began to emerge.

Many great mines of the area were opened up in this period on the south rim of the Basin west of Sudbury. To the east the ore dips below overburden, however, and Falconbridge remained a mere dot on the map until the turn of the century when Thomas Edison detected a magnetic occurrence by dip needle.

Thomas Edison – Quest for Falconbridge Ore

Edison had developed an alkaline storage battery which featured a nickel electrode. He was keen to find a source of nickel to supply battery production and was attracted to Sudbury by an exhibit of the Ontario Bureau of Mines at the 1901 Pan American Exposition in Buffalo. Edison reasoned that nickel should be present in the Falconbridge area because of its geological similarity to other parts of the Basin’s rim where mines were already being worked.

He visited Sudbury in 1901, set up an office with a crew of fifteen and spent several seasons magnetic surveying, digging and drilling in the area. He sank a shaft at the Falconbridge occurrence but was stopped by quicksand at 80 feet. In response to another magnetic showing in the Blizard area in 1903, he drilled deeper than 1000 feet without locating ore. Wherever there were magnetic indications he sank pits and holes in a persistent endeavour to find ore but made no important discoveries, and finally gave up.

E.A. Longyear – Proving Edison’s Hunch

The Falconbridge claims lay unworked but not forgotten until 1915 when they expired and reverted to the Crown. They were restaked the next day for the Bennett-Longyear interests of Minneapolis which had been eyeing them for years. The E.J. Longyear Co. had been contract drilling in Ontario since before the turn of the century. R.M. Bennett, the president of Michigan Development Co., retained Longyear as development managers. They formed Falconbridge Mines Ltd. to hold the claims. A drilling program ensued (Fig. 1) and in 1916 the Falconbridge orebody was intersected 15 feet below the quicksand that had stopped Edison in 1901!

The Falconbridge find was the largest since the great discoveries on the southwest rim 30 years earlier. The property was valued at $2.5 million but in 1918 there was no buyer. Again the claims lay unexploited and languished untouched for another 10 years until Thayer Lindsley came along.

Thayer Lindsley – Taking Falconbridge Commercial

Thayer Lindsley (Fig. 2) was a mining graduate from Boston and Harvard who became interested in Canadian mineral exploration. In 1924 he was a roving geologist for McIntyre Porcupine Mines and in 1926, with partner Joe Errington, he staked Vermillion Lake and founded Sudbury Basin Mines. In 1928 he started Ventures Ltd. and promoted Sherritt Gordon Mines in Manitoba and Beattie Gold Mines in northwestern Quebec.

Lindsley and J. Gordon Hardy, a Scottish engineer and vice-president of Ventures, negotiated a deal with Bennett and Longyear, bought the Falconbridge claims for the requested $2.5 million, and on August 28 formed Falconbridge Nickel Mines Ltd. to hold them. Ventures and Sudbury Basin Mines bought shares in the new company and were its principal financial agents.

Work started immediately to commercialize the company. In addition to sinking a shaft, starting a mine and building a smelter, one of the more challenging essentials was to establish refining operations. All nickel refining technologies available at the time were controlled by Inco, at least in North America. Noak Victor Hybinnie, a Norwegian metallurgist, had licensed his electrolytic process to Kristiansand Nikkelraffineringssverk A.S. in Norway which was then floundering due to low nickel demand and prices in the aftermath of World War I. Lindsley seized the opportunity, bought the refinery and with it acquired both the license and the know-how for the Hybinnie process, particularly that of his consulting metallurgist, Anton Gronningsater.

Also a Norwegian, Gronningsater had been in Canada in the early 1900s but returned to Norway in 1909 to take charge of the Kristiansand refinery. He came back to Canada in 1919 and was working with the Bennett-Longyear interests when Lindsley bought Falconbridge Mines Ltd. Renamed Falconbridge Nikkelverk A.S., the refinery was overhauled, doubled in capacity to 5 million pounds per year and started up in late October 1929 refining European feedstocks. A 300 ton-per-day smelter with blast furnace and two converters was built at Falconbridge in 1929 and started up February 4, 1930. Canadian nickel from Falconbridge matte reached the market through Nikkelverk in July 1930 and Brandels, Goldschmidt & Co. of London was contracted to sell it on the European market, together with co-produced copper, cobalt and precious metals.

Thus, began Falconbridge Nickel Mines Ltd. which became and remains today, as Falconbridge Ltd., the second largest nickel producer in the world. Succeeding decades saw a stepwise pattern of mine discovery and plant expansion as nickel demand cycled upward.

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1930s—The Depression and Falconbridge’s First Decade

Falconbridge not only survived the depression, but maintained aggressive activities including exploration and discovery, shaft sinking, mine development, smelter and refinery expansions and addition of new milling, concentrator and sinter plant facilities (Fig. 3).

Thayer Lindsley and Gordon Hardy were the first president and vice-president of Falconbridge respectively. In 1936 they reversed roles, which gave Lindsley more time to look for new orebodies. A canny mine developer who could reputedly “smell” ore just by looking at a geological map, Lindsley was ultimately able to capitalize on his hunch that great opportunities lay undiscovered in the Levack area on the northwest rim of the Basin.

By World War II nickel production had increased sixfold from 3 to 18 million pounds per year and the company had paid a steady stream of dividends beginning in 1933.

1940s—World War II and Growing Demand for Nickel

The decade had barely begun when the Germans occupied Nikkelwerk on April 9, 1940. For strategic reasons, Falconbridge matte was refined by Inco at Port Colborne, Ontario, throughout the war.

At Falconbridge the war years saw further expansion of mine, mill and smelter facilities, purchase of the Strathcona property from Strathcona Nickel Mines Ltd., certification of Local 598 of the Sudbury Mine Mill and Smelter Workers’ Union, and commencement of diamond drilling on the McKim property.

In 1945 Gordon Hardy retired and L.K. Brindley became president. Lindsley attracted Horace J. Fraser to join the company as mine manager in 1946, thereby ushering in a new era in the evolution of Falconbridge. Fraser, a geologist from Saskatchewan with a Ph.D. in economics from Harvard, had been a professor at Caltech and during the war was with the Office of Economic Affairs in Washington, D.C.

In 1947 Lindsley resumed the presidency, and Fraser, his assistant and general manager, became vice-president the following year. Development of the McKim Mine continued with drilling and shaft sinking. The Fecunis Lake property was purchased and refinery capacity was doubled from pre-war levels upon recovery from the soft markets of the post-war period.

1950s—The Korean War and Further Expansion of Capacity

Nickel demand had no sooner recovered from the post-war lull than the Korean conflict spurred further concern in the United States over nickel supply. In 1953 Falconbridge negotiated contracts to supply the U.S. government stockpile with guaranteed quantities of nickel for 10 years at incentive prices.

Further mine and plant expansions were undertaken to increase production capacity to 55 million pounds of nickel per year before the end of the decade. The McKim Mine came on stream in 1950 while drilling, shaft sinking and mine development were pursued throughout the decade at a sequence of mines including Hardy, East, Fecunis Lake, Mount Nickel, Longvack, Boundary, Onaping, and Strathcona. Capacity of the Falconbridge mill was nearly doubled to 3000 tons per day and new mills were completed and started up at Hardy and Fecunis Lake.

Horace Fraser became president of Falconbridge in 1957 and of Ventures in 1958. The same year a further smelter extension was blown in and by 1959 annual capacity of nickel in matte had exceeded the 55 million pound goal.

Exploration and Research—the Lindsley Legacy

Fortunately for Falconbridge, both Lindsley and Fraser believed that exploration and research are vital pillars of corporate strength and growth. Exploration offices were opened throughout Canada and in many other locations in Europe, Africa, Australia, Latin America and the Pacific.

Research laboratories were established at both the Falconbridge smelter and the Nikkelwerk refinery operations, and what was to become the Falconbridge Metallurgical Laboratories were located near Thornhill just north of Toronto. These labs have collaborated in research and pilot testing on many process and product developments that have subsequently been commercialized at the company’s operations.
New Interest in Laterites

Until the 50s the company’s interests were focussed on the sulphide ores of the Sudbury Basin. In mid-decade, however, Horace Fraser, recognizing the essence of long-term survival in nickel production, guided Falconbridge toward recovery of nickel from laterites. Exploration activities were established in the Dominican Republic, and research was begun at the Thornhill laboratories into the behaviour of Dominican ores. It was the beginning of a development that was to have a fundamental impact on the makeup and future of the Falconbridge organization.

1960s—Ventures Absorbed and Research Expanded

The sixties witnessed a tremendous expansion and broadening of Falconbridge interests and activities on a world scale. Principal features of this growth included the absorption of its parent, Ventures Ltd., successful piloting of ferronickel from Dominican laterites, development of refining alternatives to the Hybinette process at Nikkelverk, bringing the large Strathcona mine and mill complex into production, and other activities in international exploration, mining technology and metallurgical process innovation (Fig. 4).

By the 1950s, Ventures Ltd. had become an international group of some 150 companies reflecting the exploration genius and enthusiasm of Thayer Lindsley. Falconbridge was the largest operation of the group and when Horace Fraser became president he spearheaded a review and rationalization of the Ventures organization. When it was merged with Falconbridge in 1962 Ventures had been trimmed by sale and consolidation to roughly twenty subsidiary and associated companies. The merger gave rise to the Falconbridge Group of companies, in which McIntyre Porcupine Mines bought a large interest. Some of the better-known operations include United Keno Hill, Giant Yellowknife, Kiena Gold, Westfrob, Opmiska Copper, Lake Dufault, Kilembe, and what are now Indusmin and Fahramet. Ventures thus became history but Thayer Lindsley’s commitment to exploration and research have stood Falconbridge in good stead.

Falconbridge Dominicana—Dominican Laterites Mastered

Falconbridge Dominicana is a classic example. Sustained development throughout the decade resulted in delineation of the mountainous Falcondo orebodies. In addition, two successive pilot plant operations demonstrated a unique recovery process based on furnace reduction of briquetted ore and electric melting of reduced calcine to produce ferronickel. By the close of the decade a commercial plant had been designed and was under construction in the Dominican Republic.

The opening in 1972 of the $200 million Falcondo operation in the Dominican Republic set the pace for further diversification and modernization of Falconbridge operations. Falcondo, the largest single capital investment the company had made to that point, added over 60 million pounds per year of nickel capacity as high-grade ferronickel, initially as ingots. An improved, smaller product form, Ferrocones, was introduced in 1978.

Marketing Organization—Falconbridge International

Historically all Falconbridge metals were sold in Europe with the exception of nickel delivered under contract to the U.S. government stockpile. With the expiry of the contract in 1963, the company established a marketing group in Toronto to develop new customers in North America. In 1971 Falconbridge International Ltd. was formed to oversee marketing and sales worldwide. That same year Falconbridge Europe S.A. was established in Brussels. In 1976 the organization was further expanded by formation of Falconbridge U.S. Inc. in Pittsburgh and Falconbridge International Ltd., Bermuda. The latter now negotiates toll and custom refining contracts with suppliers of nickel and cobalt materials.

Chlorine Leach Plant at Nikkelverk

To improve working conditions and decrease SO₂ emissions at Nikkelverk, research was undertaken in Canada and Norway on the development of process technology to displace Hybinette electrolefining in sulphate electrolyte. Chlorine leaching of matte merged as a major advance in hydrometallurgical technology not only for its environmental benefits but also for the enhanced separation and recovery of cobalt from nickel. This development ultimately resulted in the chlorine leach plant at Nikkelverk and electrowinning from chloride electrolyte in a new product form called Nickel Crowns, introduced in 1977. Hybinette electrolefining cells and the production of nickel squares were progressively displaced by electrowinning cells and increasing output of Crowns.
Thus, the ratio of SO₂ emitted per unit of nickel in matte has fallen dramatically, as shown. Today sulphur emissions are barely one-tenth those in 1950 per unit of nickel produced, a creditable achievement by any measure.

Kidd Creek
In recent years the most significant corporate change has been the acquisition in 1986 of Kidd Creek Mines Ltd. from The Canada Development Corporation. This purchase doubled Falconbridge assets to more than $2.6 billion. With one of the largest sources of zinc in the world, the state-of-the-art mining and metallurgical facilities, together with associated copper, silver, cadmium and gold, Kidd Creek is of major importance to the Falconbridge group. In addition to Kidd Creek, the group today also includes the integrated nickel operations, Indusmin, Falconbridge Dominicanica, United Keno Hill Mines, and Falconbridge Investments (Zimbabwe), a producer of gold.

Overview
This review summarizes historical highlights in the evolution of Falconbridge from its origins in the Sudbury Basin more than a century ago and its incorporation in 1928. It now embarks on its seventh decade with a long-term record of sustained growth as an organization of international presence and stature.

In 1985 it processed its 100 million ton of ore with roughly 50 million tons in reserve, a very different situation from the one million tons of reserves in 1930 and a reflection of the strong commitment to exploration throughout the company's history. A succession of orebodies has been developed and mined in the Sudbury Basin, most recently the Fraser Mine in 1981.

Surface treatment plants for milling, concentrating, smelting and refining have been expanded, modernized and replaced by successive technological improvements and innovations.

A group of associated companies now notably including Kidd Creek, constitutes a sound basis for the company's future.

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The development of the Sherritt ammonia pressure leach process

D.G.E. Kerfoot

Sherritt's Early Years

In September 1945, when the Lynn Lake nickel orebody was discovered Sherritt Gordon Mines was a small copper mining company with rapidly depleting ore reserves at its single producing mine at Sherridon in the Cold Lake area of Manitoba, 100 miles north of The Pas.

The company has been formed in 1927, largely through the efforts of Eldon L. Brown, a 27-year old mining engineer, with the financial backing of Thayer and Halstead Lindsley and the Gordon family. The Lindsleys were American mine promoters who had been successful in the Sudbury area, and the Gordons had extensive interests in the lumber industry in northern Ontario. Carl Sherritt, who gave his name to the company, was an American trapper and prospector who had staked the copper property and sold his interest to the new company. Carl Sherritt died in a flying accident at The Pas in 1928.

The Sherritt Gordon Mine started production in 1931, shipping copper concentrate to the new Hudson Bay copper smelter at Flin Flon, Manitoba. In 1931 the price of copper was 18c/lb. Within a year the price had fallen to 5c/lb and the mine was forced to close. It remained closed through the Depression years.

The Lynn Lake Discovery

When operations resumed in 1937, with the copper price at 10c/lb, Eldon Brown instituted an active exploration program in the area to the north of Sherridon, and this continued through the war years. In September 1941, Austin McVeigh, a Sherritt prospector working in the Granville Lake area north of the Churchill River over 150 miles north of Sherridon, sampled an outcrop of sulphide-bearing rock near Lynn Lake which assayed 1.5% nickel and 1% copper. This was the first significant discovery of nickel in Manitoba; the much larger Thompson orebody was not discovered by Inco until 1956. The find was kept quiet at the time since, with the Sherritt Mine at full production for the war effort and manpower depleted by requirements of the armed forces, the company was in no position to follow up a discovery in this remote area. Austin McVeigh carried out magnetometer work in the area in the prospecting season of 1943 and detected a number of magnetic anomalies.

Finally, in the summer of 1945, with the war in Europe over, McVeigh started staking in the Ralph Lake area near Lynn Lake and a diamond drill was flown in. Drilling on the strongest magnetic anomalies at Ralph Lake found only magnetite. Eventually, in September, the drill was moved to test several weak magnetic anomalies close to Lynn Lake and by the end of the month an intersection with good grade ore had been made. Over freeze up Sherritt staked claims to cover the area. A staking rush ensued as soon as the news leaked out, but Sherritt had secured all the nickel deposits.

A 165-mile winter road was broken through from Sherridon during the winter of 1945/46, and a camp was established at Lynn Lake in the spring of 1946. The drilling program located a number of nickel copper sulphide deposits, culminating in the finding of a high-grade orebody in January 1947 which finally established the presence of sufficient ore at Lynn Lake to warrant production.

At the beginning of 1946 it was estimated that the copper mine at Sherridon, which had operated at full capacity throughout the war, had ore reserves to support only four more years of production. Consequently Eldon Brown, now president and managing director of the company, decided that the mining and milling plant from Sherridon would be utilized to equip the new mine at Lynn Lake. At the same time he obtained the agreement of the shareholders that all subsequent revenue from the copper mine should be applied to the cost of developing the Lynn Lake nickel mine. As a result, the company paid no dividends to its shareholders from 1946 until 1960.

Early Metallurgical Testwork

Metallurgical testwork was started on drill core samples in the Sherridon laboratory at the beginning of 1946 and quickly demonstrated that high-grade nickel (10% to 14% Ni, 1% to 2% Cu) and copper (30% Cu, 1% Ni) concentrates could be produced by flotation. Reg Drake, the Sherridon mill superintendent, sent some samples of the nickel concentrate to Professor Frank Forward at the Department of Mining and Metallurgy at The University of British Columbia for a preliminary evaluation. In the meantime, Eldon Brown approached the major nickel producers, International Nickel and Falconbridge Nickel, with a view to reaching an agreement on custom treatment of the nickel concentrate. However, while these companies could offer temporary custom concentrate treatment arrangements, they were not prepared to enter into the long-term commitment that Eldon Brown needed to bring the mine into production. Consequently, he was faced with the prospect of establishing a completely integrated operation to produce refined nickel metal if the Lynn Lake project was to be viable. The conventional approach would be to establish a smelter and electrowinning on the lines of International Nickel's Sudbury and Port Colborne operations. However, the remote location, the lack of local supplies of fuel and power and the relatively small ore reserves compared with the Sudbury area, combined to make this option unattractive.

Testwork at U.B.C.

At The University of British Columbia (U.B.C.), Forward had had considerable success in his attempts to apply the Caron process to the treatment of samples of roasted Lynn Lake nickel concentrate. In this process, which had recently been commercialized at Nicaro in Cuba to treat nickel laterite ore, the roasted sulphide concentrate was first reduced with hydrogen at 450°C to 500°C, to convert


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the nickel and copper to metallic form and most of the iron oxides to magnetite. Nickel and copper were then leached in ammoniacal ammonium carbonate solution under ambient conditions. Copper was removed from the leach solution by electrowinning, and the nickel was precipitated as basic nickel carbonate which was calcined to oxide and reduced in hydrogen to a marketable nickel production. Forward had in fact spent several months in 1945 with the Freeport Sulphur Company working on this process. In June 1946 he was able to submit to Reg Drake an encouraging report on his preliminary testwork, together with seven small nickel ronderles produced from the Lynn Lake concentrate samples.

In early 1947, a pilot flotation plant was established in the Sherridon mill and 780 tons of quarried ore were shipped over the winter road from Lynn Lake for the testwork. By this time Forward’s report on the preliminary testwork at U.B.C., together with the negative response of the major nickel producers, had created interest in potential alternatives to smelting for treatment of the nickel concentrate. Forward was invited to Sherridon for discussions with Eldon Brown and obtained funding from Sherritt for a laboratory program to examine potential treatment methods. It was believed at this stage that if the hydrometallurgical approach, already explored at U.B.C., could be developed to give adequate nickel, cobalt and copper extractions it might have a significant cost advantage over conventional smelting and electroly refining methods. However, it was also recognized that the time available to develop a totally new process would be very short. Because it was essential to start production at Lynn Lake as soon as possible after the Sherridon mine closed, it appeared that five years at most would be available to develop and prove a new process.

The Sherritt-funded test program at U.B.C. was carried out under Forward’s direction, primarily by Val Kudryk, a graduate student in metallurgical engineering. Initially, activity was concentrated on adapting the Caron process, and a technically viable flowsheet was identified. After a dead roast of the concentrate to remove most of the sulphur, the calcine was treated by the established Caron process of a reduction roast under hydrogen, followed by leaching in ammoniacal ammonium carbonate solution to extract the nickel, copper and cobalt. Copper was cemented from the leach solution with metallic nickel, and nickel was precipitated as basic nickel carbonate by boiling the leach solution to expel ammonia. An interesting feature of this work arose from the fact that relatively poor nickel extractions were obtained, even with lengthy retention times, when the calcine was leached at atmospheric pressure using an air sparge for oxidation. Pressure leaching tests were then carried out using pure oxygen at pressures of 1 and 2 atmospheres and these showed that 95% nickel extraction from the calcine could be achieved in 1 to 2 hours.

**Discovery of the Ammonia Pressure Leach**

During the course of the work on the Caron process in June 1948, Forward and Kudryk stumbled on the discovery that nickel and copper sulphides could be leached directly in the ammoniacal ammonium carbonate solution
in an oxygen atmosphere. The discovery occurred as a result of a test designed to extract metallic nickel from a sample of high-grade nickel-copper matte, which was being used to supplement the limited supply of Lynn Lake concentrate. Leaching was carried out by rotating 5 g to 10 g of matte in 500 mL of ammoniacal ammonium carbonate solution in an oxygen atmosphere in a sealed Winchester bottle for 48 hours. To their surprise, they observed not only the nickel metal, but also all the nickel sulphide, had oxidized and dissolved. When the technique was applied to the nickel-copper concentrate, the nickel, copper and cobalt dissolved as metal ammines, and sulphide was oxidized to a soluble salt, leaving the iron as an insoluble oxide residue. It was soon established that the ammonium carbonate was not required, and could be replaced by the ammonium sulphate formed in the reaction.

It had by now been realized that the modified Caron process was not attractive economically. Efforts were therefore switched to this simpler and more promising approach, to improve the leach extractions by increasing the temperature and oxygen pressure, and to develop methods of separating and recovering the metals from solution. The simplest method of recovering the nickel appeared to be to boil off the free ammonia and then acidify the solution with sulphuric acid to precipitate crystalline nickel ammonium sulphate which could be separated by filtration. The remaining solution was then heated to decompose “polysulphides” and precipitate copper sulphide. Ammonium sulphate could be recovered from the barren solution by evaporation.

The new flowsheet was studied exhaustively both at U.B.C. and at Sherritt’s own laboratory at Sherridon during the second half of 1948, and, by the end of the year, the results were judged sufficiently promising to warrant the setting up of a pilot plant to test the leaching process in continuous operation. By this time a 25 ton/day flotation pilot plant had been established at Lynn Lake, providing a more plentiful supply of nickel concentrate.

Sherritt had so far spent $3.1 million on the development of the Lynn Lake Mine, and Eldon Brown was staking the future of the company on bringing it into production. The transfer of some of the mining plant from Sherridon, where operations were beginning to wind down, had begun in the 1946-1947 winter freighting season. The first shaft at Lynn Lake was scheduled for completion early in 1949.

The Chemical Construction Corporation

In December 1948, therefore, a meeting was set up between Sherritt and the Chemical Construction Corporation, a subsidiary of American Cyanamid, to discuss the design and engineering of a small pilot plant, in which Chemico was known to have wide experience. By a remarkable coincidence, it turned out that Chemico had been studying techniques for pressure leaching metal sulphides in acid solution. Of even more significance, they had established that nickel could be deposited from ammoniacal solutions as a metal foil or plate on the vessel walls by reduction with hydrogen under pressure.

Because it appeared that the hydrogen reduction method could potentially greatly simplify the production of nickel from the nickel ammonium sulphate produced by the ammonia leach process, plans were made for it to be incorporated into the Sherritt process flowsheet. Early in 1949 the Chemical Construction Corporation was engaged to design and build a small pilot plant for the ammonia leach process. At the same time Chemico undertook to continue the development work on the nickel reduction process with the object of producing a nickel powder product.

The First Pilot Plant

In April 1949, Vladimir Mackiw, who was to play a key role in the development of the nickel refining process, joined Sherritt at Sherridon. In June he was one of three Sherritt staff who, along with Frank Forward and Val Kudrky from U.B.C. and three engineers from Chemico, assembled in Ottawa for the pilot plant testwork. The first pilot plant was set up in a building made available by the Mines Branch at Booth Street, Ottawa.

The circuit was designed to treat 600 lb/day of nickel-copper concentrate in a two-stage countercurrent pressure leach operating in a continuous mode in vertical autoclaves. After filtration to reject the leach residue, the leach solution was boiled to expel ammonia and then acidified to precipitate nickel ammonium sulphate. A small pressure vessel was used for copper sulphide precipitation. Initially the pressure leaching step was carried out using pure oxygen as the oxidant, but later, air was substituted after an ammonia recovery unit was installed to scrub the autoclave vent gases.

In the fall of 1949, Frank Forward returned to his academic duties at U.B.C. and Sydney Nashner of Chemico took over as manager of the pilot plant. Forward continued to take an active role in the project and kept closely in touch with developments. In 1952, Eldon Brown endowed the Sherritt Gordon Chair of Metallurgy at U.B.C., thereby permitting him to devote more time to the process.

The first pilot plant was operated intermittently over a period of fourteen months. The leaching variables were investigated in depth, and a number of changes in procedures made. The most important development in this phase was the discovery by V.N. Mackiw that copper could be precipitated from the leach solution, prior to nickel removal, by controlling the leach to leave sufficient trithionate and thiosulphate ions in the liquor so that copper sulphide would precipitate when the solution was boiled at atmospheric pressure. This breakthrough opened the way potentially for the direct recovery of nickel by reduction from the leach solution without the need to precipitate nickel ammonium sulphate.

Concurrently with the pilot plant work on the ammonia leach in Ottawa, Chemico, with Sherritt engineering assistance, was piloting the hydrogen reduction process at the American Cyanamid facilities at Stamford, Connecticut. It was established that nickel could be deposited onto fine seed particles, in preference to plating on the vessel walls, and that the powder could be grown to a marketable product by successive depositions, or densifications.

The Move to Slater Street

By the spring of 1950, it was judged that, while the overall process offered the promise of being technically and economically viable, a much better understanding of the process chemistry and engineering requirements would be essential before a commercial plant could be designed with confidence. The decision was made to expand the pilot plant operation and a disused foundry at 422 Slater Street, Ottawa, was acquired and converted. Sydney Nashner returned to Chemico in New York to work on the com-
merical plant design and Charlie Hames, who had been with Sherritt since 1938, became manager of the newly created Sherritt Gordon Metallurgical Research Division. This group was to expand rapidly from about 30 in 1950 to over 140 at the peak of activity in 1953. Twenty-two different nationalities were represented in the group, which included many recent immigrants from Europe.

It was at this point that Eldon Brown engaged a New York consulting firm to provide an independent engineering assessment of all phases of the Lynn Lake Project, and the Battelle Memorial Institute was retained to report specifically on the ammonia leaching process.

The Second Pilot Plant
The pilot plant equipment was transferred from the Mines Branch to Slater Street in November 1950. A horizontal autoclave was installed for the second leach stage in the new circuit. A vertical autoclave was used for the first leach stage in all the pilot plant campaigns. The second pilot plant operated at a rate of 600 lb/day concentrate, on a semi-continuous basis, for about three months in the early part of 1951. The pilot work on the hydrogen reduction of nickel was transferred from Stamford to Ottawa at this time, and high pressure batch reduction autoclaves were included in the second pilot plant circuit.

The Battelle team reported favourably on the performance of the second pilot plant in the Spring of 1951, and recommended that individual unit operations should be tested on a larger scale to provide engineering data for scale up purposes. Individual units of semi-commercial size were installed and operated over the subsequent six months.

Financing for the Project
Up to this stage all the costs of developing the Lynn Lake mine, and the new refining process had been covered by revenue from the Sherridon mine. A total of $5.3 million had been spent by the end of 1950, and the total was projected to reach $10.3 million by the end of 1951. Because the copper mine was by now virtually exhausted, Eldon Brown had to seek outside financing. In the fall of 1950, he had negotiated sales contracts with the U.S. General Services Administration and with four major U.S. steel companies for the first five years of nickel production from Lynn Lake, at the prevailing producer price. It was estimated that it would cost $35 million to complete the development of the mine, and construct a concentrate refinery in Alberta to produce 17 million lb/y nickel. The ore reserves at Lynn Lake stood at 14 million tons, averaging 1.22% Ni and 0.62% Cu, corresponding to a metal value of about $12.50/ton of ore.

Early in 1951 Eldon Brown approached several New York finance houses, but quickly learned that to obtain the level of financing he required, he would need the backing of a major mining company. After failing to interest several Canadian companies in the project, he approached Newmont Mining Corporation through his brother-in-law, John Drybrough, who was President of Newmont Canada. He obtained support in principle from Newmont president, Fred Sears, Jr., and Plato Malozemoff, who was screening new ventures for Newmont. Malozemoff subsequently joined the Sherritt Board as Newmont's representative.

With the backing of Newmont, Brown was able to obtain a commitment from C.D. Howe, Federal Minister of Commerce and Industry, and from the Canadian National Railway to construct the crucial 150-mile railway line from Sherridon to Lynn Lake. With the rail link assured, and the favourable report from Battelle on the new process, Sears and Brown were able to raise the necessary financing through J.P. Morgan & Co. of New York in the spring of 1951.

Nickel Reduction and Oxydrolysis
Having successfully unravelled the chemistry of the leaching process and established the technique for producing copper-free liquor, Valdimir Mackiw and his colleagues turned their attention to the chemistry of the nickel reduction step. While Chemical Construction had succeeded in outlining a procedure for depositing nickel onto fine seed particles, an economic process for generating the seed itself had not yet been found. Initially the seed particles were produced by fine grinding the coarse nickel powder product, which was both difficult and costly. The Sherritt team set out to find a method of producing the particles by direct reduction from solution. In the absence of a catalyst, such as fine nickel powder, deposition of nickel occurred only on the vessel walls. The key to success lay in finding a soluble catalyst which would initiate self nucleation in a nickel-containing solution.

The breakthrough came from the observation that self nucleation sometimes occurred in solutions made up from pilot plant nickel ammonium sulphate, but never occurred in solutions made from the pure salt. Detailed analysis of the solutions established that the active catalyst was ferrous sulphate, which was present in trace amounts in the pilot plant nickel ammonium sulphate. Systematic study of other variables affecting self nucleation established that the nickel: ammonia molar ratio and the ammonium sulphate concentration must also be controlled within certain ranges. Finally it was found that a solution of the desired composition for nucleation could be produced very economically and simply by using second-stage leach liquor to redissolve the small amount of nickel which always plates on the walls of the reduction autoclave.

Initial attempts to precipitate nickel powder directly from pilot plant copper-free liquor produced nickel with an unacceptably high sulphur content. The high sulphur level was traced to the presence in solution of residual unsaturated sulphur species, such as thiosulphate and polythionate, which were reacting with hydrogen in the reduction step. It was soon found that these species could be oxidized to sulphate by treating the solution with oxygen at a temperature above 120°C.

However, the problem of high sulphur levels in the nickel powder persisted and it was eventually discovered that the leach solutions also contained significant levels of the sulphamate ion. Nickel sulphamate was actually crystallized from leach solution and identified by X-ray diffraction analysis. A much higher temperature treatment, at 250°C or higher, proved necessary to hydrolyse the sulphamate to ammonium sulphate. A combined process step, known as oxydrolysis, was adopted to ensure complete conversion of all the sulphur containing anions in the solution to sulphate prior to nickel reduction.

The complete process for producing nickel powder from nickel-copper concentrate had now been outlined in the laboratory and in the preliminary pilot plant work.
The Third Pilot Plant
In the fall of 1951, a new larger scale pilot plant with a capacity of 300 lb./day of concentrate was designed and built. This circuit included the countercurrent leaching and copper removal steps which were operated in an integrated continuous mode, and the oxidoxylation and nickel reduction steps which were operated using the same vessel on an intermittent basis. The purpose of this pilot campaign was to generate engineering design data for the commercial plant. Newmont’s continued financial support also hinged on the successful operation of this pilot plant. At Plato Malozemoff’s request, Sherritt again retained Battelle to provide assistance in interpreting the data generated in the pilot plant.

The third pilot plant was operated continuously, under very close observation, for about five weeks in early 1952. An immense amount of data was collected during the operation and this formed the basis for the design of the 235 ton/day concentrate commercial plant. The design for the refinery was frozen in July 1952, with completion of construction scheduled for December 1953.

The Freight Haul to Lynn Lake
Mining operations at Sherridon came to an end in September 1951, and during the freight haul of the following winter the main mining and milling plant, together with 73 homes, were moved by tractor-hauled sleigh to Lynn Lake. By the time the railway reached the mine in 1953, over 40 000 tons of material, including 208 buildings from Sherridon, had been hauled in over the 165-mile winter road. This operation sparked wide media coverage, and as Eldon Brown remarked in an interview with Time Magazine in June 1951, “Sherridon is one mining town that won’t end up as a ghost town”.

Fort Saskatchewan
In January 1952, the Chemical Construction Corporation was awarded the contract for the design and construction of the refinery. The site selected was an 800-acre parcel of land outside Fort Saskatchewan, Alberta, 15 miles from Edmonton. The Alberta site was dictated by the requirement for natural gas as the feedstock for ammonium production, this being prior to the development of widespread pipeline networks. A site on the Canadian National Railway was mandatory, because this was a condition under which the CNR had agreed to build the rail link to Lynn Lake. A further consideration was that the site should have good access to the Port of Vancouver to facilitate the export of the ammonium sulphate by-product for which at that time there was no local market.

The Fourth Pilot Plant
Towards the end of 1952 the fourth and final pilot plant was assembled in Ottawa. This circuit was as nearly as possible a small-scale replica of the commercial plant and was operated with the object of identifying and overcoming problems likely to be encountered during plant startup. The most significant process problem encountered resulted from the high calcium content of the Lynn Lake concentrate, which led to severe scaling and slow leaching. This was prevented by adding carbon dioxide to the leach to precipitate calcium carbonate.

A further purpose of this pilot plant was to train a crew of key operators for the commercial plant. Many of the 140 staff employed in Ottawa would subsequently move to Fort Saskatchewan to form the nucleus of the refinery operating crew.

The fourth pilot plant provided the first integrated testing of the oxidoxylation, nickel reduction, nickel-cobalt sulphide precipitation and ammonium sulphate crystallization operations. The compaction of nickel powder into briquettes for market was also successfully demonstrated during this campaign. The circuit was started up in April 1953 and ran through to September. A total of 130 tons of concentrate was processed and 12 tons of nickel metal was produced.

The campaign was highly successful, and provided the final confirmation of the process for Newmont, Battelle and Sherritt. Charlie Hames and 70 employees were subsequently transferred to Fort Saskatchewan to prepare for commissioning of the refinery. Vladimir Mackiaw became manager of the Research Division.

Refinery Construction Delays
The railway reached Lynn Lake at the end of October 1953, and ore production and concentrate shipment started on schedule in November. Unfortunately the construction of the refinery was not progressing so well. It became evident late in 1953 that both the construction schedule and the capital cost of the plant had been seriously underestimated. Construction of the refinery was hampered by shortages of skilled labour, steel supplies and shop facilities in the Edmonton area, which was still experiencing the boom following the discovery of oil at Leduc in 1947. Construction was not in fact completed until May 1954. The over-run on the capital cost of the refinery was $7.2 million, which could not be covered by the existing financing arrangements.

In January, Eldon Brown was able to arrange with the G.S.A. for advance payment of the nickel concentrates which were already being stockpiled at Ford Saskatchewan. In March, Newmont and the original financing group, led by J.P. Morgan & Co., agreed to put up an additional $3 million with provision for a further $3 million, which was not in fact needed before the operation became self-sustaining. The final cost of the refinery complex at Fort Saskatchewan was $23.5 million. In addition the development of the process cost approximately $3 million between 1947 and 1955.
Refinery Commissioning
Leaching of concentrate started in May 1954 as soon as construction was completed. It took 3500 tons of concentrate and five weeks to fill the leaching circuits. The first nickel powder product, which incidentally met specification, was produced on July 21, almost two months after the first concentrate was fed to the leach. The plant reached 90% of capacity by the end of 1954 and reached design capacity during 1955.

Postscript
Sherritt's Ottawa operation was closed in 1955 and the Research Division was transferred to Fort Saskatchewan. The pilot plant equipment was also shipped to Alberta where it was reassembled to form the basis of a cobalt refinery, which is still very much in operation.

Much of Sherritt's current metallurgical and product technology can be traced back to work done during the development of the ammonia leach process. Pressure leaching of sulphide ores and concentrates, using continuous horizontal autoclaves, has provided the basis for a thriving pressure hydrometallurgical process licensing business which currently offers processes for treating nickel mattes and concentrates, zinc concentrates, and refractory gold ores and concentrates. The nickel reduction process perfected in the Ottawa pilot plant was subsequently licensed worldwide and by 1980, 40% of free world refined nickel metal was produced by this technique. The understanding of nickel nucleation and reduction with hydrogen from solution acquired in the pilot plant was later used to develop a specialty composite powders business which supplies products for a variety of aerospace and other specialized applications.

The Lynn Lake mine closed after its ore reserves were depleted in June 1976. The town of Lynn Lake successfully survived the closure, thanks to the 1961 discovery of a copper-zinc orebody nearby, which supported Sherritt's Fox Mine from 1970 to 1985. After exhaustion of the ore reserves of the Fox Mine, the old nickel-copper concentrator, which had been 'mothballed' in 1976, was converted into a gold mill to serve the MacLeellan Mine which came into production in 1986.

The Nickel Refinery at Fort Saskatchewan has continued to operate very successfully as a custom refiner of nickel mattes and concentrates. Over the years the production capacity has been increased from the original 17 million lb/y to over 50 million lb/y of nickel. Today the Sherritt Nickel Refinery is part of a major chemical and metal processing complex. The Fort Saskatchewan site supports a world-scale ammonia and urea fertilizer business which has developed from the ammonia and ammonium sulphate produced in the nickel refinery, as well as a rolling mill producing nickel strip from powder.

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Preserving the historical record of Canadian mining and metallurgy

Norman R. Ball

Many people in the mining and metallurgical industries were brought up to embrace Henry Ford's pronouncement that "History is bunk" but it is not generally known that the same Henry Ford created and paid for one of the finest technological museums in the Western World. I am aware that many people feel that industry history is something you ask aging executives to do or that history is a clip-and-paste compendium of pictures re-discovered every 25 years or on the occasion of issuing a press release on plant expansion. Moreover, there are many who fear historians as somewhat deranged individuals interested only in corruption, injustice and sensational labour strife. Good history lies somewhere between the controlled P.R. handout and the hysterical attack. Writing good history is a difficult creative act based on the application of experience, insight, wide knowledge and careful reasoning to available evidence. To date, the Canadian mining and metallurgical industries have achieved a great deal but left too little evidence and it is this lack of evidence which must be corrected before it is too late. It is the purpose of this article to stimulate thoughts with regard to archives, one of the more important vehicles for the preservation of such evidence.

Knowledgeably selected archival documents provide essential raw materials for high-quality historical research and writing. It is possible to write history of a sort without archival documents; evidence may be gleaned from other scraps, but products relying solely on recycled scrap often face very severe quality restrictions. The selection and preservation of archival material is crucial to a more enlightened awareness of the role and development of Canadian mining and metallurgy. Paradoxically, this is also one of the least understood facets of the industry, even in public-relations circles.

The relationship between archives and the public image and awareness of mining and metallurgy is far more important than many suspect. It is the author's opinion that without improved archival preservation, Canadian mining and metallurgy will continue to be largely misunderstood, if even thought about at all, in many influential circles. Moreover, preservation of corporate and individual records, indeed involvement with the historical record, is not a costly frill. On the contrary, individuals and corporations can often make money, as well as favourable images, by participating in preserving the historical record.

To understand the relationship between archives and the images of an industry and profession, it is necessary to begin by examining a number of related questions. Who are the users of archives? What do they do and what is their collective impact? What do they look for, find, or fail to find and with what net results?

Aside from genealogists, most users of archives are part of what I call the culture business. The Canadian culture business includes our schools — elementary, secondary and post-secondary — and our museums, from the smallest to the internationally famous such as the Royal Ontario Museum and the various National Museums. It also includes uniquely Canadian institutions such as the National Film Board and the Canadian Broadcasting Corporation as well as the rest of the communications media.

The major end products of research will consist of, or contribute toward, items such as films, radio and television programs and documentaries, educational kits and text books, museum exhibits, restoration and reconstruction of the buildings and artifacts, policy plans and directives as well as theses, books and articles for various journals, magazines and newspapers. Despite the apparent diversity, the component parts and the final products of the culture business share one very important feature: they are all part of an extremely influential network which consciously and unconsciously tells us who we are, what to be proud of, what to aspire to and whether or not we have ever achieved anything in various fields. The culture business moulds the national self-image and does it so effectively that we are very rarely aware of what it is doing, a point to which I shall return.

To continue with the questions posed earlier: What do people look for in archives? The answer is best understood by examining it in two parts. First, as regards medium or document type, they look for whatever will provide information: letters, project proposals, diaries, notebooks, transcripts, written recollections and minutes of various meetings are traditional archival sources. However, archives are also for preservation, and researchers use photographs, sketches and paintings, films, videotapes, computer tapes and tape recordings of a variety of interviews, speeches, radio programs and oral reminiscences. In brief, the researchers examine whatever they can get their hands on to take them beyond the information found in the important, but traditional, periodicals and monographs of a library.

It is more difficult to define the subject areas of archival research, as these vary so much from institution to institution and archive to archive. As an archivist, the author has dealt with individuals interested in areas such as engineering, science, mining, agriculture and medicine. The author is also very atypical of Canadian archivists in dealing with non-government records relating to science, technology and engineering. Most archivists and researchers deal with more traditional areas of historical inquiry such as the arts and politics, as well as social and economic development. A lesser number handle current and new concerns which may or may not become the entrenched historical fields and public images of tomorrow: ethnic studies, women's history, native studies, labour history and environmental studies. It is also important to note the areas which in terms of wide usage are rarely researched: Canadian engineering, Canadian technology, Canadian industrial development (except as a source of tales of labour strife) and Canadian mining. In brief, the areas of concern, activity and professional contribution and exper-


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prise of the members of The Canadian Institute of Mining and Metallurgy, are of little interest to the majority of archival researchers. Why the exclusion? Even a partial and tentative answer is very complex and in this brief article I will deal with only two of the reasons.

Researchers in the culture business — as with the engineering and science business — tend to tackle projects which interest them and which appear likely to succeed. Areas such as mining and metallurgy lose dramatically on both counts. Aside from the usual desire to appear “with it” and informed at luncheons and airports, most of the participants in the Canadian culture business are decidedly uninterested in anything smacking of Canadian engineering, technology and science. They are the past products and present perpetuators of an educational and cultural system which, by looking at the achievements of other countries while ignoring our own, subtly conveys the message that we have created, contributed and achieved but little in the scientific and technical spheres. Moreover, the few who might want to do something would face great difficulties: their peers and colleagues would question their judgment, and they would not find it easy to hire competent research staff, but, most serious of all they would find information and research sources hard to come by. The usual corporate and industry puff pastries issued by public relations departments are of little or no use; they are generally too brief, superficial, undocumented and one-sided. The documentation needed to go further — the type of documentation usually found by more traditional scholars in archives — is not easily found in Canadian archives. Thus, anyone in the Canadian culture business who wishes to break outside of the traditional mould will face serious difficulties.

Can it be changed? Yes. Will it be changed? I really don’t know. It will depend on too many factors to enumerate here. However, it is largely a question of perceptions, care and perhaps a touch of arrogance. Do the Canadian mining and metallurgical industries realize how effectively they have been excluded from the culture business paid for by their taxes and generous contributions? Do they care? Do they really believe they are so obviously indispensable, well-known, loved and appreciated that a few P.R. handouts for school kids on plant tours, and a few magazine ads about clean water, trout streams and trees or grain growing on tailings will counter the effects of an enormously large and well-financed culture business which much more subtly tells Canadians that in the larger scheme of things mining and metallurgy isn’t important or does more harm than good. Does the industry really believe that it can be strong, healthy and creative and can maximize its benefits to society in a culture and intellectual environment which does not understand it, ignores it, or is hostile to it?

When I studied chemistry at university, I was taught to pursue and base my actions on those things which could be physically observed, measured and calculated. There were some mysteries, and some things were a trifle messy, but in general, it was a quantifiable orderly world. As a historian, both in formal training and practice, I have become only too acutely aware of the power and influence of misinformation and the irrational, and of underlying cultural assumptions and prejudices. As a participant in the culture business, I have come to see inertia not as a calculable easy-to-handle Newtonian phenomenon, but as an intellectual quality which is readily defensible by its holders, destructive in its effects and virtually immune to normal logic. Putting mining and metallurgy in the culture business will involve going beyond the logical analysis familiar to so many within the industry.

Given the importance of archives, then, there are three issues which must be considered: what to save, how to go about it, and the benefits, including financial ones.

To the extent that archives have a popular image at all, they are seen as repositories for unpublished political letters and diaries and nothing else; this is false, but understandable. Although they leave most but not all of the book collecting to librarians, modern archivists collect a wide range of materials which share one common feature: they may be used to aid in understanding or illustrating some facet of the past, both the distant and the recent past. Politics has most certainly dominated traditional views of the past. However although archivists have paid little attention to the history of science, technology and engineering, the practitioners within these same fields have often been even more negligent. Consequently, many of the most central and fundamental records in Canadian science, technology and engineering — materials which one would assume to be very common — are very scarce or completely absent in Canadian archives. Moreover, almost all of the scientific and technical records found in archives relate to government activity. What is the consequence of this for records relating to mining and metallurgy?

The exceptions notwithstanding, it is a valid generalization that in Canadian mining history the main function of government, aside from regulation and taxation, has been to promote basic geological exploration and mapping. Private enterprise has dominated mining and metallurgy and the conversion of its output into usable, saleable products. However, as noted earlier, of the small amount of the Canadian archival dollar and interest devoted to science and technology, most goes to preserving a record of government activities. Thus, when one looks at the mineral industries one finds that we have an already short stick, namely archival preservation; and of that very short stick, the shortest end is given to the efforts of private enterprise — namely mining and metallurgy — and the longer end to the efforts of government in basic geological exploration and mapping. There is a reason to the perverse logic I have been pursuing — or should I say the application of logic to a perverse system? The reason is to answer the question: What should be saved for archival preservation? The answer in Canadian mining and metallurgy is practically anything you can get your hands on for, comparatively speaking, virtually nothing has been preserved.

We need to preserve a record of the activities of individuals, societies, educational institutions and companies. To do this we must start with, but go far beyond, the printed annual reports and printed research articles. Many mining companies have left no printed records, just a few newspaper clippings. Parts of older Canadian technical journals have simply disappeared. However, we all know that the printed reports tend to dwell on what happened and not how or why. We need minutes, reports of meetings, committee reports, diaries and letters; in brief, we need the unpublished and more private materials which come closer to telling what really went on. We need the plans, drawings and sketches not only for the things that worked, but also for the things that did not. We need to know more about the equipment used. Remember the trade catalogues you were flooded with? Many of them contain now scarce information and are very hard to find.
If visually appealing documentaries are to be made, we need film footage, photos of camps, mining sites, smelters, furnaces, assay offices, research laboratories and equipment. We also need voices and the sounds of the industry on tape. There are some who have had interesting careers, witnessed and participated in a great deal, but have no records such as I have discussed. If you are one such person, give us your thoughts, the benefits of your experience, your reminiscences. Put them down on paper or tape, as long or as short as you please.

Assuming one is all fired up about preserving archival records and has them on hand, what does one do next? Phone, visit or write to an archives. Let them know what you have, where it is and what you would like done with it or ask for their suggestions. There is a reference book published by the Bureau of Canadian Archivists, *Directory of Canadian Archives* (available from Association of Canadian Archivists, c/o Room 349, Public Archives of Canada, 395 Wellington Street, Ottawa, Ontario, K1A 0N3) which lists archives in Canada with phone numbers and mailing addresses. Not all archives will be interested in the records you have, but this does not mean they are not historical. If you need assistance in finding a receptive archives, write to the Science and Engineering Archivist, manuscripts, Public Archives of Canada, 395 Wellington Street, Ottawa, Ontario, K1A 0N3.

I am sometimes asked the very blunt but legitimate question: "If I go to the trouble of hunting up and putting papers in an archives, what is in it for me?" The precise answer varies from case to case, but for purposes of this article there are three major issues: first, professional pride and perhaps even responsibility and ethics; second, corporate citizenship; third, that most universal of all lubricants, money.

The participants in Canada’s mineral industries have an enviable record of achievement both in and out of Canada. It is a part of Canadian culture and achievement which we should all be allowed to feel proud of and try to emulate rather than having it hidden away in a closet as if excellence, innovation, inventiveness, ingenuity and determination in the face of unfavourable odds are somehow not part of the Canadian character, and are deviant and embarrassing behaviour. However, there are those who might argue that anything aimed at increasing public awareness of corporate and individual achievement smacks too much of suturing about, puffing oneself up and generally engaging in the base behaviour which we should reserve for politicians and charlatans. Think again. Perhaps as a professional you have an obligation not only to keep the incompetents and the charlatans out, but also to help attract the very best of the youth who are deciding what direction their education and careers should take. If so, then you have an obligation, individually and corporately, to see to it that your profession and industry receives the regular recognition it merits. To do otherwise is irresponsible or at least not very wise.

Good corporate citizenship, that ill-defined and much-abused concept, is another reason for contributing to the archival record. I have never fully understood why the corporations and eminent individuals of Canada’s mineral industries will sponsor athletes, dancers, artists and musi-cians, but not historians, archivists and reputable industrial museums. Nor am I able to comprehend or admire the mentality which purchases corporate art by the yard or the ton, according to the fashion of the day, but orders the historical heritage of the company pulped or melted. Equally unfathomable are those who will readily fund conferences on the performing and graphic arts, but would not give a nickel to fund a conference on the history of Canadian mining or engineering, or underwrite the research costs to prepare genuinely useful educational material on the history of mining and its place in Canadian society, culture and history.

All of the above largess is done in the name of good corporate citizenship. What explains the denials and exclusions? Lack of funds is more excuse than answer. Wider sharing or equitable rotation is a possibility. Does the industry have so much to hide that any attention is too much? I do not believe that, but many do. Is the industry and profession so smug that it sees no need to use its corporate largesse in anything but the most traditional way? Where are the innovative leaders as opposed to those who are merely sitting in the driver’s seat? Or is the ultimately detrimental modesty which I tried to say earlier rearing its ugly head? The Canadian mineral industries quite unfairly seem to be tossed from no public press to bad public press. There is much for the industry to be proud of, but too often the only defenders are those hired to do public relations and they are naturally somewhat suspect. Would it not make good sense to use a portion of the charitable or good citizen ship dollar to facilitate more open examination and wider dissemination of mineral industry achievement and contribution? As an outsider, informed, but an outsider nonetheless, I am confident that in the long run the benefits and good press would predominate. Do the insiders who know more think that I am wrong?

To this point, I have behaved as if in dealing with material goods and money I am asking the industry to be guided by the Biblical injunction that it is better to give than to receive. Not quite, for if they are wise, those who give to archives (and museums) shall in return be given money or tax credits which are just as good. Archives sometimes make outright purchases. If so specified at the time of donation, archival records can be evaluated for purposes of tax credit, often at figures which quite amaze and please donors. Similar rules apply to museum artifacts such as the historic machinery and equipment often sold for scrap in the name of economy.

There are a number of reasons for preserving archival records. There are also reasons for not preserving them, but I hope that this article has helped to deal with two major reasons for the loss of much of the historic fabric of Canada’s mineral industry: individual and corporate inertia (continuing in the old comfortable easy way) and a lack of understanding of the important relationship between the mineral industries business and the culture business. The reader who has got to this point has invested some time in considering the issue. I too have invested considerable time and effort as well. It is now up to the reader to forget about it, to put it off — and we both know that is a euphemism for forgetting about it — or to capitalize on our investment.