

SGAT **bulletin** **Bi-annual**

Vol.15•December 2014•No.2

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PRESIDENT'S COLUMN

WITH PROFOUND SORROW, I WOULD LIKE TO SHARE WITH YOU THAT THE SUDDEN AND UNTIMELY DEATH OF SHRI BINOD CHANDRA PATNAIK, GENERAL SECRETARY ON 18TH NOVEMBER 2014 HAS CREATED A VOID IN SGAT. MAY ALMIGHTY GOD BESTOW ALL HIS BLESSINGS TO HIS BEREAVED FAMILY AND FRIENDS TO WITHSTAND **THIS** IRREPARABLE LOSS. MAY HIS SOUL REST IN PEACE. THE SINCERE, SELFLESS ATTITUDE OF SHRI PATNAIK FOR ORGANIZING VARIOUS ACTIVITIES RELATED TO FUNCTIONING OF SGAT WILL BE REMEMBERED FOR TIME IMMEMORIAL.

THIS VOLUME OF SGAT BULLETIN IS DEDICATED IN THE MEMORY OF SHRI BINOD CHANDRA PATNAIK, GENERAL SECRETARY.

DR. S.K. SARANGI
PRESIDENT & EDITOR

VALUE ADDITION OPTIONS FOR THE MINERAL SAND RESOURCES OF ODISHA, WITH PARTICULAR REFERENCE TO OXIDES OF TITANIUM LIKE ILMENITE

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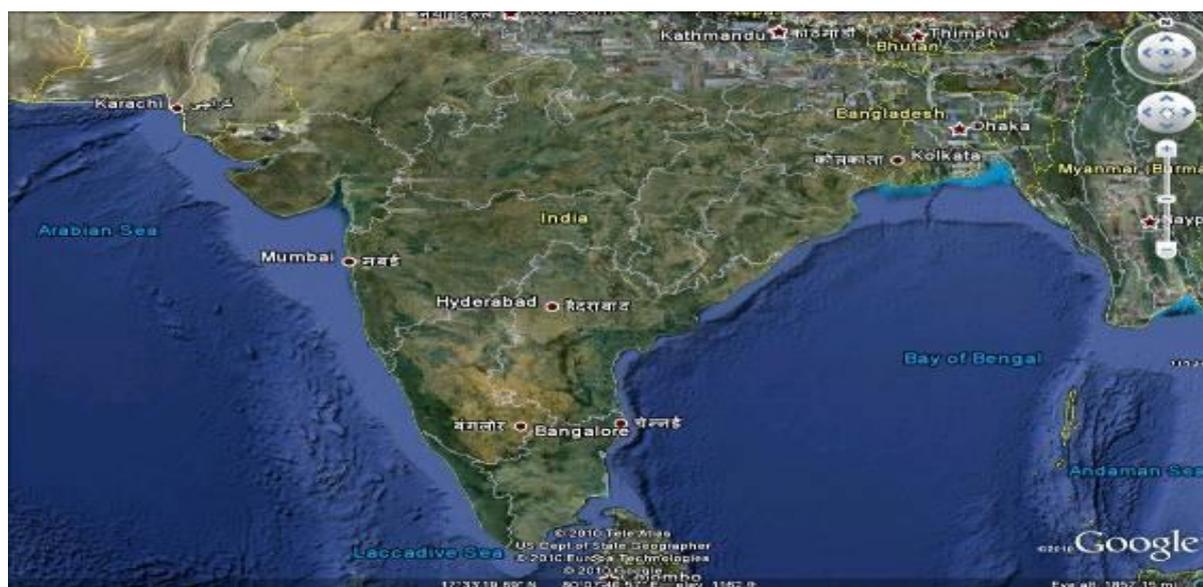
ABSTRACT

Hinterland geology, hot and humid tropical climate and favourable coastal geomorphology makes 2100 km of the Indian Coast line, including 105km of Odisha, a potential target for placer mineral sands. The Odishan coastal placers dominantly comprise of Ilmenite (45 Million Tonnes), Garnet (33 Million Tonnes) and Sillimanite (21 Million Tonnes) out of the total heavy mineral resource of 106 Million Tonnes. Ilmenite offers wide scope for value addition to TiO_2 Pigment, which is the market driver for titanium feedstock. The Odishan ilmenite have lower TiO_2 of around 48-50% and require beneficiation/upgradation to intermediate products like Synthetic Rutile (SR) or Titania Slag. For the SR (a chemical process of corrosion/leaching), there are issues of (i) handling logistics for the specific type of coal required in Bechar SR, (ii) technology transfer for Benilite SR, and (iii) environmental impact and cost involved in disposal of the huge chloride contaminated iron oxides/hydroxides generated. For the Ti-Slag (a pyro-metallurgical process), though the chemistry of Odishan ilmenite is ideal and both the products of this process (Ti-Slag and Pig Iron) are equally good and saleable, the cost of power in this extremely power intensive route, holds the key. It is suggested that if cost of power can be reasonably handled, the Ilmenite-TiSlag- TiO_2 Pigment value addition chain can become the success formula for the huge potential of Odishan Ilmenites.

1. INTRODUCTION:

India has a close to 6000km coast line and boasts of a number of potential geological provenance with positive hinterland geology combined with favourable coastal geomorphologic settings for the formation of

economic placer sand deposits. Atomic Minerals Directorate for Exploration and Research (AMD) has identified 2100 line km of the Indian Coast line as mineral sand bearing. Out of Odisha's 470km long coast line, AMD has identified cumulative stretches of 105 line kms as mineral sand bearing.



To a limited extent, economic exploitation of the coastal placer sands are taking place in

Odisha, Andhra Pradesh, Tamilnadu and Kerala while there still exists large

opportunities, particularly after the Oct-1998 policy guidelines of Department of Atomic Energy, permitting private sector participation in the exploitation of placer mineral sands of India. There are also some inland placers and palaeo-shore-bars in India. Except for the inland placers of Tuticorin and Tirunelveli districts in Tamilnadu, all other inland placers are not economically viable at present.

The major significant placer sand minerals in India are Ilmenite, Rutile, Zircon, Garnet and

Sillimanite. However, its only at one recently started Indian operation that the processing facilities for separation of all these 5 minerals are available.

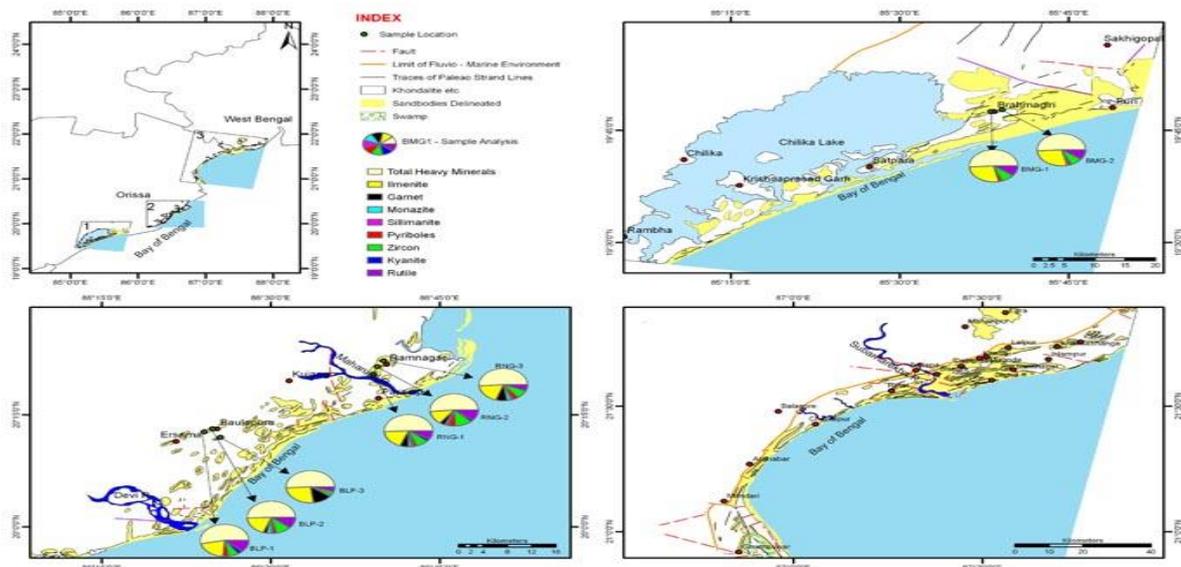
2. MINERAL SAND RESOURCES OF ODISHA:

As can be found from the below table, Odisha occupies a very significant position in the placer mineral sand resources of India (All figures are in Million Tonnes):

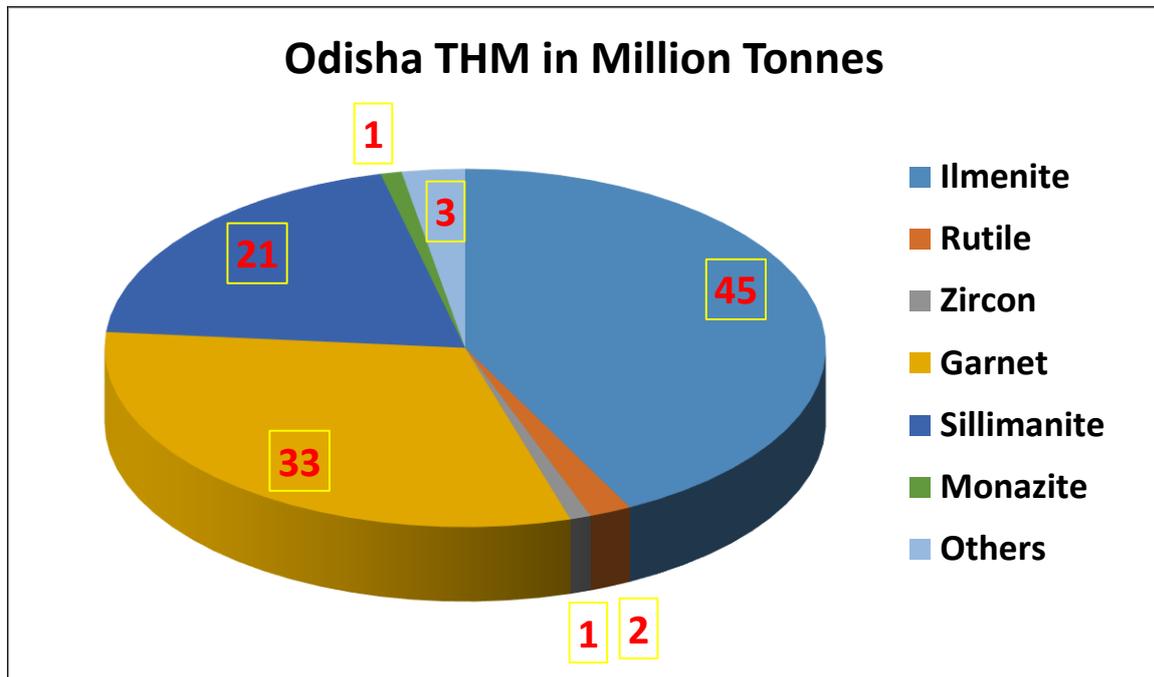
Major Constituents	Indian Resource	Odisha Resource	Odishan Resource as a % of total Indian Resource
Ilmenite	348	45	13%
Rutile	18	2	11%
Zircon	21	1	5%
Garnet	107	33	31%
Sillimanite	130	21	16%
Monazite	8	1	13%
Others	8	3	38%
TOTAL (THM)	640	106	17%

These 17% of total Indian Mineral Sand resource is principally confined to 4 segments of the Odisha Coastline:

- Gopalpur-Rusikulya Segment
- Chilka-Puri-Konark Segment
- Mahanadi Delta Segment
- Subarnarekha Segment



3. INDUSTRY DRIVERS FOR ILMENITE VALUE ADDITION:



The three major constituents in the Odisha Mineral Sands are Ilmenite (45 Million MT), Garnet (33 Million MT) and Sillimanite (21 Million MT).

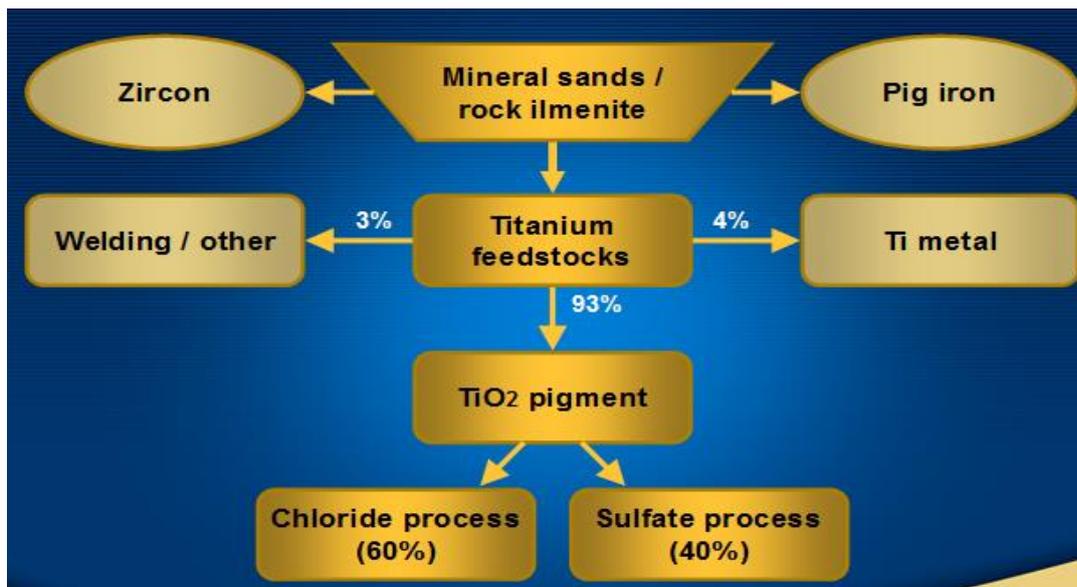
Coarse grained industrial garnets are used in pressure jet cutting and not much scope exists for value addition. Similarly, though the chemical grade of sillimanite is very much suitable for the manufacturing of high grade alumina refractory bricks, the extreme fineness of particles in the Indian placer sands make its use almost restricted, so much so that in India today, of all the mineral sand players, only one recent operator has built floatation facilities for separation of sillimanite. Rest all do not find it commercially prudent to separate sillimanite, and instead allow these to go back with the tailing sand for back filling. At the same time, some R&D initiatives have been seen recently for sintering/pelletising of sillimanite for use in the refractory industry.

In contrast to Garnet and Sillimanite, the value addition possibilities for Indian Ilmenite in general and Odishan Ilmenite in particular, throw up some very excellent opportunities and challenges for the future.

4. VALUE ADDITION POSSIBILITIES FOR ILMENITE:

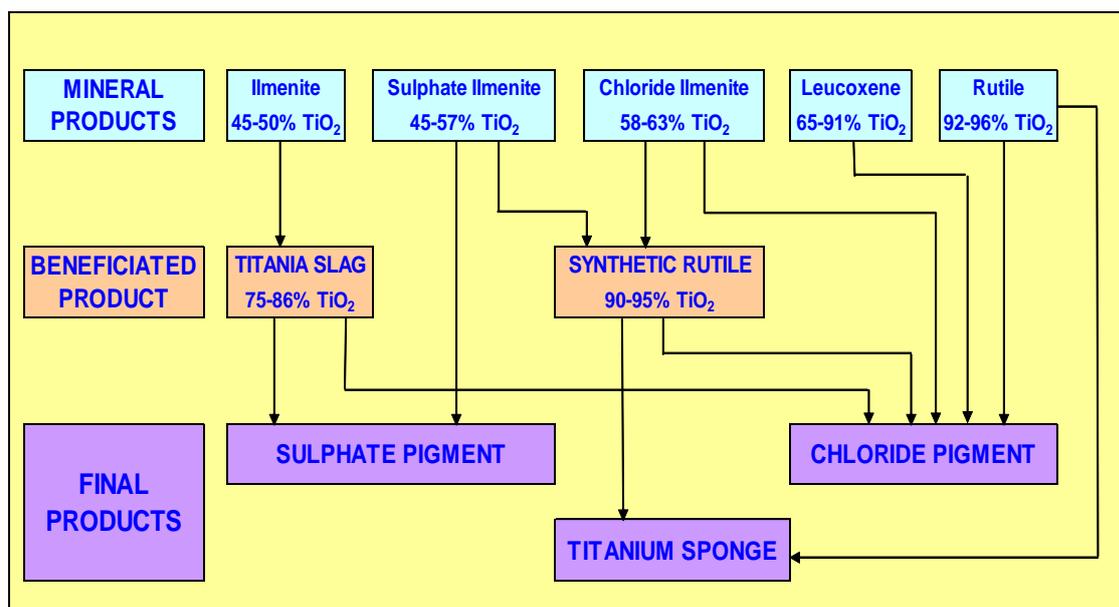
Ilmenite, which is an oxide of Titanium, has good scopes for value addition to upgrade the Ti content from the naturally occurring stage of 48-50% TiO₂ to as high as close to 100% TiO₂, which finds application primarily in the following three market segments:

- Pigment
- Welding Rods
- Metal



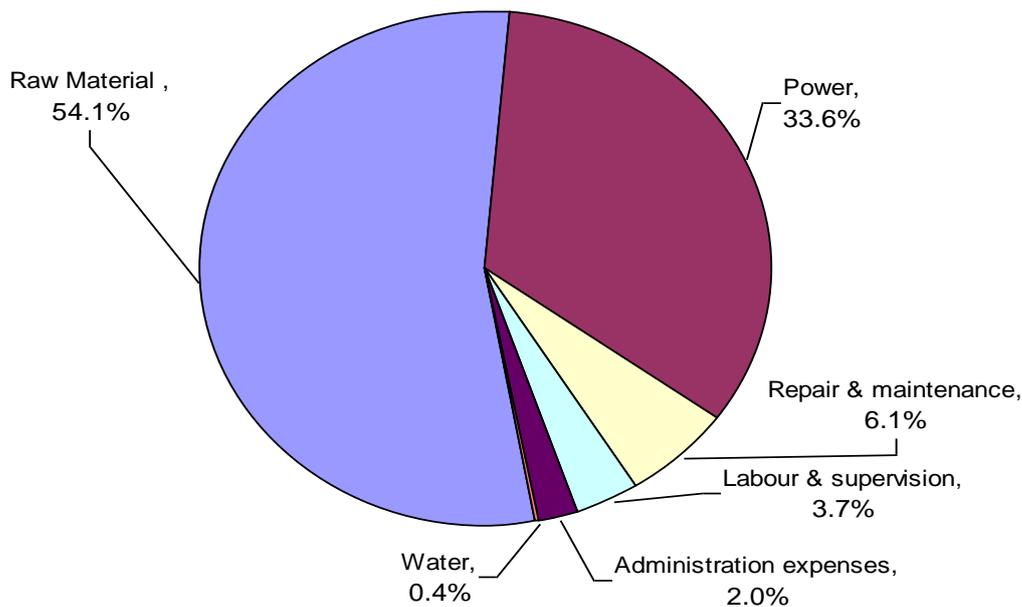
Since more than 90% of titanium feedstock finds use in the TiO₂ Pigment industry, the overwhelming market driver for the value addition of Ti bearing minerals has to be for pigment and related applications industry. As

can be noted from the below chart, there basically exist two options for beneficiation/up-gradation for low grade ilmenites of Odisha; the Synthetic Rutile (SR) route or the Ti Slag route:



Ilmenite is FeTiO₃ or to make it simple FeO.TiO₂. Therefore in the typical Odishan Ilmenite having a TiO₂ level of 48-50%, the balance almost 50% is ferrous oxide. Beneficiation/up-gradation of Ilmenite would essentially therefore mean removal of this close to 50% FeO from the raw ilmenite.

In Ti Slag making, ilmenite is smelted in an electric furnace at around 1650/1700°C temperature to produce a heavier phase at bottom (Pig Iron) and another lighter phase at top (Titanium Slag). Both the products are saleable. However, since this process is very power intensive, and power consumption alone contributes to 34% of the production cost, cost of power becomes a game changer for Ti Slag making process.



In the Synthetic Rutile (SR) route, the close to 50% FeO in Ilmenite is removed through a complex chemical process involving corrosion or acid leaching (depending on whether the process is Becher Process or Benilite Process), producing a high Ti (around 90% TiO₂) product called Synthetic Rutile (SR) and another huge volume by-product of Iron Oxide/Hydroxide in a chloride brine slurry. To date, no commercially viable significant use for such chloride contaminated iron has been developed. Moreover, disposal of such materials require extra precaution and therefore involves substantial additional costs.

5. OPTIMUM OPTIONS FOR ODISHA:

Ilmenites of Odisha having TiO₂ levels of 48-50% are technically equally amenable for upgradation/beneficiation through chemical process (corrosion or leaching) to produce Synthetic Rutile (SR) or through pyro-metallurgical process (smelting) to produce Ti Slag.

There are two commercially active SR process; the Lurgi Becher and the Benilite. The Becher SR process has major process issues in the form of handling and transportability of the special coal required. Benilite Corporation of USA, which had the original patent for the Benilite SR process, has since closed down and the patent expired.

Therefore, the knowhow for Benilite SR process now rests with employees/ex-employees of Kerr-McGee, raising a risk issue of reproducibility at higher scales of operations, for any new venture. SR process has also another issue of handling and disposal of the huge chloride contaminated iron oxides/hydroxides, generated while upgrading the Ilmenite.

The pyro-metallurgical process of smelting the Ilmenite to produce Ti Slag and a highly premium quality pig iron, seems ideally suited for the Odishan Ilmenite, which has TiO₂ and FeO almost at a proportion of 1:1. For not only one gets two equally valuable products, the additional worry and cost of by-product disposal, as in the case of SR, is completely absent. However, the moral of the Ti-Slag story is almost exclusively dependant on the cost of power, and there lies the secret to success.

6. CONCLUSION:

Odisha has a mineral sand potential of 106 million MT total heavy minerals (THM), comprising of Ilmenite, Rutile, Zircon, Garnet and Sillimanite. The 45 Million Tonnes of Ilmenite resource is of comparatively lower grade with around 48-50% TiO₂ levels, but has potential value addition options because of the market driver TiO₂ Pigment's commercial

position as a life style commodity. Two alternatives for intermediate upgradation/beneficiation namely the chemical processing to Synthetic Rutile (SR) or the pyro-metallurgical processing to Titania Slag (Ti-Slag) are available. However, considering the Ilmenite chemistry and environmental implications, the Ti-Slag route is thought to be better suited for Odishan Ilmenite, provided of course the cost of power is kept within acceptable limits.

Ilmenite-TiSlag-TiO₂ Pigment therefore looks like holding the secret to the success of Odishan Ilmenite resources.

7. DISCLAIMER AND ACKNOWLEDGEMENTS:

Permission by the Trimex Management to present this paper is thankfully acknowledged.

Some of the figures and inputs for this paper have been drawn from publications available in public domain namely (i) Exploration and Research for Atomic Minerals, Vol-13, Aug-2001, AMD Hyderabad, (ii) K. Jagannadha Rao et al, Current Science, Vol. 94, No. 8, April-2008, (iii) Publications of TZMI, Perth, Australia.

All these above are very thankfully acknowledged.

It is being unambiguously declared that the views expressed in this paper are that of the author himself and not necessarily of Trimex Sands for whom the author works.

ACID COAL MINE DRAINAGE: AN OVERVIEW

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ABSTRACT

Mining of coal is carried out in many parts of the world because of its large demand as an important industrial commodity. Exposure of coal and the associated minerals to the atmosphere because of mining results in the weathering of some of the constituent minerals contaminating the water regime in and around the mining area by the weathering products. The weathering of different sulfide minerals particularly pyrite, a common metallic mineral constituent in coal deposits, leads to the generation of acid which can contaminate mine drainage water and adversely affect the environment in the long run. Assessment of the acid generation phenomenon and prediction of mine drainage characteristics are helpful in deciding suitable mitigation measures to combat the environmental effects. This paper attempts to present a comprehensive picture of acid coal mine drainage phenomenon with respect to the genesis of the problem, assessment and prediction of mine drainage quality and, the approaches towards prevention and mitigation of the problem.

Key words: Coal mining, pyrite oxidation, acid mine drainage, acid base accounting, Abiotic remediation, biological remediation.

1. INTRODUCTION

Coal is an important industrial commodity largely due to its use in energy and steel sectors. It is the major fuel used for generating electricity worldwide. Because of these reasons and also its abundance, coal is mined in large quantities in various parts of the world and coal mining is an important economic activity today. According to some recent statistics [World Coal Association, Coal Statistics 2013], the global coal production in 2012 was 7831 Mt and coal provided ~ 30 % of global primary energy requirement, generated ~ 41 % of the world's electricity and was used in almost 70% of the total global steel production.

The use of coal as fuel, on one hand, is one of the major reasons of green house gas production contributing to environmental pollution. On the other hand coal mining, like many other mining activities, has various adverse impacts on the environment. Surface coal mining in particular requires large areas of land to be temporarily disturbed leading to environmental challenges such as soil

erosion, dust, noise and water pollution and impact on local biodiversity. Coal, in addition to carbonaceous matter, contains different minerals. Surface mining activities expose these minerals to an oxidizing environment because of which a complex series of chemical weathering reactions are spontaneously initiated since the minerals contained in the spoil are not in equilibrium with the oxidizing environment [Deutsch, 1997]. The reactions, however, take place at much faster rates compared to natural "geologic weathering" which takes place over extended periods of time (i.e., hundreds to thousands of years). The accelerated reactions can release substantial quantities of acidity, metals, and other soluble components into the environment contaminating the mine drainage. This phenomenon, termed as Acid Mine Drainage (AMD), gives rise to several problems of environment degradation, especially pollution of aqueous environments. AMD contamination to shallow ground water system and soil water zone makes the water bodies corrosive and unsuitable to support various forms of

aquatic life [Jennings et al. 2008]. As far as coal mining is concerned AMD occurs mainly in abandoned mines and has been a detrimental by-product for many years because of the increasing number of closed mines and the eventual closure of whole coal fields. Coal stocks, coal handling facilities, coal washeries and coal waste heaps can also be a source of acid production. At present, AMD continues to pose a potential problem in many areas, despite improved prediction techniques and mitigation methods. Several case studies have been reported highlighting the impact of mine drainage on water quality [Tiwari and Dhar 1994, James et al. 2000, Swer and Singh, 2004, Ochieng et al. 2010, Munnik 2010, Mohapatra and Goswami 2012]. The aspects of prediction, prevention and mitigation have been discussed individually in detail in several publications [Kleinmann, 2000], [Johnson and Halberg, 2005], [Akeil and Koldas 2006], [Sheoran et al., 2011]. In this article an attempt has been made to present a comprehensive picture on the genesis of the AMD, prediction of its quality and approaches to its prevention and mitigation.

2. GENESIS OF THE PROBLEM

Acid mine drainage is produced mainly when sulfur bearing materials, especially sulfides, are exposed to oxygen and water because of mining. As far as coal mining is concerned AMD is due mainly to weathering of pyrite although other minerals present in the ore body can also contribute to the phenomenon. Since acid present in coal mine drainage is due to weathering of mineral constituents it is worthwhile to have a look at the composition of coal with respect to mineral constituents.

2.1 Origin and Composition of Coal

Coal is a sedimentary material that develops from the settling of plant materials and subsequent transformation under the pressure of the overburdens. It comes in four major types i.e. lignite (brown coal), sub-bituminous (black lignite), bituminous (black coal) and anthracite and contains moisture, volatile substances, carbon, sulfur and inorganic matters [World Coal Institute 2004], the amount of the constituents varying with the type of coal, location of deposit etc.

i. Sulfur in coal

The total sulfur present in coal and associated strata occur both organic and inorganic forms the latter as sulfate sulfur and sulfide sulfur [Rawat and Singh 1983]. Organic S is complexed within the coal plant material and is organically bound within the coal. This form is only found in appreciable quantities in coal beds and in other carbonaceous rocks. Sulfate S is usually only found in minor quantities in rocks of humid areas, but can be present in substantial amounts in more arid areas where leaching is limited. Sulfide S is the predominant S species in the majority of overburdens.

ii. Inorganic constituents of coal

The inorganic constituents are mainly iron, aluminum, sulfur and phosphorous present in different mineral forms. Although more than 120 different minerals have been identified in coal samples, only about 33 of these occur in most coal samples; of these, only about eight generally are abundant enough to be considered as major constituents. Some of these minerals are shown in Table 2 [Schweinfurth 2009].

Table 1.1 Common major minerals (In general order of abundance) found in coal.

Mineral name	Chemical composition
Quartz	SiO ₂
Clay minerals:	
Kaolinite	Al ₂ Si ₂ O ₅ (OH) ₄
Illite	KAl ₄ (AlSi ₇ O ₂₀)(OH) ₄
Montmorillonite	(¹ / ₂ Ca,Na) _{0.7} (Al,Mg,Fe) ₄ [(Si,Al) ₄ O ₁₀] ₂ (OH) ₄ •nH ₂ O
Chlorite	(Mg,Al,Fe) ₁₂ [(Si,Al) ₈ O ₂₀](OH) ₁₆ . (May have Mn. (Clays may also contain Be, Cr, Ni, and other trace elements.)
Pyrite	FeS ₂ (May contain As, Cd, Co, Hg, Ni, Sb, and Se.)
Calcite	CaCO ₃
Siderite	FeCO ₃ (May contain Mn).

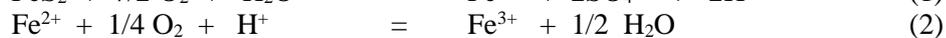
Iron is the major constituent of the mineral matter and occurs mainly as sulfide. The iron disulfide minerals pyrite (FeS₂) and, less commonly marcasite (a metastable phase of FeS₂), are the principal sulfur-bearing minerals in bituminous coal [Montano 1981]. Other iron bearing minerals detected in some coal samples are pyrrhotite (FeS), szomolnokite (FeSO₄•H₂O) and FeCO₃ (siderite). Ferric sulfates commonly observed in several coals are coquimbite and jarosite. Aluminum in coal comes mainly from the clay minerals illite and kaolinite.

Sulfide mineral oxidation, primarily of pyrite and pyrrhotite, due to weathering is the main cause for the generation of acid mine drainage. Arsenopyrite (FeAsS),

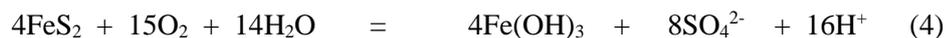
chalcopyrite (CuFeS₂) and other sulfide minerals containing Fe, Cu, As, Sb, Bi, Se and Mo may also produce acidic solutions upon oxidation; but these minerals are uncommon in coal beds.

2.2 Chemistry of Acid Formation

Pyrite oxidation, which is mainly responsible for acid generation, is a multi-step process involving an oxygen-dependent reaction [Lowson 1982] in which oxygen attacks the mineral producing ferrous iron which in turn is oxidized into ferric state by oxygen. Hydrolysis of the ferric iron produces ferric hydroxide (hydrated ferric oxide) and hydrogen ions (Reactions 1 – 3).

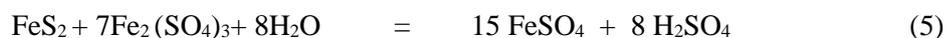


The overall reaction is



At low pH, the rate of oxidative dissolution is controlled by the concentration of ferric iron, which interacts with reactive surface

sites more effectively than oxygen [McKibben and Barnes, 1986].



Reaction (2), i. e. the conversion of ferrous iron generated through reaction (1) or (5) is the key reaction in promoting the ongoing oxidation of pyrite and is the rate limiting step of the overall acid formation scheme [Singer and Stumm, 1970].

Hydrolysis of other metals also results in generation of acid. These reactions commonly occur when acidic water containing substantial amount of metals

blend with cleaner waters resulting in precipitation of metal hydroxides.



3 QUALITY OF MINE DRAINAGE

The quality of mine drainage depends on the speed of the weathering reaction, propagation and concentration of the reaction products. Factors influencing the quality of mine drainage may be classified as primary, secondary and tertiary [Anonymous 1994] whereas downstream factors also influence the quality and quantity of mine drainage. Although oxygen is mainly responsible for initiating the weathering of pyrite the propagation and the speed of the weathering reaction depend on many other factors some of which are briefly discussed here.

3.1 Primary factors

Primary factors influencing the acid generation are those which affect the rate of sulfide oxidation reaction i.e. mineral characteristics (morphology, particle size), presence of water, pH and oxygen concentration of the water phase, ferric iron activity and presence of bacteria to catalyze the oxidation reaction.

i. Pyrite morphology

Ore mineralogy, including the effects of associated sulfide impurities is known to affect the reactivity of pyrite [Cruz et al. 2001]. Pyrite has a structure based on a face-centered cubic array of ions with a NaCl type structure. In some cases Ni, Co, Cu and other metals are found to occur in the pyrite structure substituting for iron [Prewitt and Rajamani 1987]. The trace metal content [Wiesermann and Rimstidt 1984], specific surface area [Moses and Herman 1991] and surface structure [Eggleston et al. 1996] have been reported to be some of the controlling factors in the oxidation kinetics of pyrite. Nature of origin of pyrite affects its reactivity. For example sedimentary pyrite is more reactive than hydrothermal pyrite [Borek

1994]; Pyrite also occurs several different morphological forms and grain sizes ranging from micro-size to millimeter (or large) crystals. Among the easily recognizable forms, 'framboidal' and 'euhedral' forms are quite common in coal whereas coarse grains can also exist [Ryan 1997]. The "framboidal" form is characterized by a small grain size (< 1µm) and large surface area and the 'euhedral' crystals are of larger size (1-10 µm). The former is much more chemically reactive than the latter coarser form [Evangelou, 1995]. Morison [Morison, 1988] defined nine classes of pyrite morphology, end members being framboidal and euhedral structures. The reactivity of fine grained pyrites reflects the fact that acid generating reactions occur at the mineral surface. Previous research [McKibben and Barnes 1986] indicates that differences in grain size (i.e., surface area) can exert significant control on pyrite oxidation rates. A later study using potassium dichromate as oxidant [Chirita 2003] has supported this.

ii. Presence of water and oxygen

In order for pyrite to oxidize, both oxygen and water must be present. Water serves as both a reactant and a medium for bacteria in the reaction process and also transports the oxidation products [Forstner and Salomons 1988]. A ready supply of atmospheric oxygen is required to drive the oxidation reaction. Oxygen is important to maintain the rapid bacterially catalyzed oxidation at pH below 3.5 [Anonymous 1994]. Oxidation of sulfides is significantly reduced when the concentration of oxygen in the pores of the mining waste units is less than 1 or 2 percent.

iii. Presence of microorganisms

The conversion of ferrous to ferric iron in the overall pyrite reaction sequence can be greatly accelerated by a species of bacteria, *Thiobacillus ferrooxidans*. This bacterium and several other species thought to be involved in pyrite weathering are widespread in the environment. *T. ferrooxidans* has been shown to increase the iron conversion reaction rate by a factor of hundreds to as much as one million times [Singer and Stumm, 1970; Nordstrom, 1979]. Different bacteria are better suited to different pH levels and other edaphic factors (the chemical and physical characteristics of the water and soil environments). At pH levels between 3.5 and 4.5, iron oxidation is catalyzed by a variety of Metallogenium, a filamentous bacterium. Below a pH of 3.5 the same reaction is catalyzed by the iron bacterium *Thiobacillus ferrooxidans* [Anonymous 1994]. Thus, once pyrite oxidation and acid production has begun, conditions are favorable for bacteria to further accelerate the reaction rate. At pH values of about 6 and above, bacterial activity is thought to be insignificant or comparable to abiotic reaction rates. The catalyzing effect of the bacteria effectively removes constraints on pyrite weathering and allows the reactions to proceed rapidly. The role of microbes in pyrite oxidation is described in more detail by Kleinmann et al. [Kleinmann et al. 1981] and Nordstrom [Nordstrom 1979].

iv. Depositional environment

Pyrite is formed in reducing environments (provided by easily decomposable organic matter) with a continuous supply of sulfate and iron. The major steps in sequence for pyrite formation are: reduction of sulfate to sulfide (hydrogen sulfide) by bacteria, reaction of hydrogen sulfide iron minerals to form iron monosulfides and reaction of iron monosulfides to form pyrite [Berner 1970]. Paleoenvironment of the coal deposit – that is, what the region was like when the organic-rich sediment that became the coal was first deposited, plays an important role with respect to its pyrite content and also pyrite morphology. Paleoenvironments of coal can be characterized into three general categories: marine; freshwater; and brackish. In marine

sediments the rate of sulfate reduction is low due to limited supply of organic carbon. Pyrite formation in such environments is slow and gives rise to framboidal type [Berner 1984]. Studies of Pennsylvanian age coal bearing rocks have shown that paleoenvironment can be used to broadly define acid drainage potential [Brady et al. 1988; Hornberger et al. 1981]. Rocks formed in brackish water conditions are generally most prone to acid production [Caruccio et al., 1977, Williams et al., 1982]; freshwater systems usually produce non-acid water, and marine systems produce variable drainage quality. In some coal measures, the paleoenvironment varies laterally and vertically within a single mine site and is a controlling factor in the inherent distribution of pyrite and carbonates.

3.2 Secondary Factors

Secondary factors control the consumption or alteration of the products from the acid generation reaction. They act to either neutralize the acid produced by oxidation of sulfides or may change the effluent character by adding metals ions mobilized by residual acid. The most important secondary factor is the neutralization of acid by alkalinity released from carbonate minerals in the mine waste, such as calcite (CaCO_3) and dolomite [$\text{CaMg}(\text{CO}_3)_2$]. Neutralization of acid released by reaction with carbonate minerals is an important means of moderating acid production [Forstner and Salomons 1988]. Products from the oxidation reaction (hydrogen ions, metal ions, etc.) may also react with other non-neutralizing constituents. Possible reactions include ion exchange on clay particles, gypsum precipitation, and dissolution of other minerals. Dissolution of other minerals contributes to the contaminant load in the acid drainage. Examples of metals occurring in the dissolved load include aluminum, manganese, copper, lead, zinc, and others [Ferguson and Erickson 1988].

3.3 Tertiary factors

Tertiary factors include the physical characteristics of mining waste, the spatial relationship between wastes, and the hydrologic regime. The physical characteristics of importance are particle size, physical weathering tendency and permeability. The particle size relates to the amount of surface area that is exposed to oxidation. The smaller the particle size, the more total surface area is exposed, and the greater potential for oxidation. Weathering increases the surface area of the material. The increase in surface area and the physical weathering and fracturing of strata increase the permeability of the waste material.

The sequence of stacking different wastes may affect the water quality of mine drainage. By contributing to the alkalinity of the percolating water, calcareous material placed atop pyritic material will reduce both the potential for oxidation and the acidity generated [Forstner and Salomons 1988].

The hydrologic regime of a mine can influence the quality of mine drainage. When acid generating material occurs below the water table, acid production is retarded because of low availability of oxygen owing to its slow diffusion in water. This is reflected in the portion of pits or underground workings located below the water table. Where mine walls and underground working areas extend above the water table, the flow of water and oxygen in joints may be a source of acid [Anonymous 1994].

3.4 Downstream factors

Physical processes such as dilution and precipitation and chemical processes such as neutralization will permit a stream to assimilate acid drainage, but not without incurring a great deal of acid damage to the preceding stream area [Forstner and Salomons 1988].

4 PREDICTION OF ACID DRAINAGE QUALITY

The impact of AMD on the environment can be reduced through advanced planning for its prevention and control if the quality of the AMD is predicted accurately. Prediction of the potential and magnitude of AMD helps in identifying the risks and liabilities and selecting effective waste handling strategies. Methods to predict acid mine drainage can be divided into five groups [Ferguson and Erickson 1987].

4.1 Geographical mining comparisons

Geographical mining comparisons involve examining and sampling drainage from abandoned or operating mines near the project under study. Simple extrapolations are made for similar mining operations in related geological environments.

4.2 Paleoenvironmental and geological models

Paleoenvironmental models examine the regional depositional environment of the sulfide minerals. As mentioned earlier, pyritic sulfur formed in marine or brackish water may have a greater tendency to generate AMD than that formed in fresh water environments. Geological models of a sedimentary sequence or igneous/metamorphic ore body are valuable in estimating the location and size of potentially acid generating zones.

4.3 Geochemical Static Tests

These are screening methods to determine the acid generating capacity and acid neutralizing capacity and to predict the extent of acid generation from the values of these parameters. Many predictive static tests have been applied in recent years to evaluate the acid-forming potential of samples [Chotpantararat 2011].

4.3.1 Paste pH

Paste pH is a relatively rapid, simple method used to assess the readily available acidity/alkalinity of a sample and is easily conducted in the field [Sobek et al. 1978]. The sample is mixed with deionized water at a ratio of 1:1 or 1:2 shaken for 2 h and the pH of the solution is measured. Samples having paste pH < 4 are considered potentially acid forming. Because the test is of short duration (<12 hr) and non-vigorous (de-ionized water), only soluble salts and reactive minerals are assessed. Minerals typically assumed to be assessed by paste pH tests include acid generating sulfate salts such as melanterite, reactive sulfides such as greigite, and high surface area pyrite and carbonate [Weber et al. 2006].

The paste pH test has several limitations. It only indicates the number of free hydrogen ions in the prepared sample. Since pyrite oxidation reactions are time dependent, the paste pH results provide inadequate information on the reactivity of pyrite contained in the sample and hence, on the propensity of a sample to produce acid mine drainage. In fact, the paste pH of un-weathered, high-sulfur sample is likely to be near that of the de-ionized water, while a weathered sample with relatively low percent sulfur, but which includes a small amount of residual weathering products, may have a significantly depressed paste pH [Kania 1988].

4.3.2 Acid Base Accounting

Acid-base accounting (ABA) is an analytical procedure that helps to evaluate the properties of overburden rock prior to coal mining relative to its potential for immediate and future pollution by taking into account its acid-producing and acid-neutralizing capacity [Skousen et al., 2002]. To predict the post-mining water quality, the amount of acid-producing rock is compared with the amount of acid-neutralizing rock, and a prediction of the water quality at the site (whether acid or alkaline) is obtained. The method, [Sobek et al. 1978], is the most common practice for prediction of post mining drainage water quality. ABA is based on the principle that the possibility for a site to produce acid mine drainage can be

predicted by quantitatively determining the total amount of acidity and alkalinity the strata on the site can potentially produce.

Components of acid base accounting measurements adopted for use in coal mining, processing and other applications are:

i. Acid production potential (APP)

APP, also termed as maximum potential acidity (MPA) is the maximum amount of sulfuric acid that can be produced from the oxidation of sulfur minerals in the rocks or overburden material. APP is determined by multiplying the total sulfur percentage of the sample by 31.25, the stoichiometric conversion factor determined from equation (1). Although acid production is associated mainly with pyritic sulfur, total sulfur value is often used to calculate MPA [Kania 1998] because its measurement is easier compared to that of pyritic sulfur. APP is reported either as kilograms of CaCO₃ equivalent per metric ton of sample, or tons of CaCO₃ per thousand tons of waste.

ii. Neutralization potential (NP)

The NP is a measure of the amount of acid neutralizing compounds (mostly carbonates, exchangeable alkali, and alkaline earth cations) present in the coal and overburden. The NP is calculated from the quantity of acid neutralized by a specific quantity of the sample (after digestion in HCl followed by back-titration with NaOH to pH 7) [Sobek et al., 1978] and, as in the case of APP, is reported either as kilograms of CaCO₃ equivalent per metric ton of sample, or tons of CaCO₃ per thousand tons of waste [Kania 1998]. Prior to neutralization a Fizz rating test [Sobek et al. 1978] is performed to decide approximately the quantity and concentration of acid to be used in the test. There are several procedures for determining NP which are slightly different individually but all follow the steps described above.

iii. Net neutralization potential (NNP)

The NNP also termed as net acid production potential (NAPP), is a measure of the sample's overall acid generating capacity. It is the difference between the inherent acid neutralization capacity (NP) and the maximum potential acidity (MPA) of the sample expressed in kg H₂SO₄/t equivalent.

$$\text{NNP} = \text{MPA} - \text{NP}$$

A negative NAPP indicates that the sample has a net neutralizing capacity and a

positive NAPP indicates that the sample has a net acid generating capacity.

iv. Neutralization Potential Ratio (NPR)

$$\text{Neutralization Potential Ratio (NPR)} = \text{NP/MPA}$$

A generally accepted guideline for defining strata as either acid-generating or alkaline is given in Table 3.1.

Table 3.1 Accepted guide line for interpretation of NP and NNP values in predicting AMD quality

	Acidic	Undecided	Alkaline
NP, Tons/1000 t	<10	10 - 21	> 21
NNP, Tons/1000 t	< 0	1 - 12	> 12
NPR	< 1	1 - 3	>3

3.1.3 Net Acid Generation (NAG) test

Net acid generation (NAG) test [Miller et al. 1997], a modification over the ABA method, determines the acid rock drainage characteristics based on the complete oxidation of the sample's sulfide content (as well as ferrous iron from siderite dissolution) using hydrogen peroxide. Acid that is produced by oxidation is consumed simultaneously by carbonates and/or other acid consuming components of the material. After measuring the pH of the solution (NAG pH) any net acid remaining after the reaction is determined by titration with standardized NaOH solution. It is observed that both net acid production potential (NAPP) and net acid generation (NAG) test results used in combination provide a more reliable routine screening technique than either test alone, increasing confidence in acid potential classification [Stewart et al. 2006].

3.1.4 ASTM Carbon/Sulfur Methods

The overburden sample is combusted under oxygen flow for complete conversion of any carbon or carbonaceous material to CO₂ and any S compounds (organic, sulfate, or sulfide) to SO₂. The SO₂ is measured by various methods e.g. ASTM D 3177 and ASTM D 4239 A, B and C. The

total sulfur value computed from such measurements is used in the acid base accounting process.

4 GEOCHEMICAL KINETIC TESTS

The kinetic tests (leaching tests) consist of the simulation of mine drainage production from samples of strata to be affected by mining, followed by chemical analyses of effluent quality produced from these simulated conditions. These tests are designed and conducted to deal with the reaction kinetics, the rates and mechanisms of the chemical reactions which lead to the production of acidic or alkaline mine drainage and they incorporate dynamic elements of the physical, chemical, and biological systems and processes which control the production of mine drainage. Kinetic tests are commonly run as a companion to static testing to measure the weathering behavior of geologic material when exposed to field conditions. Kinetic test may be run in a laboratory or in the actual field. The quality and quantity of leachate is subsequently evaluated to offer a supporting interpretation to static testing. Brief description of some of the important kinetic tests follows [Kleinmann 2000].

4.1 Humidity cell test

It is the most popular kinetic test. The test [Sobek et al. 1978] simulates accelerated weathering of the sample. The sample is placed in a chamber and moist air and dry air are passed in succession for three days each followed by distilled water on the seventh day. This one week cycle is run for twenty weeks as per ASTM D 5744-96 [Lapakko and Antonson 2006].

4.2 Soxhlet extraction

Soxhlet extraction tests [Renton et al. 1988, Sobek et al. 1982] are conducted in a soxhlet apparatus in which $-125\mu\text{m}$ size rock sample is subjected to cyclic flushing with boiling water for several hours. Thereafter the sample containing thimble is stored in a drying oven for two weeks at 105°C that brings about oxidation of the pyretic material. Usually the acid potential is determined after five or six cycles of leaching and oxidation.

4.3 Column tests

Column tests are conducted by staking the waste in a cylinder or similar container, wetting the sample with water or required solution and analyzing the drain water for desired constituents. Wetting and drying cycles may from several days to weeks. Results of research indicated that column tests for well-stored tailing materials > 0.5 cm in diameter accurately represent field conditions [Bradham and Caruccio 1990].

4.4 British Columbia research confirmation test

The test is usually carried out if the sample is shown to be potentially acid producing in static testing. The acid potential derived by assuming total oxidation of sulfur (sulfides) in static prediction testing may not be realized under field conditions. To determine the degree to which the sulfur content of a sample might be oxidized, and to assess if this amount of acid is sufficient to overcome the neutralizing capacity of the sample, a biological oxidation test can be carried out.

A pre-acidified pulp containing the finely ground test sample is inoculated with an active culture of sulfide-oxidizing bacteria such as *T. ferrooxidans* and agitated under ideal conditions for bacterial oxidation. The pulp pH is monitored till it becomes stable indicating the end of oxidation. The same weight of sample is then added in two installments after 24 and 48 hours and the pH values are measured 24 hours after each addition. If the pH is above 3.5 at these times, the sample is classified as a non-acid producer, since this pH value does not support the growth and oxidative activity of the bacteria. If the pH remains below 3.5, a potential for acid generation is indicated [Bruynesteyn and Hackl 1984].

4.5 COASTECH modified biological oxidation test

This is a modification of the British Columbia Research Confirmation Test which overcomes some of the disadvantages of the latter [Lawrence et al. 1986]. The initial sample weight is selected on the basis of the sulfur content as shown in Table 4.1 [Lawrence, R.W., and Sadeghnobari, A., 1986b] (basis: 50 ml nutrient solution). At the end of the test, following the full addition of extra sample, if the pH is still below 3.5, sodium hydroxide solution (3 to 6 N) is added to the pulp, stoichiometrically equivalent to the acid added initially to bring the pH into the biochemical oxidation range. The final pH is recorded after 1 hour. This procedure removes the bias towards an acid classification.

Table 4.1 Sample Weight for Various Sample Sulfur Contents, Coastech Modified Biological Oxidation Test

% Sulfur	Sample Weight, g
<4	10.0
4-8	7.5
8-15	5.0
>15	3.0

4.6 Batch Reactor (Shake Flask) Tests

In the Batch Reactor test, like the British Columbia Confirmation test, a slurry made of mine sample and water (usually distilled water with added nutrients if necessary) is shaken continuously for the test period and water samples are taken at regular intervals to determine water quality parameters such as pH, sulfate, and metals in solution. Data from the tests are used to estimate the rate of sulfide mineral oxidation and release of contaminants, such as metals [Anonymous 1994].

The batch reactor is relatively simple and allows examination of multiple factors, such as pH and temperature, which can be tested simultaneously. The influence of bacteria and control measures may be used as test parameters. The primary difficulty with the method is that the duration of the test may not exceed the lag time prior to acid formation (Lapakko 1993b). Other concerns are that the water volume in the flask may inhibit acid formation and bacteria may not acclimate in the test conditions [Anonymous 1994].

4.7 Field Scale Test

Field Scale Testing, use large volumes of material to construct test cells in ambient environmental conditions, typically at the mine site in question. In these tests sample size may be as much as 1000 metric tons or more, depending on space availability. Particle size of the test material is not usually reduced for the test to better approximate field conditions. The sample is loaded on to an impervious liner to catch solutions and a vessel is used to collect the leachate which is analyzed for pH, sulfate, dissolved metals, and other parameters. Consideration of climatic conditions is important when evaluating results from field scale tests. Climatic effects, especially precipitation, determine the flushing rate but do not influence either reaction rate or the subsequent chemical composition of the leachate. (British Columbia AMD Task Force 1989).

Field scale tests have the advantage of being conducted under the same environmental conditions as the waste or other units they are simulating. They also

allow monitoring of the influence of bacteria and control measures. Drawbacks to field tests are that they require long test durations. Unlike other kinetic tests, field test do not accelerate environmental conditions, which tend to assess the potential to generate acid more quickly. Consequently, field tests will provide information on acid generation potential for a mine waste unit for that amount of time that they are started before waste emplacement begins. For some operations this may be 10 years or more and test results may be used to optimize reclamation design (Lapakko 1993b).

4.8 Simulated weathering tests

Another way is leaching the rock with a strong oxidant or with water for a long time. Hydrogen peroxide has been used as an oxidant to speed the reaction process. There is no standard leaching technique; the various methods have different flushing cycles, use different amounts of water per flushing cycle, use different particles size of the overburden material etc.. A relatively short term, standardized accurate technique is needed to resolve questions concerning acid potential in situation where acid base accounting fails to provide adequate answer.

4.9 Advantages of kinetic tests

A major advantage of kinetic tests for the prediction of mine drainage quality is that, since these types of tests produce an effluent of simulated mine drainage quality, the effluent may be tested for the same water quality parameters as the actual mine drainage to be produced from the proposed mining operation. The water quality parameters typically included in the leachate analyses are pH, acidity, alkalinity, sulfates, iron, manganese and aluminum. These are the same water quality parameters typically monitored as required by NPDES permit conditions for an active mining operation. If the physical, chemical, and biological conditions of the kinetic tests are representative of those found in the mine environment, the concentrations of the water quality parameters in the leachate may be used to

predict or estimate the concentrations of these parameters in the actual mine drainage from the proposed mining operation.

4.10 Disadvantages of kinetic tests

Unfortunately, there are several potential disadvantages of some kinetic tests for the prediction of mine drainage quality. A major problem with these types of tests is the interpretation of the results of the experimental process and the extrapolation to the actual environment of the proposed mine site. It is easy to be fooled by the laboratory results, where test procedures are simplified or some component of the physical, chemical, or biological systems is either underestimated or overestimated. Then, the apparently precise laboratory analyses may not be accurate, and in fact, may be meaningless. Further, it is difficult enough to test a single representative sample of a lithologic unit. Some kinetic tests attempt to combine numerous strata in the same test apparatus to simulate the configuration of the active mine or backfilled mine spoil. That is a very difficult, if not an impossible task, within the average laboratory kinetic test apparatus.

The overall objectives of these test procedures are to determine aspects of the ARD evolution trend such as the overall acid nature, lag period to acid onset, ARD duration and geochemical acid-base kinetics

5. MANAGEMENT OF ACID MINE DRAINAGE

In comparison to many other pollution problems, AMD causes more serious threat to the environment because the pollutants present in it, especially metal ions, will remain in the environment in one form or another without being broken down. Without proper treatment there will be no controls as to where these polluting species, either concentrated or dispersed, will deposit and the local environment will suffer the effects of the pollution over an extended period of time. This warrants the treatment of AMD and there are a number

of methods which have successfully been applied commercially or are in the stage of development. A review by Johnson and Hallberg [Johnson and Hallberg 2005] gives a vivid account of such methods. Various case studies have demonstrated the application of such methods [Baruah et al 2010].

It is not within the scope of this article to give a detailed picture of the control and remediation measures for AMD. They are mentioned only briefly.

5.1 Source Control

It is always desirable to prevent or minimize the generation of acid mine drainage waters so that the problem is less serious at a later stage. Several options, collectively known as 'source control', are available for this. Source control primarily aims at preventing or controlling pyrite oxidation which is the principal cause of acid generation. The effectiveness of prevention depends on the nature of the mine and the geological characteristics of the strata. Active research on controlling pyrite oxidation can be represented by three classes of methodology [Kleinmann 1990]. Kleinmann LP, 1990, At-source control of acid mine drainage; International Journal of Mine Water, vol. 9 (1-4), p 85-96.

5.1.1 Barrier method

These methods are intended to isolate pyrite from weathering reactants or from the hydrologic transport system. Since oxygen and of course water are the most essential components to initiate and propagate acid drainage, preventing the contact of one or both of them with the mine waste can prevent or reduce AMD. Preventing water from reaching underground mines involves the use of diversion ditches and pipes to divert water from acidic areas. Mine reclamation is another approach. Mine reclamation practices can influence post mining water quality and the extent to which acid mine drainage is generated. Regrading and revegetation of a mine site often improve the drainage water quality probably due to physical restriction of the system.

Compaction of the material reduces exposure of pyrite surface to oxygen and water. Covering the spoil with soil or a soil substitute and vegetation also increases evapotranspiration and restricts migration of water and possibly oxygen, into the pyritic zone. However, conventional reclamation is not a long term solution and cannot control strongly acid producing materials. For example, Fugill and Sencindiver [1986] reported that initial decrease in acid loads from vegetated coal refuges lasted less than two growing seasons.

Another approach is flooding and sealing of abandoned underground mines. While the dissolved oxygen in the flooding water is consumed by microorganisms present therein, it is not replenished due to sealing of the mine. Underwater storage of mine tailings can also prevent contact between minerals and dissolved oxygen (Li et al 1997 cf). Land based storage in sealed waste heaps is another method. The spoil material is stored over an impermeable base and covered with layer of organic matter, clay, and soil at different layers.

As far as isolation from oxygen is concerned techniques that isolate pyrite from water also can be applied, but they do not typically reduce oxygen sufficiently to bring down the pyrite oxidation rate [Kleinmann 2006].

5.1.2. Chemical method

Chemical methods alter the composition of the spoil water solution, limit reactant availability or serve to passivate the pyrite surface to limit its oxidation. The AMD system is disrupted by the addition of chemicals to raise the pH of the medium thereby producing an environment unfavourable for pyrite oxidation. Near neutral pH inhibits the activity of iron-oxidizing bacteria and cause ferric iron to precipitate as hydroxide that forms a coating on the pyrite surface. Most widely used abatement chemicals are alkaline compounds caustic soda (NaOH), Lime stone (CaCO_3), quick lime (CaO), hydrated lime ($\text{Ca}(\text{OH})_2$). Both laboratory and field studies have shown lower AMD production

at high pH Nicholson et al. [1988]. Another method involving blending of acid generating material with acid consuming materials to form an environmentally benign composite was reported by Mehling et al. [1997]. A solid phase phosphate (such as apatite) was added to pyritic mine waste in order to precipitate ferric iron as insoluble ferric phosphate. The oxidation of sulfides by ferric iron is thus prevented as the availability of ferric iron for the purpose is less.

5.1.3. Bacterial inhibition method

These methods represent a separate class of treatment that disrupts the cyclic ferrous to ferric oxidation process catalyzed by iron and sulfur bacteria (Reaction). Various laboratory and field studies have been carried out using anionic surfactants such as sodium lauryl sulfate (SLA) and alkyl benzene sulfate (ABS) which highly toxic to iron and sulfur oxidizing bacteria. However, such methods appear to be expensive as they afford only short term control of the problem and require repeated application of the chemicals [Loos et al 1989],

5.2 Remediation

Because of practical difficulties source control or preventive methods are not enough to tackle the problems of AMD and various remediation or 'mitigation control' measures are adopted for it. Such methods can be classified as active (direct) and passive (indirect). The direct approach requires continuous inputs of resources to sustain the process whereas the passive methods require relatively little resource input once in operation. These methods are also classified as 'abiotic' (without direct participation of microorganisms) and biological (where micro-organisms play a significant role).

5.2.1 Abiotic active technologies

The most widespread methods under this category involve **aeration** and **addition of a chemical-neutralizing agent** (Coulton et al., 2003b). Various neutralizing agents are

lime, slaked lime, calcium carbonate, sodium carbonate, sodium hydroxide and magnesium oxide and hydroxide. The objective is to accelerate the rate of chemical oxidation of ferrous iron by raising the pH and aeration or optional addition of an oxidizing agent, thereby precipitating many of the metals as hydroxide and carbonate and formation of an iron rich sludge.

Although effective, active chemical treatment has the disadvantage of high operation costs and problems of disposal of bulky sludge. Some of the refinements added to this technology are multi-stepped addition of reagents accompanied by pH control for selective precipitation of some metals [Aube and Payant, 1997] and addition of flocculants to increase density of the sludge and improving its settling characteristics [Coulton et al., 2003a]. The major advantage of this is the substantial reduction of cost of storage and disposal of the high density sludge.

5.2.2 Abiotic passive methods

Abiotic passive methods to increase alkalinity of acidic mine water are mainly limestone-based.

i. Anoxic limestone drains (ALD)

A common method is the use of ALD [Kleinmann et al. 1998] which typically comprises a trench filled with crushed limestone rocks surrounded by impervious materials. The trench is generally isolated under a geomembrane (or plastic) liner covered with soil/clay. Contaminated un-aerated acidic mine effluents flow by gravity towards these limestone drains where the passive treatment is initiated by dissolution of carbonates in the bed thereby generating some bicarbonate alkalinity and raising the pH [Bernier et al. 2002]. The sole purpose of these systems is to provide alkalinity that will change acidic water into net alkaline water while maintaining the iron in reduced form to avoid the oxidation of ferrous iron and precipitation of ferric hydroxide on the limestone (armouring).

ii. Open limestone channels (OLC)

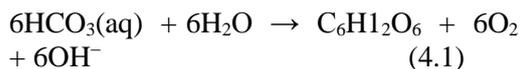
A more recent practice is to use OLC [Anonymous 2000] where the AMD is treated in an oxic (open to the atmosphere) environment. Armouring of the limestone with iron hydroxide reduces limestone dissolution by 20 to 50%. Steep channels are constructed which result in sufficient water velocity and turbulence that keep precipitates in suspension and prevent clogging of the interstitial pore space; thereby reducing the armouring of the limestone and enhancing the performance of the system.

iii. Limestone Ponds (LSP)

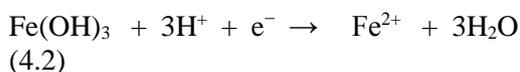
These are a new passive treatment idea in which a pond is constructed on the upwelling of an AMD seep or underground water discharge point [Skousen et al. 1998]. Limestone is placed in the bottom of the pond and the water flows upward through the limestone. Based on the topography of the area and how the water emanates from the ground, the pond can be built to accommodate water several feet deep (from 4 to 10 feet deep) with 1 to 3 feet of limestone. The pond is sized and designed to retain the water for 1 or 2 days for limestone dissolution, and to keep the seep and limestone under water. In case of armouring, the precipitate from the limestone surface can be dislodged by periodically stirring. If the limestone is exhausted by dissolution and acid neutralization, then more limestone can be added to the pond over the seep.

5.3 Biological Remediation

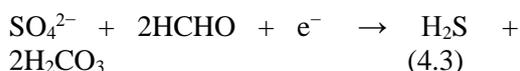
Bioremediation methods are based on the abilities of some microorganisms to generate alkalinity and immobilize metals, thereby essentially reversing the reactions responsible for the genesis/formation of AMD. Microbiological processes that generate net alkalinity are mostly reductive processes and include denitrification, methanogenesis, sulfate reduction, and iron and manganese reduction. Photosynthetic microorganisms, by consuming a weak base (bicarbonate), and producing a strong base (hydroxyl ions), also generate net alkalinity.



Reduction of ferric ion compounds in the solid phase decreases solution acidity by consuming H^+ , as in reaction (4.2) where the electron e^- is generally supplied by an organic substrate. Bacteria



that catalyse the reduction of sulfate to sulfide generate alkalinity by transforming a strong acid (sulfuric) into a relatively weak acid (hydrogen sulfide) as per reaction (4.3) and in turn many toxic metals present in AMD are removed in the form of insoluble metal sulfide.



5.3.1 Passive biological systems

Majority of the bioremediation options of AMD treatment are passive systems which, as mentioned earlier, take advantage of naturally occurring chemical and biochemical processes, mainly through use of sulfate-reducing bacteria or limestone or both to neutralize acidity and precipitate metals from contaminated mine area in a controlled environment [Ford 2003] instead of a direct chemical treatment. Passive treatment methods function by retaining contaminated mine water long enough to decrease contaminant concentration to accepted levels. Efficient passive treatment systems create conditions that favour rapid removal of target contaminants. There are several types of passive treatment systems some of the common ones are given below:

i. Constructed aerobic wetland

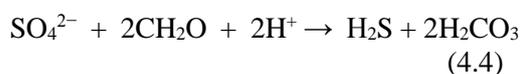
A wetland is a complex assemblage of water, basin, substrate and plants. Natural wetlands are characterized by water-saturated soils or sediments with supporting vegetation adapted to reducing condition in their rhizosphere (the narrow region of soil that is directly influenced by root secretions

and associated soil microorganisms). Constructed wetlands are man-made ecosystems which perform similar to natural ones.

A constructed wetland is a shallow basin filled with substrate and planted with vegetation tolerant of saturated condition. Water is introduced at one end and flows on the substrate and discharged at the other end through a weir or other structure, which controls the depth of water. Sometimes a liner is also included beneath the substrate. Constructed wetlands have been designed [Sheoran 2008] for the treatment of AMD having acidity up to 100 mg/l on the basis of acidity removal rate of 20 g/m²/day. A retention time of one day was shown to be sufficient to remove all pollutants. The retention time could possibly be even shorter depending on the specific AMD characteristics of the site.

ii. Anaerobic wetlands/compost bioreactors

In these systems the reaction that are responsible for AMD mitigation take place under anaerobic conditions. The reaction systems are usually enclosed entirely below ground level so that there is no air contact. Hence the term 'compost bioreactor' is more appropriate than wetland. The microbially catalyzed reactions that occur in compost bioreactors generate net alkalinity and biogenic sulfide as per reaction [Johnson and Halberg 2005]



And therefore these systems can be used to treat mine waters that are net acidic and rich in metal.

Systems that comprise of a combination of composite aerobic and anaerobic wetlands are also used to achieve full-scale treatment of AMD.

iii. Permeable reactive barriers

Permeable reactive barriers (PRB) operate on the same basic principles as of compost bioreactors. Its construction involves digging of a trench in the flow path of

contaminated ground water. The void in the trench is filled with reactive materials such as a mixture of organic material and limestone, permeable enough not to obstruct the flow of water. Within the PRB reductive microbiological processes take place generating alkalinity that helps removal of metals as sulfide, hydroxide and carbonate.

iv. Iron-oxidizing bioreactors

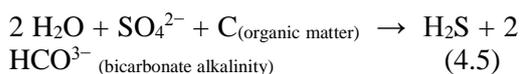
These are packed bed reactors where the rate of bacterium-assisted oxidation of ferrous iron to ferric iron is greatly accelerated. The microorganism is immobilized onto a solid matrix which forms the basis of the packed bed reactor or biological contactors. A much widely studied microorganism is the bacterium *Acidithiobacillus ferrooxidans* with which iron oxidation rates up to 3.3g/l/h has been recorded [Long et al. 2003].

v. Successive alkalinity-producing systems (SAPS)

SAPS combine treatment concepts from both wetlands and ALDs. Oxygenated water is pre-treated by organic matter removing O_2 and Fe^{+3} , and then the anoxic water flows through an ALD at the base of the system.

5.3.2 Active biological systems

These are Microbial Reactor Systems (MRS) which consists of a sulfidogenic (sulfate reducing) bioreactor and metal sulfide precipitators. In the former bacterial activity reduces sulphate (SO_4^{2-}) to soluble H_2S (and HS^-) and produces HCO_3^- .



In the latter, HCO_3^- partially neutralizes the incoming water, while H_2S and HS^- react with dissolved metals present in the AMD, resulting in precipitation of the metals as sulfide minerals. Partially treated AMD from the metal sulfide precipitators then flows into a limestone reactor for further neutralization. Water flows from the limestone reactor at near neutral pH into the

sulfate reducing bioreactor, where it provides a source of sulfate for the bacterial activity. Treated AMD is then discharged from the bioreactor, while the H_2S , HS^- and HCO_3^- produced in the bioreactor are circulated into the precipitator [Tayler et al. 2005].

6. CONCLUSION

Weathering of sulfide mineral present in coal deposits, mainly pyrite and mascalrite, in the presence of oxygen, water and iron oxidizing microorganisms result in formation of acid drainage in coal mines. As a result of weathering undesired amounts of acid and metal ions are produced and introduced to surrounding water bodies posing both short term and long term threats to the environment. The weathering process and the extent of contamination from the weathering products are affected by different factors.

Prediction of occurrence and characteristics of AMD is possible though various chemical and biological tests classified as 'static' and 'kinetic'. The static tests are short duration methods that take into account the reactions which take place during that duration and usually don't represent the weathering reactions in actual field conditions. On the other hand the kinetic tests simulate the weathering reactions that occur in nature, i. e. in actual field conditions. Proper combination of static and dynamic tests gives a reasonably accurate prediction of the quality of AMD that would be helpful in formulating proper waste management strategy.

The problem of AMD is solved through both 'source control' and 'remediation' approaches. The former aims at preventing or minimizing the generation of AMD whereas the later aims at treatment of the AMD by various methods. Remediation methods are resorted to when source control measures are inadequate to address the problem. The face of acid mine drainage treatment is changing. With research and new technology, mines are moving away from the more conventional forms of treatment and towards the use of ecologically engineered wetlands. The

costs of abatement of acid mine drainage are high, but the benefits to society and the environment are far greater in the long run.

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HIGH VALUED UTILIZATION OF AN INDUSTRIAL WASTE

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ABSTRACT

Present day world suffers from power crises for social maintenance of living beings. Most of the electric power crises are met by thermal power generation plants; which generates a lot of wastes i.e. fly-ash which creates environmental hazards, needs disposal/utilization. Present work has been undertaken to reveal the possibility of high valued utilization of fly ash. Thermal plasma spray coating of fly ash mixed with graphite is deposited on metal substrate. It is found that coating-substrate interface bond strength is dependent on input power of plasma torch. Maximum adhesion strength of 35MPa is obtained at 16kW power level.

1. INTRODUCTION

Utilization of Fly ash is a big challenge for us in present time. Coal fired power plants are main source of Fly ash production. In many aspects such as Mine filling, bricks, cements, constructions etc. fly ash is utilized; still the percentage is not enough on safety concern [1, 2]. Our present research has added some positive value to the percentage of utilization. All the materials have three types of utilization, i.e. low valued, medium valued & high valued. We have chosen high valued utilization of fly ash, i.e. it is used as a coating material. Fly ash is a low-grade industrial waste that has potential as a feedstock for plasma spray coating purposes. Chemical analysis of fly ash reveals that, silicon oxide (SiO₂) and aluminium oxide (Al₂O₃) are its major constituents. Hence, can be used as a ceramic coating material [3-5]. In modern technology many machine components are operated under extreme conditions, such as high pressure, temperature & corrosive atmosphere etc. Due to these factors the industries faces a lot of losses for the degradation of machine parts. Ceramic coating has emerged as a major solution to the machine parts operating under harsh conditions for longer operational life period; and such coatings are being used for a remarkable number of applications [6]. Mainly ceramic coatings are used as thermal barrier coating for its excellent wear, corrosion and high temperature

resistance. However, it is not always possible to replace structural parts made of metals and alloys with ceramic ones, since ceramics have poor mechanical properties. Moreover, in many applications, properties desired at the surface are different from those required at the core, viz. wear resistance and fatigue resistance at the outer surface.

Ceramic coating technology, the process of preparing or depositing a ceramic layer on a substrate offers the best solution in such situations. Ceramic coating have found use, not as a substitute for metals and alloys but in complementing metal characteristics by imparting additional thermal insulation and wear resistance. Moreover coating technology allows the design engineer to select the base material and other design considerations and then prepare the surface for meeting requirements simply by making a ceramic coat on it. Ceramic coatings are required in various industries for different applications and hence the requirements and specification of the coatings vary with a particular application. Ideal ceramic coating should have good mechanical properties over a wide temperature range and resistance to wear.

Plasma spray process is the most versatile one for preparing ceramic coatings. In this process any material which melts without decomposing or vaporizing can be sprayed. The plasma spray technology has the

advantage of being able to process various low-grade-oreminerals to obtain value-added products and also to deposit ceramics, metals and a combination of these, generating approximately a homogenous composite coating with desired microstructure on a range/variety of substrates [7, 8]. Its other advantages are there is no restriction on size and shape of the job, fast coating process. Graphite is mixed to act as reducing agent and/may form carbides which will affect the coating properties [9, 10].

2. EXPERIMENTAL DETAILS

Fly ash, taken from CPP-I of Rourkela steel plant mixed with graphite powder with 10 & 20 weight % is coated on Aluminium and copper substrates (of size 25x25x3 mm⁰). The interface bonding of the coating with substrate is mainly due to mechanical anchoring. Hence surface roughness is important to obtain a good interlocking between the substrate and the coating. So the substrates were subjected to sand blasting and surface roughness of 3.5 R.A was produced.

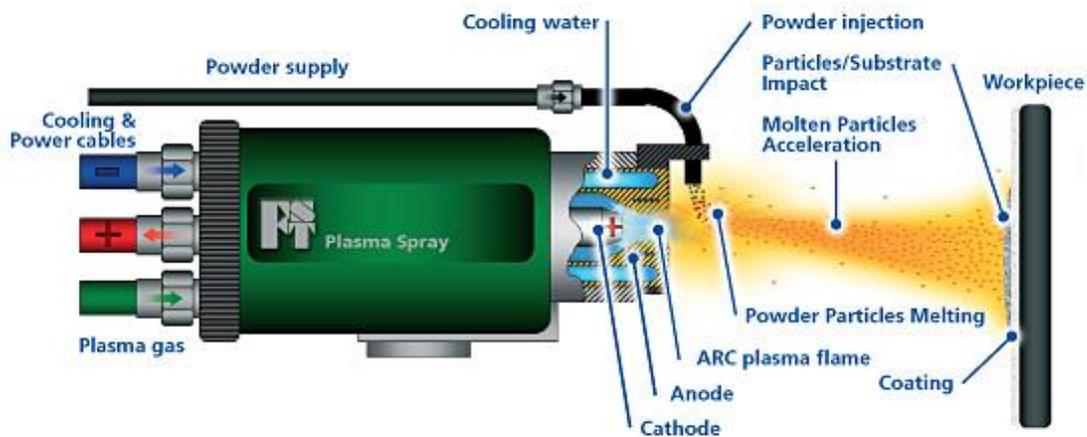


Fig 2.1: Schematic Diagram of Plasma Spraying Process.

Table 2.1: Operating parameters during coating deposition.

Operating parameters	Values
Plasma arc current (amp)	250,300,400,500
Arc voltage (volt)	40
Torch input Power(kW)	10,12,16,20
Plasma gas (argon) flow rate (lpm)	20
Secondary gas (N ₂) flow rate (lpm)	2
Carrier gas(Ar) flow rate (lpm)	12
Powder feed rate (gm/min)	15
Torch to base distance (TBD) (mm)	100

3. RESULTS & DISCUSSION

3.1 Particle Size Analysis

The particle size of the sprayed powder plays an important role for deposition of the coating. The particle size analysis of raw material used for coating is carried out with laser particle size analyser and is found that the particle size ranges from 50 to 150 microns. However the major amounts of particles are in the range of 70 to 110 micron.

3.2 Chemical Composition Analysis

Table 3.2: Chemical analysis of the fly ash used for the coating.

Element/compounds	Percentage
SiO ₂	56.00
Al ₂ O ₃	28.4
Fe ₂ O ₃	4.5
TiO ₂	1.5
CaO	1.33
MgO	1.28
Na ₂ O	.61
H ₂ O	.2
LOI	6

To the above composition, Graphite was added at 10 & 20 weight percentage.

3.3 Measurement of Coating Thickness

Coating thickness of Fly-ash & Fly-ash + Graphite (10 & 20%) was measured on the polished cross-sections of the samples, using an optical microscope [11]. The thickness values obtained for coatings deposited at different power levels for Aluminium and Copper substrates are shown in the graph. IT can be observed that, with increase in coating power level, coating thickness increase up to 16 kW power and is greater for Cu than Al substrates.

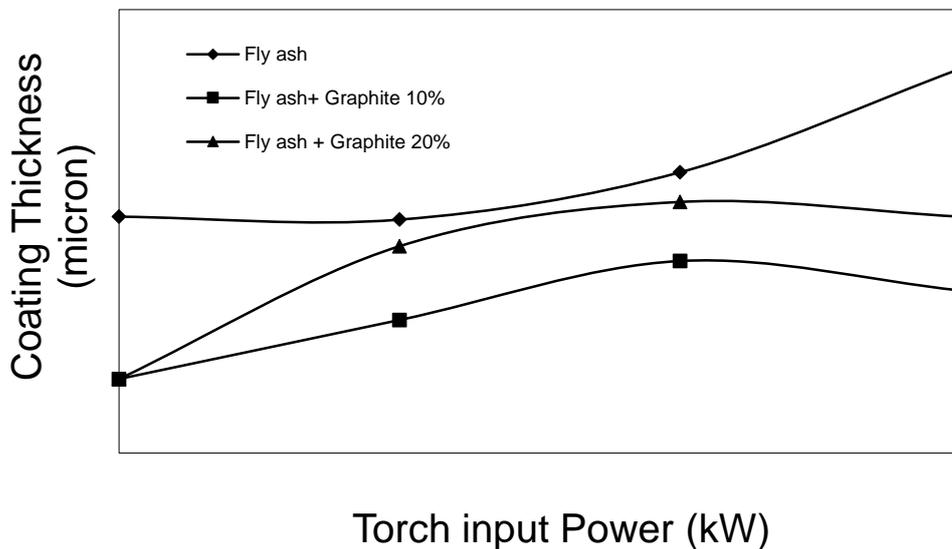


Fig. 3.3.1: Coating Thickness Comparison for Al Substrate.

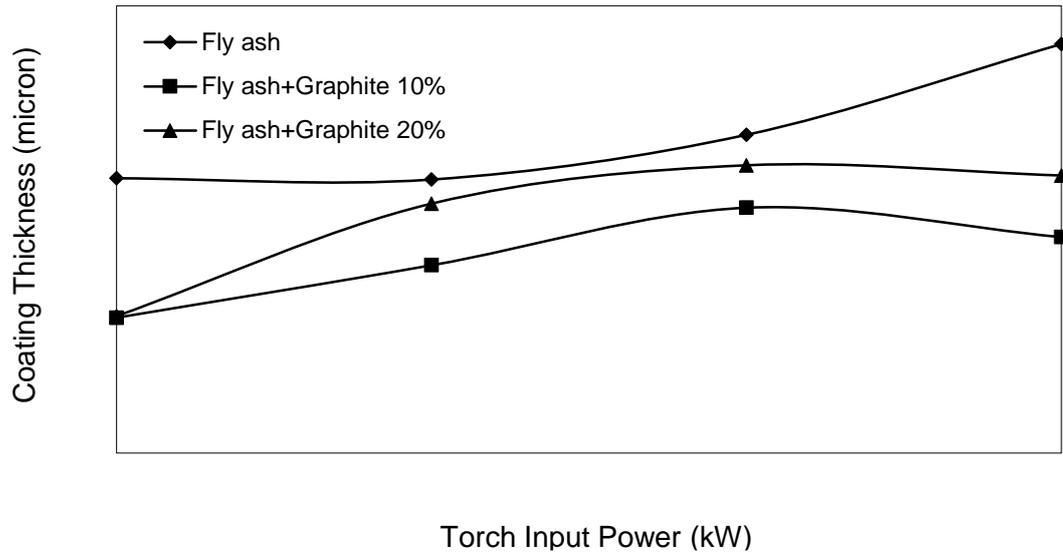


Fig. 3.3.2: Coating Thickness Comparison for Cu Substrate.

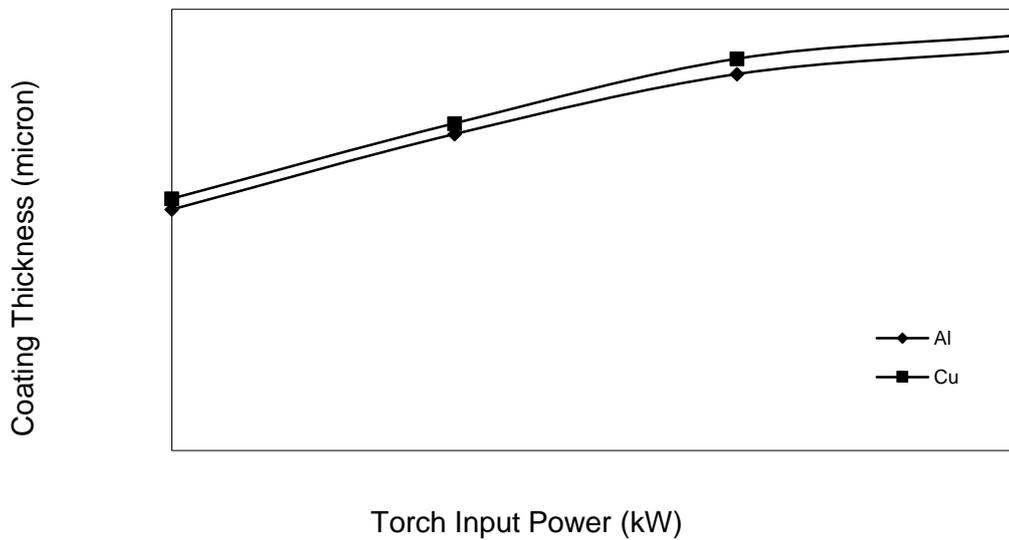


Fig. 3.3.3: Coating Thickness Comparison between Al & Cu substrates.

3.4 Measurement of Adhesion Strength

Coating pull out method was used to evaluate the interface bonding strength. Cylindrical samples of 25x10 mm are prepared for testing in Tonometer. The coated samples were fixed on studs by using a resin (epoxy 900-C) adhesive [11].

The adhesive was so selected that its bonding with the surface of the coating would be stronger than the bonding of the coating with the substrate. The specimens are allowed to set for 24 hours and then the test was carried out. A maximum adhesion value of 35MPa is obtained at 16kW power level for Cu substrate.

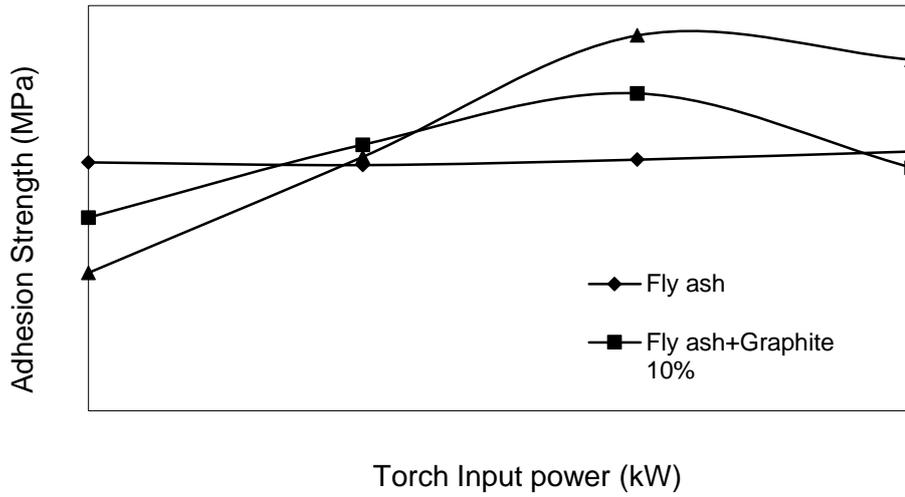


Fig. 3.4.1: Adhesion Strength Comparison for Al Substrate

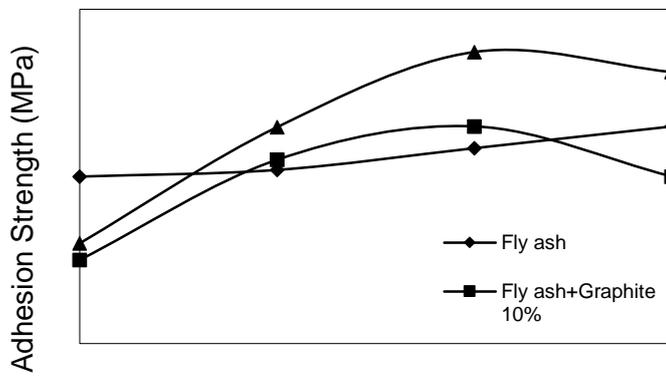


Fig. 3.4.2: Adhesion Strength Comparison for Cu

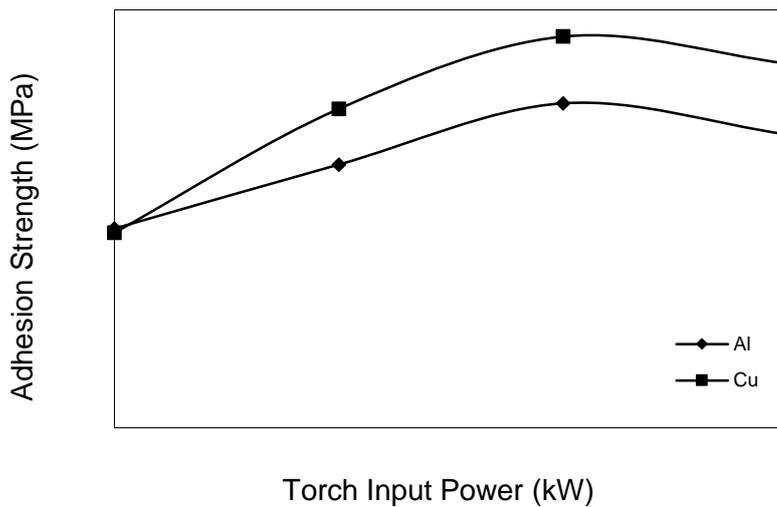


Fig. 3.4.3: Adhesion Strength Comparison between Al & Cu

4. CONCLUSION

Fly ash can be coated on metal substrates to produce ceramic coatings. Coating properties are affected with addition of graphite.

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DEVELOPMENT OF BIO-PROCESS FOR NICKEL METAL RECOVERY FROM NICKELIFEROUS CHROMITE OVERBURDEN

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ABSTRACT

Nickeliferous chromite overburden (COB) in the Sukinda Valley of Odisha containing 0.7-1% nickel awaits a suitable process methodology to extract the valuable metals ingrained in their mineral phases. The conventional pyro- and hydro- metallurgy methods consume intense energy, require high capital investments, cause environmental hazards, etc., thus becoming industrially and environmentally unfeasible. Employing microorganisms towards biotechnological means of extraction may overcome the above downsides. A group of microorganisms called 'dissimilatory iron reducing bacteria (DIRB) inherently reduce Fe^{3+} to Fe^{2+} coupled to oxidation of organic carbon source for their energy and growth requirement. Hence, in the first step of present study, COB of 5% pulp density was subjected to biological pre-treatment using DIRB capable of transforming the mineral phase (Goethite \rightarrow Hematite, Magnetite) at 1L scale. In second step enhanced nickel recovery was achieved by reductive acid leaching of the bioreduced COB with glucose as the organic reducing agent in sulphuric acid. A total of 88.8% nickel was recovered from the nickeliferous COB subjected to bio-reduction and reductive acid leaching. Nickel was recovered from leach solution through solvent extraction using Na2DHPA as extractant in the third step. Finally, electrowinning of the nickel laden aqueous phase obtained a metallic nickel deposit with 99.6% purity. Hence, in this study, an efficient and eco-friendly process is proposed involving bioreduction, acid leaching, solvent extraction and electrowinning (BR-AL-SX-EW).

Keywords: Nickeliferous chromite overburden, Dissimilatory iron reducing bacteria, Flowsheet

INTRODUCTION

The global annual demand for nickel has increased with the economic and industrial development. However, the production of nickel in correlation with demand is quite low. The available world land-nickel resources are mainly found in the lateritic and sulphidic form. Around 72% of the nickel resource is found in lateritic form

while the remaining 28% is present in sulphidic form. The current major nickel production from sulphidic nickel has led to its rapid depletion (Fig.1). Hence, the research focus has been diverted towards nickel production from lateritic nickel to decipher out an efficient, eco-friendly and cost-effective process of extraction.

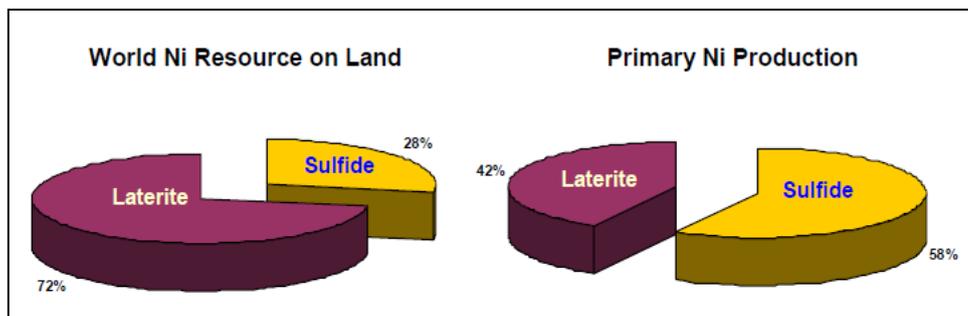


Figure 1: World Nickel Resource and Production (Source: Dalvi et al., 2004)

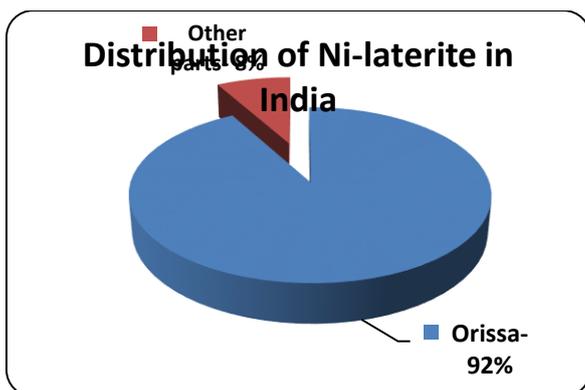


Figure 2: Distribution of nickel laterite reserve in India

In India, 92% of lateritic nickel reserve is concentrated in the state of Odisha (Fig.2). Also, around 6 MTs of nickeliferous Chromite overburden (COB) are generated annually in the Sukinda Valley, Jajpur but they lack a suitable technology for nickel extraction. Currently, the nickel need in India is met majorly through import which costs \$16,038/tonne as of April' 14 and to a little extent from other sources such as nickel bearing scraps. These COB dumps occupy large land space and cause environmental problems while awaiting a suitable technology to recover the nickel value.

Mineralogically, the COB is predominated with goethite phase along with other mineral phases of hematite, magnetite and quartz. The nickel present in this nickeliferous COB is firmly ingrained in the goethite matrix [Fe (III)] withstanding its recovery by conventional methods. Moreover, conventional methods of hydrometallurgy and pyrometallurgy consume high energy and require high capital investment accompanied with various environmental hazards. To overcome these downsides, biological means of extraction is the focus of current

day research. The emphasis of our work was to establish a suitable eco-friendly bio-process towards nickel extraction from COB. Hence, a group of dissimilatory iron reducing bacteria (DIRB) was exploited to biologically reduce the COB with enhanced nickel extraction on reductive acid leaching (glucose as reducing agent).

For the first time, in our laboratory at CSIR-IMMT, DIRB has been employed to reduce COB (Pradhan et al., 2012). DIRB have inherent capacity to utilize Fe (III) as terminal electron acceptor with organic carbon source as its electron donor for its growth. This Fe (III) bio-reduction activity by DIRB would result in a phase transformation (Goethite → Hematite, Magnetite) causing exposure of nickel. The exposed nickel can further be effectively extracted on reductive sulphuric acid leaching using glucose as reducing agent. Hence, through this work, a novel conceptual bio-process flowsheet was developed to profitably extract nickel from the lateritic COB as illustrated in Fig.3.

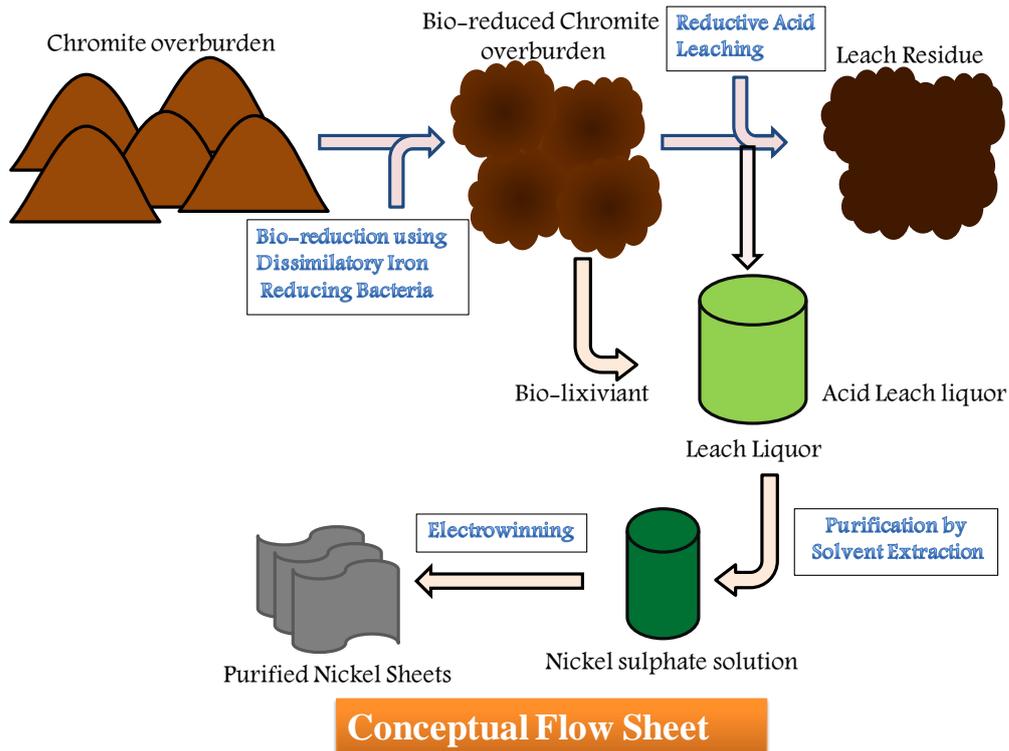


Figure 3: Conceptual bio-process flowsheet for nickel extraction from COB

MATERIALS AND METHODS

Ore Sample and DIRB consortium

The nickeliferous COB sample was kindly provided by Odisha Mining Corporation (OMC). Chemical analysis of the COB specifies the presence of 0.78% Nickel, 0.09% Cobalt, 0.61% Mn, 2.6% Chromium, and 40.52% Iron.

A 7 day old culture of adapted, enriched DIRB consortia isolated from the stagnant pond sediments as used in our previous studies was employed for this study too (Esther et al., 2012; Esther et al., 2014). Sterile Mineral Salt Medium (MSM) containing (g/L): KH_2PO_4 – 0.8; K_2HPO_4 – 3.0; KCl – 0.2; NH_4Cl – 1.0; MgCl_2 – 0.2; CaCl_2 – 0.1 and yeast extract – 0.05 with 10 mM glucose (as electron donor) was used for enrichment, adaptation and experimental studies.

Bio-reduction of COB using DIRB

Bio-reduction of COB was performed in a 1L volumetric flask having sterile contents of 50g ore sample (5% w/v pulp density) and MSM. The 7 day old culture of adapted, enriched DIRB consortia was

inoculated (10% v/v) and a layer of paraffin oil was introduced to maintain anaerobic conditions. Sampling was performed every week to monitor the Fe (III) reduction activity via Fe (II) estimation. Fe (II) was estimated using the titration method as described in our previous studies (Esther et al., 2014).

Acid leaching of DIRB-treated COB

The DIRB-treated, bio-reduced COB was subjected to acid leaching using 6N sulphuric acid at 65°C and 10% (w/v) pulp density to extract the nickel. The influence of organic reducing agent on acid leaching was studied using 2% (w/v) glucose. Sampling was performed every hour for 6hrs.

Solvent Extraction and Electrowinning of Pregnant Leach Solution (PLS)

The pregnant leach solution containing the extracted metals was subjected to solvent extraction to selectively extract the nickel metal. The PLS was equilibrated with 0.5M NaD2EHPA in a single stage with O: A ratio of 1: 4 at pH 5. The mixture was then allowed to segregate and the aqueous phase

was separated. The aqueous phase was then stripped with 5% H₂SO₄ in a single stage with O: A ratio of 1: 1. The organic components of the purified aqueous phase were adsorbed using activated charcoal before subjected it to electrowinning for metal deposition. Electrowinning was carried out at a current density of 200 A/m² for 30 min

RESULTS AND DISCUSSION

Bio-reduction of COB using DIRB

The bio-reduction of COB was co-related with the production of Fe (II) in the media. The rate of Fe (II) was found to increase with time reaching 0.83 g/L in 100 days (Fig.1). With further increase in duration, Fe (II) was found to reduce indicating the saturation in solution resulting in adsorption to the Fe (III)-oxide or bacterial cell surfaces. This adsorption has been reported to limit the availability of Fe (III)-oxide for bacteria via reduced surface area

(Roden and Urrutia, 2002). Chemical analysis of the bio-reduced COB indicated an extraction of 22.3% nickel during the bio-reduction stage. Mineralogical and morphological studies of the bio-reduced COB confirmed the phase transformations which can be referred from our previous study (Esther et al., 2012; Esther et al., 2014).

Acid leaching of bio-reduced COB

Acid leaching of bio-reduced COB resulted in 17% nickel extraction which was found to be enhanced to 64.17% on addition of glucose. However, the reducing agent glucose was found to have little or no effect on acid leaching of original COB as can be seen in Fig.4. Hence, it can be rightly proposed here that phase transformation due to bio-reduction of COB by DIRB can improve the metal extraction rates.

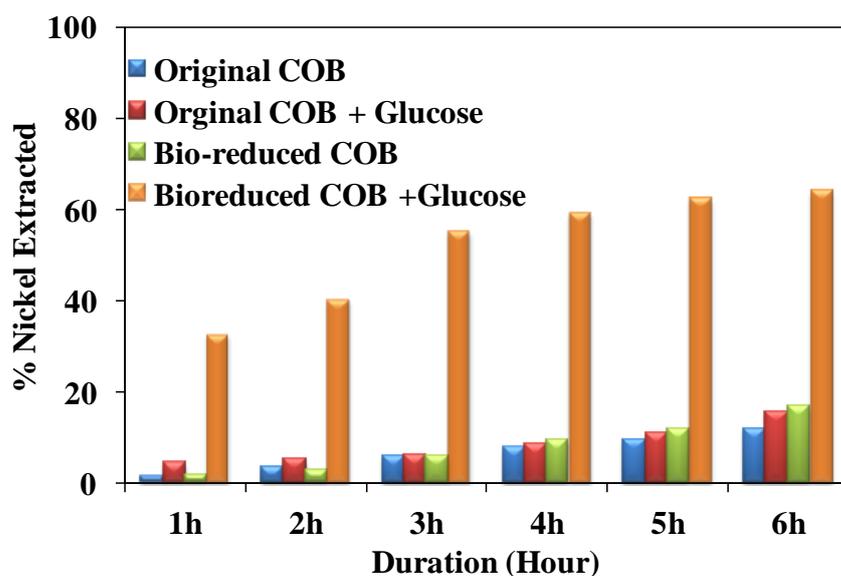


Figure 4: Nickel extraction on reductive acid leaching

Solvent Extraction & Electrowinning of leach liquor

A preliminary study of solvent extraction of PLS was performed to extract nickel using 0.5 M D2EHPA as extractant. 100% extraction efficiency was obtained in one

stage with O: A ratio of 1: 4 at a pH of 5. The stripped PLS was thus subjected to electrowinning to obtain metallic nickel value. Chemical analysis of the nickel deposit highlighted its purity of 99.6%.

Generalised Bio-process Flowsheet

Having obtained efficient results at every stage from nickeliferous COB to nickel metal, a generalized bio-process flowsheet (BR-AL-SX-EW) was conceptualized as in Fig. 5. The flowsheet describes the various steps from bio-reduction of COB using DIRB in the first step followed by reductive acid leaching in the second step. The leached COB residue is separated

physically from the leachate. The leachate is then purified to remove the iron content that may hinder the subsequent steps of metal extraction. In the third step, the purified pregnant leach solution was subjected to solvent extraction and stripping to produce nickel sulphate solution which was then electrowon in the final step to obtain nickel metallic deposit.

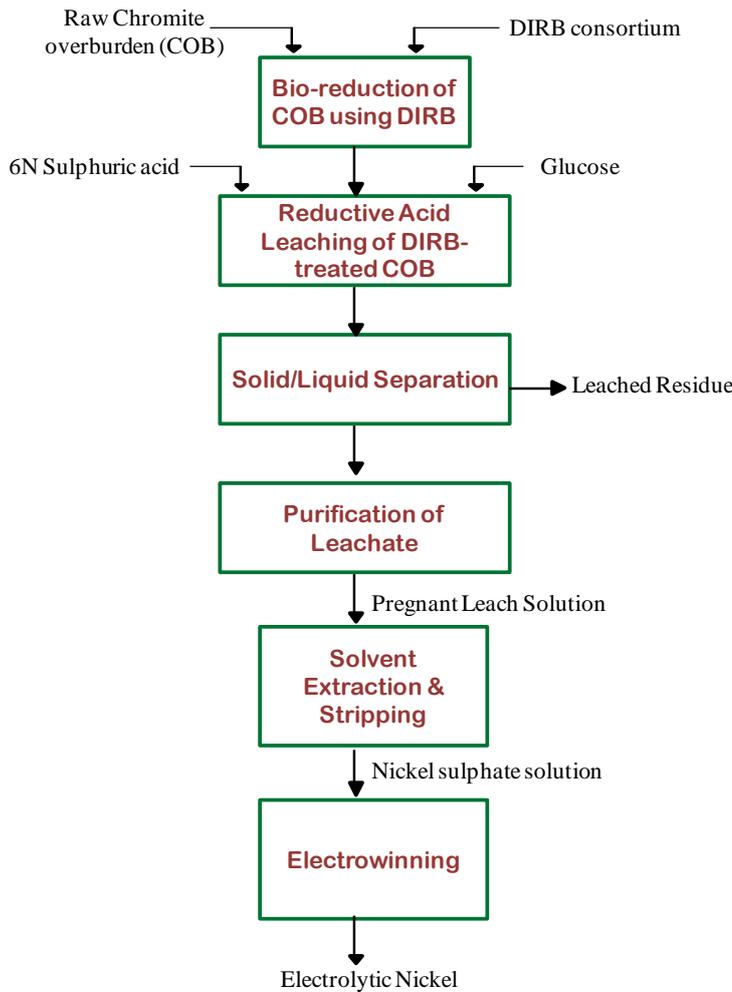


Figure 5: Generalised bio-process flowsheet

CONCLUSION

A novel, eco-friendly bio-hydrometallurgical process (BR-AL-SX-EW) was developed to efficiently extract nickel value from nickeliferous COB. Organic reducing agent, glucose served as a significant organic reductant during acid leaching contributing to the enhanced extraction of nickel. 86.5% cumulative nickel was extracted on bio-reduction (22.3%) and reductive acid leaching (64.2

%) of COB. Metallic nickel deposits of 99.65% purity were obtained on solvent extraction and electrowinning of pregnant leach liquor. Further, optimization and scale-up of this process can help retrieve nickel from our native resources improving the economic development of our country which is the vision of every citizen towards building a developed nation.

ACKNOWLEDGEMENT

The first two authors thank CSIR-Emeritus Scientist Scheme (ESS-4) for their grant-in-aid towards this project. The authors thank our director, Prof. B.K. Mishra for granting permission towards publication of this work.

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DESIGN MODIFICATIONS FOR EXISTING SPONGE IRON PLANT WITH THE UTILIZATION OF STACK GAS HEAT

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ABSTRACT

The present work deals with a suggested approach for utilizing the heat associated with the stack gas of the sponge iron plant. Utilization of this heat in the process may reduce the coal consumption which is the only source of energy for this plant. Heat integration is applied on the actual plant data to check the feasibility of utilization of the heat associated with waste gas exiting from ESP. Certain design modifications has been incorporated without disturbing the SL/RN process technology involved in the sponge iron production. The design modification accounts for the preheating of feed coal and air supplied to the kiln using waste gas exiting from ESP. The additional equipment required are a duct to carry waste gases from electrostatic precipitator to rotary kiln, a Fluidized bed dryer to preheat the kiln feed, Gas-gas heat exchanger for preheating the air and a return duct to carry waste gases back to chimney. Preheating of coal and air reduces the coal consumption by 4.3% and the air requirement by 23.74 tph, in comparison to the existing system.

Key Words: *Sponge iron plant, Energy integration, Fluidized bed dryer, gas-gas heat exchanger, stack gas, design modifications.*

1. INTRODUCTION

Since last two decades the non-coking coal based sponge iron plants are gaining importance, for the demand of sponge irons as raw materials in the production of steel [1]. At present, there are 120 large and small sponge iron plants operating in India and most of these plants are coal based DR plants. Sponge iron manufacturing units are seen profitable since the beginning of the 1990s. However, due to the hike in the cost of the raw materials and decrease in sales, these plants find difficulty to last for more years. Lack of inclusion of modern technology added further difficulties to the situation. Although at present, the conditions have improved considerably, still the sponge iron industries pass through several problems, such as absence of proper integration of heat energy, outdated technology and non-optimal operations of equipments which make these industries less profitable ventures. Therefore there exists a large scope to apply modern technology to these sponge iron industries to make them more competitive by cutting down their internal losses [2].

Many investigators have worked on the saving of energy for sponge iron plants. Agrawal et al. [1,2]; Devi and Mazumder [3]; and Misra and Ipicol [4]; considered the manufacturing process of sponge iron plants and recommended many improvements for the operation processes. Different aspects such as waste gas composition, dust loss, air requirement, and efficiency of the process were studied by Jean et.al [5]. Due to chemical reactions and combustion, the heat generated inside the kiln is measured to be 174.28 GJ/h while heat value of coal input is found to be 323.2 GJ/h. This results in a thermal efficiency of 46% for the process. It is also seen that 33% of heat generated in the kiln is lost with waste gas. That is why many authors have suggested for the recovery of heat from the waste gas by using a waste heat recovery system which can be used for generation of power.

Agarwal and Sood [6] have studied that, only 35% of the total heat generated from coal is used in the reduction process and rest goes as waste. Elsenheimer and Serbent [7] have concluded that the energy contained in waste gas is in form of sensible

or chemical energy which amounts up to 40%. They have suggested a number of options to recover the energy but have not discussed about the quantity of energy that can be saved by installing these options.

Although many plants have now attained the desired level of operational efficiency, but from the energy analysis point of view various units are still below the optimum limits. The principal cost factor in coal-based process is energy cost as energy requirement for rotary kiln ranges between 14.63 and 20.9 GJ/t [8]. Therefore it is felt that energy conservation in these plants is highly essential for which heat integration is necessary to minimize the losses. Thus, the present study aims at developing a methodology to break the bottlenecks of the process used in sponge iron industries. That is why the heat of waste gas is planned to

be recovered by preheating the raw materials. As a result a considerable saving in the amount of coal can be achieved.

2. A BRIEF DESCRIPTION ON CONVENTIONAL COAL BASED SPONGE IRON PLANT

The longitudinal section of typical coal based sponge iron plant is shown in Figure 1. The feed consisting of iron ore, coal and dolomite are charged into the rotary kiln with the help of a conveyor. Air is injected to the rotary kiln along the length at various places with the help of blowers which is adjusted separately. In addition, air is also injected to the kiln at the outlet by central burner pipe which serves as process air inlet during normal operation as shown in the Figure 2[9].

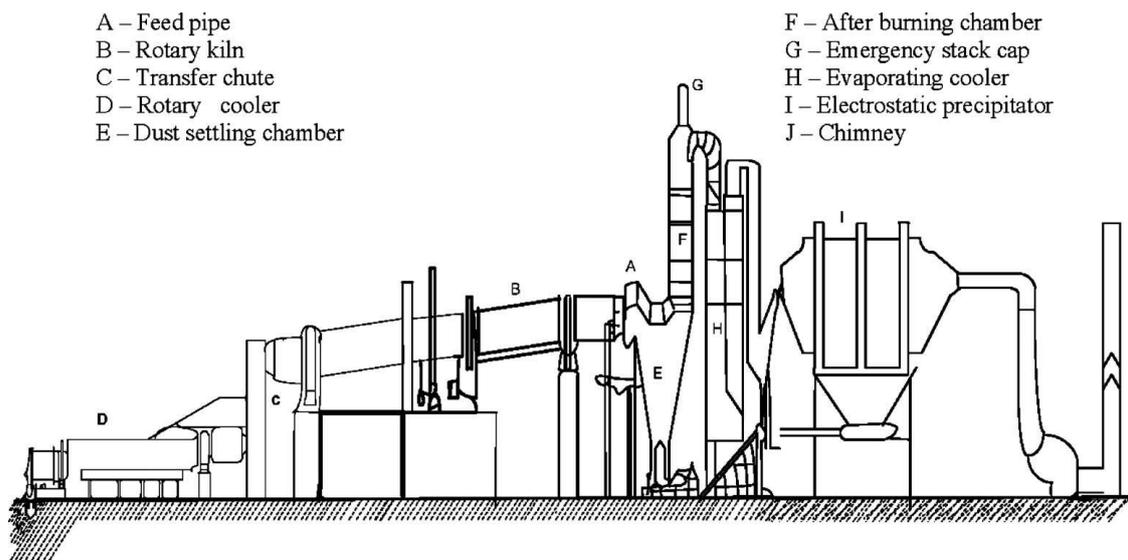


Figure 1 : Longitudinal section of sponge iron plant

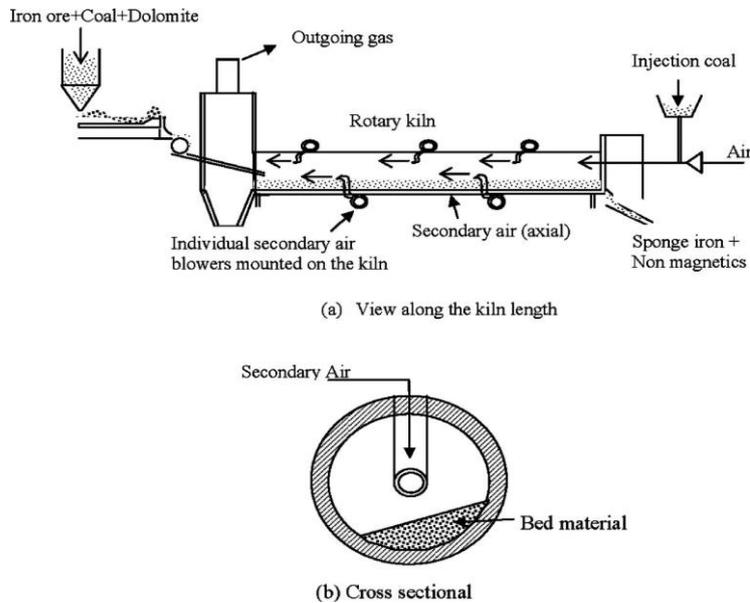


Figure 2 :Normal injection of air into the kiln

The inside of the kiln is lined with refractory materials. It is supported on three piers called support rollers with an inclination of 2.5° with the horizontal as shown in Fig.-1. The girth gear is used for the rotation of the kiln. Due to its inclination and rotation, feed materials present in the bed of kiln move along the axis. Around two third amount of total coal is fed with iron ore and it is called feed coal. It produces CO and CO₂ gases when combustion takes place in the presence of air inside the kiln. These exothermic reactions increase the temperature of coal and bring it to reaction temperature. The remaining one third amount of total coal is injected from the discharge end which

reacts with CO₂ to produce CO. This CO acts as the reducing agent to produce Fe from iron ore.

The residence time of iron ore inside the kiln is about 10 hours. During this time iron ore is reduced optimally. Hot sponge iron along with semi burnt coal exits the kiln from the discharge end at 1000°C which consists of Fe, char and ash. These products then enter the rotary cooler where water is sprayed over the cooler and the temperature is brought down to about 120 °C. After cooling the products are separated into iron, char and other non- magnetic impurities using electromagnetic separators.

Table 1: Feed Stream Properties

Raw Material	Particle Size (mm)	Feed Temp(°C)	Flow Rate(tph)
Iron Ore	5-18	30	6
Coal	<18	30	3.3
Dolomite	2-6	30	0.2

Usually, waste gas consists of N₂, CO₂, CO, H₂, H₂O, O₂ and CH₄ with compositions of 59.69%, 35.91%, 0.32%, 0.17%, 3.7%, 0.08%, and 0.13%, respectively. Waste gas comes out of the kiln at a temperature about 900°C and is then passed through the After Burner Chamber (ABC) and Dust Settling Chamber (DSC). DSC reduces velocity of waste gas, removes large dust particles by gravity and retards pressure variation. Thus

waste gas attains uniformity with respect to temperature and concentration of combustibles. After DSC the waste gas is again allowed to move upward to ABC. From ABC the stack gas is allowed to flow to the waste heat recovery boiler (WHRB) where it is cooled to 250°C thereby causing the generation of steam. Further, stack gas coming out from the WHRB is passed through the electrostatic precipitator (ESP)

for removal of particulate matter. Desired temperature of the stack gas is also maintained below 250°C. Finally, filtered stack gas is vented out to the atmosphere at a temperature of 250°C and remaining material of ESP is collected as dust.

Feed of specific size and composition as shown in Table 1 is fed to the rotary kiln. Mass balances around the main components of the plant are shown in Table 2. The overall mass balance of the plant is shown in Figure 3.

Table 2 : Mass balance of each equipment

Rotary kiln			
Input	tph	Output	tph
Iron ore	6.5	Sponge iron	4.513
Feed coal	3.529	Char	0.601
Slinger coal	2.429	Ash	2.406
Air	20.78	Spent lime	0.07
Dolomite	0.2		

ABC+DSC			
Input	tph	Output	tph
Dusty gas	25.855	Wet scraper	0.7
ABC air	4.018	Dusty gas	29.17

ESP			
Input	tph	Output	tph
Dusty gas	29.17	Dust	1.679

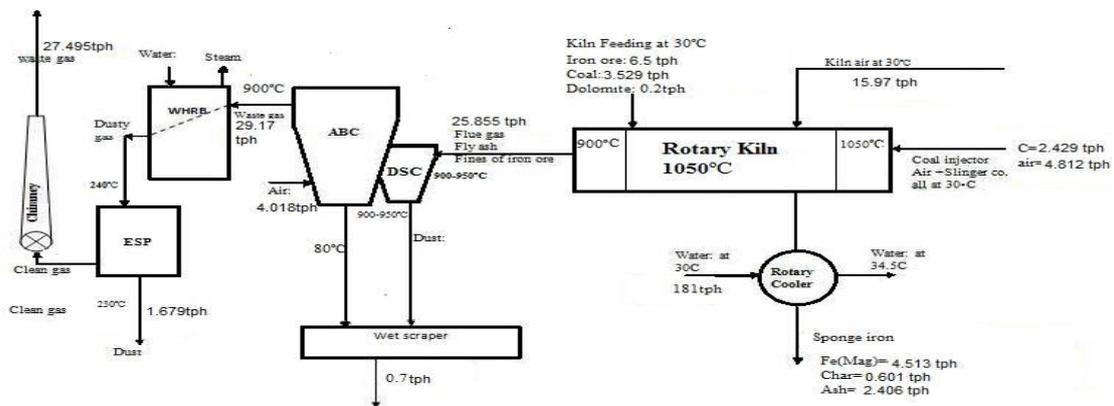


Figure 3: Flow Sheet of Typical Sponge Iron Plant

3. SUGGESTED MODIFICATIONS IN THE CONVENTIONAL PLANT FOR UTILIZATION OF WASTE HEAT

3.1 Potential regions for Heat Integration in the sponge iron plant

The process used in sponge iron industry produces a substantial amount of heat that

remains unutilized. One of such example is the heat associated with waste gas which is at a temperature of around 250°C is released to the atmosphere through chimney.

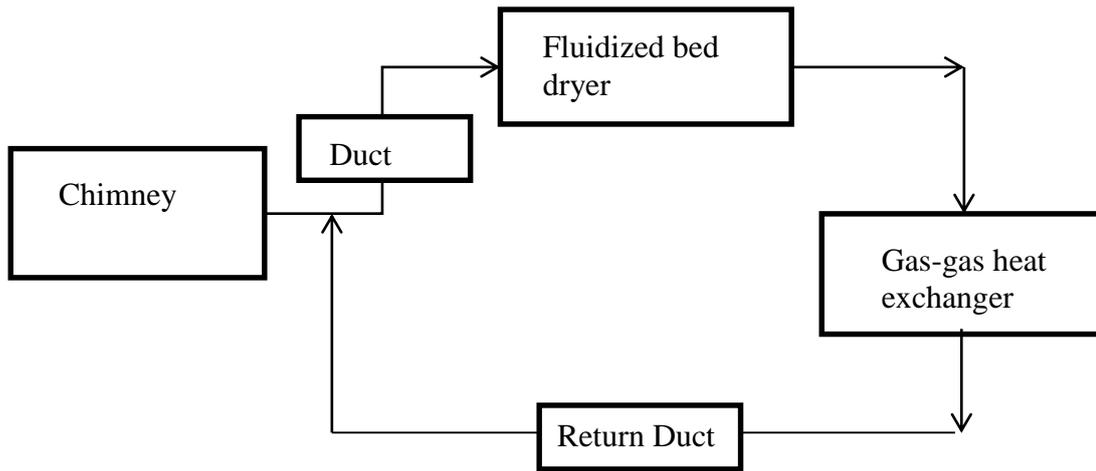


Figure 4: The partial schematic diagram of the modifications required in the process

Therefore in present work, heat of waste gas is planned to be utilized for preheating the inlet streams such as feed material and air. The partial schematic diagram of the modification is shown in Figure 4 where following equipments are included in the process.

- (1) A duct to carry the stack gases from ESP outlet to rotary kiln.
- (2) Fluidized bed dryer for preheating of coal.
- (3) Gas-gas heat exchanger for reheating air
- (4) A return duct to carry the stack gases back to chimney.

3.2 Design of equipment required for Energy Integration

3.2.1 Duct

The waste gas from ESP is brought to the rotary kiln through a duct and a return duct is used to carry the waste gas from kiln to the chimney. Both ducts are to be insulated.

The distance between ESP outlet and rotary kiln is about 35m in plant site for which an insulated duct of 35m length is used. The inside diameter of duct which is a hollow cylinder is considered as 0.309m [10].

3.2.2 Theoretical design of Fluidized bed Dryer

Once the waste gas is carried up to the rotary kiln through the insulated duct, the heat of gas is used to preheat the kiln feed. For this purpose a fluidised bed dryer with height of 1m and diameter of 0.8m is designed using the following equations. The schematic diagram is shown in the Figure 5.

- a) Minimum fluidization velocity:

The lower limit of the superficial velocity of the gas that causes the bed materials to fluidize is calculated using the following expression (Eq.-1) which is known as minimum fluidization velocity [11].

$$\frac{d_p u_{mf} \rho_g}{\mu} = \left((28.7)^2 + 0.0494 \left[\frac{d_p^3 \rho_g (\rho_s - \rho_g) g}{\mu^2} \right] \right)^{0.5} - 28.7 \quad (1)$$

- b) Terminal velocity of the particle:

The maximum value of the superficial velocity of the gas beyond which the bed

materials get entrained is determined using the expression as mentioned below in Eq.

(2). This is called as the terminal velocity of particles [12].

$$u_t = \left[\frac{4d_p(\rho_s - \rho_g)g}{3\rho_g C_D} \right]^{0.5} \quad (2)$$

Where, C_D is the drag coefficient and is calculated as per the following

$$C_D = \frac{24}{Re_p} \left[1 + (8.1716e^{-4.0655\phi_s}) Re_p^{0.0964+0.5565\phi_s} \right] + \frac{73.69(e^{-5.0748\phi_s}) Re_p}{Re_p + 5.378e^{6.2122\phi_s}} \quad (3)$$

c) Fluidization velocity during Fluidization :

calculated considering the relation between the maximum and minimum expanded bed heights of the fluidized bed [13].

The fluidization velocity of the gas to be used during the fluidization is

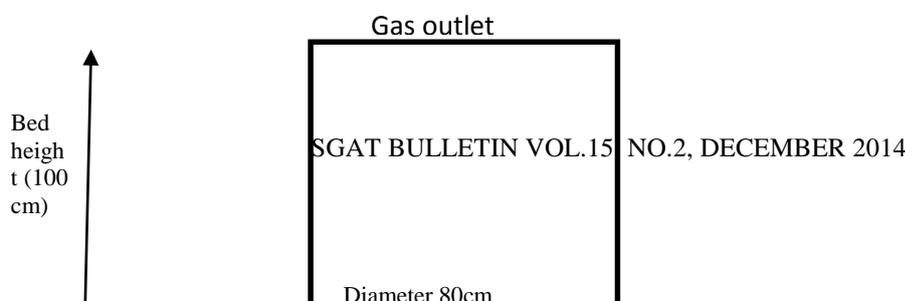
$$\frac{H}{H_{mf}} = 1 + \frac{10.978(U_f - U_{mf})^{0.738} \rho_p^{0.376} dp^{1.006}}{U_{mf}^{0.937} \rho_f^{0.126}} \quad (4)$$

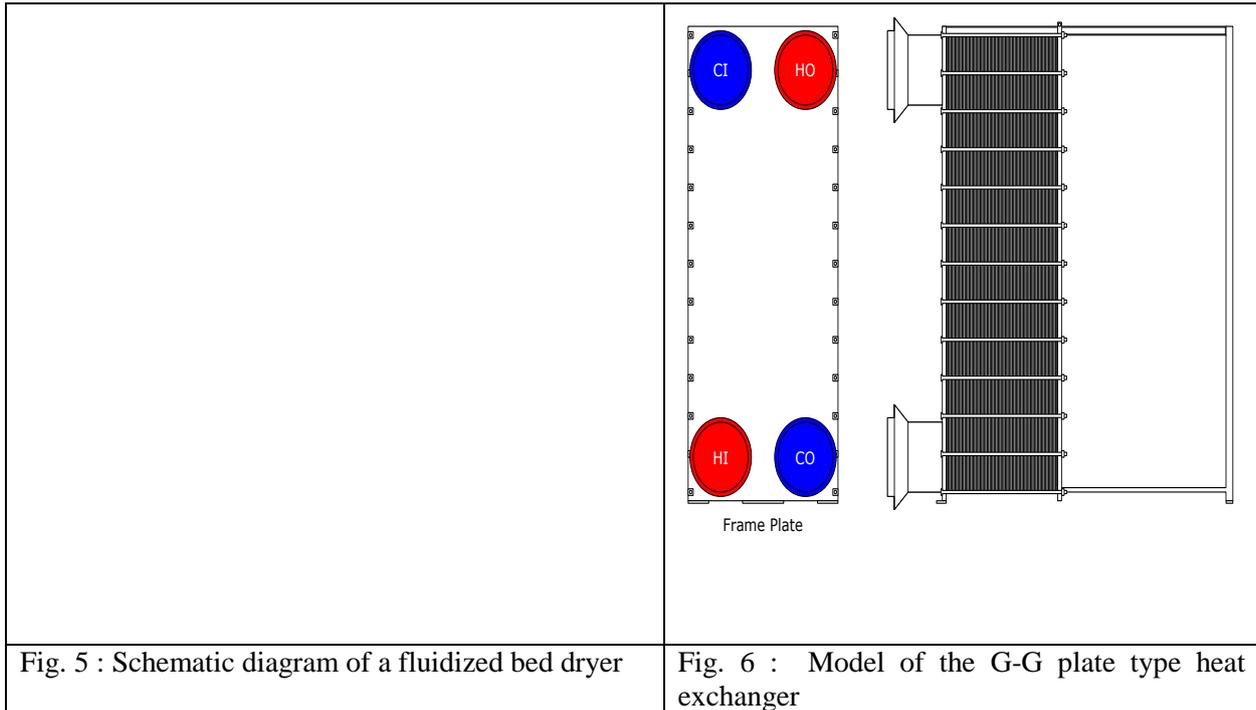
$$1.2 < \frac{H}{H_{mf}} < 1.4 \quad (5)$$

For the design the value of $\frac{H}{H_{mf}}$ is selected as 1.3.

Table 3 :Input data configuration for plate type HX

Run mode	Rating
Number of units in parallel	1
Pass specification	Symmetric passes
Number of hot passes	1
Number of cold passes	1
Channels (per pass)	40
Hot side U/Z arrangement	U
Cold side U/Z arrangement	U
Flow configuration in first hot channel	Counter current
Front plate type	Frame plate
Plate manufacturer	Alfa laval
Chevron angle	28.00 deg
Area per plate	2.699 m ²
Plate material	316 stainless steel (17 Cr, 12 Ni)
Port diameter	495 mm
Horizontal distance between ports	765.05 mm
Vertical distance between ports	2740 mm
Channel spacing	12 mm
Channel width	1100 mm





3.2.3 Design of Gas-Gas heat exchanger

After heating the feed coal, the waste gas is used to preheat the kiln air. For this purpose a plate type heat exchanger is designed using a software package HTRI (Xphe, ver. 5.0). The complete design procedure is discussed below.

The input configuration data used to design a plate and frame heat exchanger is shown in Table 3 where flow rates of waste gas and air are considered as 7 kg/s and 5 kg/s, respectively. The inlet temperature of the waste gas is 191°C, which heats the air from 30 to 114°C. The physical properties of cold and hot fluids are generated through software VMG Thermo. After feeding all the input data and physical properties, the software is run and the solution is obtained. Results are thus calculated by iterative process to minimize the overdesign. The permissible limit for overdesign is considered as 2%. The model of the gas-gas (G-G) plate type heat exchanger is shown in Figure 6.

4. ESTIMATION OF THEORETICAL AMOUNT OF COAL REQUIRED FOR THE OVERALL PROCESS

The Sponge iron production process consumes a lot of energy and the chief source of energy for the process is coal. Therefore the coal consumption is decided by energy demand for the process which depends on heat gained by the entering feed to get heated to reaction temperature, heat involved in the process of reduction, heat lost through the rotary kiln wall and latent heat required by moisture of feed material to be evaporated [14]. The estimation of total heat requirement for conventional process consists of the following expressions.

In the preheating zone inlet air and iron ore are heated upto 1020°C and then reduction takes place in the respective zones. The sensible heat gained by air and ore are obtained from combustion of coal which is computed using the following equations.

$$Q_a = m_a \times C_a \times (T_r - T_a) \quad (6)$$

$$Q_o = m_o \times C_o \times (T_r - T_a) \quad (7)$$

The heat of reaction of the reduction and combustion processes are computed using

energy balance calculations. It is referred as Q_{rxn} .

The actual heat lost through the wall includes the heat lost through the kiln shell, inlet and outlet hoods, post combustion chamber and inlet area of the cooler. Therefore, the total heat loss is considered as twice of that of the kiln [5, 10].

$$Q_{loss} = 2\pi D L h_r \quad (8)$$

The coal is also required to be preheated up to the reaction temperature and sensible heat involved in this process is calculated as:

$$Q_c = m_c \times C_c \times (T_r - T_a) \quad (9)$$

The average moisture contents in coal and iron ore are 3.58% and 2.3% by weight respectively, and heat required for evaporation of is computed as

$$Q_w = (0.023 m_o + 0.0358 m_c) \times \lambda \quad (10)$$

The combustion efficiency is assumed as 67.5%. It is understood that as 67.5% of total fixed carbon available in non-coking coal burns completely to give the necessary heat and remaining 32.5% do not burn. The major fraction of this unburnt carbon is lost to the atmosphere through waste gas as smoke, and rest is discharged with sponge iron from the rotary kiln. The final empirical relation for estimating coal requirement is expressed below in Eq.-11. Thus net heating value, NHV is calculated as 27382.267 kJ/kg from the following equation.

$$Q_a + Q_o + Q_{rxn} + Q_{loss} + Q_c + Q_w = m_c \times NHV \times (0.675) \quad (11)$$

5. RESULTS AND DISCUSSION

In the actual plant, feed mixture (iron ore, coal, and dolomite) and air are fed to the rotary kiln at 30°C where these materials are heated upto reaction temperature of 1020°C. This temperature is achieved by burning of coal inside the kiln. However, by preheating the kiln feed and air with the heat of waste gas outside the kiln, some amount of coal requirement may be reduced. The preheating of coal and air can be carried out using a fluidised bed dryer and gas-gas heat exchanger respectively.

The temperatures of kiln feed and air are increased to 120°C and 114.5°C, respectively through preheating by heat of waste gas. Thus, heating of kiln feed and air are further carried out from 120°C to 1020°C and 114.5°C to 1020°C respectively inside the kiln using heat of combustion of coal. Thus, total coal consumption is calculated and compared with those obtained from usual design. It is observed that coal consumption is reduced from 6.144t/h to 5.95t/h with the inclusion of fluidized bed dryer and gas-gas heat exchanger. The coal consumption is computed using equation-11.

It is seen that when the consumption of coal drops, the quantity of air required to burn it also drops. Therefore the solution for this problem requires a trial and error method. Based on the trial and error method, the final amount of coal required is estimated to be 5.88t/h and the corresponding air requirement comes out to be 23.74 t/h. Detailed computations for this is shown in Table 4. The outlet temperature of the kiln feed is kept constant at 120 °C due to the conveyer problems.

Table 4: Results of iterative approach for coal consumption

S.No.	Coal consumption (tph)	Air requirement	Kiln feed outlet temperature(°C)	Air outlet temperature(°C)
-------	------------------------	-----------------	----------------------------------	----------------------------

		(tph)		
1	6.1442	24.8079	120	109.94
2	5.9541	24.0405	120	113.29
3	5.8973	23.8112	120	114.33
4	5.8838	23.7566	120	114.58

From Table 4 it is observed that the coal consumption is reduced by 4.3%. Due to the drop down in air requirement the heat required to preheat the feed which is supplied by the waste gas, will also drop. Further, due to decrease in the amount of air, the amount of waste gas decreases to 26.3698t/h in comparison with the earlier value of 27.68155t/h. With this modification temperature for waste gas

drops down to 150⁰C which is also a high value of temperature for the waste to be vented out through chimney. The present modification plans to reduce the coal consumption and waste gas generation. To achieve this modification, a fluidized bed dryer is used for preheating kiln feed and a gas-gas heat exchanger is used for preheating air.

Table 5 : Design parameters of fluidised bed dryer

Parameters	Value(m/s)
Minimum fluidization velocity	2.18
Terminal velocity of the particle	22.79
Fluidization velocity	3.609

5.1 Fluidized bed Dryer

The waste gas enters the fluidized bed dryer at 219⁰C with a temperature drop of 1⁰C. The waste gas at 219⁰C is used to pre heat the feed coal from 30⁰C to 120⁰C. The air enters at a velocity of 3.609 m/s through the bed of Geldart-D particles. The design

parameters are shown in the Table 5. The Minimum fluidization velocity and Terminal velocity of the particles are determined to be 2.18 and 22.79 m/s using eq (1-3) respectively. The coal at 120⁰C is then fed to the kiln and the waste gas at 191⁰C is then sent to the gas-gas heat exchanger for further use.

HTRI		Final Results				Page 1	
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Xphe E Ver. 5.00 13-Jun-14 23:57 SN: Friendsl		SI Units					
Rating - Single Pass Countercurrent Flow							
Process Data		Hotside		Coldside		Plate Type 1 Geometry	
Fluid name		Flue gas		air		Manufacturer	(--) Alfa Laval
Fluid condition		Sens. Liquid		Sens. Gas		Plate ID	(--) AX35
Total flow rate	(kg/s)	7.000		5.000		Chevron angle	(deg) 28.00
Weight fraction vapor, In/Out	(--)	1.000	1.000	1.000	1.000	Surface area enlargement factor	(--) 1.21
Temperature, In/Out	(Deg C)	191.00	150.00	30.00	114.00	Plate thickness	(mm) 0.600
Temperature, Average/Skin	(Deg C)	170.50	164.28	72.00	162.52	Area per plate	(m2) 2.699
Skin temperature, Max/Min	(Deg C)	190.12	142.20	183.67	32.51	Plate material	(--) 316 Stainless steel (17 Cr, 12 Ni)
Pressure, Inlet/Outlet	(kPa)	100.002	99.998	90.002	89.201	Thermal conductivity	(W/m-C) 18.00
Pressure drop, Total/Allowed	(kPa)	4.307e-3	94.476	0.802	56.661	Plate Type 2 Geometry	
Port pressure drop, In/Out	(kPa)	-2.92e-4	9.235e-4	0.000	0.000	Manufacturer	(--)
Port velocity, In/Out	(m/s)	4.863e-2	4.857e-2	12.56	16.18	Plate ID	(--)
Nominal channel velocity	(m/s)		1.771e-2		5.27	Chevron angle	(deg)
Average film coef.	(W/m2-K)		661.42		55.88	Surface area enlargement factor	(--)
Heat transfer safety factor	(--)		1		1	Plate thickness	(mm)
Fouling resistance	(m2-K/W)		0.00000		0.00000	Area per plate	(m2)
Fouling thickness	(mm)		0.000		0.000	Plate material	(--) 316 Stainless steel (17 Cr, 12 Ni)
Equivalent shear stress	(kPa)		6.992e-6		1.475e-3	Thermal conductivity	(W/m-C)
Maldistribution parameter	(--)		0.16		0.16	Pack Configuration	
Overall Performance Data							
Overall coef., Design/Clean/Actual	(W/m2-K)	52.307	51.423	/	51.423	Group #	1
Heat duty, Calculated/Specified	(MegaWatts)	22.1296	0.0000			Plate Type 1	1
Effective mean temperature difference	(Deg C)	66.92	0.995	=	66.46	Plate Type 2	1
See Runtime Message Report for Warning Messages.				Channels 40			
				Hot pass # 1			
				Cold pass # 1			
Unit Geometry		Common Plate Geometry				Heat Transfer Parameters	
Total effective area	(m2) 426.448	Channel width		(mm) 1100.00	Overall wall correction factor	(--)	0.9988 0.9730
Number of units, series/parallel	(--) 1 / 1	Channel spacing		(mm) 12.000	Average Prandtl number	(--)	54.86 0.71
Number of passes, hot/cold	(--) 1 / 1	Equivalent diameter		(mm) 19.835	Reynolds number	First group	(--)
Number of channels, total/per pass	(--) 80 / 80	Average plate pitch		(mm) 12.600		Last group	(--)
Number of plates, total/effective	(--) 81 / 79	Port diameter		(mm) 495.00			4431 4496
Number of plate types	(--)	Tightened pack length		(mm) 1008.60	Pressure Drop, % of Total		
Number of channel types	(--)	Horizontal distance of port centers		(mm) 765.05	Channel	(--)	85.2 100.8
Flow configuration, first hot channel	(--) Countercurrent	Vertical distance of port centers		(mm) 2740.00	Other	(--)	14.8 -0.8
Flow path across plate	(--) Diagonal				Thermal Resistance, %		
Port Geometry		Hot Inlet	Hot Outlet	Cold Inlet	Cold Outlet	Hotside	Coldside
Location (front/back)	(--)	Front	Front	Front	Front	7.77	92.02
Location (top/bottom)	(--)	Bottom	Top	Top	Bottom		
Location (left/right)	(--)	Left	Right	Left	Right		
Diameter	(mm)	495.00	495.00	495.00	495.00	Fouling	Plate
Connection diameter	(mm)	495.00	495.00	495.00	495.00	0.00	0.21
Connection material	(--)	316 Stainless steel (17 Cr, 12 Ni)				Over Design, %	
						1.08	

Figure 7: Design specification for G-G heat exchanger

5.2 Gas-gas heat exchanger

Once the coal is preheated, the waste gas is sent to the plate and frame heat exchanger where the preheating of kiln air is carried out. The effective heat transfer area of the heat exchanger is 426.448 m². The overall heat transfer coefficient is estimated to be 52.307W/m² K. The air is preheated up to 114.8°C with an over design of 1.08%. The

design specifications are shown in the Figure 7.

After heating the air, the waste gas at 150°C passes through the return duct to the chimney and is released to the atmosphere thereof. The detailed material balance for the process with design modifications along with the equipment is shown in Figure 8.

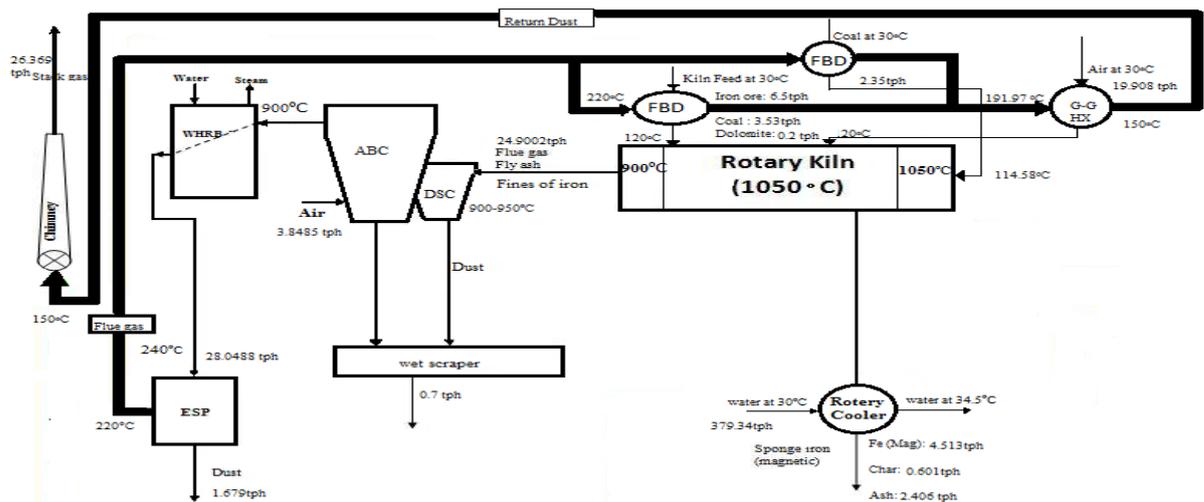


Figure 8: Modified Flow Sheet of Sponge Iron Process

6. CONCLUSION

The results, obtained by the inclusions of the modifications are compared with the actual process with respect to coal and air consumption. The salient features of conclusions for the present work are as follows:

- The heat integration of the system is carried out by recovering the heat from the waste gas at 220°C. For this purpose modifications are considered in the existing system where the heat available with waste gases are integrated in the process to preheat kiln feed mixture and air.
- The additional equipment required for this modification include a duct to carry the waste gases from ESP to rotary kiln, a Fluidized bed dryer for preheating kiln feed mixture from 30°C to 120°C, a gas-gas heat exchanger for preheating kiln air from 30°C to 114.58°C and a return duct to carry the waste gases back to chimney.
- The waste gas at 219°C is used to pre heat the feed coal from 30°C to 120°C. The air enters at a velocity of 3.609 m/s through a static bed of 44cm. This reduces the coal consumption from 6.144t/h to 5.88 t/h, which is 4.3% less in comparison to the existing system. The corresponding air requirement comes out to be 23.74 t/h

- The plate and frame type heat exchanger is designed. The effective heat transfer area of the heat exchanger is calculated to be 426.448 m² which is used for preheating of air from 30°C to 114.8°C. The overall heat transfer coefficient is estimated to be 52.307W/m² K with an over design of 1.08% using software package HTRI (Xphe, ver. 5.0).
- Further, due to decrease in the amount of air, the waste gas amount also decreases to 26.3698 t/h in comparison to the amount of 27.68155 t/h for the existing system.

ACKNOWLEDGEMENT

The work has been carried out in National Institute of Technology, Rourkela, Odisha, for the project funded by CSIR, Govt. of India. The authors acknowledge the financial support provided by CSIR, Govt. of India and other supports provided by the Director, NIT Rourkela.

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ISOTHERMAL FORGING
A State-of-art of forging process for manufacturing of aero engine Components.

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ABSTRACT

Titanium and titanium alloys are used extensively both as structural materials and engine component materials in aerospace applications because of their high specific strength which results in significant weight reduction.

Due to increasing demand for higher performance of titanium alloys in aerospace sector requires new processing technique to be developed to tailor the microstructure. This requirement gives a reason to develop a unique forging method known as isothermal forging. The isothermal forging technology emerged as development efforts in the early 1970s and became a production reality shortly thereafter.

Basically this is a forging process where the forging stock and the die temperature maintained at equal/ near equal temperature throughout the forging cycle. Slow deformation within tight temperature ranges minimize flow stress, minimize deformation heating, and allow super plastic behavior of titanium alloys. The process itself claims to produce net/near-net shape components, leading to optimum utilization of materials, product Uniformity and optimum mechanical properties.

In aero engine industries engine inlet fan blades and compressor blades are manufactured to near net shape from many years .However blades in recent advanced aircraft engines i.e Al-31FP, Pratt & Whitney F119-PW-100, RD33 engine are made from difficult to forge alloys with very thin aerofoil with less tolerance for thickness and geometrical deviations. Due to increasing demand for thinner aerofoil, sharper edge radii, close tolerances and high strength to density and modulus to density ratio Isothermal forging of Ti-alloys came in to picture for producing fans and Compressor blades of aero engine.

PROCESS DESCRIPTION

The process defines that **the dies are maintained at the same temperature as the forging stock**. This eliminates the die chill completely and maintains the stock at a constant temperature throughout the forging cycle. The process permits the use of extremely slow strain rates, thus taking advantage of the strain rate sensitivity of flow stress for certain alloys. Alloys forged using these processes include titanium alloys, such as Ti-6Al-4V, Ti-6Al-2Sn-4Zr-2Mo, and Ti-10V-2Fe-3Al.

The process is capable of producing net shape forgings that are ready to use without machining or near-net shape forgings that require minimal secondary machining.

The process is

- Capable to Produce, forging of very thin section having close tolerances.
- It decreases the traditional material allowances between the forged shape and the finish machined shape.
- Increasing the utilization factor of materials.

The biggest advantages to use isothermal forging in aerospace industries are

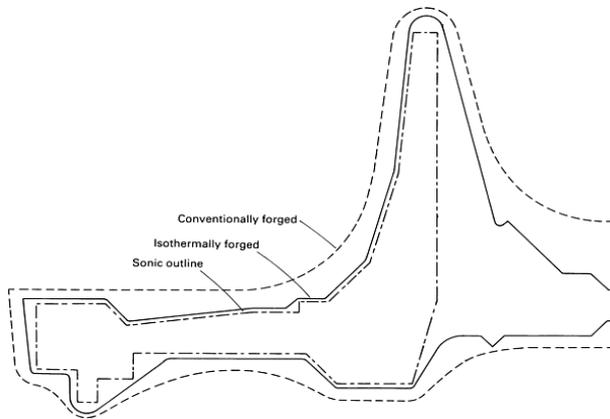
1) Reduced material cost

Isothermal forging is designed with smaller corner and fillet radii, a smaller draft angle, and smaller forging envelope reduce the additional material. About 40% reduction in material ultimately reduced the material costs.

2) Reduced machining

Less material removal is required during machining comparable with conventional forging. No machining or only finish

machining cuts are required to produce the final part. Sometimes elimination of complex machining



Material allowance comparison between conventional forged and isothermal forged part.

3) Uniformity of product

Isothermal forged parts are isotropic in nature because of lower or nonexistent

thermal gradients with respect to die. Hence producing fine equiaxed structure & free of slip banding.

<u>Micro structure of conventional forging part</u>	<u>Micro structure of Isothermal forging cylindrical part</u>

4) Close tolerance thin forgings

Close tolerance forging of +/-0.04mm is possible.

5) Decrease of forging pressure

Due to super plasticity of deformation less forging pressure required to deform the stock material.

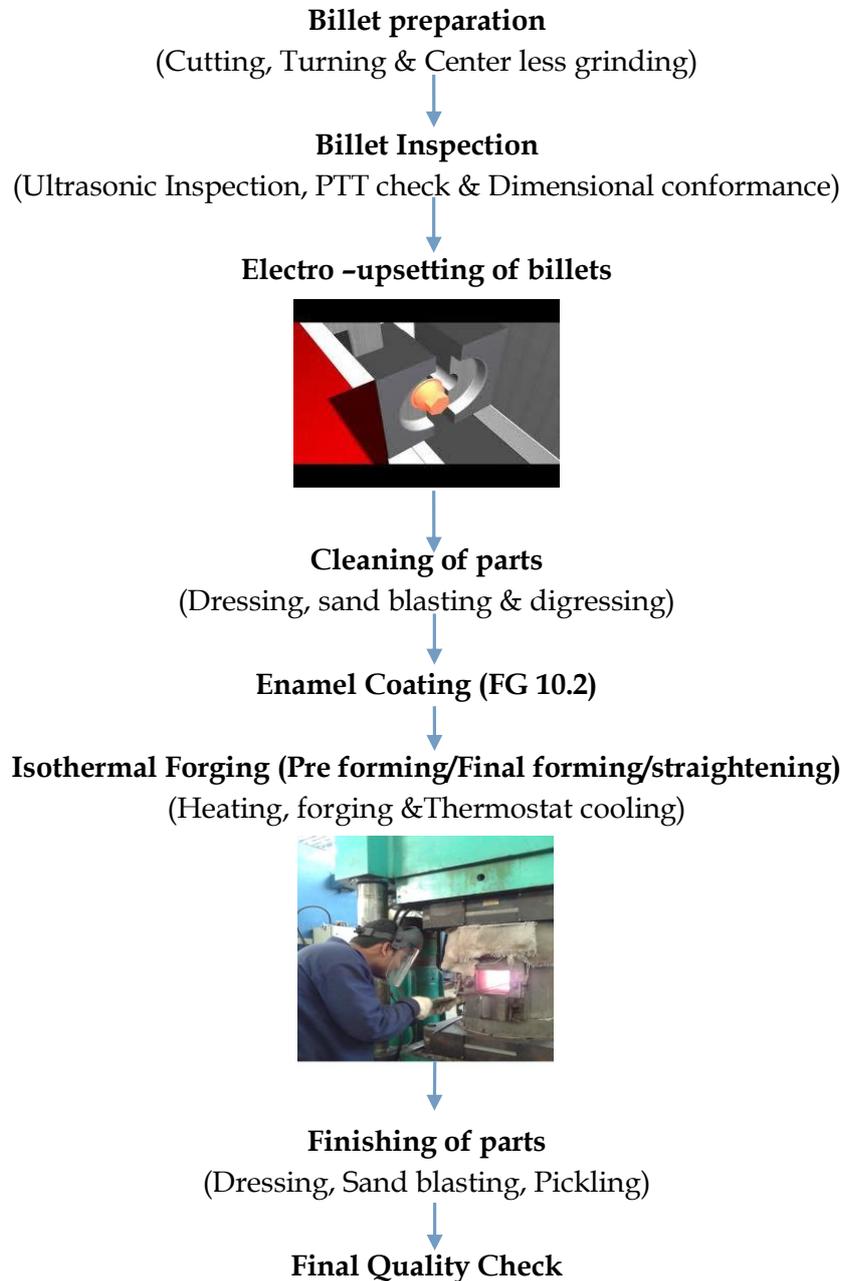
Glimpse of Isothermal forging In HAL

Al-31 FP Engine consists of four stages of low pressure compressor rotor blades i.e inlet fan blades required high fatigue properties due to its operating in severe fluctuating guest loads from inlet air. Titanium alloy is used to manufacturing of these fan blades due to high strength to weight ratio through isothermal forging

routes .from these four blades two blades are winglet shape design i.e. 2nd stage and 3rd stage of LPC rotor blades due to its thinner aerofoil and larger shape. Apart

from three compressor blades, 9 types of engine components made of titanium alloys are manufactured by this process.

The details processing steps for isothermal forging of low pressure compressor blades are



✚ Billet preparation

The raw material i.e titanium billets are cut as per technological length and then turned in diameter wise with proper chamfering i.e 3X45 °. For better finishing of surface, billets are grinded in center less grinding machine.

✚ Billet inspection

The billets are checked in immersion ultrasonic test for defect if any. The samples of billets are tested for find out P.T.T (polymorphic transformation

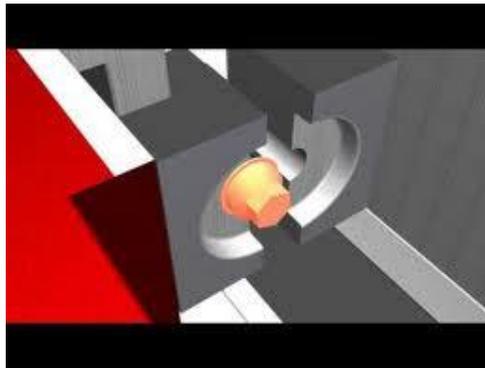
temperature).forging temperature: P.T.T – 35 °C.

one end of a round bar by localized heating the bar end electrically and pushing it against a flat anvil or shaped die cavity.

✚ Electro –upsetting of billets:

Equipment: 250 KVA Electric upsetters.
Materials: Titanium(BT-6).i.e Ti-6Al-4V

Electro-upsetting is the hot forging process of gathering a large amount of material at



✚ Cleaning of parts

Electro upsetted parts are then dressed to remove burrs and defects on the surface of the billets. Then it is sand blasted i.e electro corundum sand to clean the surface.

for the application of a protective lubricant coating onto titanium, nickel and super alloys billets for forging and extrusion. FG 10.2 coatings assist lubrication and contain superior protection vs oxidation properties for high temperature forming operations in combination with die lubricants.

Cleaning time: 15-30 min

Advantages

Degreasing: Degreasing is carried out to clean the surface and make the defect clear visible if any in the part.

- Protects billets in the furnace, which reduces gaseous diffusion and intergranular defects.
- Lead free and solvent free
- Assists to control the metal flow, which reduces finishing costs
- Reduces press force during forging

Bath Content: Caustic soda: 5-15 Gr/lit
Calcium Bicarbonate: 20-25 Gr/lit
Trinitro phosphate: 30-70 Gr/lit
Liquid Glass: 10-20 Gr/lit

Temperature of Bath: 60-90°C
Degreasing time:10-15 min
Washing by hot and cold water bath

Application Thickness: 30 - 70 microns for blades and other parts

✚ Enamel coating

Application Procedure:

Billets are coated by dipping or by spraying in the prepared glass slurry i.e FG10.2 having viscosity in the range of 15-18 second. FG 10.2 is a dispersion of a specially developed, rigorously controlled glass frit in clay-water slurry. It is designed

Billets are heated in a dryer for 20-30 min at temperature of 100-200°C and then dip or sprayed by enamel slurry. Then the coated billets are dried in air for 1.5-2.0 hours and then inside drying oven for 15-20 min at temperature of 50-100°C

Enamel coated parts



✚ Isothermal forging

The coated billets are then loaded in the furnace for heating before deformation

Heating temperature (P.T.T -35°C) $\pm 10^{\circ}\text{C}$

Soaking time: 25 min-30 min

Max. Heating time: 80 min.



Equipment: - 630 ton hydraulic press

Die heating temperature: (P.T.T -35°C) $+ 10^{\circ}\text{C}$, $- 30^{\circ}\text{C}$

Die heating medium: Resistance heating or Induction heating type.

Pressure: 50-110 Kgf/cm²

Holding time under pressure: 50-90second

Thermostat Cooling

After forging the billets are transferred in the thermostat furnace heated at $550-600^{\circ}\text{C}$

Holding time: 5-15 min, Rotating speed of thermostat bed: 0.5-0.7RPM

Transferring time of forged blade from die to thermostat < 6 second.

Cooling of blades: In air

✚ Finishing of parts

Forged blades are then process for fins cutting and followed by sand blasting which removes the surface enamel and make the surface clear. The defects are then removed and the blades are sends for pickling.

Pickling solution: Nitric acid: 720 ml/L

Hydro fluoroc acid (Hf) 180-200 ml/L

Temperature of solution: $< 30^{\circ}\text{C}$, pickling time 3-5 min.

Final quality check

After all the operation completed, the blades are checked for geometry condition in pomkl.

Pomkl is an optical measuring device, which measures the projected shadow of blades profile through contact pins on a transparent sheet having scale. The allowance on roots and profile are clearly

visible in the pomkl and is very fast to measure the profile geometry than the template and fixtures. The blades are then segregated according to their geometry and profile condition and which is known as set making. It helps better ways to distinguish the blades profile allowances.

The photo given below shows the shadow of contact pins of fixture projected on the transparent screen of pomkl.



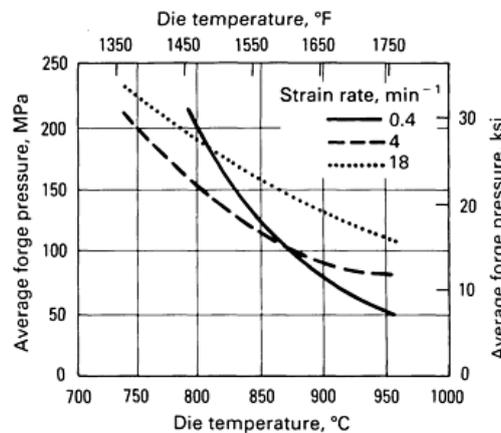
Isothermal forging Process Design:

Because of tighter forging designs and the requirements for strict uniformity and consistency, stringent controls on the following process parameters are necessary.

Forging parameters such as forge temperature, strain rate, pre form microstructure, forging pressure, and dwell

time are all important factors in deciding the degree of dimensional sophistication and the resultant microstructure of the finished part. Some of critical parameters are

- **Die temperature:** Effect of die temperature on forging pressure at various strain rates for Ti-6Al-4V is shown in the graph.



Decrease in die temperature from 955 to 730 °C result in doubling the forging pressure and affect the shape capability available. It will also have an impact on the selection of die materials and economics and for some alloys; the surface microstructure is affected by die temperature.

- **Lubrication:** Isothermal dies lubrication plays an important role because of the precision of the Forgings, the existence of net surfaces, and the high interface temperatures.

Lubrication i.e enamel is giving in the form of coating on the surface of billet which act as parting agent and make the metal flow easily on the surface of dies at high temperature.

Lubricants also protect the surface of the blade and give the blade super finish for higher die temperatures, glass frits with

proper additives or boron-nitride coatings are Wider use.

- **Pre Form Design:** For complex parts pre forming operation is carried out prior to final forging. Pre form design must also take into consideration, the amount of deformation needed during the finish forge operation to obtain the desired mechanical properties.

Die Systems:

Proper selection of isothermal die material depends on the operating temperature, forging pressure requirements, and anticipated die life.

A special kind of alloys known as ZC6Y (I.e. Ni-V-Mo) is used as die material. The die is manufactured in the process of Vacuum casting and machined by EDM process.



Isothermal Blade Die (top & bottom) with die holder Die assembly in 630tonhyd.press

During designing of the dies for isothermal forging, calculation of the die shrinkage allowance is important due to tight tolerances associated with the parts. When fairly tight draft wall and/or complex contours are in the forge design, segmented dies with a holder system are used to

achieve accuracy while maintaining the ease of removing the forging from the dies.

Isothermal forging is one of the emerging technologies used for manufacturing of aero engine components of AL-31FP engine which produce optimum quality product with less /nil rejection.

MANUFACTURING OF SINGLE CRYSTAL BLADE CASTING FOR AERO ENGINES

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ABSTRACT

Something exciting has been happening during these last several years in the field of aero engine blade castings. Initial development was polycrystal castings and then came DS (Directionally solidified) castings and the latest one is single crystal castings.

A new precision casting technique, based on directional solidification, which imparts significantly improved ductility and thermal shock resistance to high temperature creep resistant, nickel-base super alloys, has been carried through from research to production.

This controlled solidification technique has been used to produce single crystal gas turbine components. The improvement in physical properties is achieved by controlling the solidification process to produce a complete single crystal throughout the part, with a preferred [001] crystallographic orientation. This single crystal structure and orientation is established parallel to the major stress axis of the part with the use of a "seed crystal".

HP turbine blade is cast by this method. The Single Crystal (SX) castings are produced basically by keeping the molten metal in the mould in the liquid stage itself and allow it to solidify in the desired manner. In the conventional castings the molten metal starts solidification as soon as it comes into contact with the mould walls. This is avoided in case of SX castings. The mould is kept at high temperatures (1500-1600 °C), almost close to the pouring temperature, and then solidification is allowed to start at a predetermined place, usually the bottom of the mould. Solidification starts at the 'starter' at the bottom, which is a Ni-W button (there is provision in the shell to place this). Every shell has 6 blades. 8 coatings form the ceramic shell. The 'starter' is checked for its crystallographic orientation before placing in the mould.

Key words: *single crystal casting; directional solidification.*

INTRODUCTION

1. CONDITIONS FACED BY HPTR BLADES

- High turbine entry temperature
- Non uniform temperature distribution on the profile
- Centrifugal stresses
- Creep
- High Cycle Fatigue
- Low Cycle Fatigue
- Hot Corrosion

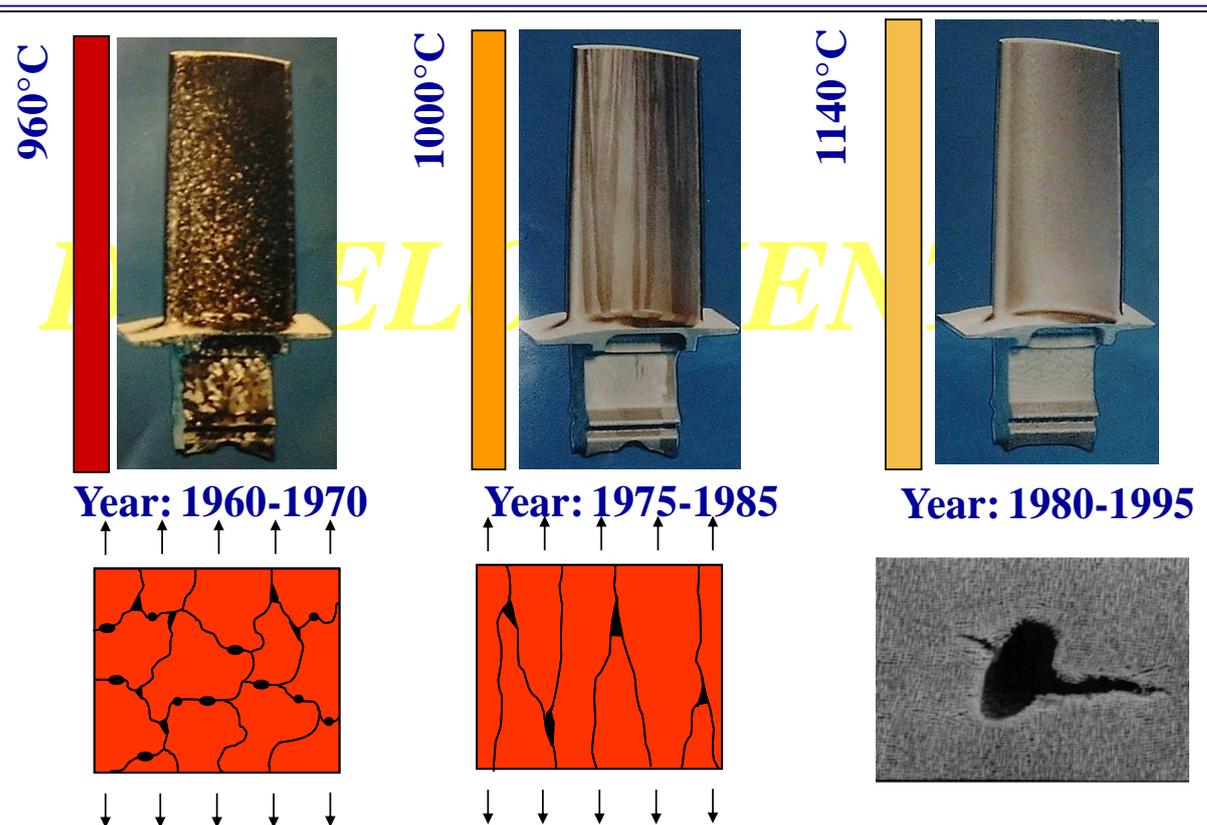
2. GAS TURBINE PERFORMANCE IMPROVEMENT TECHNIQUES

- Primary goals in the development of aircraft gas turbines are increased operating temperatures and improved efficiencies, i.e. higher TET
- High TET benefits thrust, thermal efficiency and specific fuel consumption

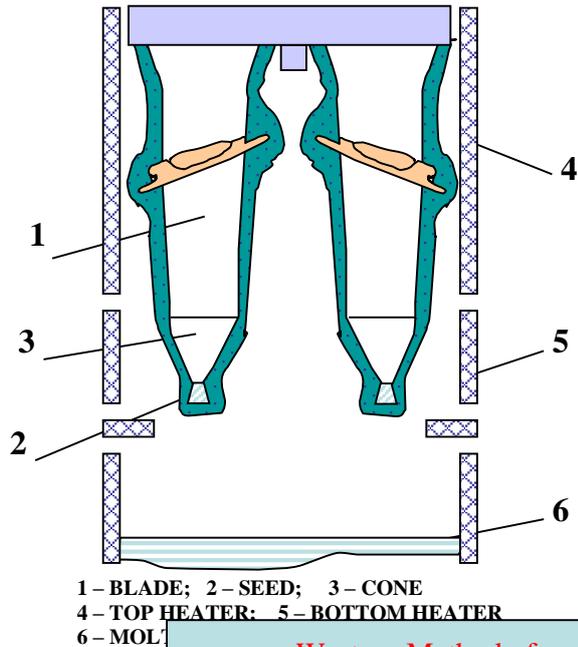
- Higher TET is achieved by: - Efficient design for air cooling of blades i.e. serpentine convection and film cooling (TET ~ 1340°C)
- Improved super alloys and casting processes – Directional solidification and Single crystal solidification (TET ~ 1600°C)
- Improved Thermal barrier coating processes.

3. COMPARISON OF CASTING PROCESS

EQUIAXED	DIRECTIONALLY SOLIDIFIED (DS)	SINGLE CRYSTAL
MULTI GRAIN	All grains oriented in one direction =>(001) crystallographic orientation	Single grain =>(001) crystallographic orientation
GRAIN BOUNDARIES IN ALL THE 3-AXES	No grain boundary is normal to principal stress axis	No grain boundary at all in any axis
MODERATE CREEP RUPTURE LIFE	6-fold increase in thermal fatigue to grains with low modulus of elasticity	10-fold increase in thermal fatigue life
MODERATE THERMAL FATIGUE LIFE	3-fold increase in creep rupture life due to absence of transverse grain boundaries	9-fold increase in creep rupture life due to absence of grain boundaries
MODERATE CORROSION RESISTANCE	2-fold increase in corrosion resistance	4-fold increase in corrosion resistance
[O] & [N] CONTENTS AROUND 50 PPM EACH	[O] & [N] Contents around 5-10 ppm each	[O] & [N] Contents around 5-10 ppm each

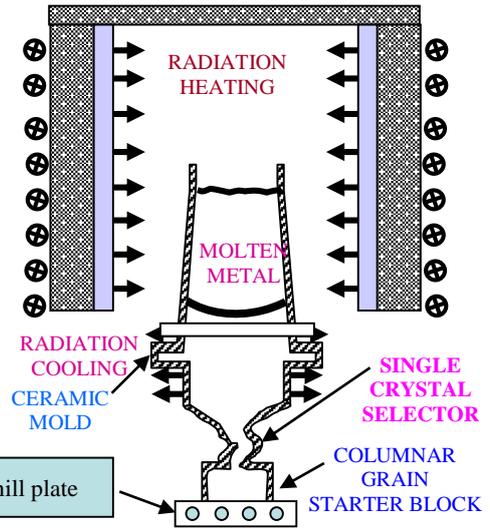


COMPARISON OF RUSSIAN PROCESS AND WESTERN PROCESS FOR PRODUCTION OF SINGLE CRYSTAL BLADE CASTINGS



1 - BLADE; 2 - SEED; 3 - CONE
4 - TOP HEATER; 5 - BOTTOM HEATER
6 - MOL

Western Method of DS/SC Solidification



RUSSIAN METHOD	WESTERN METHOD
Ceramic shell is preheated to within 10-15 °C of pouring temperature in mould heater.	Ceramic shell is preheated to 1600°C
Ceramic shell has a single crystal seed at the tip	Ceramic shell has a helix at the bottom which sits on water cooled copper chill plate
After pouring, the cast shell is gradually withdrawn from the mould heater and dipped in molten Aluminium bath	After pouring, the cast shell is gradually withdrawn from the mould heater
Growth of single grain of (001) orientation from the seed crystal	Growth of single grain of (001) orientation from the helix
Entire casting operation takes 1.5 hrs for 2 shells	Entire casting operation takes 2 hrs for 1 shell

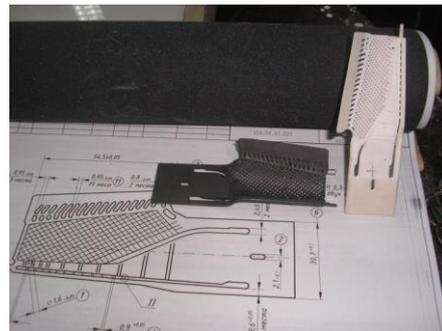
PROCESS FOR MANUFACTURING OF SINGLE CRYSTAL BLADE CASTING



A) BLADE CORE MANUFACTURING:



Ceramic core Making



Sintered ceramic core

CORE MASS:

1. ELECTRO-CORRONDUM
2. TITANIUM DI-OXIDE
3. ALUMINIUM POWDER (ACD-4)
4. QUARTZ SAND (Marshalite)
5. GRAPHITE POWDER
6. PLASTICIZER

CORE INJECTION:

- The core mass is injected into the die and the raw core is cleaned and loaded in a ceramic box packed with alumina powder and loaded for sintering.

SINTERING CYCLE:

- STARTING TEMP. NOT EXCEEDING 50 °C
- RAISE UPTO 300 °C AS PER 50 °C IN HOUR
- FROM 300 TO 1350 °C ARBITRARY AND HOLD FOR 10-12 Hr.
- FURNACE COOLING UPTO ROOM TEMPERATURE

B) BLADE PATTERN MANUFACTURING



Pattern Assemblies

INJECTION PARAMETER OF PATTERN:

- TEMP. OF PATTERN MASS =75-80 °C
- INJECTION PRESSURE = 4-5 Kg/Cm²
- HOLDING TIME =40-50 SEC
- LUBRICATION SOLUTION=30% CASTOR OIL+ 70% SPIRIT

C) BLADE SHELL MANUFACTURING



HYDROLYSER FOR PAINT:

- 3TC-40 (Ethyl silicate 40)
- NITRIC ACID
- RECTIFIED SPIRIT
- WATER

FILLER MATERIAL :

- M7 / M10
- M-40
- M63
- Pure Aluminium powder

The assembled patterns are dipped in the slurry and then in stucco bath. After each coat shells are dried in a vacuum chamber under ammonia. Like this 8 coats are given. After this shells are eliquated in an autoclave.

D) BLADE CSATING MANUFACTURING:



Single crystal Vacuum melting Furnace

- Seed crystal made of Nickel – Tungsten alloy which is also a single crystal is placed at the bottom of the shell and is checked for crystallographic orientation before placing in the mould.
- Every shell has 6 blades.
- Two shells are poured simultaneously. The weight of the metal is 8.5 Kg.
- Ceramic shell is kept at a temperature of 1590-1600°C, almost close to pouring temperature.

After pouring ceramic shells are lowered gradually into molten aluminium bath. The withdrawal rate is 8.5 to 11.5 mm/Min. The Temperature of the molten Aluminium is 680-740°C. This is maintained by the radiation heat of the bottom shell heater. In molten Al bath it is kept for 5 minutes. Then shell is shifted from Al bath to shell heater which is set at 1280 degree C. Then shell is kept for 10-11 mis and then shell heater is switched off. The shell is taken out after the temperature reached below 700 degree C.

The Temperature of top heater is 1545 °C
 The Temperature of bottom heater is 1625 °C



Shell hanging arrangements



Cast Block

D) PROCESSING OF BLADE CSATINGS:



Blade Macro-Structure



(001) Grain orientation

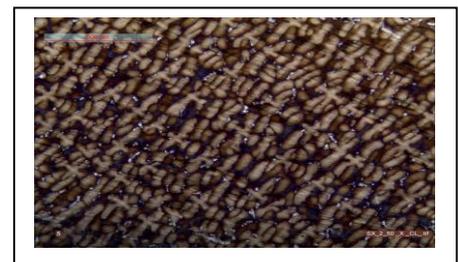


Blade after cutting

PICKLING :

1. HNO₃
2. HF
3. WATER

Holding time in solution =30-60 min for Blade
 = 20-40 min. for cone.



Microstructure of sx blade

GRAIN ORIENTATION CHECK:

X-Ray Diffractometer: Checked for grain orientation
Deviation from stacking allowed for blade: 10 Degree
Deviation from stacking axis allowed for seed crystal: 5 Degree

LEACHING (CORE REMOVAL):



Solution:

- $KF_2 = 75\%$ AND $KF_3 = 25\%$ (mixture)
- Blades with basket loaded in the molten solution at a temperature around 320-340 degree C with rocking mechanism and hold for 4-4.5 hrs.
- HOT WATER (60-80 °C) WASHING FOR 5 MIN.
- WASHED IN HOT RUNNING WATER (80-90 °C) FOR 15 MIN.
- HOT WATER (80°C) JET CLEANING WHICH IS AT 6-7 ATM. FOR 40

Blades for Leaching

Brazing & Heat Treatment:

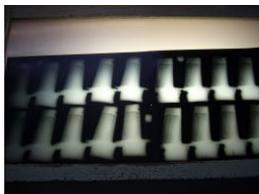
PROCESS:

- PLUG FITTING
- BRAZING OF PLUGS
- X-RAY INSPECTION OF BRAZING OF PLUGS
- THERMO-VACUUM PROCESS
 1. $1280 \pm 10^\circ\text{C}$ for 10-15 Min. (for brazing) / 70-30 min. (for HT) at vacuum 10^{-4} Mm Hg.
 2. Furnace cooling to 200°C and then removal of blade.



SX Vacuum Brazing & Heat Treatment furnace

X-RAY FOR CORE REMOVAL & METALLURGICAL DEFECT:



X-RAY FILM FOR CORE REMOVAL



X-RAY FILM FOR METALLURGICAL

QUALITY CONTROL REQUIREMENTS:

FPI, Chemical analysis, Dimensional check.

ALLOY COMPOSITION (Wt %)

Element	C	Cr	Co	W	Mo	Ti	Al	Nb	V	Hf	Ta	Re	Ni
%	0.13-0.18	4.3-5.6	8.0-10	7.7-9.3	0.9-1.1	-	5.6-6.3	1.4-1.6	-	-	3.5-4.5	3.5-4.5	Base

CONCLUSION

The yield of single crystal blade casting by using seed crystal is 25-30 %. The Single crystal blade which withstands high turbine entry temperature (TET) is because of the alloy which contains rare earth elements like Re and Ta, single crystal structure with serpentine cooling passage inside the blade in as cast condition, which is produced by using sintered ceramic core. Also the casting is given thermal barrier coating on external surface after complete machining to withstand this high TET. The next trial is taken by using helix core with seed crystal to improve the acceptance rate of the casting. By taking trial on fifty batches of blade casting the yield has been improved from 25-30 % to 30-35%.

➤ **SGAT NEWS**

- Mineral Development Awareness and Quiz (MDAQ) Programme for 5th year post graduate and 3rd year engineering students was conducted in Joda Valley Club of Tata Steel from 22-24 August 2014. Altogether, 18 teams participated in the event from Odisha, Jharkhand, Chhattisgarh and West Bengal. The SGAT MDAQ shield was won by Utkal University Geology team.
- Shri B.C. Patnaik, General Secretary and Shri B.K. Mohanty, Advisor called on Secretary, Dr. A.K. Pujari, Ph.D., IAS, Dept. of Mines, Govt. of India, at State Guest House, Bhubaneswar on 27.08.2014 and apprised about the activities of SGAT and its role in mineral development of the country as well as Odisha. Dr. Pujari was requested to be Chief Guest in the MINEXPRO – 2014.
- Workshop on Geoheritage and Geotourism Potential of Odisha was organized on 14.09.2014 at Bhubaneswar in association with Indian National Trust for Art and Cultural Heritage (INTACH). The workshop was inaugurated by Hon'ble Minister S.J. Ashok Chandra Panda, Tourism and Culture Department.
- National Seminar – Vision – 2010 for Metallurgical Industries (VMI-2020) was organized in association with Indian Institute of Metals, Bhubaneswar Chapter on 11-12 October 2014 in spite of threat from cyclone “Hud Hud”. The seminar was inaugurated by Sri Arun Kumar Mishra, Vice President, Tata Steel.
- 2nd Indian Mining Day was observed in the SGAT Hall on 01.11.2014. The celebration was conducted jointly by Mining Engineers’ Association of India, Bhubaneswar Chapter. Dr. S.K. Sarangi, President, SGAT and Er. S.K. Patnaik, Chairman, MEAI, Bhubaneswar Chapter addressed the gathering. Prof. M.C. Dash, spoke on “Green Mining” in the meet.
- A Press Meet was conducted on 01.11.2014 in SGAT Hall to spread the news on the purpose of celebrating mining day. Also the media was informed about present situation of mining industry and views of SGAT on that.

➤ **NEWS ABOUT MEMBERS**

- Dr. S.K. Sarangi, President, SGAT was conferred with “Outstanding Leadership Award” by Mining Engineers’ Association of India during All India Exploration Geologists Meet – 2014 at Hyderabad on 24.08.2014.
- Dr. B.K. Mohapatra was awarded with Master Tanay Chadha Memorial Geologist Award by Mining Engineers’ Association of India on 05.08.2014 in the Annual General Body Meeting for his significant contribution to the exploration of minerals.
- Prof. Dr. S. Acharya was felicitated by Odisha Bigyan Academy as Senior Scientists by the Hon’ble Chief Minister S.J. Naveen Patnaik on 05.08.2014 for his significant contribution to Earth Science.
- Dr. Birendra Kumar Mohapatra was conferred with Samanta Chandra Sekhar Award for his outstanding work on Physical Science by Odisha Bigyan Academy.
- Mr. Manoranjan Panda, Geologist was promoted to the post of Dy. Director of Geology who assigned the office on 03.07.2014 in the Directorate of Geology, Bhubaneswar.
- Mr. S.K. Popli, Special Secretary in Steel & Mines Dept. was transferred and posted as CEO in CESU, Bhubaneswar.
- Er. Surendra Nath Padhi was felicitated by SGAT & MEAI on 01.11.2014 on the eve of celebration of INDIAN MINING DAY.
- Dr. Hara Prasanna Misra was conferred with “Kalinga Samman – 2013” by Kalinga Foundation Trust for his outstanding contribution for popularization of science amongst the common man on 15th September 2014.
- Er. Surendra Nath Padhi, former Director General, DGMS & former Independent Director, Board of Orissa Mining Corporation Ltd., was presented with “Golden Jubilee Life Time Achievement Award” for his lifetime contribution in the field of Mines Safety and Technological Innovations in the Seminar “Sustainable Development in Mineral Industry” held at New Delhi on 21.06.2014.

NEW MEMBERS

1. **Mr. Karun Kant Dave**
COO, SESA Sterlite Ltd.
Sesa Sterlite Ltd.
Alumina Refinery Unit
Po: Lanjigarh,
Dist: Kalahandi
Odisha
2. **Mr. Raj Gurung**
Head, Mining & exploration
Sesa Sterlite Ltd.
Alumina Refinery Unit
Po: Lanjigarh,
Dist: Kalahandi
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3. **Dr. Alok Kumar Mohapatra**
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4. **Mr. Prem Chand Bakliwal**
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5. **Mr. Anil Mathur**
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Kothangudem,
Dist: Khammam -
Telegana
8. **Mr. Biraja Nandan Bhol**
Retd. Jt. Director, Geology
Parvati Niwas
Sriram Nagar
Madhupatna
Cuttack – 753010

Condolence Messages received from the following persons.

• **A.B. Panigrahi**

- My deepest condolences on the sad demise of Shri Binod Chandra Patnaik.. May God give his family the strength in this hour of grief. I pray God may his soul rest in peace.

• **B.K. Mohanty**

- Binod was to accompany me to Matikhal today and to Tikri&Damojnodi later this month in connection with EMAP. That was not to be. His passing away has left a void which can never be filled up. Binod was a dedicated, disciplined and conscientious officer, attributes are not seen among the officers of today. He has been an Architect of the land mark investigation of Bauxite deposits in Gandhamardan hill. He was ever willing and helpful to human beings with a development buyers. SGAT will miss him and me for a years to come. He was by far the best officer in the Directorate of Mining & Geology and his contribution to the development of SGAT has been immense.

May his soul rest in peace. My condolence to be the members of the bereaved family.

• **A.S. Rao**

- I am extremely sorry to note the sudden and untimely demise of our good friend and your right hand man Sri. BC.Patnaik, an eminent geologist and administrator. It is very difficult to fill the gap left by Late Sri.BC.Patnaik.

Please convey my heartfelt condolences to Smt. Patnaik and other family members.

• **U.K. Mohanty**

- Extremely sad to know about untimely demise of Mr. B.C. Patnaik. His ever smiling face and kind response to any problem or query is unforgettable. May God give his family strength to endure the loss. May his soul rest in peace.

• **M.D. Behera**

- Mourning the sad demise of BC PATNAIK. May his soul rest in peace.

• **Dr. S. K. Biswal**

- I am extremely sorry to hear the sad demises of Mr. B. C. Pattnaik, General Secretary, SGAT on 18th November 2014 due to health problem. His contribution towards earth science particularly in Odisha as well as SGAT is tremendous. I pray to Lord Jagannath to give strength to his family members for the huge loss and rest his soul in peace.

• **B.S. Pani**

- Very Sad! Binod babu was the life our Society. We will all miss him very badly. May God give strength to the family to bear this terrible loss.

• **Sanjay Patnaik**

- Deepest condolences to the bereaved family on the passing away of Mr. B. C. Patnaik. SGAT in particular and the mining fraternity in general has lost an able ambassador. May his Soul rest in peace.

• **Shoubhaya K. Tripathy**

- I share my deep condolences for the sad demise of Mr. Pattanaik B. C. May his soul rest in peace.

• **Omkar Nath Mohanty**

- It is shocking news! I join all members of SGAT in conveying our deep condolences to the bereaved family.

• **Anil Mathur**

- Our heartfelt condolences to the bereaved family. Anil Mathur & MEAI Jaipur members.
- **Rabi Narayan Mishra**
 - My deep condolences. My God, what a calamitous info., loved BC so much like so many of our colleagues. The news hurts my heart. My wishes and condolences for the bereaved family, Gurubhai.
- **V.S. Rao**
 - Very sorry to know, heartfelt condolences.
- **Hari Sarvotham**
 - Saddened by the demise of Mr. B.C. Patnaik. RIP. I pray for his soul.
- **R.N. Gupta**
 - I am very sad to hear the demise of MR BINOD CHANDRA PATNAIK, GENERAL SECRETARY, SOCIETY OF GEOSCIENTISTS AND ALLIED TECHNOLOGISTS (SGAT) AND FORMER DIRECTOR OF GEOLOGY & MINES, ODISHA
- **Bhima Rao**
 - It is a really a shocking news that the sudden demise of B.C. Patnaik. We pray god to take his soul to heaven.
- **Desh Sikka**
 - May God bless his soul and give strength to bear the loss of loved one.
- **Vivek Patnaik**
 - My deepest condolence to the family. May his soul rest in peace.
- **Jayanta Nanda**
 - Very sad news. May his soul rest in heavenly peace.
- **M.K. Somani**
- I am shocked to listen the sad demise of Sh. B C Patnaik. I pray to the almighty god to let his soul rest in peace and to give strength to his family members for the irreparable great loss.
- **S.K. Mishra**
 - Mr. Binoda was my class mate & was kith & kin to me. Although I have attended yesterday, but I am unable to believe the same, as it happened so quickly. Let God give strength to his family members. What more I can do?
- **T.M. Kelly**
 - My heart grieves to hear that such a lovely human being and dear friend of mine, Binod Patnaik Sir, is no more. It was a shock to hear of his sudden demise. I pray GOD THAT HIS SOUL RESTS IN PEACE. And also pray that his bereaved family gets the strength to bear this immense loss. His passing away has created a vacuum in SGAT & in the Geological & Mining Fraternity. We will definitely miss him.
- **Mahapatra Girija Prasad**
 - Untimely demise of Sri B.C. Patnaik, is loss to the geoscientific community of the State and country. May God give us the strength to bear on such loss. All sympathy for the bereaved family.
- **R.H. Sawkar, Geological Society**
 - We regret very much to hear about the sad demise of Shri B.C. Patnaik. We pray god to rest his soul in peace. Our condolences to the bereaved family.
- **S.K. Tamotia**
 - Please convey our heartfelt condolences to the bereaved family members. May God bless the noble soul to rest in peace.

- **M.C. Dash**
 - I deeply condole the untimely and sad demise of Mr. Binod Patnaik. I was shocked and stunned. SGAT lost a very good member and secretary.

He was such a jovial and nice person, we all will miss him. Kindly send SGAT's condolence message to Mrs. Patnaik

- **M. Mukherjee**
 - The sudden and premature demise of Mr. Binod, a person with a noble heart, is indeed shocking. It is a painful loss to our Society. Please convey my condolence to the

bereaved family. His soul will rest in peace. May the Almighty bless all his family members.

- **Bibhuti Sarangi**
 - I had been to Puri on 18th morning & returned today at 06.00 PM. I opened my mail to get this shocking news. What happened to him and how can he expired. I am really shocked. Bye.

- **O.P. Varma**
 - Shocked to know the untimely sad demise of Binod Patnaik. Our heartfelt condolences to the office-bearers of SGAT & to his family members.

OBITUARY



Binod Chandra Patnaik
(01.07.1952 - 18.11.2014)

It is with profuse sense of sorrow we announce the untimely sad demise of Binod Chandra Patnaik, who held the office of General Secretary, Society of Geoscientists and Allied Technologists, on the 18th November 2014 at Bhubaneswar. The Society has incurred an irreparable loss in his death. He had joined the Society in its early stages and took over the responsibility of General Secretary in the year 2009 and delivered with utmost sincerity. He was enthusiastic in all the developmental activities of the Society and has helped in the collection of rock and mineral specimen for its museum.

He was an endearing personality with sober temperament and quiet disposition. He was receptive to new ideas and suggestions and maintained a positive attitude above the mundane bureaucratic paraphernalia.

After obtaining M.Sc. degree in Geology from Utkal University (1975), he joined the Directorate of Geology & Mines, Govt. of Odisha in the year 1975. Carried out mineral investigation – bauxite, iron ore, gemstone and tin ores to mention a few. He underwent a training for gem testing technology in Australia under UNDP (1991). Superannuated as Director of Geology and Mines, Govt. of Odisha in 2010.

Binod Chandra Patnaik is survived by his wife, son and daughter, and a large circle of friends and well wishers.

Members of SGAT deeply mourn his death and convey their condolence to his bereaved family. SGAT pray before Almighty God to keep his soul in peace.

Dr. S.K. Sarangi
President, SGAT
Immediate Past President, MEAI

- **SUBMISSION OF PAPERS FOR SGAT BULLETIN**
(Instruction to Authors)

Research papers, review articles, short communications, announcements and letters to editors are invited on topics like geosciences, mineral exploration, mining, materials science, metallurgy, mineral industry and trade, mineral economics, environment, education, research and development, legislation and infrastructure related to mining, mineral policy and mineral development planning.

Submission of manuscript implies that the same is original, unpublished and is not being considered for publication elsewhere. Two copies, complete in all respect (with copies of figures and tables) are required to be submitted. Originals of figures and tables should be enclosed separately. Each manuscript must accompany by a computer diskette (floppy) containing the electronic version of the text. Electronic files of figures, if available, should be submitted in a separate diskette. In each case, the details of software and type of equipment used should be clearly indicated. The copies of manuscripts, strictly in accordance with the instructions to authors given below may be sent to the editor of the bulletin.

Journal Format: A-4 size

Language: English

Manuscripts: Manuscripts should be typed in double spacing with wide margins in one side of A-4 size paper either by electronic typewriter or computer (size 12 point Times New Roman font). The title page should include the title of the paper, name(s) of author(s) and affiliation(s). The title should be as brief as possible. An informative abstract of not more than 500 words to be included in the beginning. Not more than 5 key words are to be listed at the end of the abstract. Text of research papers and review articles should not exceed 4000 words. The short communication is for quick publication and should not exceed 1200 words.

Headings: Different headings should be in the following format.

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- (c) Abstract: Left aligned, bold
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Illustrations: All illustrations should be numbered consecutively and referred to in the text. They should conform to A-4 size and carry short captions. Lettering inside figure should be large enough to be accommodated up to 50% reduction. One set of hard copy of all figures (either tracing in ink or laser prints) should be provided in a separate envelope marked "Original Figures". Photographs should be of good quality with excellent contrast, printed on glossy paper. Colour photos are acceptable, provided the author(s) bear the cost of reproduction. Figure captions should be provided on separate sheet.

Tables: Each table must be provided with a brief caption and must be numbered in the order in which they appear in the text. Table should be organised within A-4 size and should be neatly typeset for direct reproduction. Tables will not be typeset by the printer, so their clarity and appearance in print should be taken into account while the author(s) prepare(s) them. Use of 10 points Time New Roman/Arial Font for table is recommended.

References :

- (a) References in the text should be with the name of the author(s) followed by the year of publication in parenthesis, i.e. Patnaik (1996); Patnaik & Mishra (2002); Nayak et al. (2001)
- (b) Reference list at the end of the manuscript should be in alphabetical order, in the following format: Sehgal, R.K. and Nanda, A.C.(2002) Palioenvironment and palioecology of the lower and middle Siwalik sub-groups of a part of North-western Himalayas. *Jr. Geol. Soc. Ind*, vol. 59, pp. 517-529
- (c) Articles from the books should follow the format given below: Windley, B.F. and Razakamanana, T. (1996) The Madagascar – India connection in a Gondwana framework. In: Santosh, M. and Yoshida, M. Eds.) *The Archaean and Proterozoic terrains of South India within East Gondwana. Gond. Res. Group Mem. No.3, Field Sci. Publ., OSAKA*, pp. 25-37

- (d) Books should be referred to as: Sengupta, S.M. (1994) *Introduction to sedimentology*. Oxford and IBH Publ. Co. Pvt. Ltd., New Delhi, 314 pp.

Submission of manuscript

Manuscripts strictly confirming to the above format should be mailed directly to Editor in his mailing address available in the bulletin. Manuscripts not confirming to the format of the journal will be returned.

All the manuscripts confirming to the standard format of the bulletin will be reviewed by specialist referees before publication.

Page proofs: One set of page proofs will be sent to the corresponding author, to be checked for typesetting only. No major changes are allowed at the proof stage. Proof should be returned within three days.

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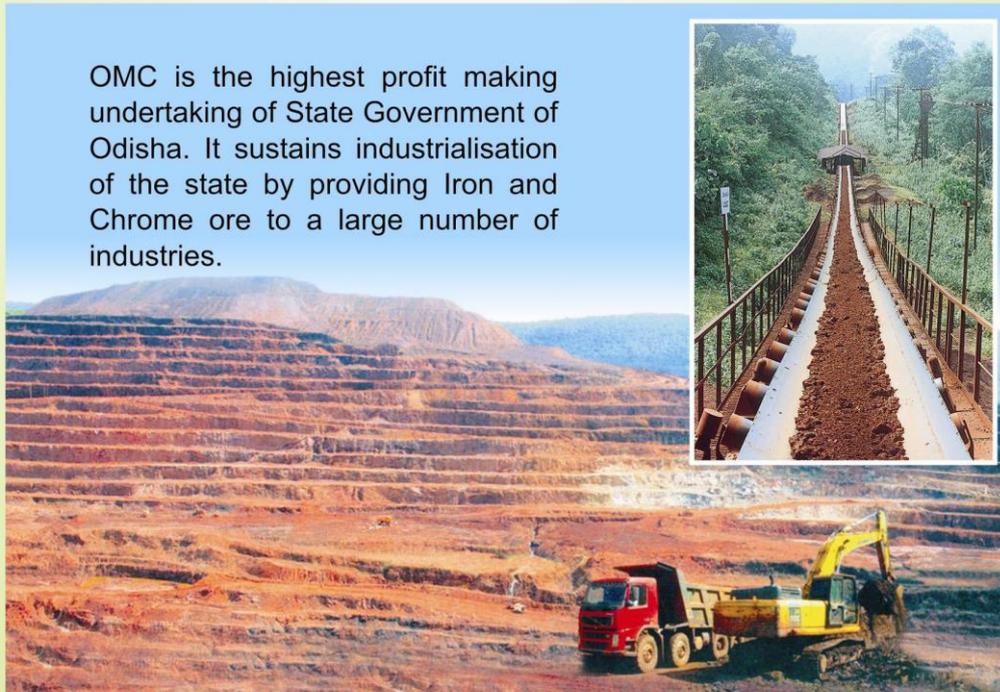
Printed and published at Bhubaneswar by Dr. S.K. Sarangi, President on behalf of the Society of Geoscientists and Allied Technologists (SGAT), Plot No. ND/12 (Part), IRC Village, Nayapalli, Bhubaneswar – 751015, Odisha, Printed at Reproprint (P) Ltd., N-5/49, IRC Village, Bhubaneswar – 751 015.

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- Rs.1.25 crores for development of Shree Jagannath Temple, Puri.
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- Rs.56 lakhs for development of Saraswati Sishu Mandir, Barsuan, Sundargard.
- Rs.40 lakhs per annum for 5 years to All Odisha Chess Association for Development of Chess in Odisha.
- Rs.32 lakhs for Renovation of Sanskruti Bhawan, Jajpur.
- Rs.22 lakhs for Renovation of Existing Stadium at Keonjhar.
- Rs.15 lakhs for organizing Adivasi Mela-2013 at Adivasi Exhibition Ground, Bhubaneswar.
- Rs.15.00 lakhs for Installation of a life size Statue of Utkal Gourav Madhusudan Das in the premises of Special Circuit House, Puri.
- Rs.14 lakhs to St. John Ambulance, Odisha State Center, Bhubaneswar for purchasing Ambulance.
- Rs.6 lakhs to women & child dev. dept. for Observation of International Day for Disabled.
- Rs.5 lakhs per annum to Swaviman for five years to conduct Anjali International Children's Festival.
- Rs.5 lakhs for participation of Odisha in 11th Pravasi Bharatiya Divas at Kochi, Kerala.
- Rs. 12 crores for development of Suakati-Dubuna Road in Keonjhar.
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