

PLANT DESIGN FOR THE PRODUCTION OF LITHIUM HYDROXIDE FROM  
LITHIUM CONTAINING ROCKS IN GHANA

A Plant Design Project Report

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## **ABSTRACT**

The ongoing drive and demand for green-energy revolution has reflected an attendant increase in the demand for strategic and critical metals and elements such as lithium, tantalum, cobalt, and rare earth elements. Specifically, the projected demands for portable electric devices, plug-in hybrid electric vehicles (PHEVs), hybrid electric vehicles (HEVs) and electric vehicles (EVs) have highlighted the probable increasing demand for lithium. Literature has shown that the extraction of lithium from various resources depends on the type of resource, general mineral processing methods, and extractive metallurgical processes. To characterize the characteristics of ores, identify important prospects and determine possible challenges related to the extraction of lithium, process mineralogy is essential.

This report aims to design a plant with an economically friendly process that can be used in the extraction of lithium hydroxide from lithium ore (spodumene), which is a very significant substance needed to produce modern day equipment and technological devices. Due to the high demand for lithium on the market, the plant is targeted to produce about 2 tonnes of lithium hydroxide an hour to contribute in bridging the demand and supply gap on the market. In summary, the insights gained from this analysis will provide motivation to investigate alternative approaches for the sustainable extraction of lithium from spodumene to produce lithium hydroxide.

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## **CHAPTER ONE**

### **1.0 INTRODUCTION**

#### **1.1 BACKGROUND STUDY**

In 1800, A Brazilian naturalist called Jose Bonifacio de Andrada Silva confirmed the discovery of spodumene, a lithium mineral, in Uto Sodermanland, Sweden (Azevedo et al., 2022). The pyroxene mineral spodumene is almost exclusively present in granite pegmatites. Lithium was found in 1817 by J. August Arfvedson at the Stockholm laboratory of J. J. Berzelius in the rock-like mineral petalite. At the Atacama Salt Flat, lithium is extracted from salar brine, making Chile the world's largest producer of lithium (Salakjani et al 2020). One of the rare metals with a wide range of uses is lithium, whose demand is predicted to rise in tandem with the growing use of electrical and electronic equipment as well as hybrid electric cars. While several well-established technologies are in use to extract lithium from its primary resources, such as minerals and brine, in the desired form, the selective extraction of lithium from seawater and bitterns requires more intensive tapping (Borden & Harris, n.d.). In 1821, two British scientists named Humphry Davy and William Thomas Brande made the purest form of lithium discovery Even with their achievement, Davy and Brande managed to extract a comparatively small amount of pure lithium. Only in 1855 did Robert Bunsen and Augustus Matthiessen come up with a method that allowed for the practical production of enough pure lithium to be used for property research (L. I. Barbosa et al., 2014).

#### **1.2 SIGNIFICANCE OF STUDY**

The energy industry is in the process of moving toward a greater use of renewable energy technologies. Metals such as aluminum, chromium, cobalt, copper, lithium, manganese,

nickel, rare earth elements (REEs), silicon, tin, titanium, tungsten, and zinc are essential to the green energy transition (Kashif Nazir, 2022). This shift does, however, take place in the context of three factors. That is a geographic concentration of known mineral reserves and downstream capabilities, a demand that greatly outpaces supply and a strong push to address energy and environmental problems.

The demand for lithium, a crucial component of lithium-ion batteries, which are utilized in energy storage systems is predicted to rise sharply. Many African countries most notably Zimbabwe, Namibia, Ghana, the Democratic Republic of Congo, and Mali have lithium resources and the potential for lithium mines (Leo-Russell, 2023). According to Atlantic lithium limited, the Ewoyaa Lithium Project being developed is expected to be the first lithium-producing mine in Ghana. The project is focused on exploiting the Ewoyaa, Abonko, and Kaampakrom lithium spodumene pegmatite deposits in western Ghana. The lithium generated can be used for several purposes which includes the production of lithium secondary batteries, ceramic electronic materials, refrigerant adsorbents, medicines using lithium isotopes and many others (Tran & Luong, 2015).

### **1.3 OBJECTIVES**

#### **1.3.1 Main objective**

To design a plant for the production of lithium hydroxide from lithium containing ore (spodumene).

#### **1.3.2 Specific Objectives**

- To determine the best method for extracting lithium from the ore.
- To develop a process flow diagram that shows the various stages involved in lithium extraction.
- To design systems and equipments that will be needed to extract lithium

- To understand and mitigate any operational and safety risks related to lithium manufacturing.
- To select an ideal location for the production plant.
- To determinate the optimal conditions to prepare high purity lithium carbonate from spodumene concentrate
- To determinate the optimal conditions to prepare high purity lithium carbonate from spodumene concentrate.
- To develop and make feasible cost analysis for the construction and operation of the plant.
- To select relevant process control and instrumentation equipment for the smooth operation of the processing plant

## CHAPTER 2

### 2.0 LITERATURE REVIEW

#### 2.1 LITHIUM

Having an atomic number three and the symbol Li, lithium is the lightest metal in the periodic table. It's a silver-white, flexible metal that falls under the alkali metal category. It is the least dense solid element when operating under normal circumstances (Salakjani et al., 2021). Lithium metal is extremely reactive and combustible, just like any other alkali metal. Because of its high reactivity and light weight, it is ideal for use in batteries and other sectors. Lithium is only found in compounds in nature; it is never found freely. Lithium can be recovered from brines, clays, and sea water in addition to being found in pegmatitic minerals, especially spodumene. With an atomic weight of 6.939 and an atomic radius of 1.33 Å, it is the lightest alkaline metal. It can float even at the point of reaction because of its density, which is around half that of water (Kudryavtsev, 2016).

#### Properties of lithium

Table 1: Properties of Lithium

Electronic Configuration	[He] 2s <sup>1</sup>
Block, Period and Group in periodic table	s-block, Group-1, Period-2
Atomic Number	3
Atomic Weight	6.941 g.mol <sup>-1</sup>
State at 20 °C	Solid
Melting Point	180.50°C, 356.90°F, 453.65 K
Boiling Point	1342°C, 2448°F, 1615 K
Density	0.534 g/cm <sup>3</sup>

Atomic radius	152 pm
Vander waal Radius	0.145 nm
Electronegativity (Pauling Scale)	0.98
Standard potential	– 3.02 V
Crystal Structure	body-centered cubic
Electron affinity (kJ mol <sup>-1</sup> )	59.633
First Ionization energies (kJ mol <sup>-1</sup> )	520.222

Source; (Gordon et al., n.d.)

## 2.2 DISCOVERY OF LITHIUM IN GHANA

Ghana is expected to become the first lithium producer in West Africa and has been identified as a potential key hub for lithium production in Africa. Iron Ridge Resources (Australia) and Piedmont Lithium agreed into a conditional binding agreement in 2021, based on sources in the mainstream media to fund and speed the Ewoyaa Lithium Project in the central region to full scale production (Abaka-Wood et al., 2022a). The estimated lithium content of the Ewoyaa Lithium Project is 14.5 Mt at a grade of 1.31% in the indicated and inferred categories, including 4.5 Mt at 1.39% in the indicated category.

A total of 25.6 million tonnes are known to be in reserve at the discovered site. The project's main goal is to extract lithium spodumene pegmatite from the Ewoyaa, Abonko, and Kaampakrom deposits in western Ghana. The mine is anticipated to produce its first lithium concentrate in the second quarter of 2025 (Gordon et al., n.d.). The Ewoyaa lithium project is situated about 100 kilometers to the south-west of Accra, the capital of Ghana. Two adjacent licences, Mankessim (RL 3/55) and Mankessim South (PL3/109), are included in the project area. As of June 2023, an estimate of 25.6 million tonnes (mt) of probable ore grading 1.22% lithium oxide (LiO) was contained in the Ewoyaa lithium project. Because the ore is shallow, the lithium project will employ traditional open-pit

mining techniques such as drill, blast, load, and haul operations due to the shallow nature of the ore body(Marcinov et al., 2023). Eight major deposits, which are spaced four kilometers apart, will be mined: Ewoyaa, Okwesi, Anokyи, Grasscutter, Abonko, Kaampakrom, Sill, and Bypass.

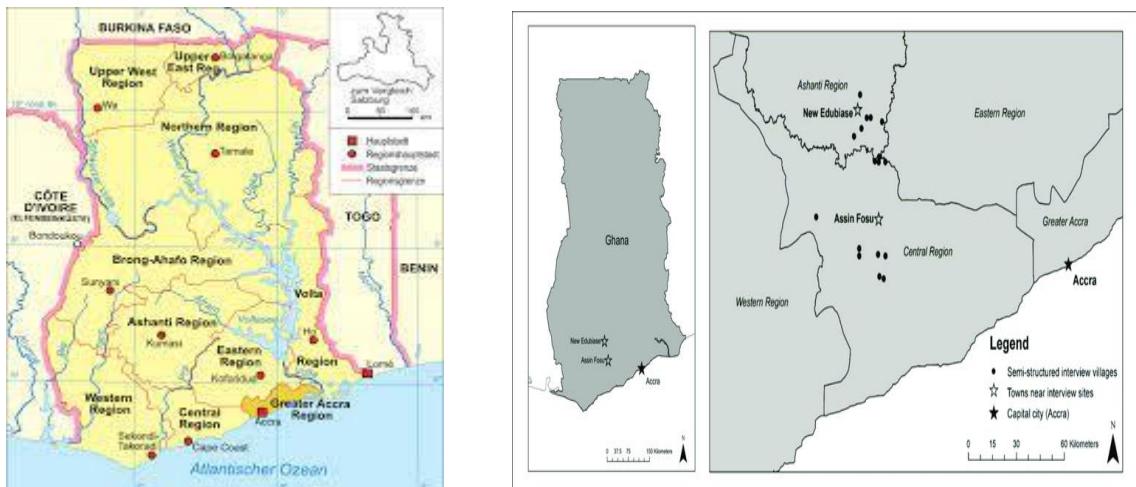


Figure 1: Geographical study of the central region of Ghana

### 2.3 LITHIUM INDUSTRY IN AFRICA

Zimbabwe has the largest percentage of lithium reserves in Africa, accounting for 1% of the world's total lithium resources. In Africa, 28% of the active lithium projects are located in Zimbabwe. Only eight nations on the continent had 18 working mines in 2020: Zimbabwe, the Democratic Republic of the Congo (DRC), Mali, Namibia, Ghana, Tanzania, Madagascar, and Mozambique (Abaka-Wood et al., 2022a).

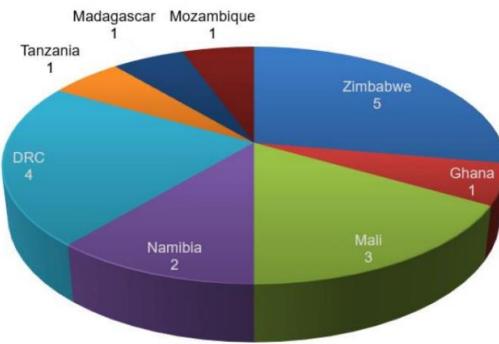


Figure 2: The figure shows the distribution of lithium projects in Africa.

## 2.4 SOURCES OF LITHIUM

Lithium can be obtained from both primary and secondary sources. The resources that are covered include ores, minerals, clays and brines or seawater and bitterns, and lithium-ion batteries for the hydrometallurgical recovery of lithium (Meshram et al., 2014). Over the years, some countries have played critical roles in the production of lithium. Countries with the major source of lithium are Australia, Argentina and Chile. The Earth's continental crust contains 20 parts per million lithium on average. Compared to some of the more well-known metals, such as tin and silver, it is more abundant. Most rocks contain trace amounts of lithium, which is often used to replace magnesium in common rockforming processes (Yelatontsev and Mukhachev, 2021).

Lithium reserves estimate vary from 4 million tons to 30 million tons. In 2011, a detailed study examining data from 103 deposits containing lithium estimated lithium reserves in 39 million tons. Such differences in reserves are due to the availability of information and the assumptions for quantifying the feasibility of recovering lithium (Riyo et al., 2022). The economic feasibility depends on the size of the deposit, the content of lithium, the content of other elements (such calcium and magnesium, which might interfere during extraction and processing), and the processes used to remove the lithium-bearing material and extract lithium from it.

Table 2: Worldwide Sources of Lithium and its Composition

Ore Deposit Type	Percentage
1) Closed-basin brines	58%
2) Pegmatites and related granites	26%
3) Lithium-enriched clays	7%
4) Oilfield brines	3%
5) Geothermal brines	3%
6) Lithium-enriched zeolites	3%

Source: (Hao et al., 2017)

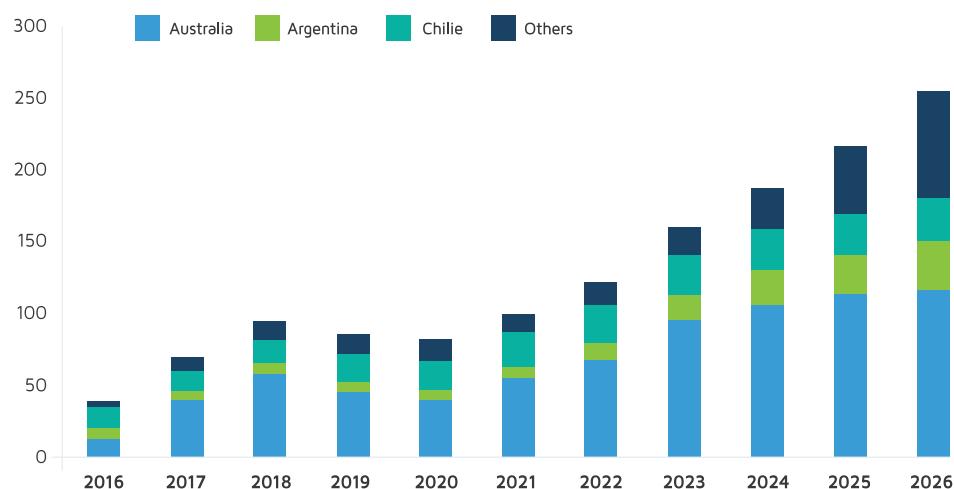


Figure 3: Major producing lithium countries over the years.

#### 2.4.1 Primary sources

Lithium can be recovered by acid, alkaline, and chlorination methods from primary resources like ores and minerals (spodumene, petalite, and lepidolite) or by adsorption, precipitation, and ion exchange procedures from brines. Issues pertaining to the utilization of alternative resources, like seawater and bitterns, are emphasized in details in this report.(Gao et al., 2023)

Table 3: Various lithium containing minerals and lithium containing composition

Lithium Source	Li Formula	Li %
1) Spodumene	$\text{LiAlSi}_2\text{O}_6$	3.73
2) Lepidolite	$\text{K}(\text{Li, Al})_3(\text{Si, Al})_4\text{O}_{10}(\text{F, OH})_2$	3.58
3) Petalite	$\text{LiAlSi}_4\text{O}_{10}$	2.09

Source:(Gao et al., 2023)

#### 2.4.2 Minerals

Lithium is present in trace amounts in granite and other igneous rocks. Lithium is mostly found in pegmatites, spodumene, and petalite, three common rocks and minerals. Other minerals include eucryptite, lepidolite, amblygonite, and zinnwaldite (Ferrell, 1985). Zinnwaldite is an impure form of lepidolite with significant MnO (3.2%) and FeO (up to 11.5% Fe as FeO) values. Pegmatites contain recoverable amounts of lithium, tin, tantalum, niobium, beryllium, and other elements.(Abdullah, n.d.)

#### 2.4.3 Extraction of lithium carbonate from Spodumene

##### Spodumene Ore

Spodumene is the most abundant mineral containing lithium, it is of fundamental commercial importance. With a theoretical lithium content of 3.73%, it is a stable lithium mineral known as aluminum silicate,(Ebensperger et al., 2005). Because of its high lithium concentration, widespread deposits, and financial viability for processing, it is presently being investigated and processed on an industrial basis. Roughly 90% of the production of lithium carbonate equivalent worldwide is accounted for by spodumene. Spodumene ore is part of the pyroxene group and naturally occur with other major gangue minerals such as quartz, alumina, annite and phlogopite (Borden, 2023). There are three polymorphs of spodumene. The form occurring in nature is monoclinic low temperature called  $\alpha$ -spodumene. The tetragonal high-temperature form is called  $\beta$ -spodumene which

is a stuffed derivative of the keatite structure (Borden, 2023). The third hexagonal high-temperature form which is a stuffed derivative of the high-quartz is known as  $\gamma$ -spodumene.



Figure 4: Lithium ore from spodumene

#### **2.4.3.1 Lithium Extraction from Spodumene using the Acidic or Alkaline Method**

##### **Spodumene Beneficiation**

Including many other minerals, spodumene is processed by first crushing the ore and then upgrading it using physical separation techniques like flotation, magnetic separation, dense media separation (DMS), and ore sorting. Prior to further extraction procedures, these are used to remove related gangue minerals such quartz, micas, and feldspar.(Chen et al., 2011)

As a result, spodumene in the  $\alpha$ -phase is difficult to treat; therefore, it must be roasted at between 1040 and 1100 degrees Celsius to convert it to  $\beta$ -spodumene and improve leaching under mild chemical extraction conditions. Chemicals that are acidic, alkaline, or chlorinated are used to roast the calcined  $\beta$ -spodumene concentration further.(Nicolaci et al., n.d.)

## **Acidic or Alkaline Roasting of Spodumene**

$\beta$ -spodumene is crushed, mixed with concentrated sulfuric acid, and then roasted at 250°C in the acid process. This procedure yields soluble lithium sulfate and an insoluble mineral residue.(Chen et al., 2011).

On the other hand, the  $\beta$ -spodumene concentrate is crushed and heated to 825–1050°C using limestone or soda ash to calcine the lithium silicates into a soluble state.

After that, the soluble lithium is leached in water and reacts with carbon dioxide to form aqueous lithium bicarbonate. When using soda ash for neutralization, the pregnant leach solution is obtained and then evaporated at around 90 degrees Celsius, causing the lithium to crystallize as lithium hydroxide monohydrate or as  $(Li_2CO_3)$  (Mamudu et al., n.d.-a)

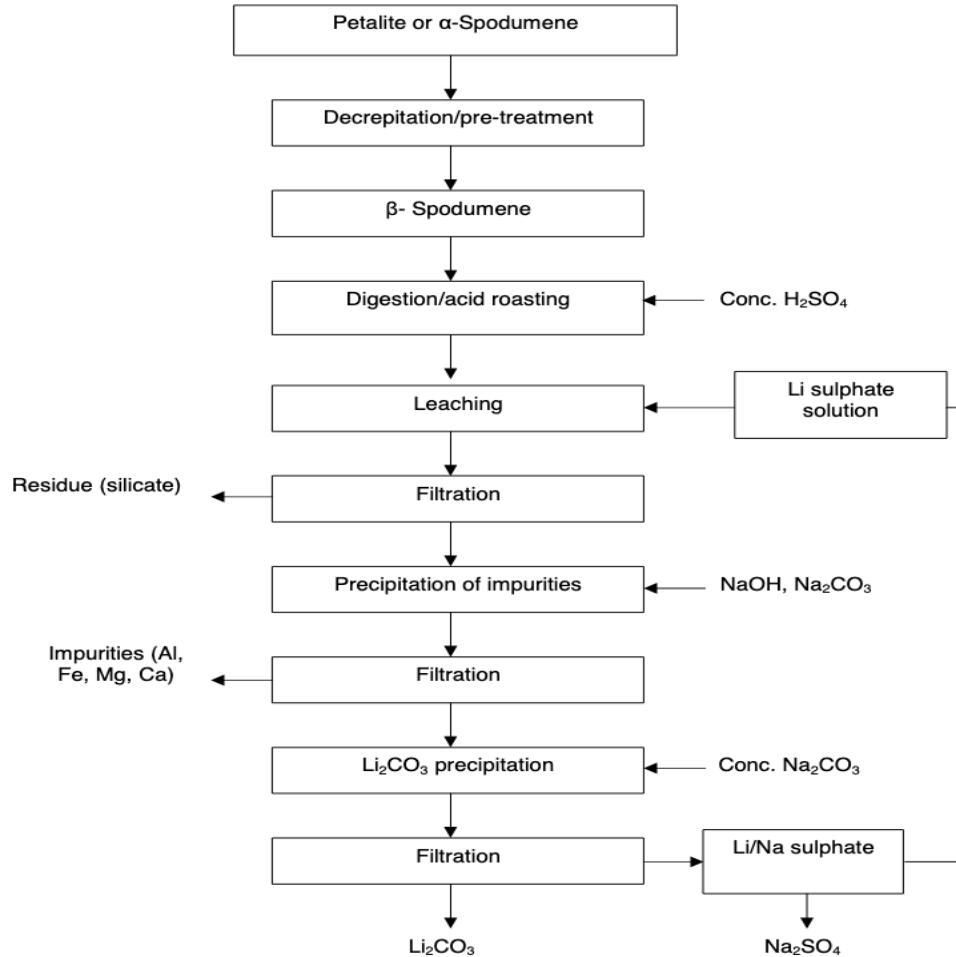


Figure 5: Flow diagram for the production of lithium carbonate using sulfuric acid

#### 2.4.3.2 Lithium Extraction Using the Chlorination Method

Chlorination roasting of spodumene takes place at temperatures above 1000°C in the presence of chlorine gas. This process produced a soluble lithium chloride (LiCl) which can be water leached and subjected to evaporation or other purification processes. Roasting β-spodumene with Chlorine gas at 1100°C for 2.5 hours could results in almost complete extraction of lithium as Lithium chloride (Miatto et al., 2020)

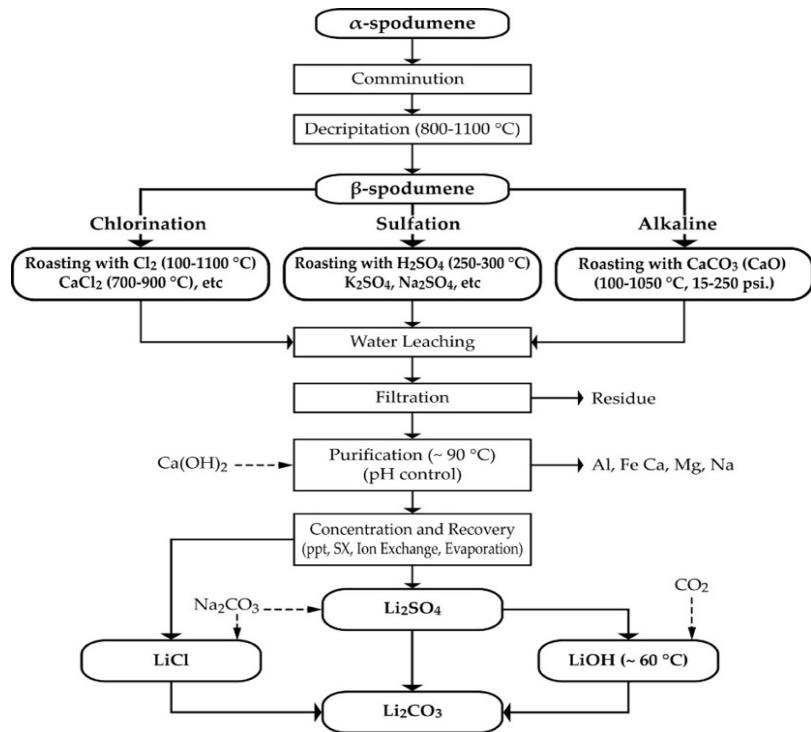


Figure 6: The flow diagram of the treatment processes of spodumene

#### 2.4.3.3 Lithium Carbonate Extraction Using the Autoclave Method

Spodumene is the most attractive mineral among these economically viable sources of lithium. Numerous methods for extracting the lithium values from spodumene ore have been suggested. Sintering and the sulfuric acid process are the only two methods that have been used in a number of these processes (Mamudu et al., n.d.-a). Because of its high efficiency, the sulfuric acid process has taken over as the primary method for producing lithium carbonate from spodumene. Nevertheless, there are inherent disadvantages to this process, including the product's high sulfate and heavy metal ion levels and its complex sodium sulfate recovery procedure. To get over these problems, an autoclave process was developed to produce lithium carbonate from spodumene has been suggested as an alternative.(Braga et al., n.d.)

## Process Description

Because of its low reactivity, the  $\alpha$ -spodumene was first roasted and then heated to a high temperature to change it into  $\beta$ -spodumene throughout this process. Subsequently, in an autoclave, the  $\beta$ -spodumene reacted with the sodium carbonate solution to generate a slurry of lithium carbonate and analcime, as follows:



The slurry was then leached with carbon dioxide and the resulting bicarbonate solution was heated to derive lithium carbonate.

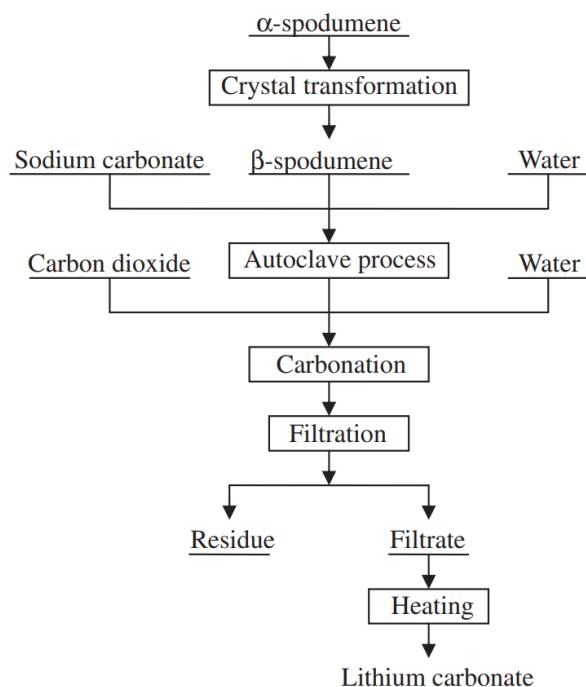


Figure 7: Process flow diagram for the production of lithium carbonate using the autoclave method

#### **2.4.4 Clay**

Clay minerals contain minor levels of lithium. It can exist in clays as inclusions, contaminants, adsorbed on the surface, lattice cavities, or by isomorphous substitution (Braga et al., n.d.). The most frequent type of substitution is isomorphous substitution. Among the clay minerals, hectorite, a type of smectite is rich in lithium and magnesium, and generally contains 0.3 to 0.6% Li. The best known hectorite deposit with 0.7% Li is in Hector, California.(Meshram et al., 2014b)

#### **2.4.5 Brine**

Seawater contains about 0.1–0.2 mg/L Li (Hamzaoui et al., 2007). Total amount of metallic lithium in seawater (globally) is estimated to be ~230 Gt. Brine sources include lithium found in salt water deposits lakes, salars, oilfield brines, and geothermal brines. Oilfield brines are underground brine reservoirs that are located with oil.(Pickles & Marzoughi, 2022) Geothermal brines are underground brines naturally heated. As of 2012, a greater percentage of lithium (83%) is obtained from brine lakes and salt pans of limited size. In those facilities, lithium is concentrated using solar energy, a very time-consuming process. Seawater contains a vast amount of lithium; however, it is mostly dilute and so its recovery is difficult and expensive. The most significant source for the commercial manufacture of lithium products is natural brines with high lithium chloride contents which are the end product of natural leaching evaporation processes.(Abaka-Wood et al., 2022b)

## Lithium Extraction from Brine

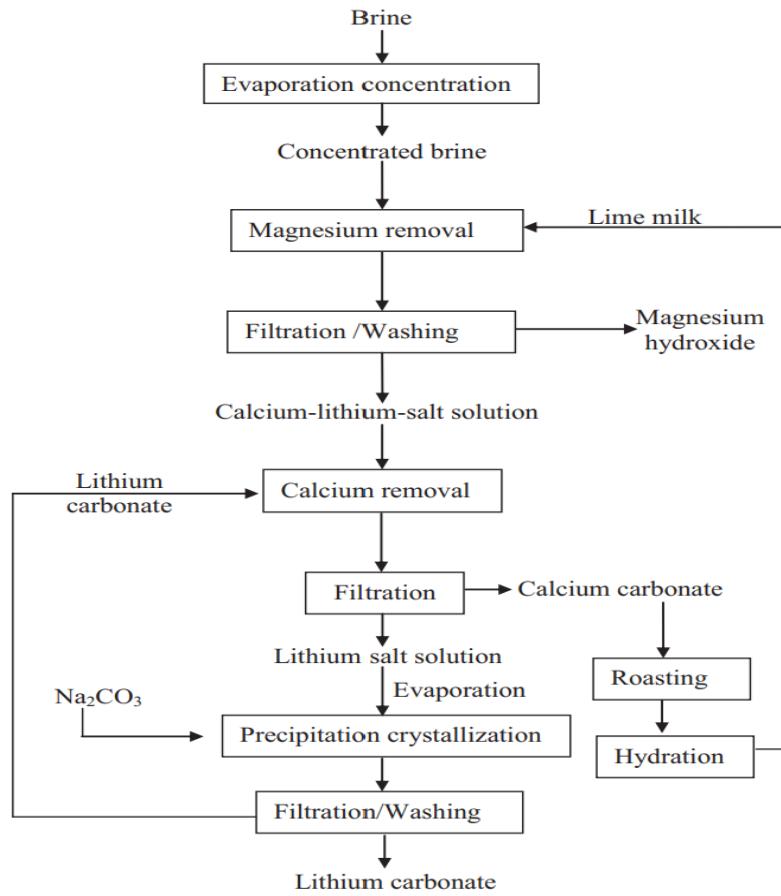


Figure 8: Process flow diagram for the production of lithium carbonate from brine

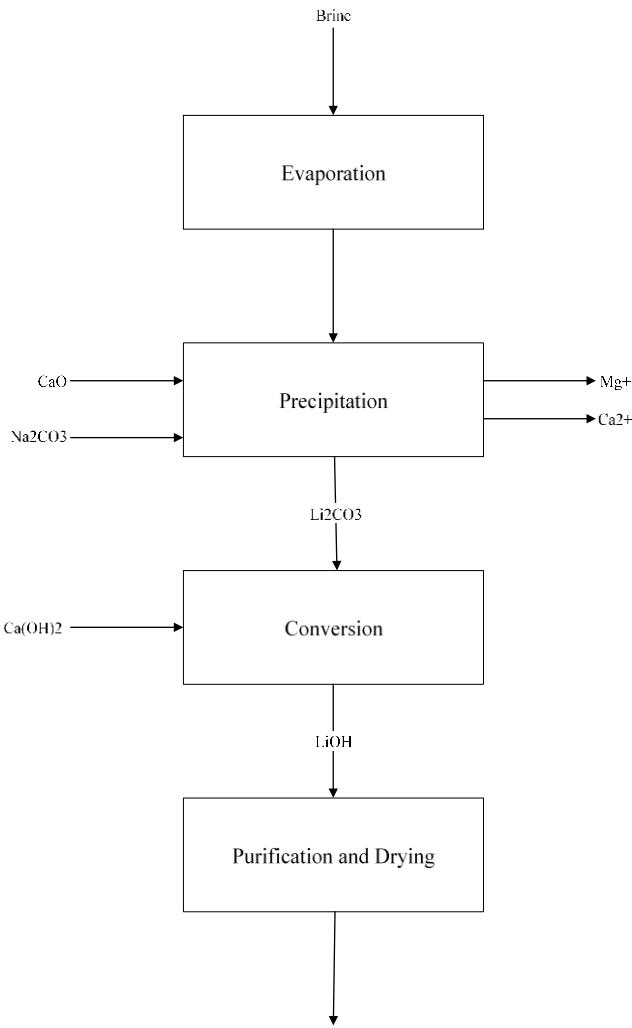


Figure 9: Production of lithium hydroxide from brine

#### 2.4.6 Secondary sources.

Spent LIBs are the most well-known secondary source of lithium and other metals among the different types of secondary resources. It is necessary to understand the general structure and makeup of the cells as well as how they change when in use in order to recycle these resources. In terms of secondary resources, lithium is recovered from lithium-ion batteries (LIBs) through industrial procedures and more recent advancements.(Tran & Luong, 2015)

## **2.4.7 Lithium-Ion Batteries.**

A battery with lithium metal, alloy, or substance adsorbing lithium ions as its negative active material is commonly referred to as a lithium-ion battery. Lithium ions are present in carbon, which is used as the anode in lithium-ion batteries. Batteries are classified as main or secondary based on their attributes and technological design.



Figure 10: Spent lithium batteries

### **2.4.7.1 Primary Batteries**

Metallic lithium is used in the construction of primary cells. In a non-rechargeable primary lithium battery, the metallic lithium is a combustible alkali metal that self-ignites at 178°C. It interacts exothermically with water or seawater to liberate hydrogen. Primary batteries are single-use because they are disposed of after usage due to irreversible discharge processes that occur in the cells.(Liu et al., 2022)

### **2.4.7.2 Secondary Batteries**

Secondary battery cells have a chemistry that allows reversing the discharge reaction and are rechargeable. LIBs are of the re-chargeable secondary type. The functional parts of LIBs are the cathode, anode, electrolyte and separator, which are housed in a protective metal casing.(Li et al., 2014)

#### 2.4.7.3 Composition of Lithium-ion Batteries

LIBs are composed of cathode material, anode material, a diaphragm, an electrolyte, and a binder, with different components bonded together. Some materials are flammable and toxic, meaning they cannot be handled together. Mechanical crushing can destroy the metal shell. (Xu et al., 2008)

Afterward, the organic membrane, spent anode, spent cathode, metal shell, and some electrode material particles can be released from each other, and they can then be separated according to physical differences (Zhou et al., 2020)

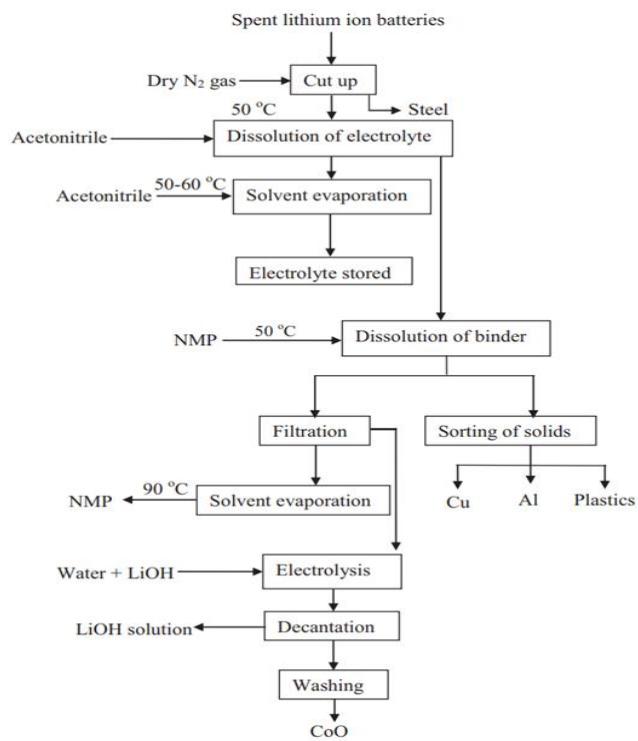


Figure 11: Production of lithium hydroxide from spent batteries

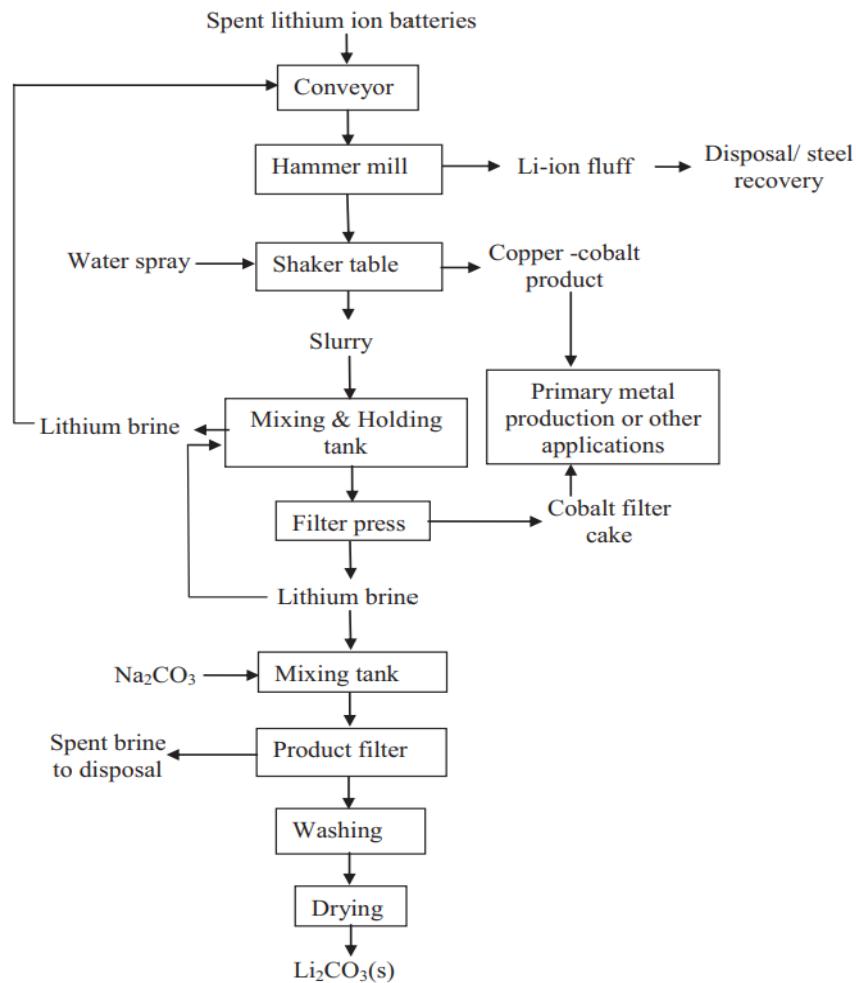


Figure 12: Production of lithium carbonate from spent lithium batteries

## 2.5 USES OF LITHIUM

Lithium has several uses and play a vital role in the global economy. It can be used for several purposes in diverse sectors. Some various uses of lithium include;

1. It is used in lithium secondary batteries
2. Ceramic electronic materials
3. Refrigerant adsorbents
4. Medicines using lithium isotopes.

5. Due to power generation, the use of lithium secondary batteries has exploded which lithium is a core raw material, is also rapidly increasing in accordance with the rapidly increasing demand for lithium secondary batteries.(Li et al., 2014)
6. Since lithium is used to multiply tritium in fusion power generation, which is expected to be the next generation energy source, the demand for lithium is further increased.
7. Lithium is used to make electric vehicles and more durable and lightweight alloys when combined with other metals. These are used in many sectors including the transport industry (high-speed trains, aircrafts, etc.)
8. Lithium carbonate is mostly produced from both ores and brines and the production figures are often expressed as lithium carbonate equivalent (LCE). Other chemicals such as lithium chloride and hydroxide are also produced in varying amounts.
9. Lithium hydroxide is a component of lubricating greases and makes these become water and oxidation-resistant.

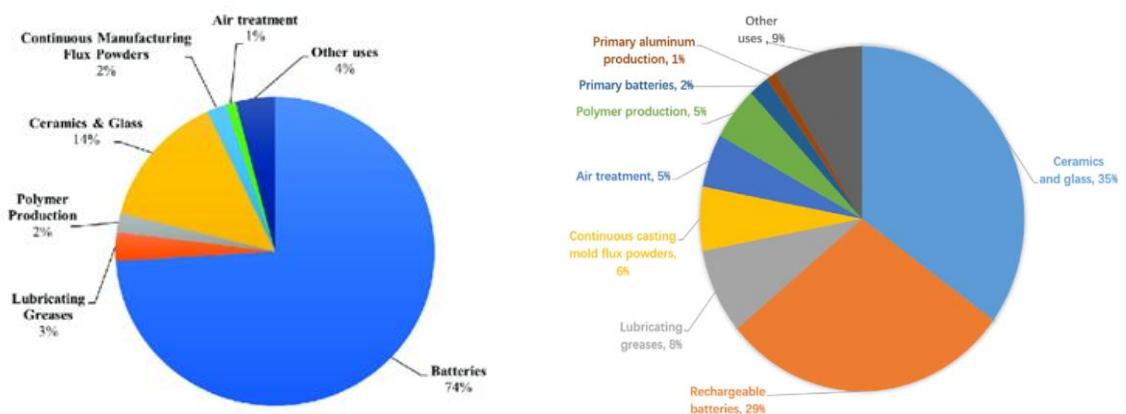


Figure 13: Pie chart showing market end uses of lithium as a percentage of global consumption in 2021 (U.S Geological Survey, 2022).

## **2.6 LITHIUM EXTRACTION METHODS**

Several ways or techniques are employed in the extraction of lithium from various sources. Extraction from primary sources includes;

### **2.6.1 From Ores (Minerals)**

Two methods are used to extract lithium from its minerals. The acid process and the alkaline process, while chlorination is also occasionally used.(Liu et al., 2022)

#### a. Using the Acid Approach

In order to process ores/concentrates, acid digestion with H<sub>2</sub>SO<sub>4</sub> may be followed for decomposition of the silicate structure at 250–400 °C which is suitable for the processing of lepidolite, amblygonite and zinnwaldite.(L. I. Barbosa et al., 2014)

#### b. Using the Alkaline or Gypsum Approach

Alkali digestion is suitable for the decomposition of spodumene and lepidolite largely by the treatment of potassium carbonate to produce lithium hydroxide. The most of the alkaline processes involve either the heating of lithium minerals with alkali salts or in more advanced hydrothermal processes by decomposition in solutions containing Na<sub>2</sub>CO<sub>3</sub>, NaOH, Na<sub>2</sub>SO<sub>4</sub> and/or other alkali salts at elevated temperature and pressure.(Tran & Luong, 2015)

#### c. Chlorination Process

A less common method is the chlorination process in which the ore is roasted in the temperature range 880–1100 °C in the presence of chlorine gas or HCl depending upon the type of mineral treated.(Mamudu et al., n.d.-b)

### **2.6.2 From Brine**

Lithium is removed from brines by ion exchange, adsorption, precipitation, and evaporation. Brine and bittern systems have drawn more attention as a means of supplying

lithium, which is in rising demand.(Vera et al., 2023) Lithium produced from brine-water requires a 30% to 50% lower production cost than lithium extracted from mines. As noted, an evaporative concentration and refining process is used to manufacture lithium carbonate from brines.(Stringfellow & Dobson, 2021)

a. Adsorption process

Different kinds of adsorbents have been employed to recover lithium from brines and seawater in a selective manner. Certain inorganic ion-exchangers, including manganese oxide of the spinel type, have exceptionally high selectivity for lithium from seawater when used in the adsorption process. These substances show strong adsorption abilities for Li<sup>+</sup> in alkaline media (seawater has a pH of -8), when alkali and alkaline earth ions are present. (L. I. Barbosa et al., 2014)

b. Precipitation Process

Aluminum salts perform the best among the different co-precipitating agents used to recover lithium from geothermal water. Lithium recovery requires a pH between 10 and 13, and NaAlO<sub>2</sub> appears to be a better option than AlCl<sub>3</sub>. A high-purity NaAlO<sub>2</sub> solution was used as a co-precipitating agent, and at pH 11.5, 98–99% Li recovery from Ca- and SiO<sub>2</sub>-free geothermal water was accomplished.(Chen et al., 2011)

c. Ion exchange/Solvent extraction process

For high magnesium and calcium containing bitterns or brines solvent extraction or ion exchange can be used. After selective stripping/elution lithium can be precipitated out. Li<sup>+</sup>/H<sup>+</sup> ion exchange, in which Li<sup>+</sup> ions in brines are adsorbed on ion-sieves to displace H<sup>+</sup> ions, is the basis for the mechanism by which lithium ion-sieve adsorbs Li<sup>+</sup> ions from brines.(Lee et al., 2022) The displaced H<sup>+</sup> ions are then released back into the brines. Brine becomes acidic if released H<sup>+</sup> ions build up in it, which prevents Li<sup>+</sup> ions from

adhering to ion-sieves. Therefore, while recovering lithium from brines using ion-sieves, it's crucial to control the pH level so that it's more than 7 (Yuzer et al., 2020).

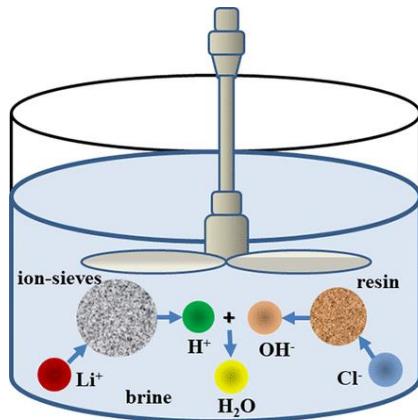


Figure 14: Alkaline resins enhancing  $\text{Li}^+/\text{H}^+$  ion exchange for lithium recovery from brines

## 2.7 LITHIUM BENEFICIATION

Any procedure that eliminates the gangue minerals from the ore to produce a higher-grade product (ore concentrate) and a waste stream (tailings) is referred to as beneficiation. Conventional beneficiation techniques such as flotation, gravity methods, flocculation-dispersion, electrostatic and magnetic separation, and use of chemical reagents are often employed to treat the finer iron ore deposits.(Tadesse et al., 2019) These techniques are energy-intensive, costly, and environmentally hazardous.

### 1. Dense Media Separation

Heavy media separation, also known as dense media separation (DMS), is a preconcentration method that can be used to extract lithium concentrations from high-grade ores. It is typically used to reject coarse gangue before grinding. In 2019. This technique makes use of the variations in specific gravity between the target mineral and the gangue minerals. Because DMS has a relatively high specific gravity of 3.1–3.2, it is frequently employed to separate lithium crystals from the primary gangue

silicates.(Abaka-Wood et al., 2022a). Given that DMS is most successful at relatively coarse size fraction, lithium minerals concentration by DMS is usually carried out on ore fraction of  $-9.5\text{ mm} + 850\text{ }\mu\text{m}$ .

## 2. Magnetic Separation

When a magnetic field is applied, the changes in the behavior of the mineral of interest's magnetic susceptibility in comparison to the gangue mineral(s) are used in magnetic separation.(Tadesse et al., 2019) This method of separation is usually used in conjunction with lithium values to exclude gangue minerals that contain iron. Because of its high iron content, zinnwaldite often has a very high magnetic susceptibility, making it suitable for magnetic beneficiation.

## 3. Froth Flotation

Using variations in the hydrophobicity of mineral particles, froth flotation uses them to separate. It is the most popular technique for processing minerals that contain lithium. The surface chemistry of the minerals, the type and concentration of the collector, the pulp pH, the pretreatment techniques, and the presence of slimes can all have an impact on the flotation recovery efficiency of lithium minerals. Comparing this process to other preconcentration techniques, however, yields "richer" lithium concentrate. (Tadesse et al., 2019).

## 4. Hydrometallurgical Processes

Direct leaching techniques are used to apply various extraction processes to lithium preconcentrates. For instance, it has been suggested that zinnwaldite and  $\beta$ -spodumene can be leached using hydrochloric acid (HCl). It was claimed that in order to obtain better recoveries with HCl, higher reagent concentrations are required, which makes the process excessively costly in terms of energy and chemical expenses. In other cases, lepidolite concentrate has also been directly

leached for 10 hours at 138°C using sulfuric acid (H<sub>2</sub>SO<sub>4</sub>). At 75°C, hydrofluoric acid produced 90% lithium recovery with good effectiveness.

## **2.8 THE ROLE OF LITHIUM IN ENERGY TRANSITION**

The term energy transition describes how the world's energy industry is moving away from fossil fuel-based energy production and consumption systems, such as those based on coal, oil, and natural gas, and towards renewable energy sources like solar, wind, and lithium-ion batteries.(Sadik-Zada et al., 2023)

Businesses are adjusting to the "energy transition" as more investors and companies seek greater clarity and confidence in accounting for long-term climate risks and opportunities. The transition to renewable energy from non-renewable energy sources like coal, natural gas, and oil is made possible by societal pressure for sustainability and technical breakthroughs (Heredia et al., 2020). Driven by long-term, structural adjustments to energy costs, supply, and demand, the energy transition also seeks to lower greenhouse gas emissions associated with energy consumption by implementing various decarbonization strategies.

Lithium has become a very critical element in transition. It is one of major elements that has had a very huge impact on energy transition due to its advantages it has over both non-renewable energy sources and other metal batteries. As the lightest metal and least dense solid element, lithium gained significance as an anode material for lithium batteries in the latter half of the 20th century. The element is an important part of high energy density rechargeable lithium-ion batteries due to its high electrochemical potential (Gielen & Lyons, 2022).

Handheld electronics, heavy-duty cordless power tools, laptop computers, and other gadgets already have a sizable market for lithium batteries. However, there may be a larger market for lithium if it is used in the batteries of hybrid and electric cars as well as

in the generation of alternative energy. Worries regarding the carbon dioxide emissions from internal combustion engines and hydrocarbon-based power plants, as well as the hydrocarbon shortage (which can result in high prices) in the upcoming years, and the United States' reliance on imported hydrocarbon fuels has increased curiosity about alternate energy sources. In the future, internal combustion engine vehicles are predicted to lose market share to electric vehicles (Graham et al., 2021).

### **2.8.1 EFFECT OF LITHIUM EXTRACTION ON CLIMATE CHANGE**

The lithium-ion battery has evolved over the past 10 years from being a technology that made mobile devices possible to one that is crucial to the global decarbonization and reduction of greenhouse emissions (GHG) (H. Barbosa et al., 2023). First and foremost, as a catalyst for the electrification of the fleets of modern vehicles, including automobiles, buses, lorries, and ferries, as well as for the development of novel, disruptive applications like autonomous robots, electric bikes, and scooters. When fossil fuel-based energy sources are replaced by cleaner but less predictable renewable energy sources like wind and solar, which are prerequisites for the reduction of greenhouse gas emissions, the lithium-ion battery has also quickly emerged as the technology of choice for various energy storage solutions (Sadik-Zada et al., 2023).

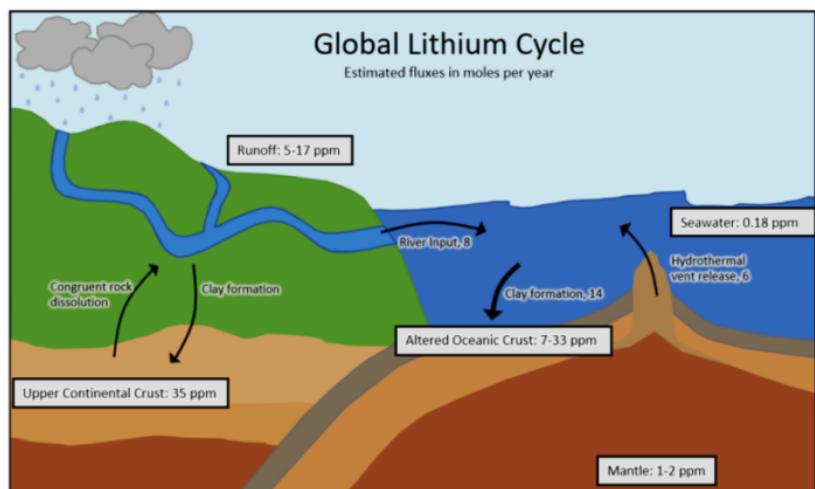


Figure 15: Global Lithium Cycle

Positive changes effects of lithium battery production have been stated above however there are drawbacks to this mining as well, such as chemical spills into rivers and groundwater and the devastation of man-made communities and natural ecosystems in the areas where the metal is mined. For example, brine mining is a common method of mining lithium that involves removing the metal from saltwater sources that are subterranean.(Heredia et al., 2020) Here are some instances of the problems associated with contaminating nearby water sources: (Salar de Uyuni and Salar de Atacama). In order to extract the lithium and other minerals, saltwater is pumped to the surface and allowed to evaporate. Even though it uses a lot of energy, this is still one of the most economical methods of mining lithium today. Regretfully, these hazardous metals have the potential to poison water supplies, endangering the biodiversity of animals as well as people. Moreover, even in minute amounts, some of the metals included in EV batteries have extremely harmful properties.(Sun et al., 2020)

## **2.9 ANALYSIS OF LITHIUM MARKET**

### **2.9.1 Lithium Demand Analysis**

As this decade began, the market for lithium chemicals could only be described as tiny compared to other metals such as copper or nickel (Calisaya et al, 2020). In 2012, the entire global market for lithium chemicals was less than U.S. \$1 billion in sales. The global lithium market size is valued at \$7.49 billion in 2022 and is expected to grow at a compound annual growth rate of 11% from 2023 to 2040. The cost of lithium is not significant in applications like rechargeable batteries, glass, grease, and medicine. Demand for lithium chemicals is relatively price inelastic. Investing in the lithium industry is a good opportunity for the next decade because of high demand and limited supply.(Maxwell, 2014)

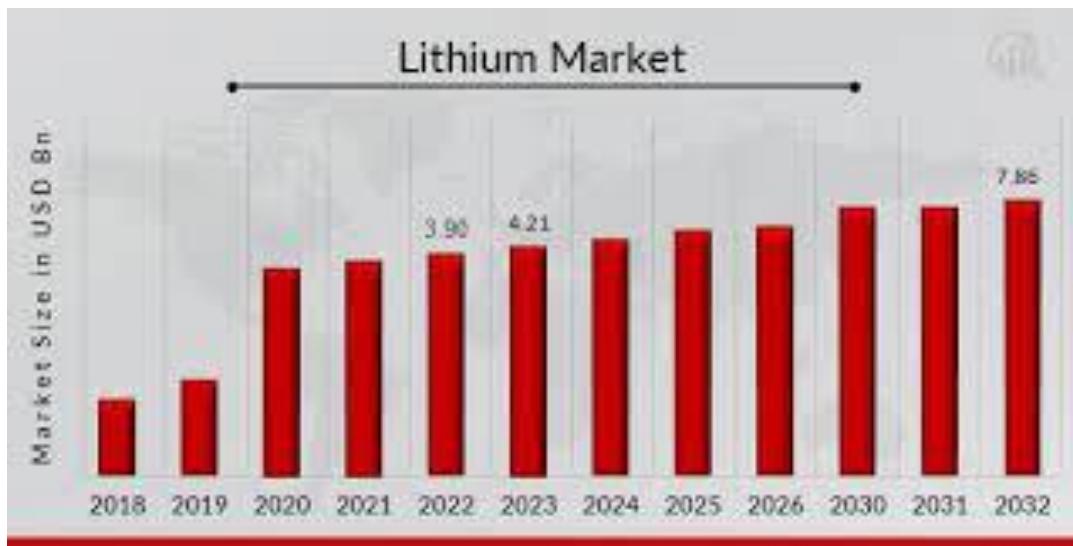


Figure 16: Graph projecting lithium analysis in various years.

In 2016, global demand for lithium chemicals was approximately 180,000 Metric tonnes (MT) of lithium carbonate equivalent (LCE). Several experts project that existing mineral deposits must produce twice as much lithium within the next 5-8 years. The transportation industry needs a lot more lithium production for full electrification.(Martin et al., 2017)

The IEA forecasts 2020 to 2040 total demand increases of lithium of 13 times to 42 times. 2021 IEA forecast growth in demand for selected minerals from clean energy technologies by scenario, 2040 relative to 2020 - Increases of Lithium 13x to 42x, Graphite 8x to 25x, Cobalt 6x to 21x, Nickel 7x to 19x, Manganese 3x to 8x, Rare Earths 3x to 7x, And Copper 2x to 3x.

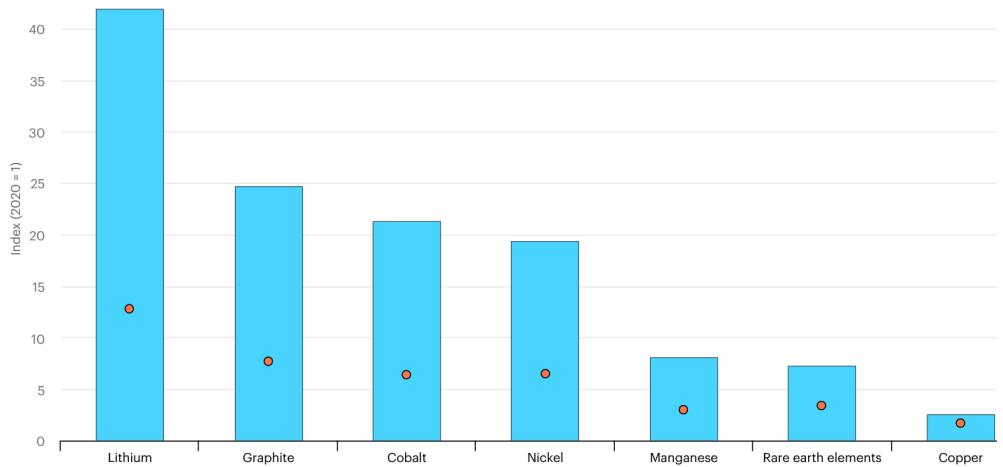


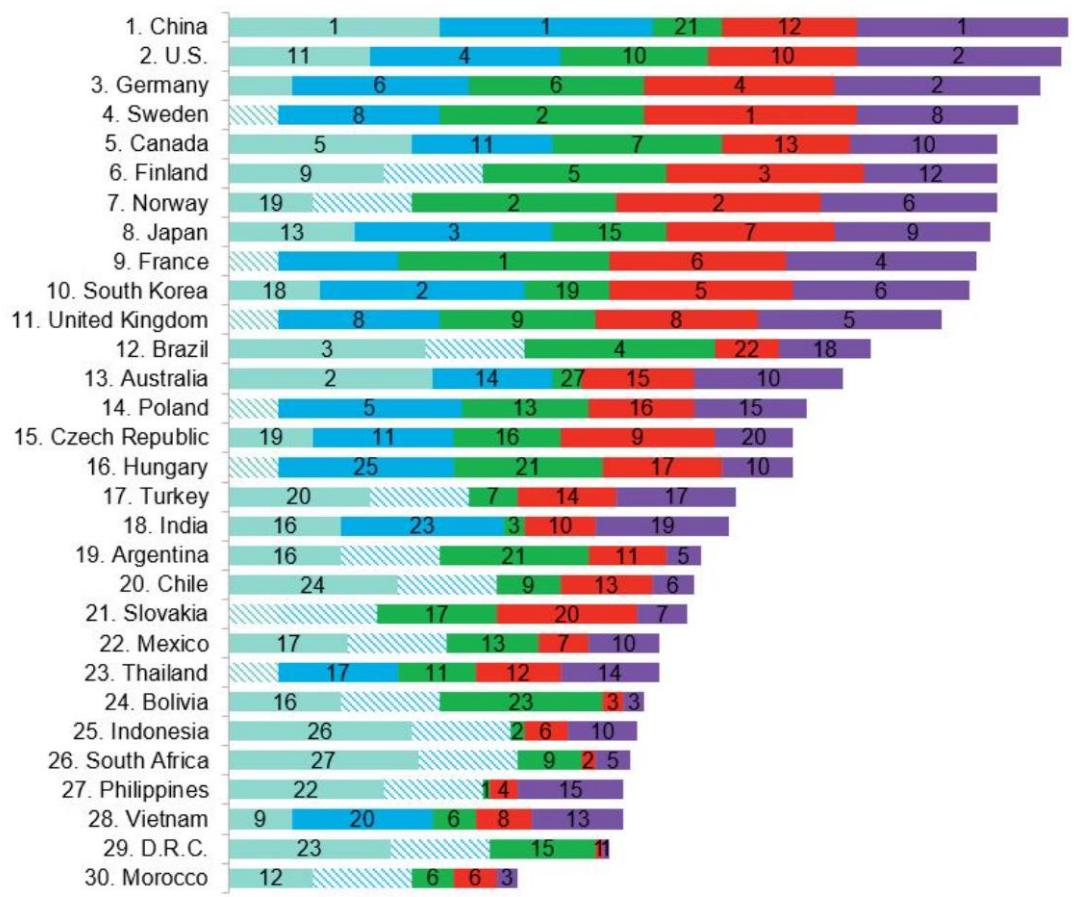
Figure 17: The demand analysis for various critical minerals (IEA)

### 2.9.2 Lithium Supply Analysis

Lithium is an important element that is utilized in many different processes, such as making glass, ceramics, and rechargeable batteries. However, the rise in the markets for renewable energy storage and electric vehicles has recently resulted in a sharp increase in the demand for lithium. Understanding the lithium supply chain globally has therefore grown in significance. (Calisaya et al., 2020).

Most lithium is produced in a small number of nations. Research shows that the top producers are Australia, Chile, Argentina, and China. These nations possess substantial lithium reserves, with the "lithium triangle" region of South America having the greatest lithium reserves. Recycling LiBs to create batteries in the form of  $\text{Li(OH)}$  or  $(\text{Li}_2\text{CO}_3)$  is another source of lithium.

Due to historical underinvestment and the rising demand for electric vehicles, lithium is in short supply. The dynamics of the lithium supply may be momentarily impacted by recent fluctuations in prices and new discoveries. Financing, obtaining permits, and establishing a balance between supply and demand are still long-term issues.(Olivetti et al., 2017)



■ Raw materials ■ Manufacturing ■ Environment ■ Regulations, Innovation & Infrastructure ■ Battery demand

Figure 18: Global lithium-ion battery supply chain ranking

## **CHAPTER THREE**

### **3.0 PROCESS SELECTION AND DESCRIPTION**

#### **3.1 PROCESS SELECTION**

Any plant design project must carefully consider the process method to be used because it can affect the overall cost-effectiveness, safety, and efficiency of the plant. Thus, when choosing a process technique in plant design, factors such as raw material availability, production capacity, process efficiency, safety, cost-effectiveness, environmental impact, adaptability, and equipment selection must be carefully taken into account. (Chen et al., 2011)

##### **1. Availability and raw material selection**

Based on the availability of raw materials, Lithium is to be extracted from spodumene ore for this project since Ghana now has abundant Lithium reserves. Spodumene ore contains 5.8%-8% Li<sub>2</sub>O. Currently, Ewoyaa, a town near Mankessim, in the central region of Ghana, has about 18.9 million tons of lithium reserves in the form of spodumene ore. The world production of lithium via spodumene is around 80,000 metric tons (MT) per year, primarily from the Greenbushes pegmatite of Western Australia and from some Chinese and Chilean sources.

##### **2. Choice of extraction method**

The sulfuric acid method requires a lot of energy but has a high rate of lithium recovery, the alkali and chlorination methods can directly react with lithium ores and use less energy, but they also require careful attention to reaction conditions, equipment safety, and operation. Due to this, the acidic method is preferred. The roasting method is where the technique differs most from others. Three widely recognized techniques exist for

roasting spodumene. The first method is termed acid roasting, and it involves baking the calcined spodumene product for an hour or 320°C (for microwave irradiation) at a temperature of around 250°C (for traditional heat treatment). The second is the alkaline technique, which employs CaO, limestone (CaCO<sub>3</sub>), or lime (Ca(OH)<sub>2</sub>). The third method is the chloride method, which uses several kinds of chlorinating chemicals.

Table 4: Roasting methods and their corresponding percentage yields

<b>Roasting Method</b>	<b>Chemicals used</b>	<b>Temperature (°C)</b>	<b>The percent yield of lithium</b>
1. Acid Roasting (Sulfation)	H <sub>2</sub> SO <sub>4</sub>  K <sub>2</sub> SO <sub>4</sub>  Na <sub>2</sub> SO <sub>4</sub>	250-300	96
2. Alkaline	CaCO <sub>3</sub> ) /lime  (Ca(OH) <sub>2</sub> ) /CaO	100-1050	90
3. Chlorination	Cl <sub>2</sub> CaCl <sub>2</sub>	100-1100  700-900	90

Source: (Chen et al., 2011)

The alkaline approach is more cost-effective and uses salt, a gentle reagent, as opposed to the acid procedure, which uses concentrated acid. The use of chlorine gas and chloride-bearing salts in mineral beneficiation is the least considered method because of the potential toxicity of the generated chlorine gas together with the products and the high cost of corrosion-resistant equipment needed for the Chlorination process. There are several benefits to using chlorine metallurgy in the processing of minerals. The two main

benefits are chlorine's high selectivity and its noticeably reduced operating temperature (Gordon et al., 2014).

### **3. Choice of Final Product Produced**

In terms of how the lithium is to be made, some applications prefer lithium hydroxide over lithium carbonate due to its increased reactivity and water solubility (Riyo et al., 2022). Because of this, it can be helpful in situations where solubility or quick reaction times are crucial. For instance, lithium hydroxide is frequently utilized in the manufacturing of lithium-ion batteries, the performance of which depends on quick and effective ion transfer. The mass percentage of lithium in lithium carbonate ( $\text{Li}_2\text{CO}_3$ ) and lithium hydroxide ( $\text{LiOH}$ ) varies. According to (Gao et al., 2023), lithium carbonate has a mass percentage of approximately 18.8% lithium. This implies that for every 100 g of lithium carbonate, there are approximately 18.8 g of lithium. However, lithium hydroxide has approximately 30.4% lithium by mass, which means that for every 100 g of lithium hydroxide, there are about 30.4 g of lithium which makes it a more economical option in some cases.

## PROCESS FLOW SHEET

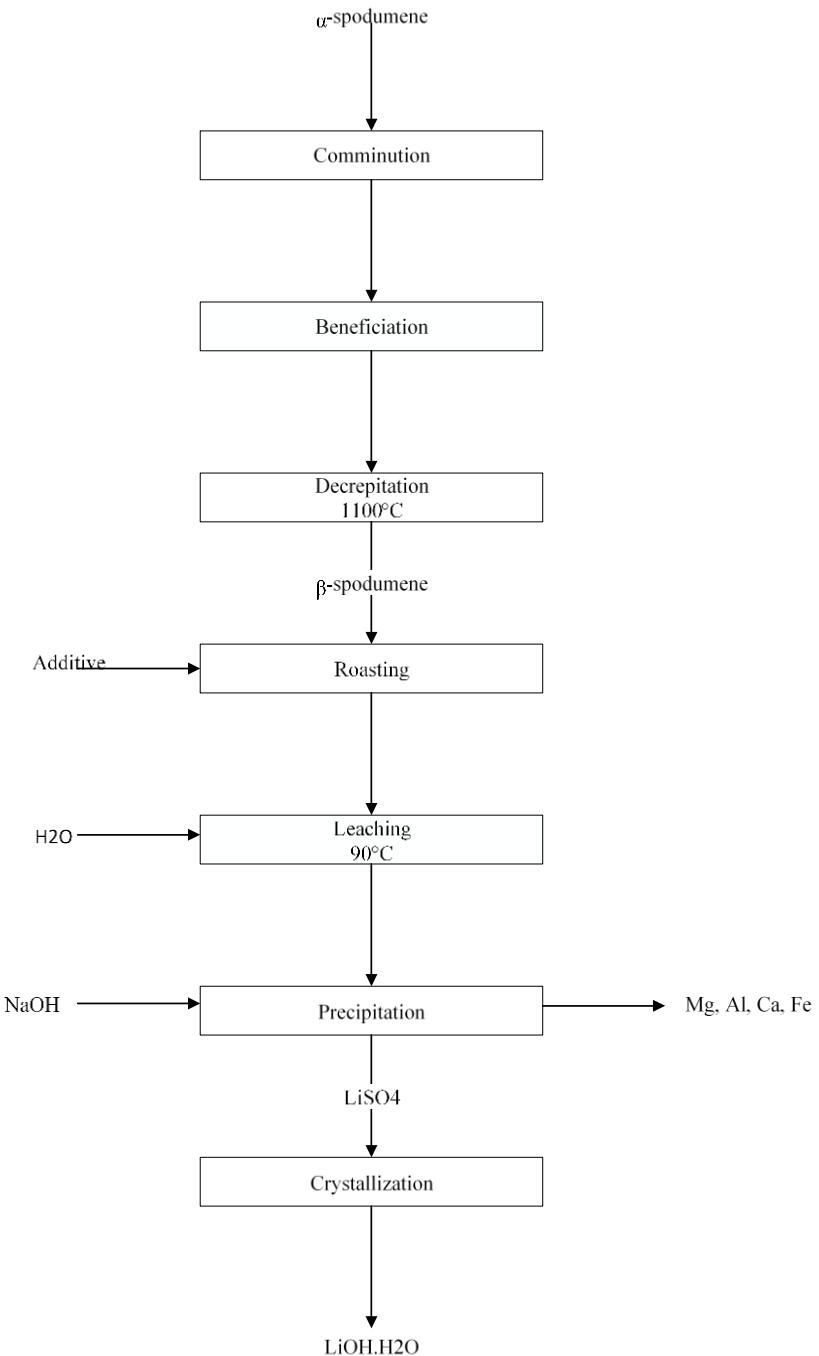


Figure 19: Block diagram for Lithium Production from Spodumene ore (Source: Schmitt et al, 2019).

## **3.2 PROCESS DESCRIPTION**

### **3.2.1 Raw Material**

The raw material used was spodumene ore.

### **3.2.2 Comminution**

The majority of ore minerals are typically closely linked to gangue minerals and are finely dispersed. Hence, in order for the minerals to be collected into distinct products, they must first be "liberated" or broken apart. Consequently, the first step in any ore dressing procedure will be to crush and grind the ore to the point where each mineral grain is essentially free. This process is known as comminution.

The process of crushing and grinding normally involves a series of steps that gradually lower the size of the lump. There are two grinding stages and three crushing stages.

- i. Primary Crushing (Coarse Crushing): A jaw or gyratory crusher is used to crush ore or run-of-mine ore up to 1 m in size down to a size of about 10 cm.
- ii. Secondary Crushing (Intermediate Crushing): Roll, cone, or jaw crushers are used to reduce ore size from 10 cm to less than 1 or 2 cm. Compared to primary crushers, these secondary crushers use more energy.
- iii. Tertiary Crushing (Fine Crushing): Ore is crushed by tertiary crushers to a size of no more than 1–2 cm.

In our case, traditional mining techniques including blasting are used to extract the ore from an open pit at Ewoyaa. After that, a truck is used to deliver the mine ore to the processing facility. A jaw crusher is used at the processing plant to smash the spodumene ore. Because of their great capacity, ability to handle huge rocks, and use in the main crushing stage, jaw crushers are frequently utilized for this purpose. Secondary crushers are used to further crush the spodumene ore to a smaller size following the main crushing.

Because cone crushers can create a finer product than jaw crushers, they are frequently utilized for this purpose. The larger particles are sent back to the cone crusher for further processing and undersized particles are sent to the next stage. A conveyor belt is used along the process to convey the particles from one stage to another.

### **3.2.3 Beneficiation**

Ore dressing or beneficiation is a process of mechanically separating the grains of ore minerals from the gangue minerals, to produce a concentrate (enriched portion) containing most of the ore minerals and a tailing (discard) containing the bulk of the gangue minerals.

But in most ore processing, the mineral comes with less gangue surrounding the ore of interest. This makes it invaluable to have machines purposely for beneficiation. Therefore, it is more economical to omit this process since it does not have any adverse effect on the entire process.

### **3.2.4 Decrepitation**

Spodumene ore's decrepitation is a required fracture of the crystal structure of the ore. Calcination is a high-temperature thermal treatment done in a rotary kiln which causes this fracturing. Decrepitation occurs during calcination at temperatures ranging from 1,075 to 1,100° C. Temperature control is crucial during this period to prevent undesired forms, which happen at somewhat higher temperatures. Decrepitation is a process used to extract lithium from spodumene ore by converting  $\alpha$ -spodumene to  $\beta$ -spodumene. Phase change from  $\alpha$ -form to  $\beta$ -form is necessary for the chemical extraction of lithium from spodumene concentrate. This is because  $\beta$ -spodumene is a porous substance with a bigger surface area and significantly higher reactivity than  $\alpha$ -spodumene, which is a rather compact mineral with moderate reactivity.

The process of decrepitation involves heating the spodumene ore to a high temperature of 1100°C using natural gas for an hour in a rotary kiln. The application of heat causes volumetric expansion which makes the ores porous. The final  $\beta$ -spodumene consequently has a larger surface area which improves subsequent chemical processing. Also, the  $\alpha$ -spodumene cannot be easily decomposed by acid hence the conversion. The quartz in the pegmatite composition undergoes no transformation. The  $\beta$ -spodumene is then cooled to 100°C. The spodumene is screened to separate the  $\beta$ -spodumene from the unreacted  $\alpha$ -spodumene. The unreacted  $\alpha$ -spodumene is recycled back to the rotary kiln. The  $\beta$ -spodumene is then ground into a fine powder using a ball mill. The finely ground  $\beta$ -spodumene is sent to the next process with the aid of a belt conveyor. The reaction for 96% conversion of  $\alpha$ -spodumene to  $\beta$ -spodumene is shown below. (Granata, 2020);

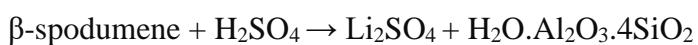


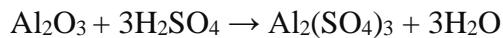
### **3.2.5 Baking or Acid Roasting**

Spodumene undergoes acid roasting after decrepitation. To create lithium compounds, spodumene concentrate is combined with sulfuric acid and roasted.

The  $\beta$ -spodumene that has been finely powdered during the decrepitation process is roasted for one hour at a temperature of around 250°C with 40% sulfuric acid. This procedure also makes use of a rotary kiln. A part of the exhaust from the decrepitation kiln is used to create heat for this kiln. The reason this procedure works so well is that during the leaching step, lithium sulfate becomes extremely stable and dissolves easily in water.  $\beta$ -spodumene, alumina, and the solid biotite mixture are transformed into their corresponding sulfates during the sulfation-roasting process.

The reactions which characterize this process are;



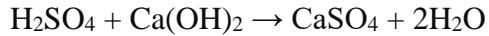


### 3.2.6 Leaching

In mineral processing, hydro-leaching is often referred to as hydrometallurgical leaching, is a particular kind of leaching procedure where metals are extracted from ores using water-based solutions. In this case, water is used for cooling. After using a chiller to bring the material's temperature from the sulfation kiln down to 100°C, recycled water from the crystallizers is combined with it in a mixing tank. To make sure that the final water content is at least 50% w/w, further water is added after that. Extra water is added to ensure that the final water content is no lower than 50% wt/wt. This means that, after mixing the additional water, the resulting mixture must have a water content of at least 50% by weight or if we are to weigh the final mixture, at least 50% of the weight should be water.

### 3.2.7 Neutralization and Precipitation

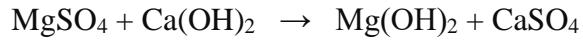
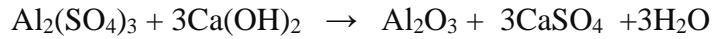
At this point, the sulfuric acid of the mixture is then neutralized with calcium hydroxide in a Neutralization procedure as shown below.



Lithium sulfate is also leached from the neutralization vessel. Due to its great water solubility, practically all of the lithium sulfate is still in solution at this stage of the process. After that, the precipitated and suspended elements (unreacted spodumene, aluminum silicate, and the produced  $\text{CaSO}_4$ , which is not very soluble in water) are eliminated using a clarifier. Iron, aluminum, and magnesium contaminants are present in the clarifier along with the majority of the dissolved lithium sulfate. Consequently, in

order to precipitate iron, aluminum, and magnesium as hydroxides, two further neutralization processes are required: one at pH 5.5–6, for iron and aluminum, and one at pH 10–11, for magnesium.

The following reaction explains the above;



Subsequently, to get rid of the precipitates, a belt filtration stage comes after each neutralization phase. The majority of the dissolved lithium sulfate in the filtrate from the second Belt Filtration is subsequently supplied to a second Neutralization process, where lithium sulfate combines with sodium hydroxide to generate lithium hydroxide in the reaction below.



### **3.2.8 Crystallization and Purification of product (Lithium Hydroxide)**

After that, the lithium hydroxide solution is fed into an evaporator or crystallizer, where water is removed and the concentrate is cooled to 40°C to enable the lithium hydroxide crystallize. After that, the mother liquor and crystalline lithium hydroxide are separated using a clarifier. Using a rotary vacuum filter, the clarifier's solids are further concentrated after the cake is rinsed with water to eliminate any dissolved contaminants. Using a rotary dryer, the moisture from the purified cake is finally removed.

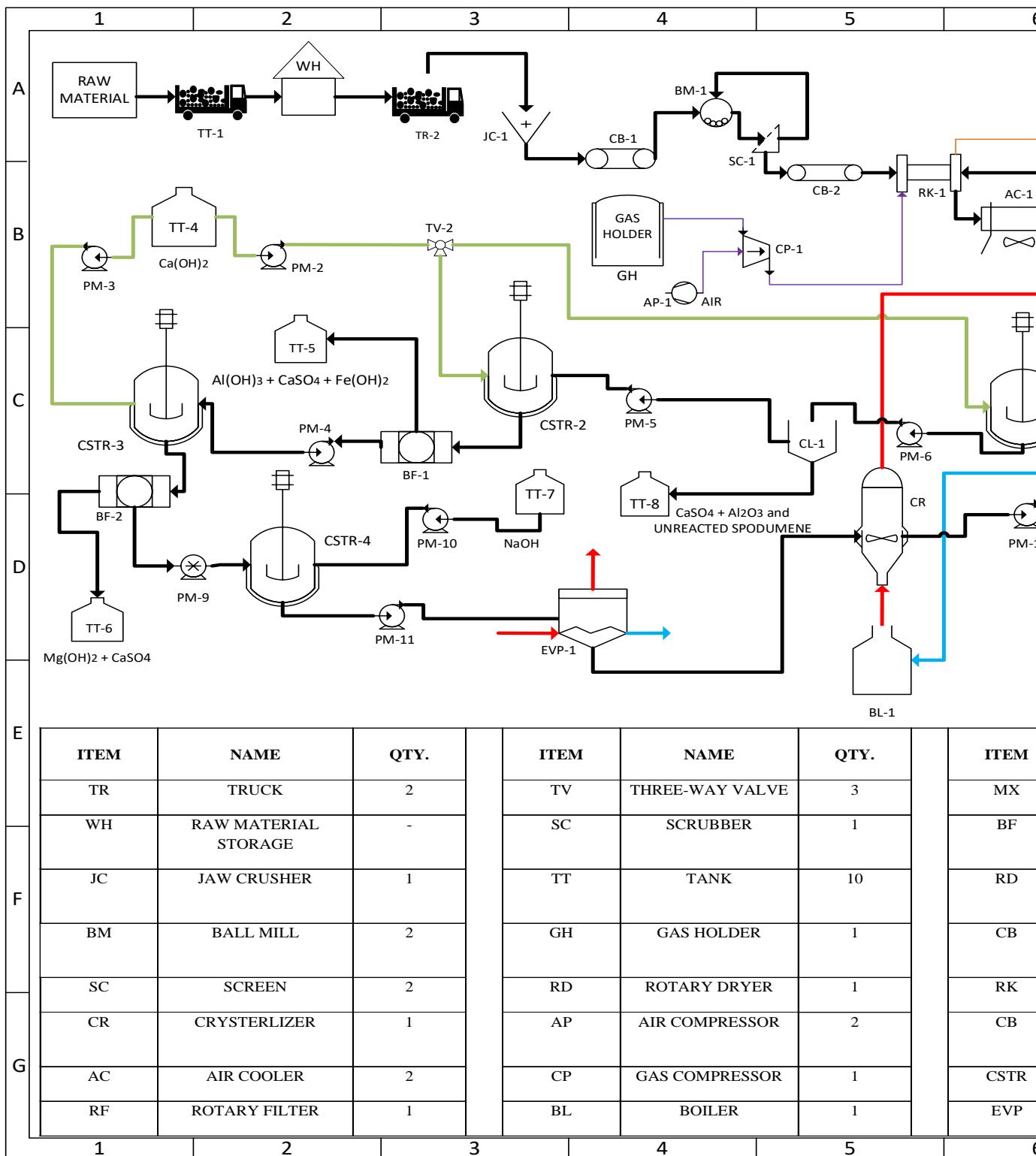


Figure 20: Process Flow Diagram

## CHAPTER FOUR

### 4.0 MATERIAL AND ENERGY BALANCE

#### 4.1 MATERIAL BALANCE

##### Plant Attainment

The plant has a processing capacity of 20 tonnes per hour (20,000 kg/h) of spodumene ore. It is a batch process that will run continuously. This will be achieved due to the continuous supply of the spodumene ore and other auxiliary materials that would be required. The plant will operate 335 days in a year which contributes to approximately 91.78 % of the total number of days in a year leaving room for 30 days downtime maximum. About 1918.07 kg (1.9 tonnes) of lithium hydroxide is expected to be produced hourly.

General assumptions made with respect to the material balance.

1. Steady State operation (no accumulation).
2. All masses are calculated on an hourly basis.
3. 20,000 kg is the total amount of ore processed per hour
4. Daily production rate of 1918 kg/h of Li(OH).

Table 5: Compositions of spodumene ore processed

Mineral Components	Percentages % by mass	Mass flow rate (kg/h)
Spodumene, (LiAlSi <sub>2</sub> O <sub>6</sub> )	80	16000
Quartz, (SiO <sub>2</sub> )	10	2000
Alumina, (Al <sub>2</sub> O <sub>3</sub> )	5	1000
Annite, KFe <sub>3</sub> AlSi <sub>3</sub> O <sub>10</sub> (OH) <sub>2</sub>	2.5	500

Phlogopite, $KMg_3AlSi_3O_{10}(OH)_2$	2.5	500
<b>Total</b>	100	20,000

The overall material balance is given by the general equation below;

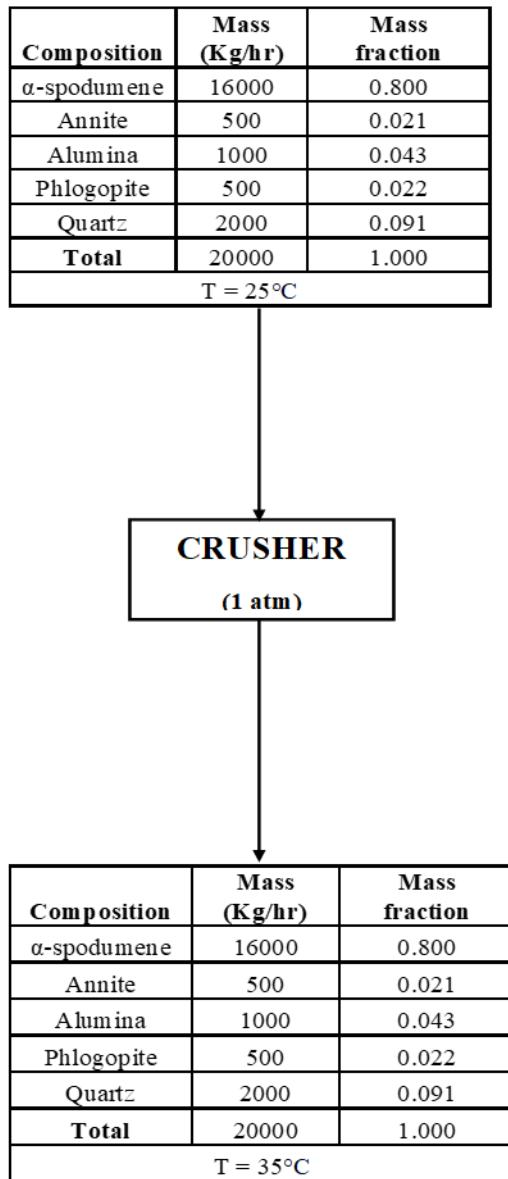
$$\text{Input} + \text{Output} + \text{Generation} = \text{Accumulation}$$

From the law of conservation of matter, input = output.

Detailed calculations are shown in Appendix A.

## Material Balance on Crusher

- No reaction and chemical change occurred in the crusher. Only size reduction of the feed.



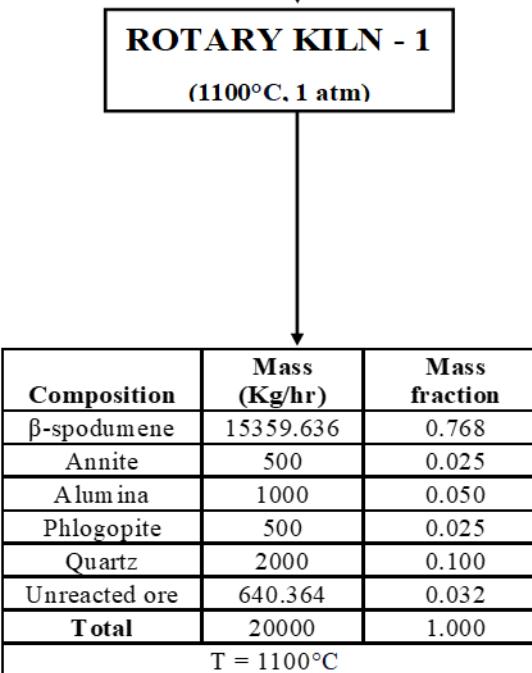
## Material Balance on Rotary Kiln 1(RK-1)

Assumptions:

- An air cooler was used to cool the product from 1100°C to 100°C
- Ratio of Raw material to (Natural gas + Air) is almost 1:2
- 96% Conversion of  $\alpha$ -spodumene to  $\beta$ -spodumene

Composition	Mass (Kg/hr)	Mass fraction
$\alpha$ -spodumene	16000	0.800
Annite	500	0.021
Alumina	1000	0.043
Phlogopite	500	0.022
Quartz	2000	0.091
<b>Total</b>	<b>20000</b>	<b>1.000</b>

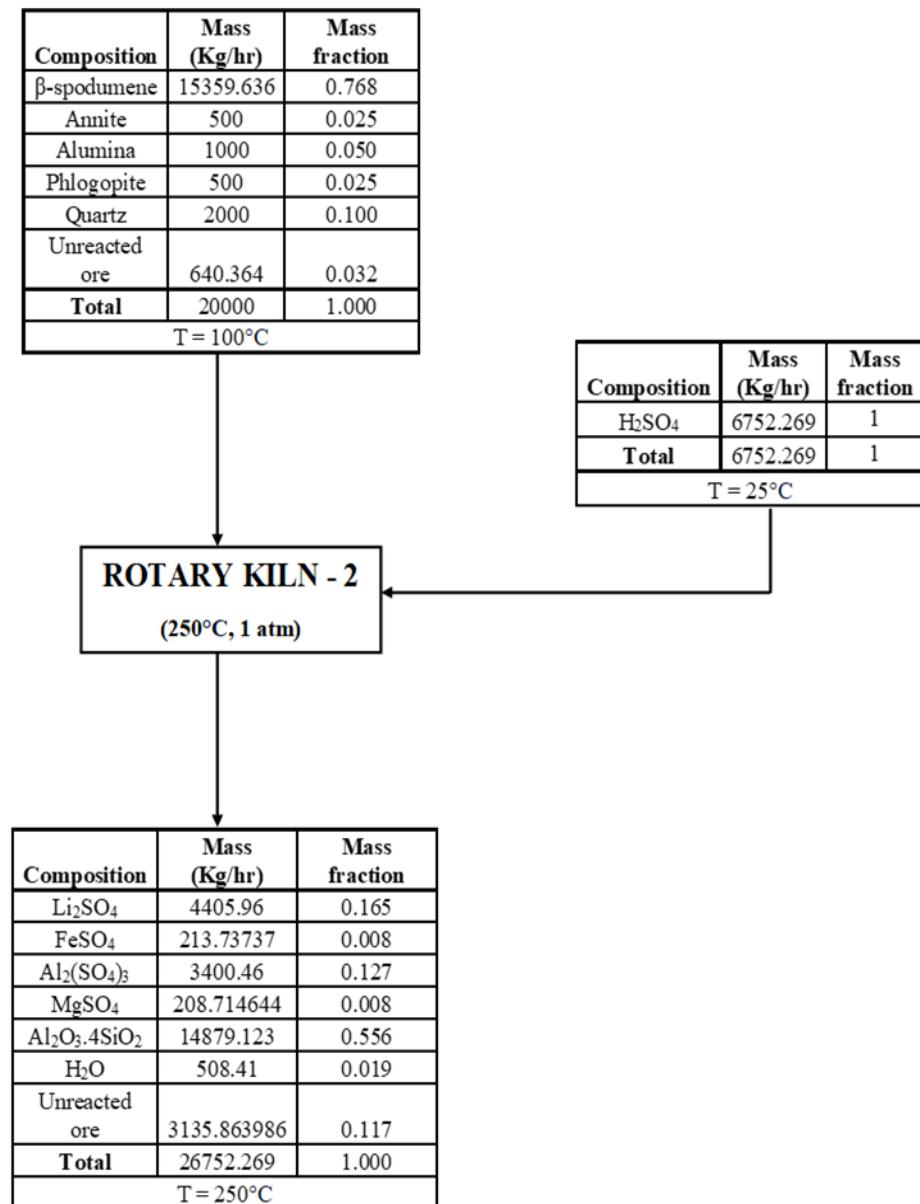
$T = 35^{\circ}\text{C}$



## Material Balance on Rotary Kiln 2 (RK-2)

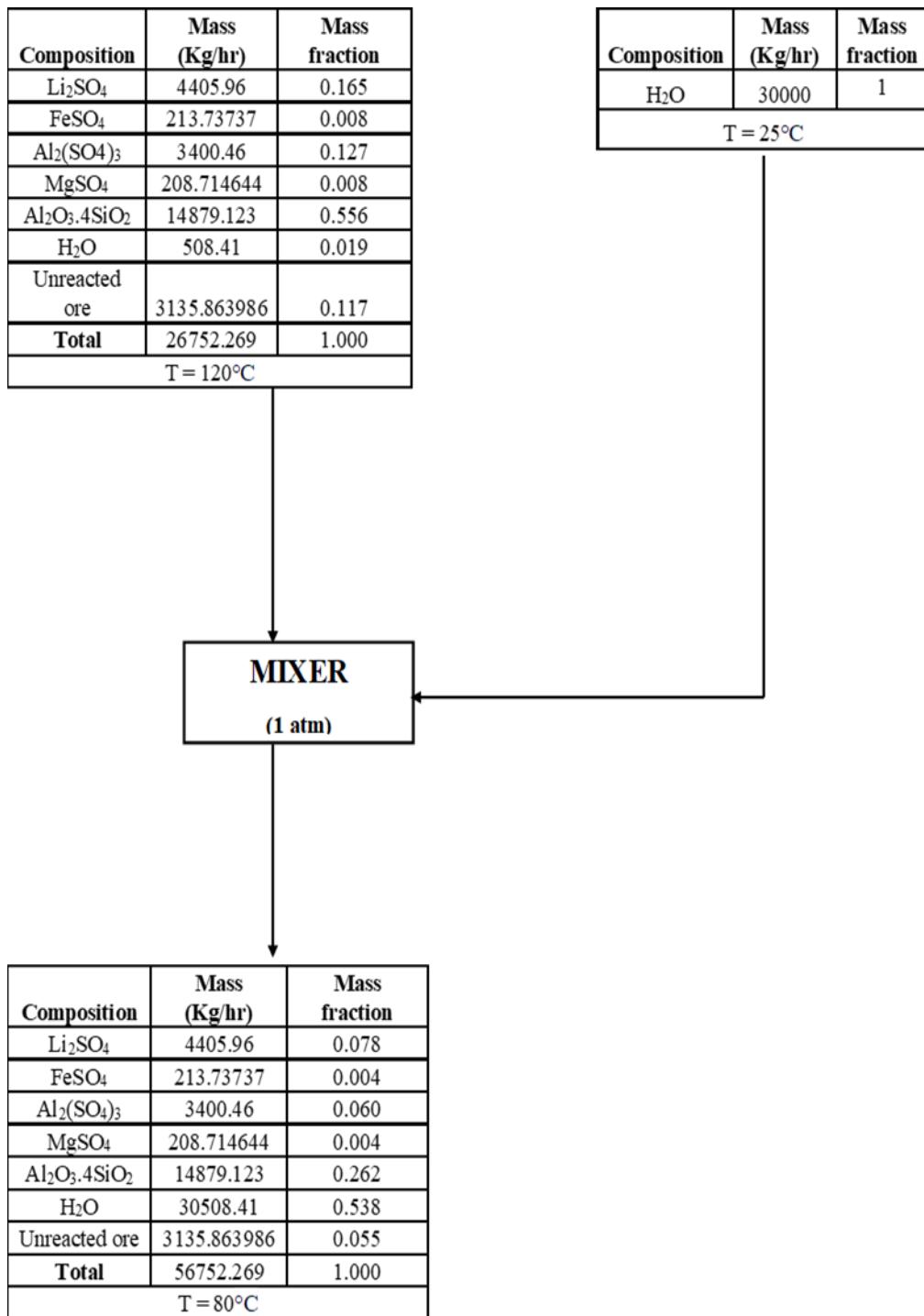
Assumptions:

- 80% of the exhaust gas from the decrepitation process is used to heat the kiln
- 6752.269 kg/h H<sub>2</sub>SO<sub>4</sub> is used for the acid roasting



## Material Balance on Mixer (MX)

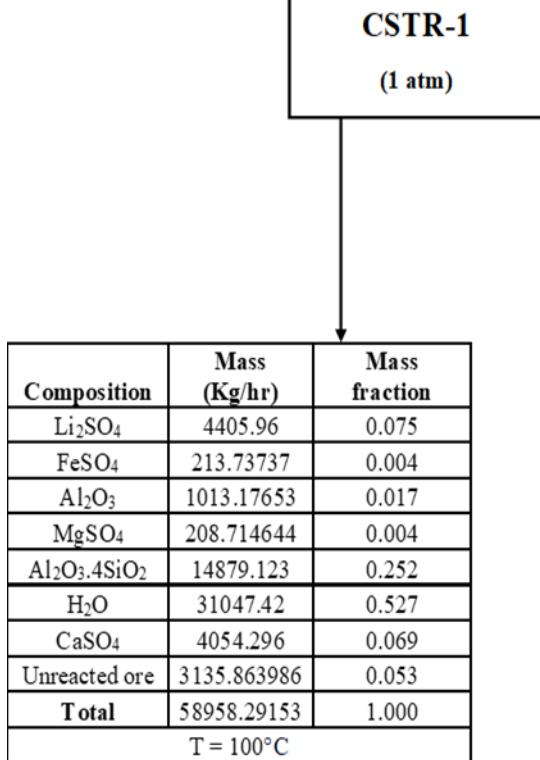
- 30000 kg/h of water is used for the water leaching process



## Material Balance on CSTR 1

Composition	Mass (Kg/hr)	Mass fraction
Li <sub>2</sub> SO <sub>4</sub>	4405.96	0.078
FeSO <sub>4</sub>	213.73737	0.004
Al <sub>2</sub> (SO <sub>4</sub> ) <sub>3</sub>	3400.46	0.060
MgSO <sub>4</sub>	208.714644	0.004
Al <sub>2</sub> O <sub>3</sub> .4SiO <sub>2</sub>	14879.123	0.262
H <sub>2</sub> O	30508.41	0.538
Unreacted ore	3135.863986	0.055
<b>Total</b>	<b>56752.269</b>	1.000
T = 80°C		

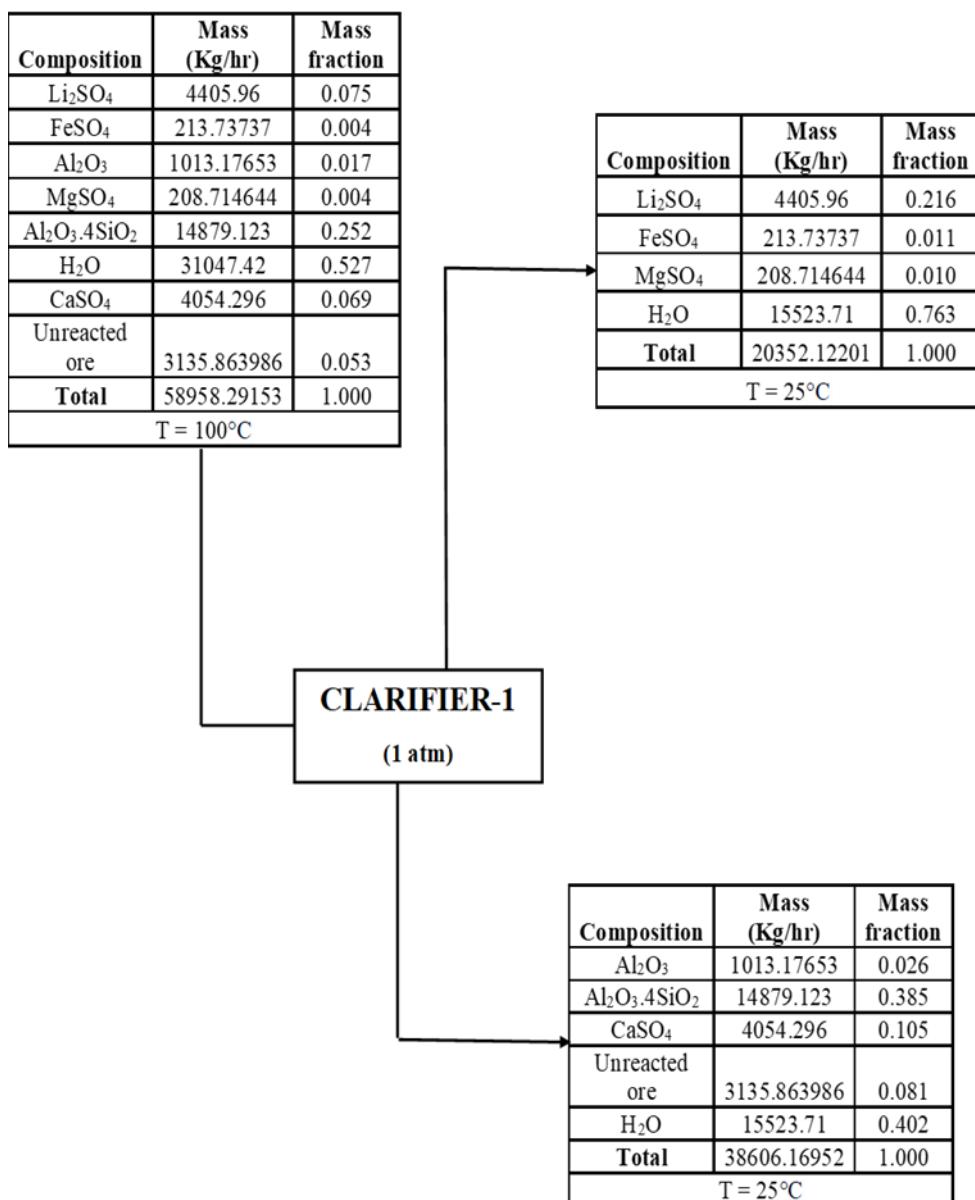
Composition	Mass (Kg/hr)	Mass fraction
Ca(OH) <sub>2</sub>	2206.014	1
T = 25°C		



## Material Balance on Clarifier (CL-1)

Assumption:

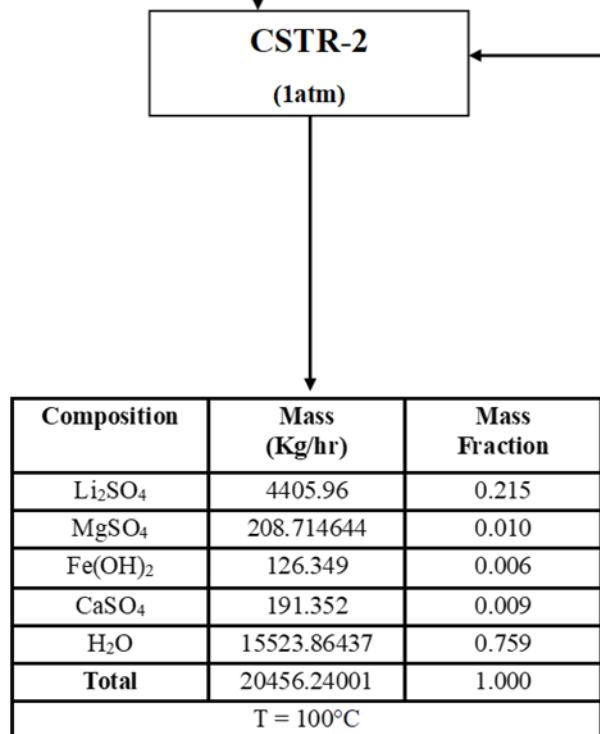
- 99.9% efficient
- Water Fraction of bottom solids is 0.5



## Material Balance on CSTR-2

Composition	Mass (Kg/hr)	Mass fraction
Li <sub>2</sub> SO <sub>4</sub>	4405.96	0.216
FeSO <sub>4</sub>	213.73737	0.011
MgSO <sub>4</sub>	208.714644	0.010
H <sub>2</sub> O	15523.71	0.763
<b>Total</b>	<b>20352.12202</b>	<b>1.00</b>
$T = 25^\circ\text{C}$		

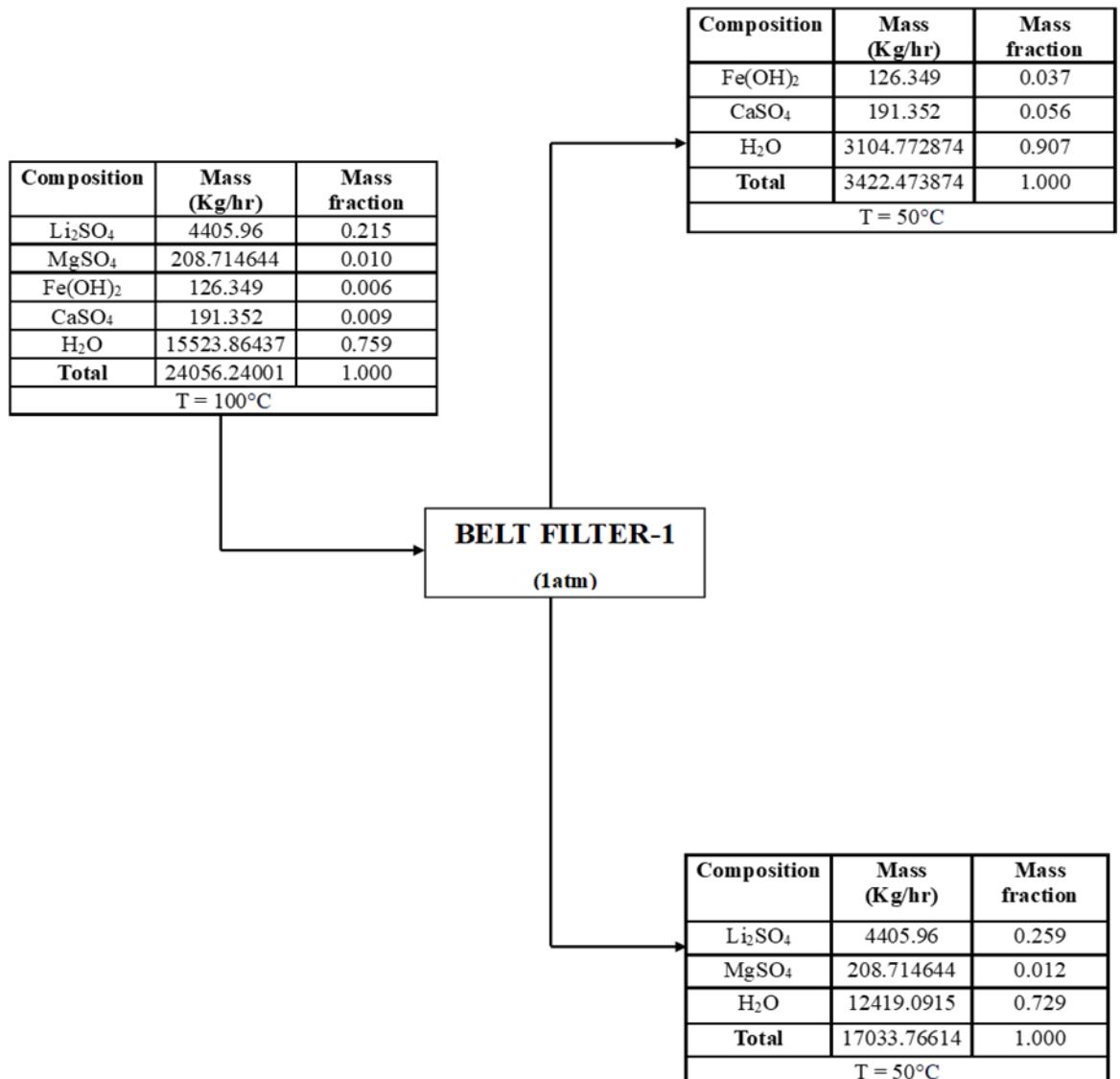
Composition	Mass (Kg/hr)	Mass fraction
Ca(OH) <sub>2</sub>	104.118	1
$T = 25^\circ\text{C}$		



## Material Balance on Belt Filter 1

Assumption:

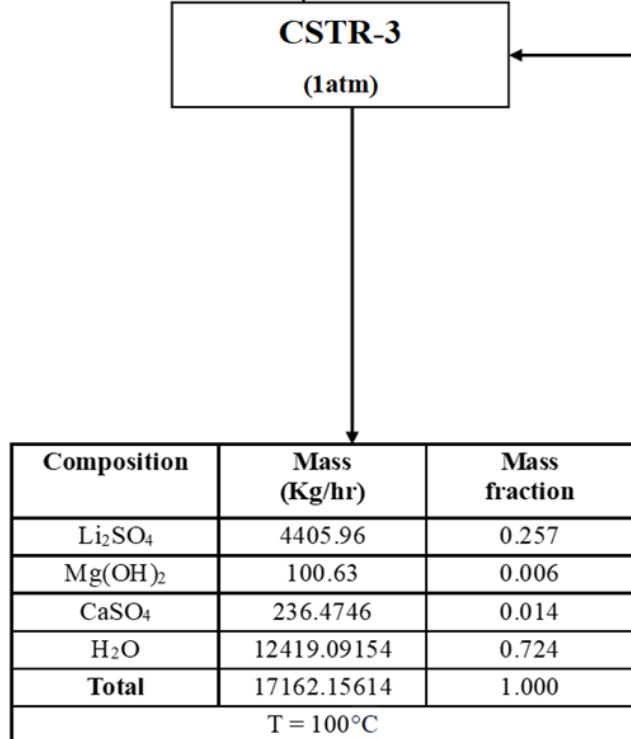
- 99.9% efficient
- Water fraction in solids is 0.2



## Material Balance on CSTR 3

Composition	Mass (Kg/hr)	Mass fraction
Li <sub>2</sub> SO <sub>4</sub>	4405.96	0.259
MgSO <sub>4</sub>	208.714644	0.012
H <sub>2</sub> O	12419.0915	0.729
<b>Total</b>	<b>17033.76614</b>	<b>1.000</b>
T = 50°C		

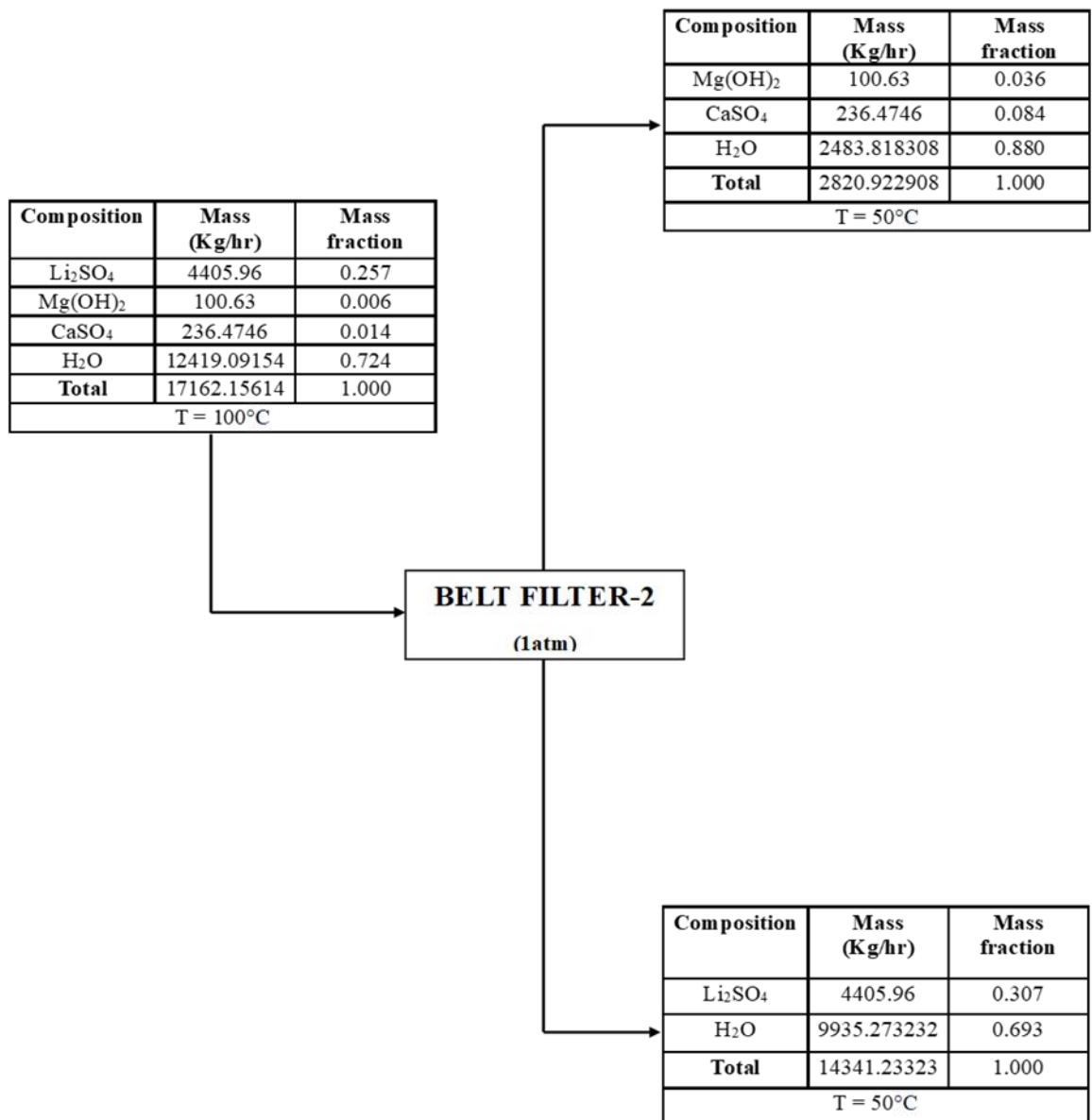
Composition	Mass (Kg/hr)	Mass fraction
Ca(OH) <sub>2</sub>	128.39	1
T = 25°C		



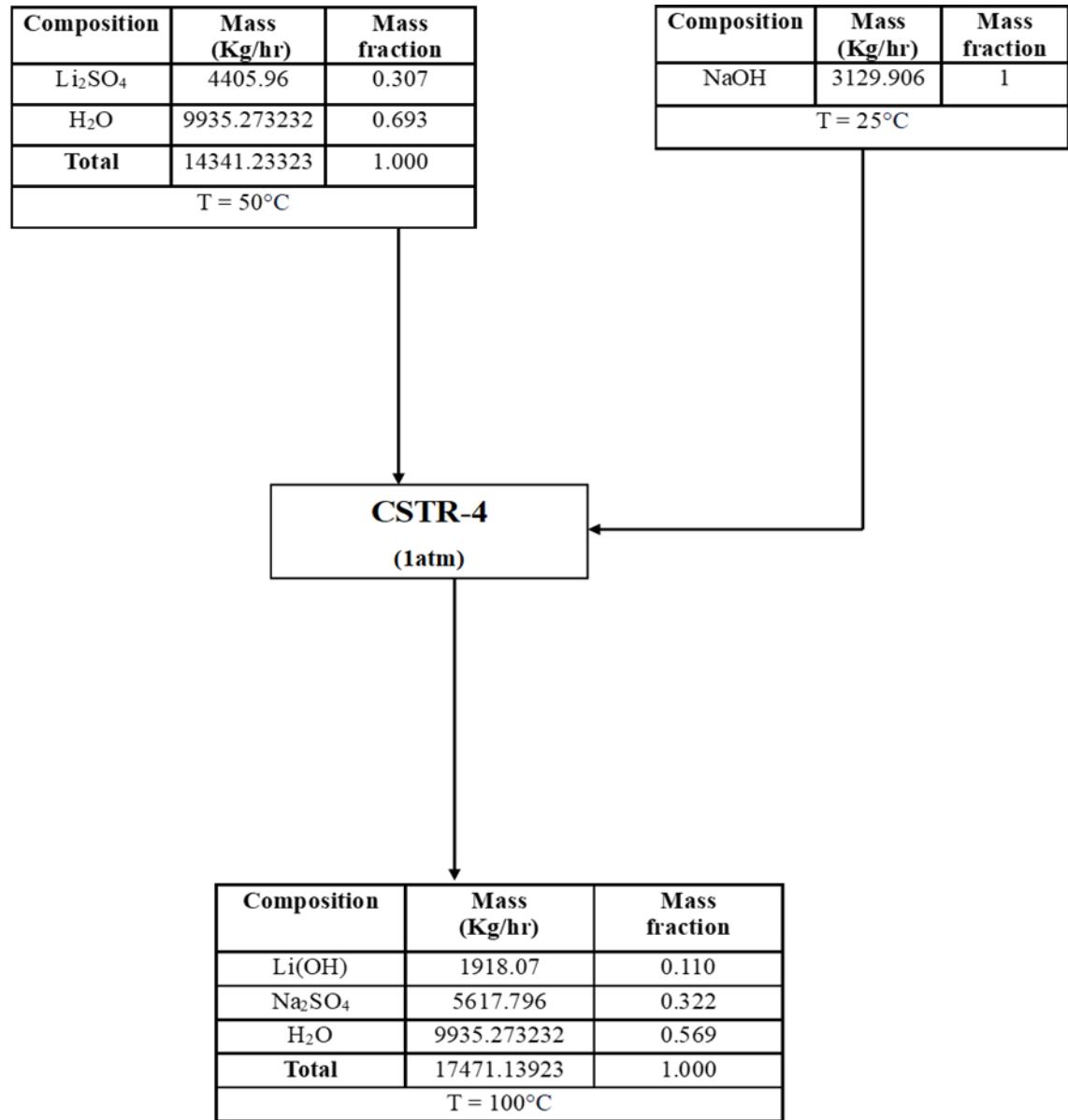
## Material Balance on Belt Filter (BF-2)

Assumption for filter;

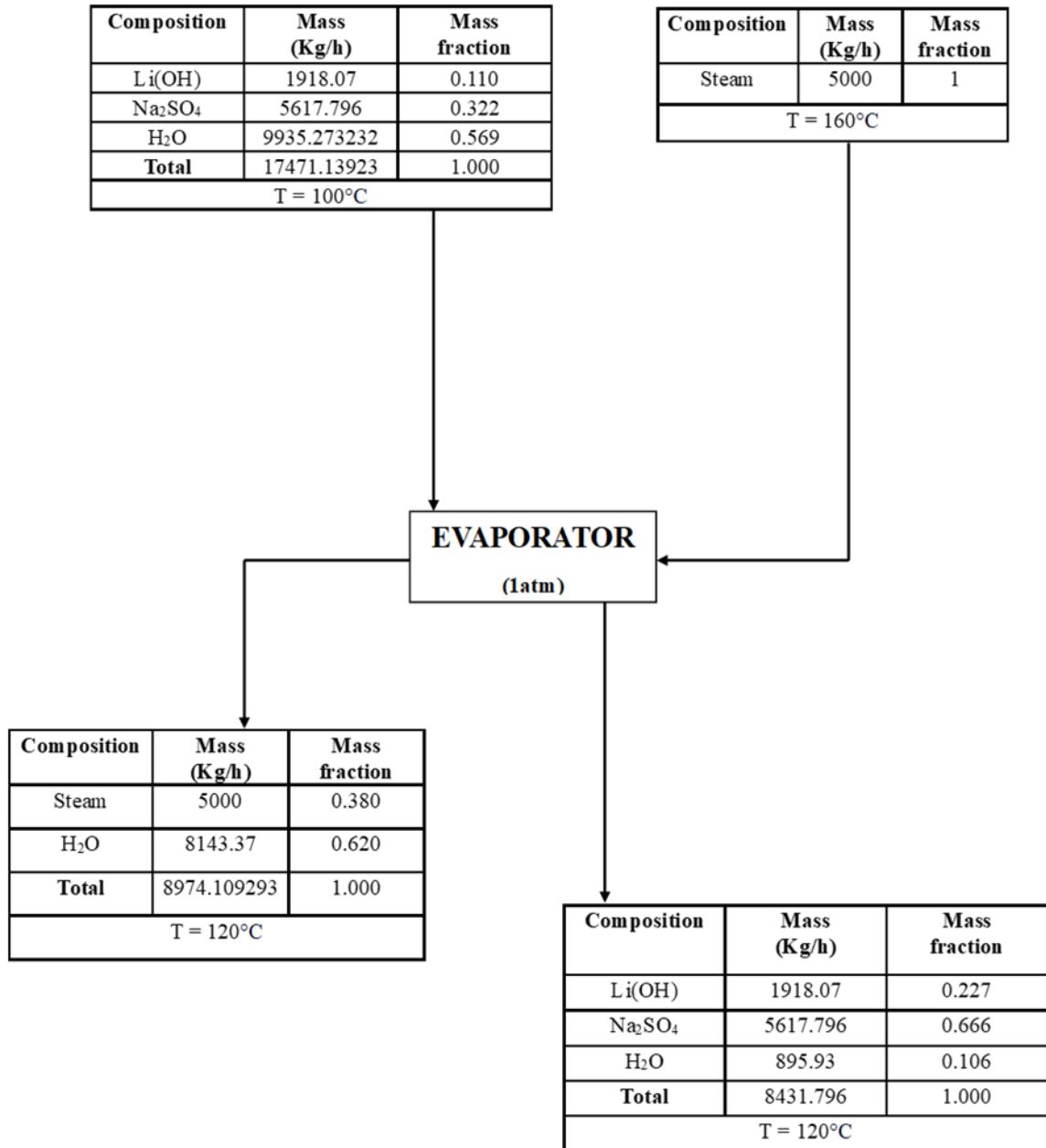
- 99.9% efficient
- Water fraction in of solids is 0.2



## Material Balance on CSTR-4

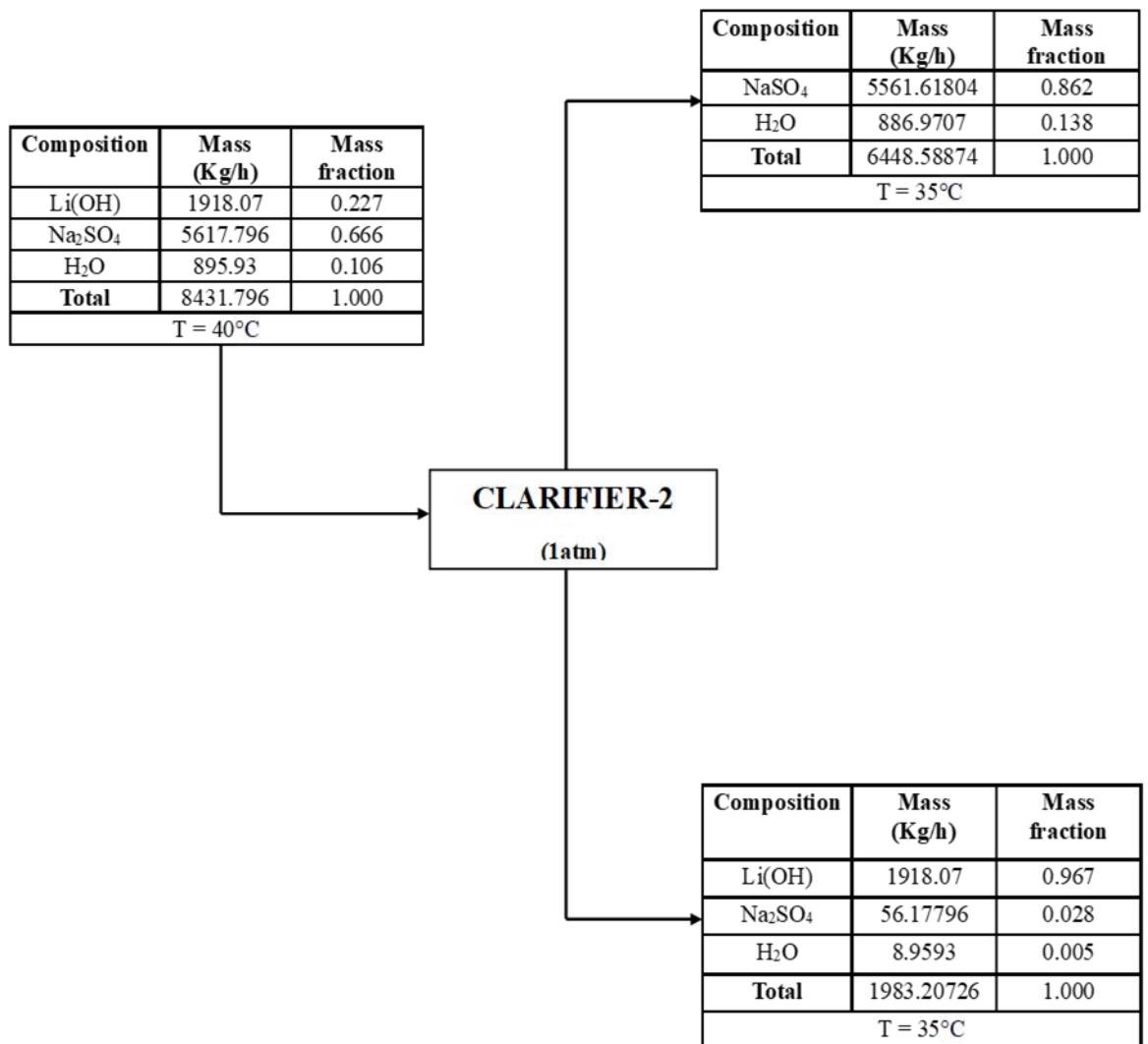


## Material Balance on Evaporator Unit



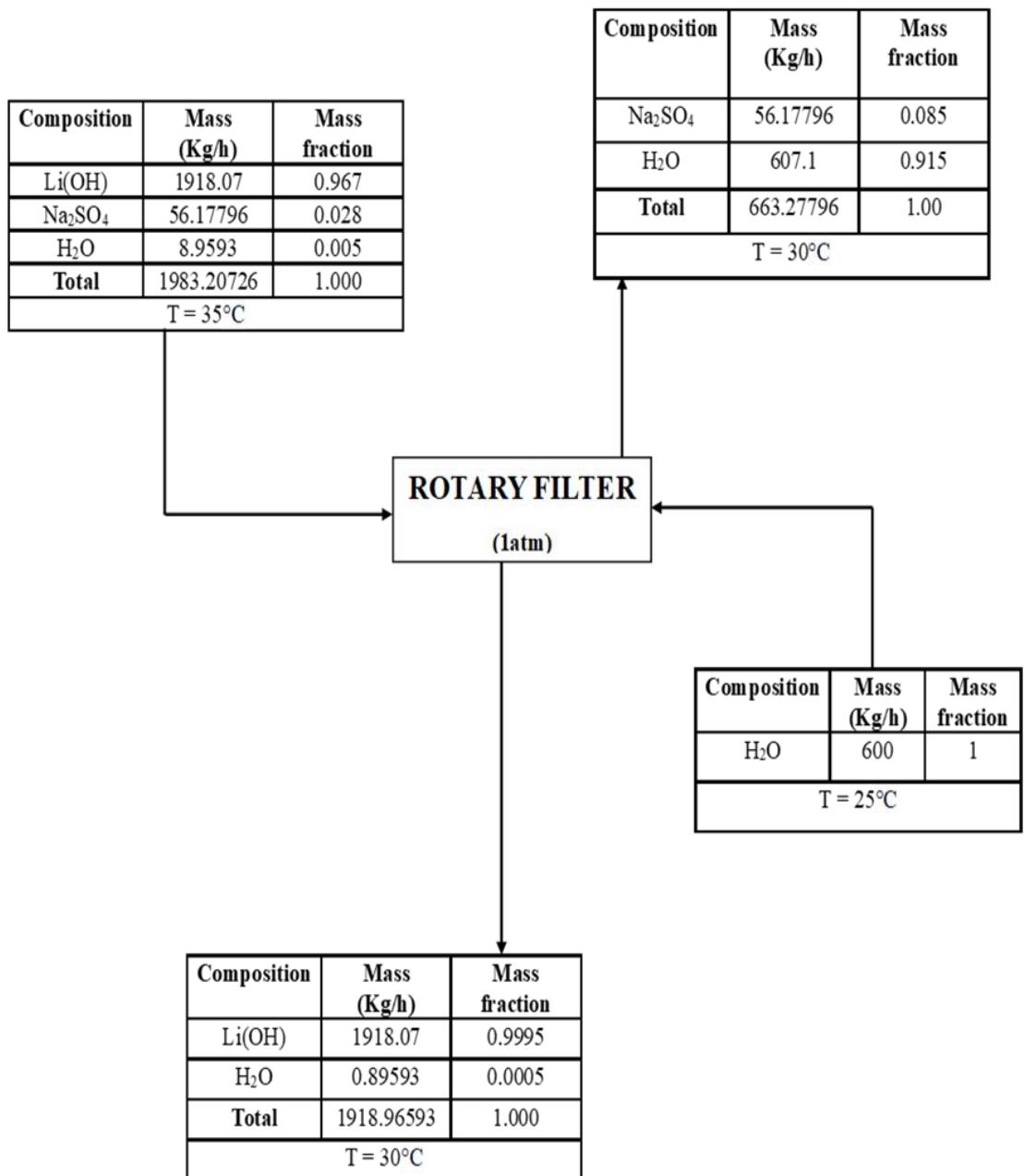
## Material Balance on Clarifier-2

- 99.9% efficient

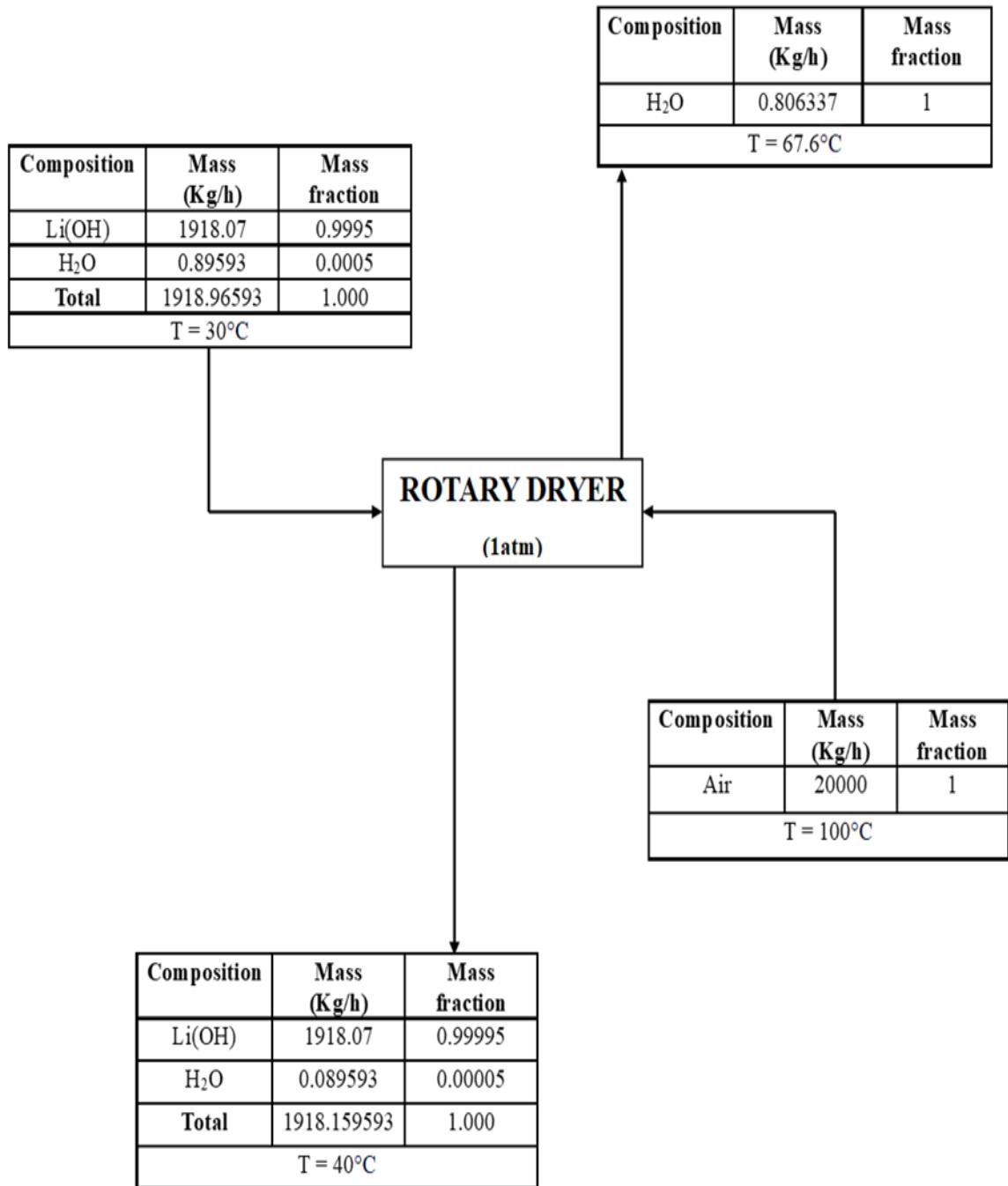


## Material Balance on Rotary Filter

- 90% efficient



## Material Balance on Rotary Dryer



## **4.2 ENERGY BALANCE**

The basis for Energy Balance;

- i. The calculations are done on a daily basis.
- ii. The reactions occurring in the continuous stirred tank reactor units are exothermic.
- iii. Equation used;  $\Delta H = \sum \Delta H_{reaction} + \sum \Delta H_{output} - \sum \Delta H_{input}$

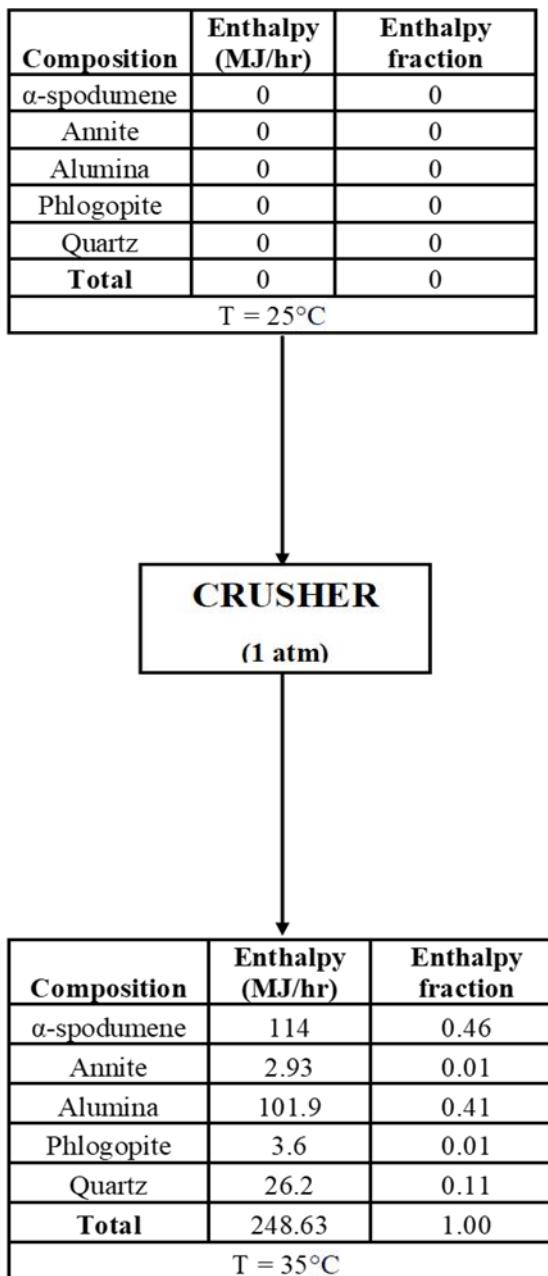
Assumptions;

- All operations are at steady state.
- A basis of 1MJ/h for every operation is taken.
- The flow dependent terms, that is, kinetic and potential energy terms were neglected.
- Reference temperature of 25 °C is assumed.
- Reference pressure of 101.325kPa (1atm) is used.

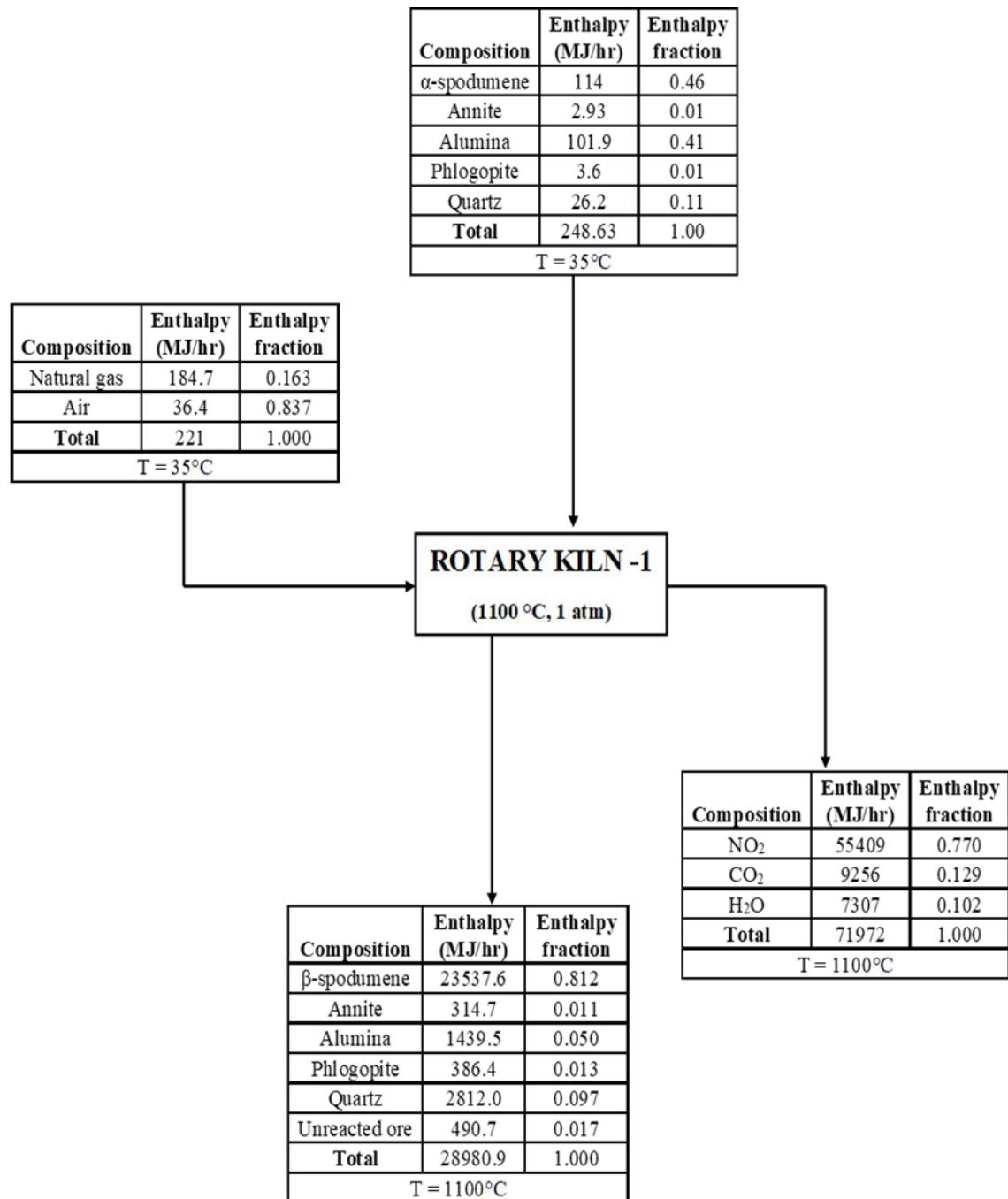
The diagrams below contain a summary of the energy balance on all the major equipment

Detailed calculations are shown in Appendix B

## Energy Balance on Crusher



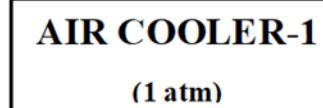
## Energy Balance on Rotary Kiln 1



## Energy Balance on Air Cooler 1

<b>Composition</b>	<b>Enthalpy (MJ/hr)</b>	<b>Enthalpy fraction</b>
$\beta$ -spodumene	23537.6	0.812
Annite	314.7	0.011
Alumina	1439.5	0.050
Phlogopite	386.4	0.013
Quartz	2812.0	0.097
Unreacted ore	490.7	0.017
<b>Total</b>	<b>28980.9</b>	<b>1.000</b>

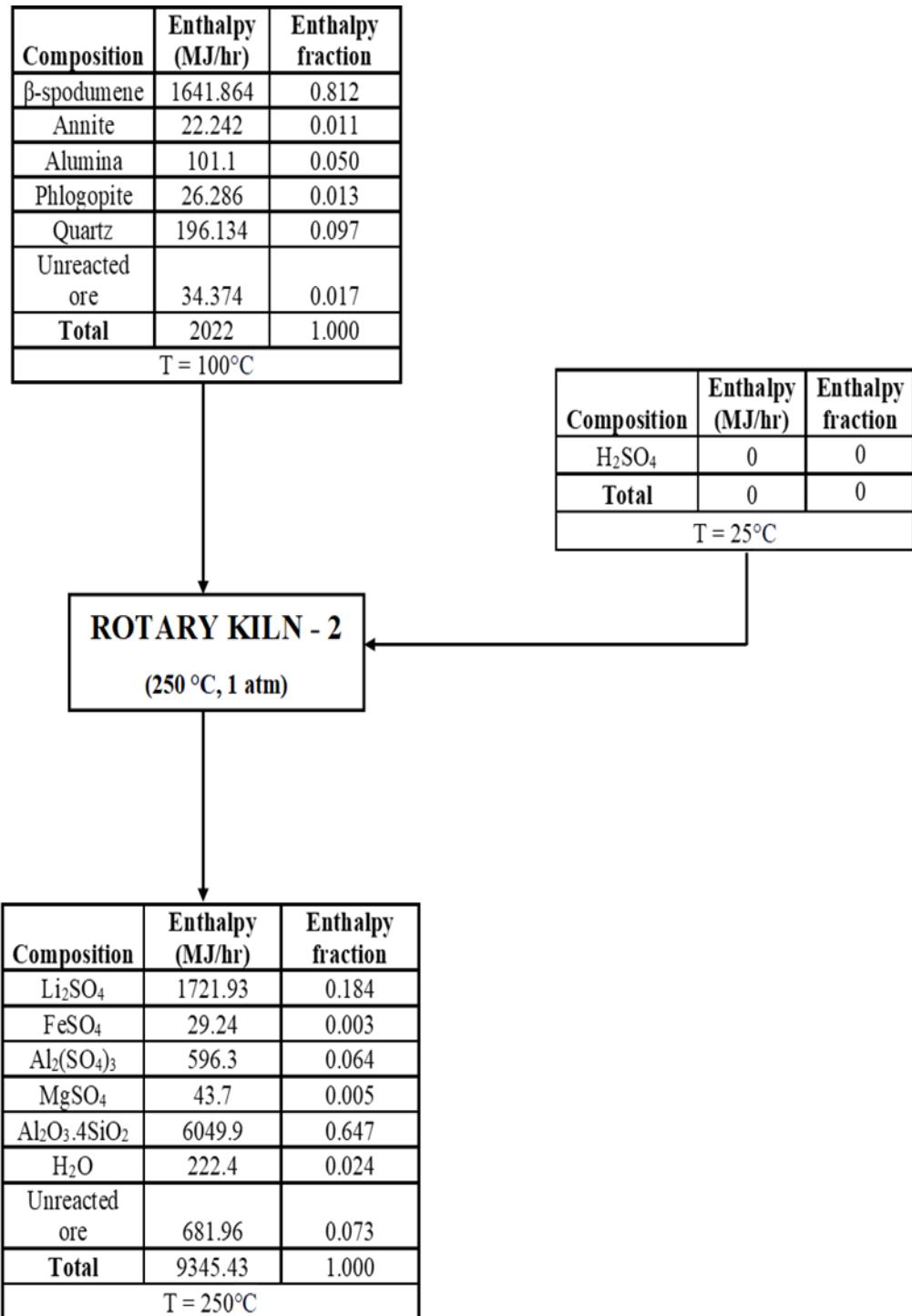
T = 1100°C



<b>Composition</b>	<b>Enthalpy (MJ/hr)</b>	<b>Enthalpy fraction</b>
$\beta$ -spodumene	1641.864	0.812
Annite	22.242	0.011
Alumina	101.1	0.050
Phlogopite	26.286	0.013
Quartz	196.134	0.097
Unreacted ore	34.374	0.017
<b>Total</b>	<b>2022</b>	<b>1.000</b>

T = 100°C

## Energy Balance on Rotary Kiln 2



## Energy Balance on Air Cooler 2

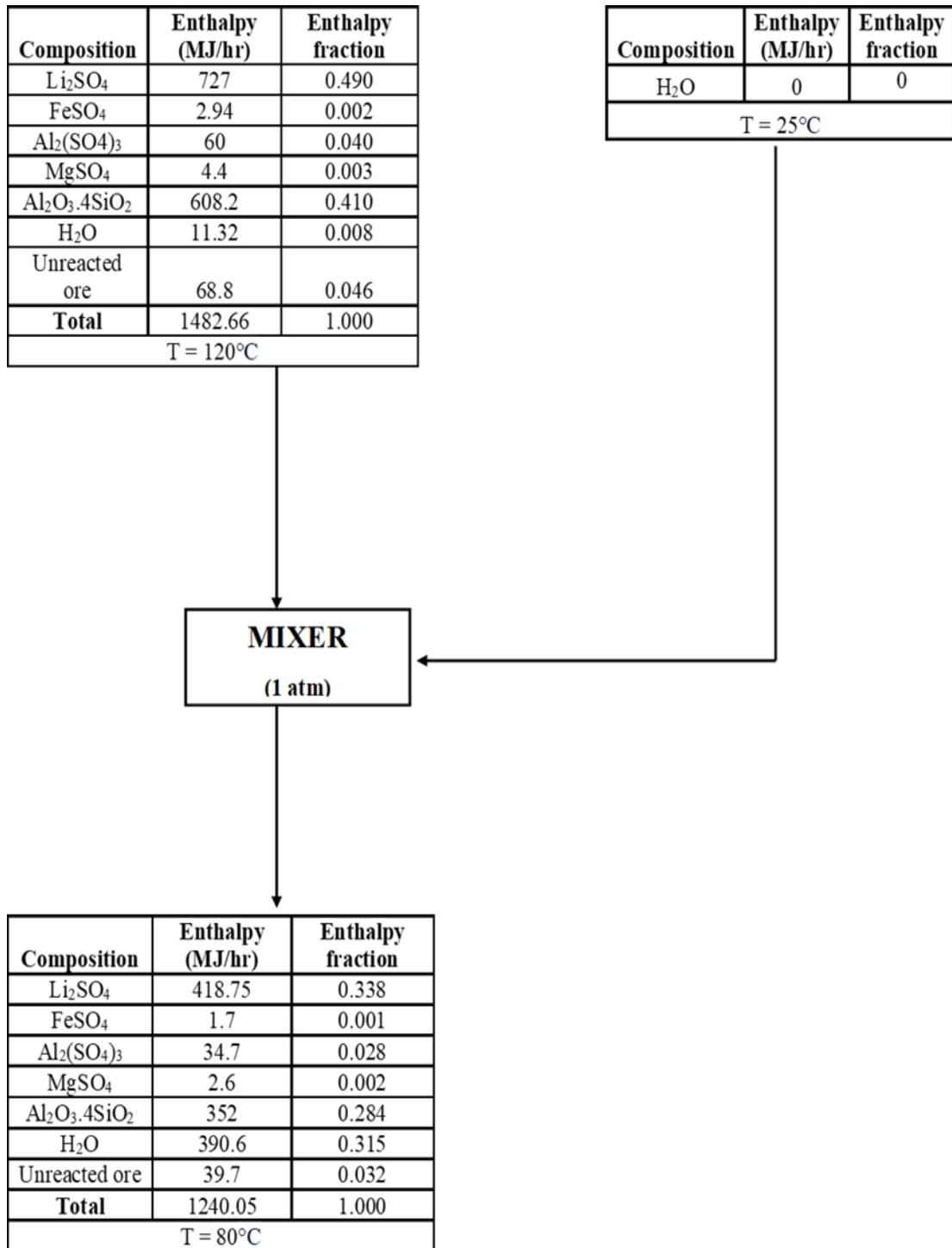
<b>Composition</b>	<b>Enthalpy (MJ/hr)</b>	<b>Enthalpy fraction</b>
Li <sub>2</sub> SO <sub>4</sub>	1721.93	0.184
FeSO <sub>4</sub>	29.24	0.003
Al <sub>2</sub> (SO <sub>4</sub> ) <sub>3</sub>	596.3	0.064
MgSO <sub>4</sub>	43.7	0.005
Al <sub>2</sub> O <sub>3</sub> .4SiO <sub>2</sub>	6049.9	0.647
H <sub>2</sub> O	222.4	0.024
Unreacted ore	681.96	0.073
<b>Total</b>	<b>9345.43</b>	<b>1.000</b>
T = 250°C		

### AIR COOLER-2

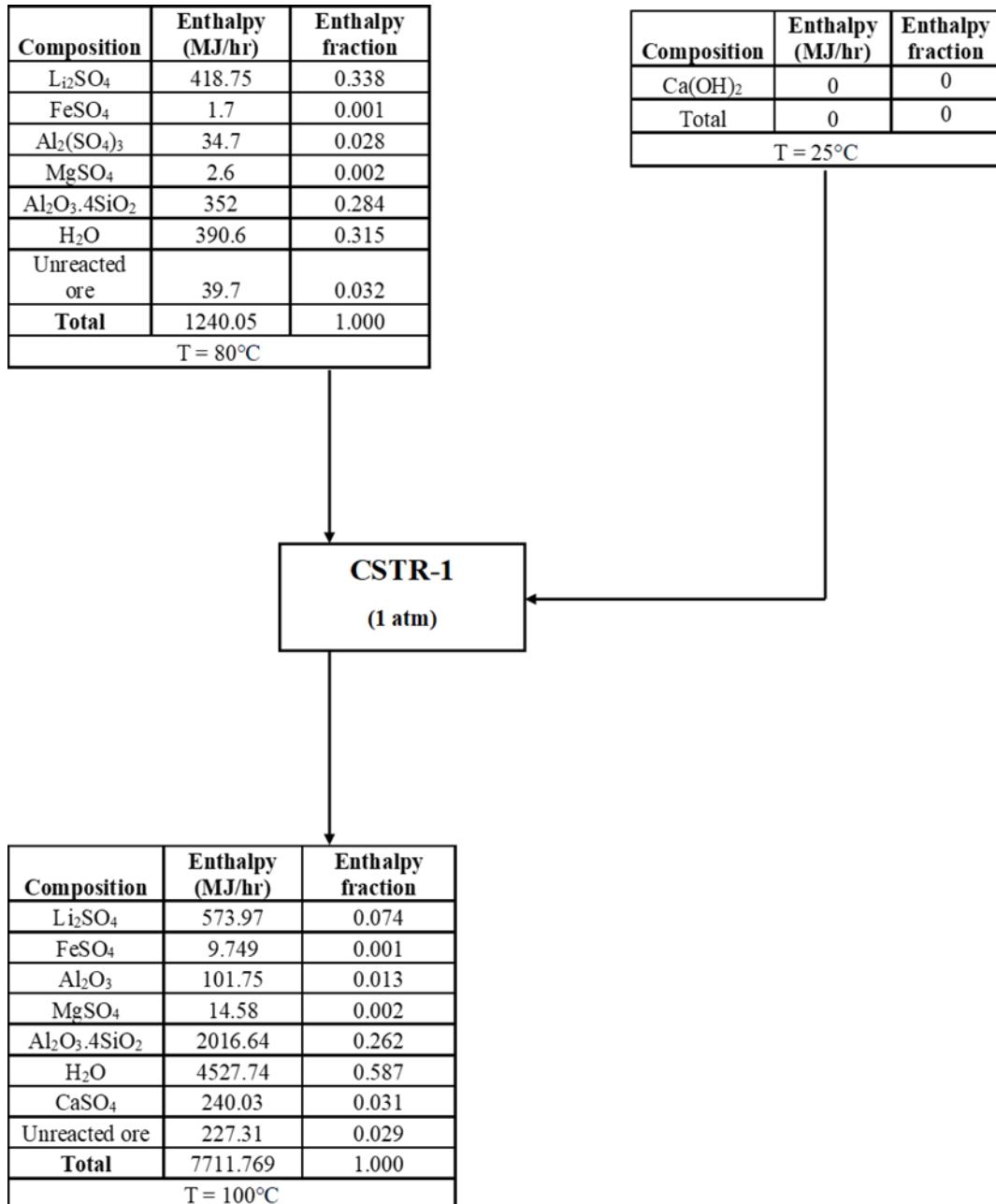
(1 atm)

<b>Composition</b>	<b>Enthalpy (MJ/hr)</b>	<b>Enthalpy fraction</b>
Li <sub>2</sub> SO <sub>4</sub>	727	0.490
FeSO <sub>4</sub>	2.94	0.002
Al <sub>2</sub> (SO <sub>4</sub> ) <sub>3</sub>	60	0.040
MgSO <sub>4</sub>	4.4	0.003
Al <sub>2</sub> O <sub>3</sub> .4SiO <sub>2</sub>	608.2	0.410
H <sub>2</sub> O	11.32	0.008
Unreacted ore	68.8	0.046
<b>Total</b>	<b>1482.66</b>	<b>1.000</b>
T = 120°C		

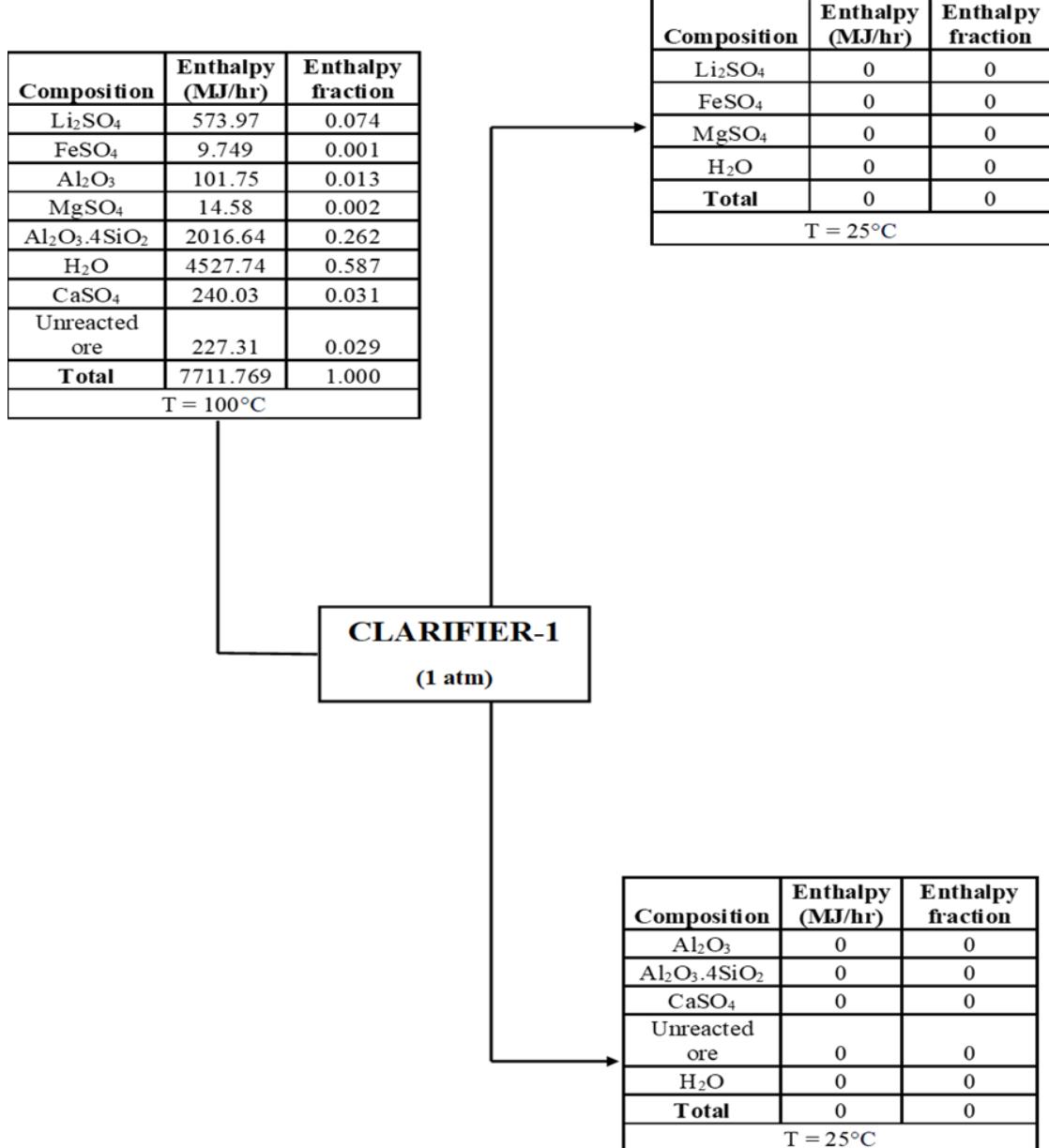
## Energy Balance on Mixer



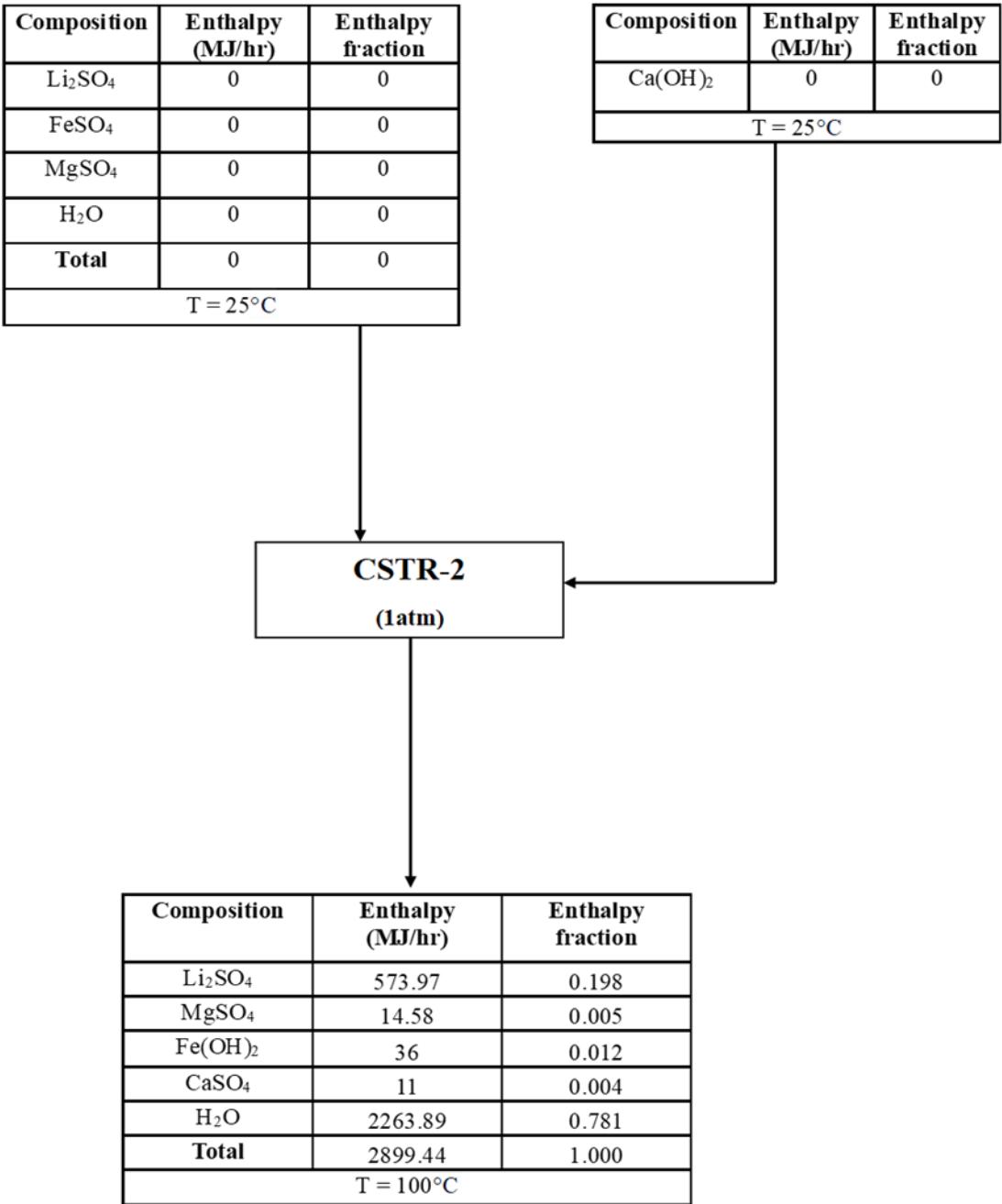
## Energy Balance on CSTR 1



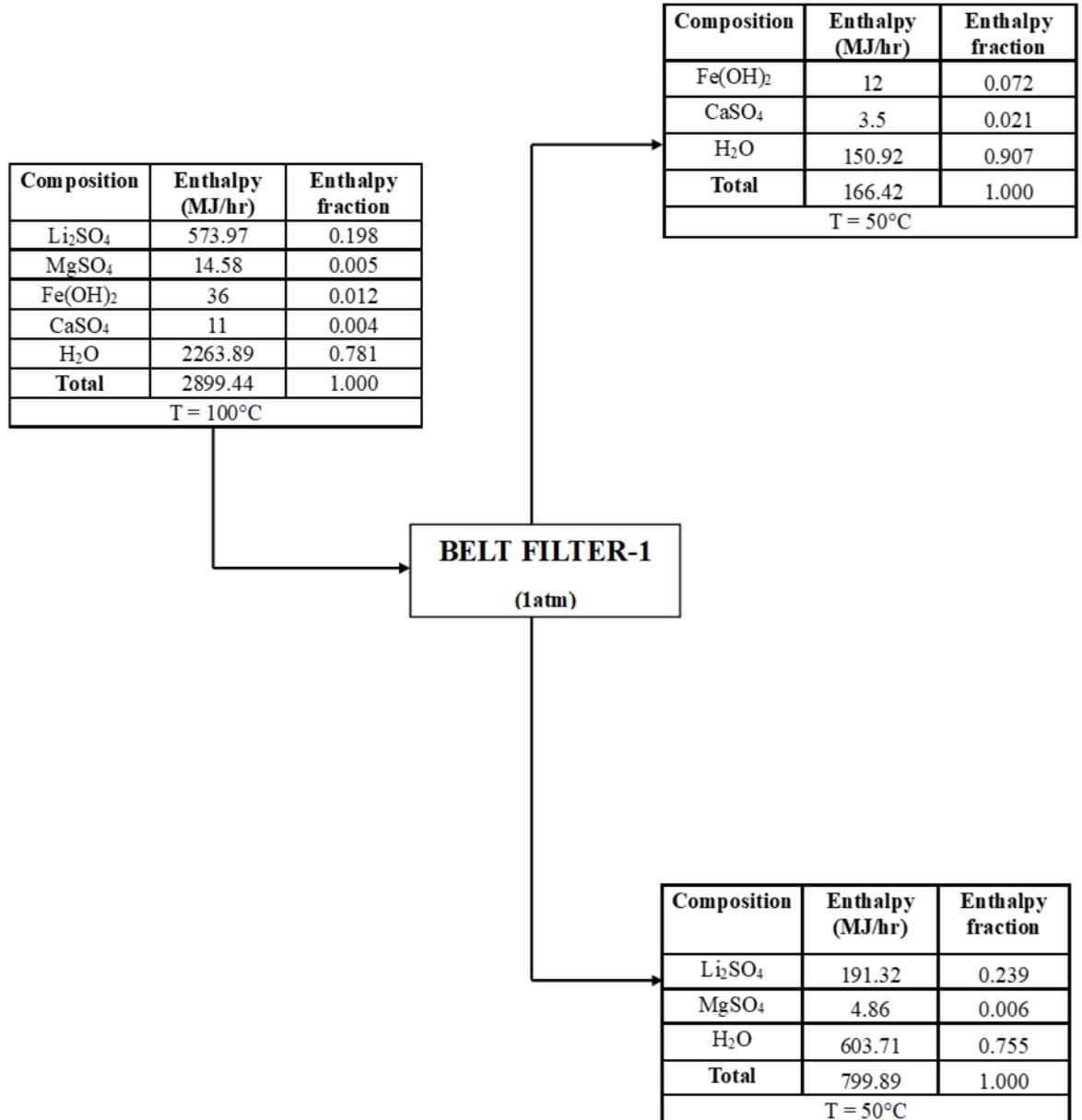
## Energy Balance on Clarifier 1



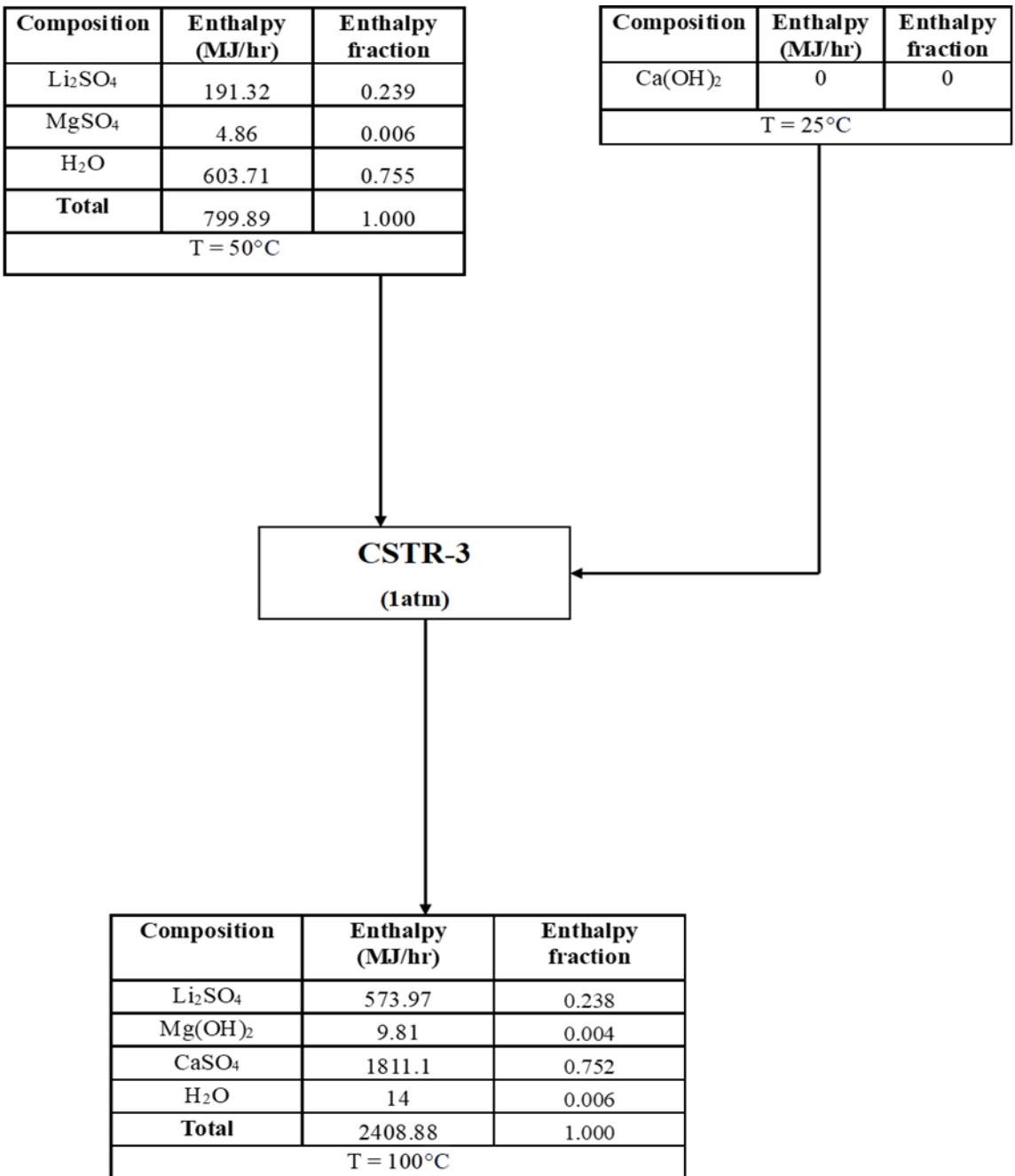
## Energy Balance on CSTR-2



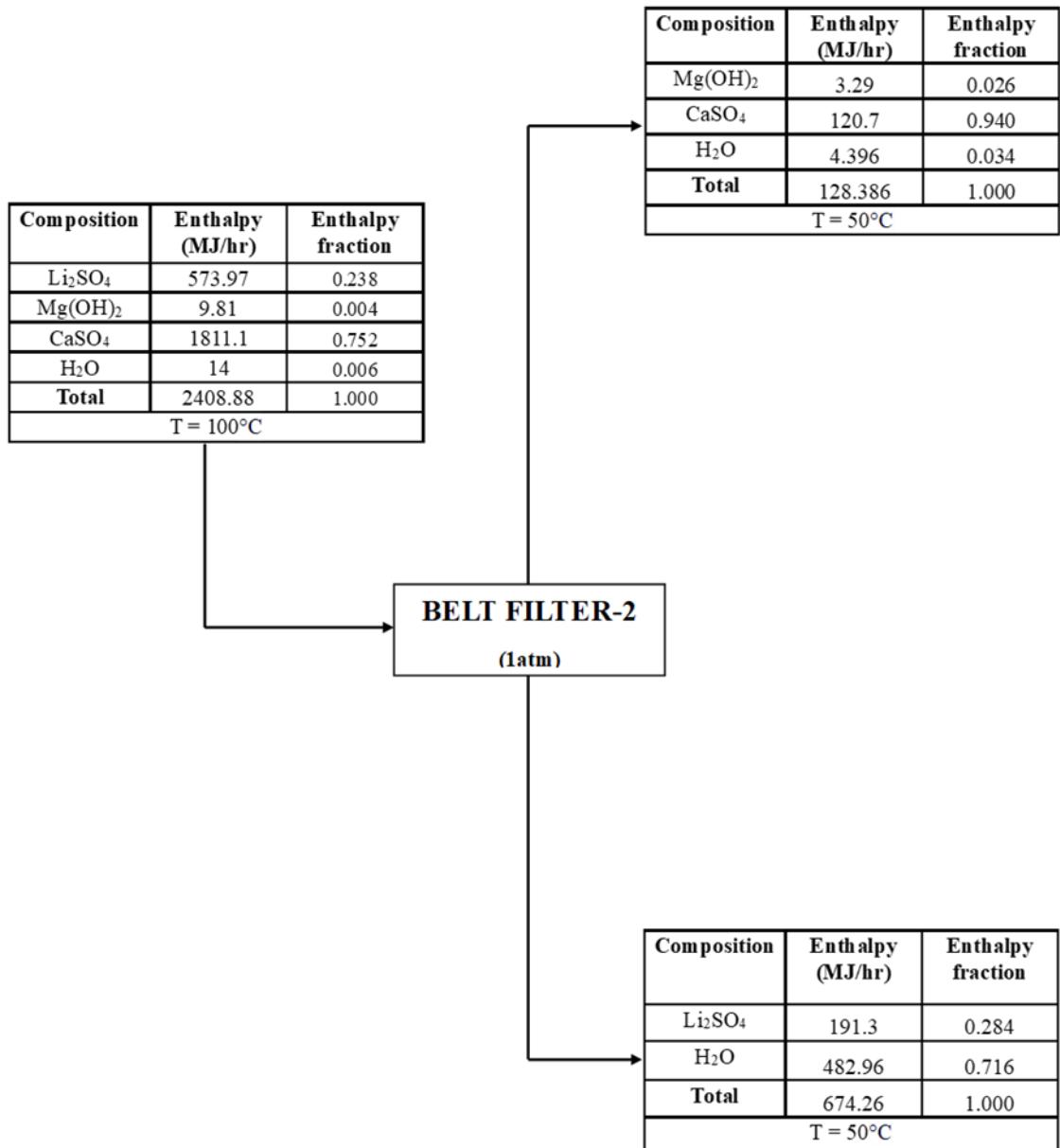
## Energy Balance on Belt Filter 1



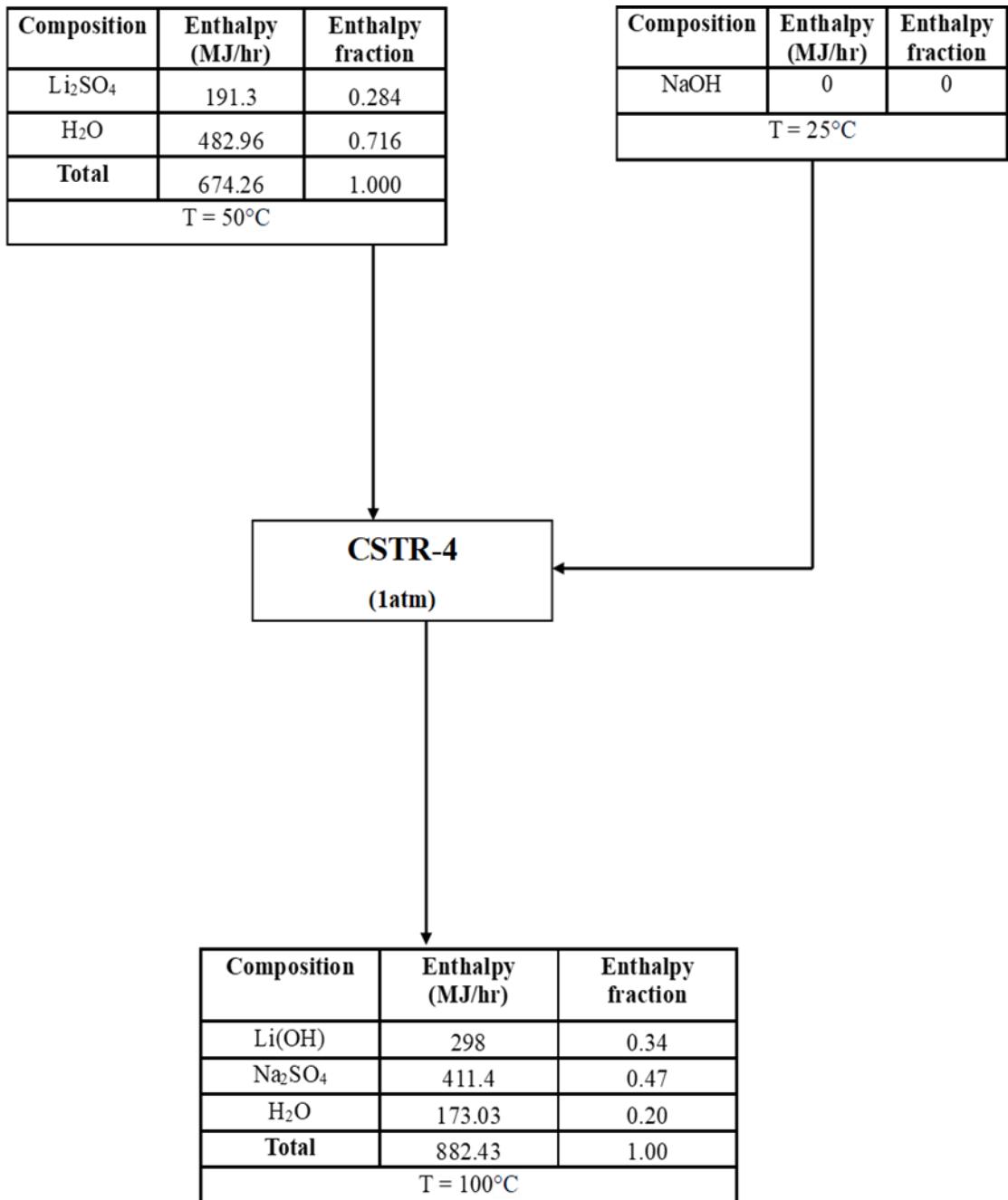
### Energy Balance on CSTR 3



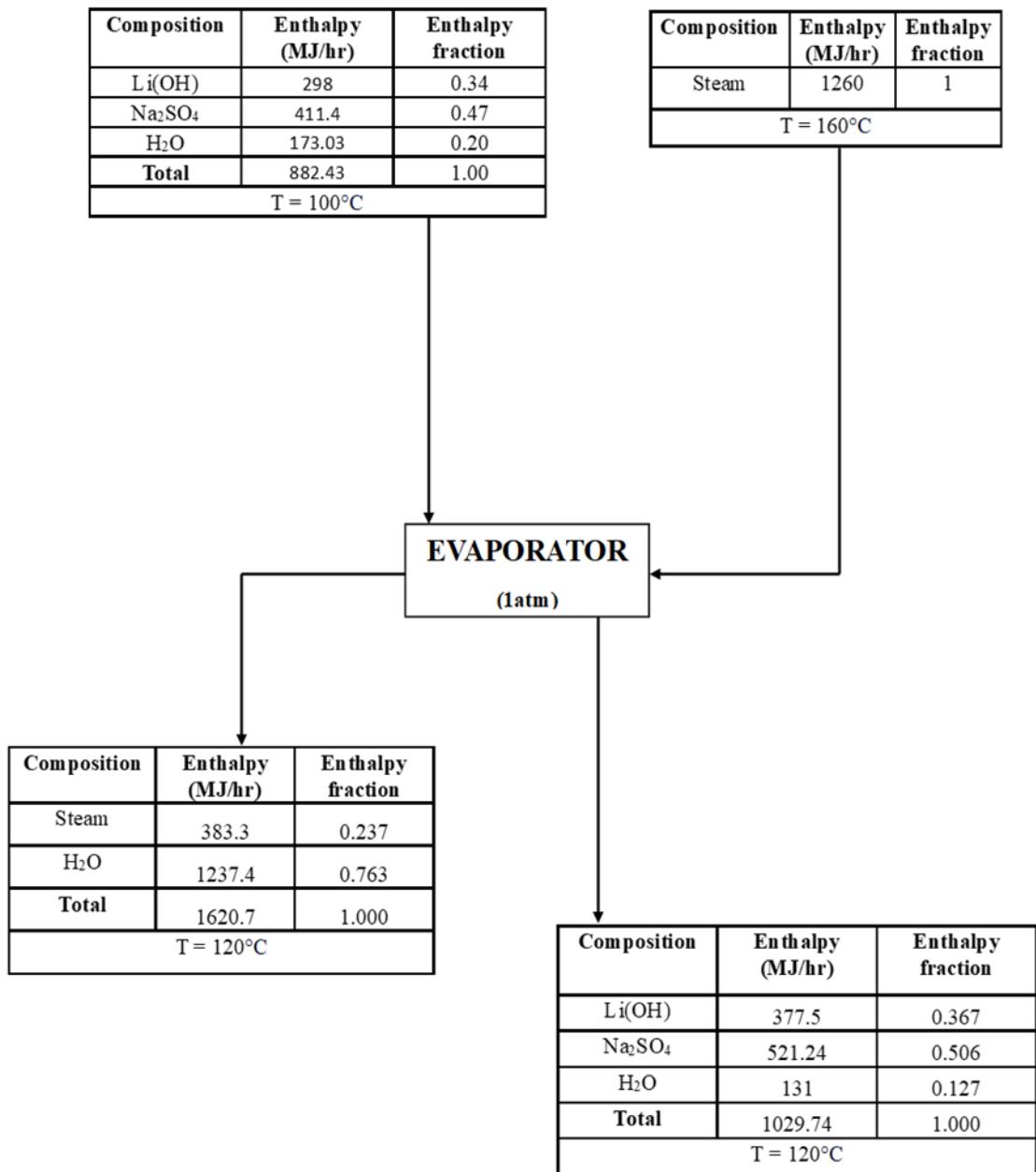
## Energy Balance on Belt Filter 2



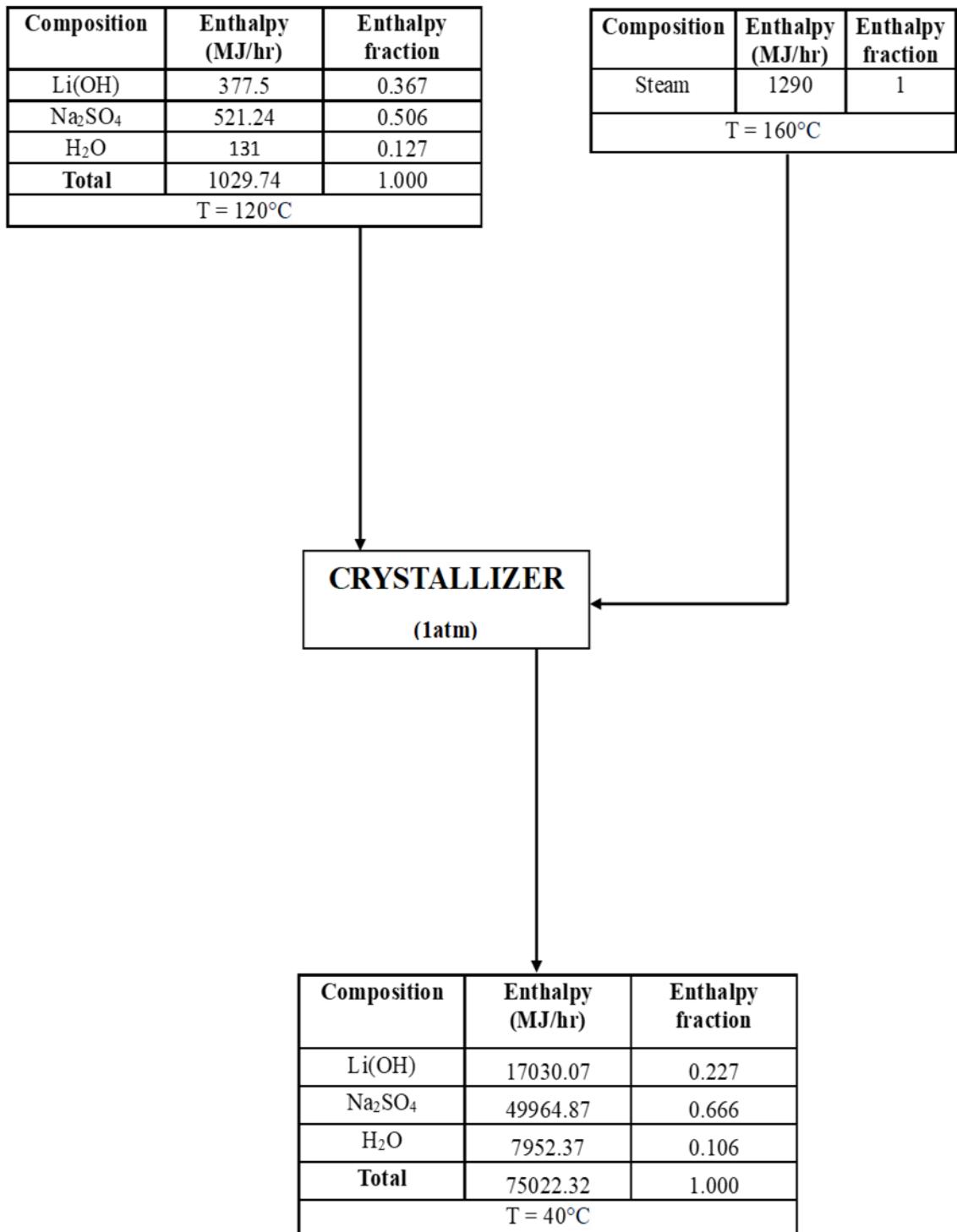
## Energy Balance on CSTR 4



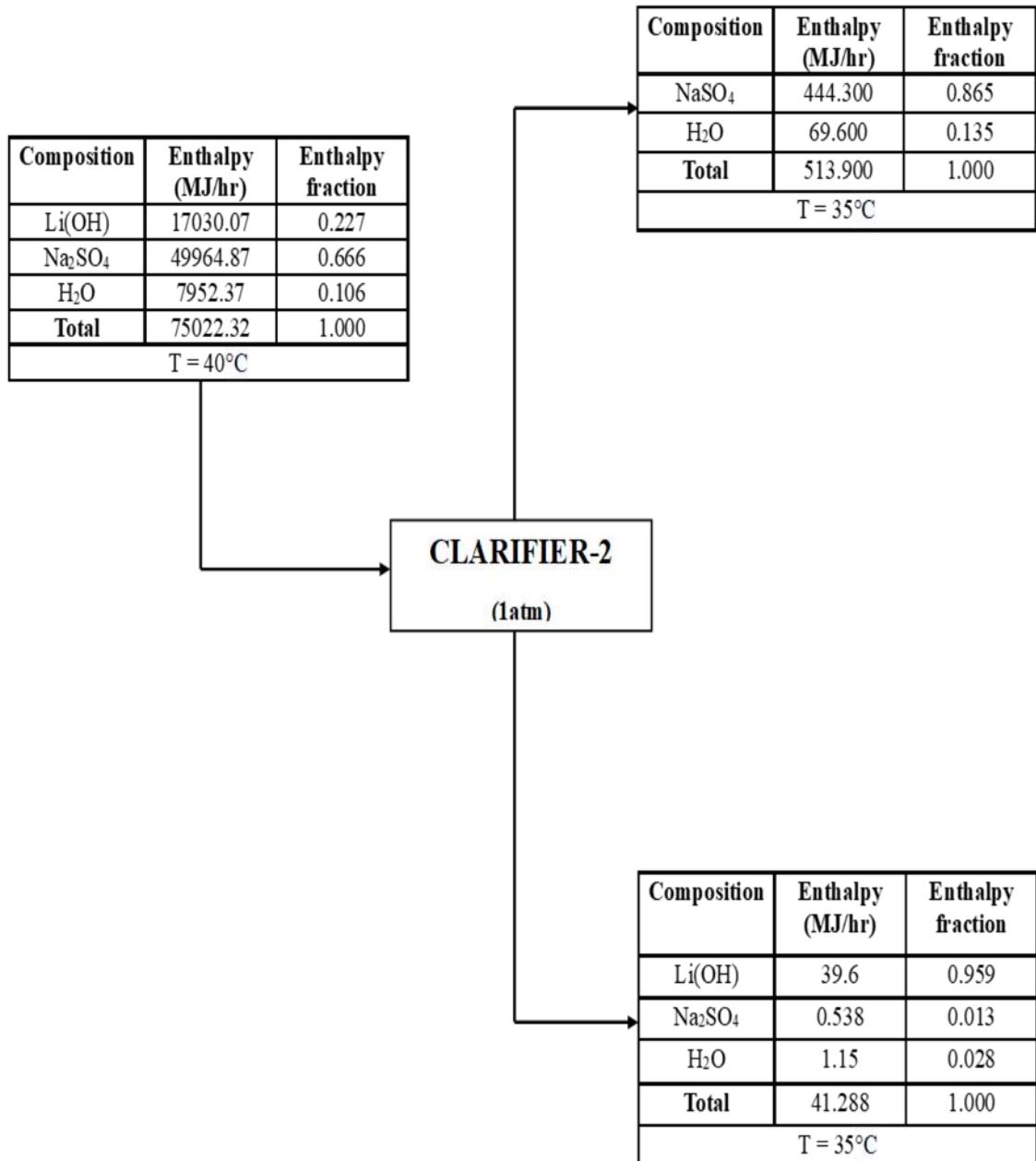
## Energy Balance on an Evaporator



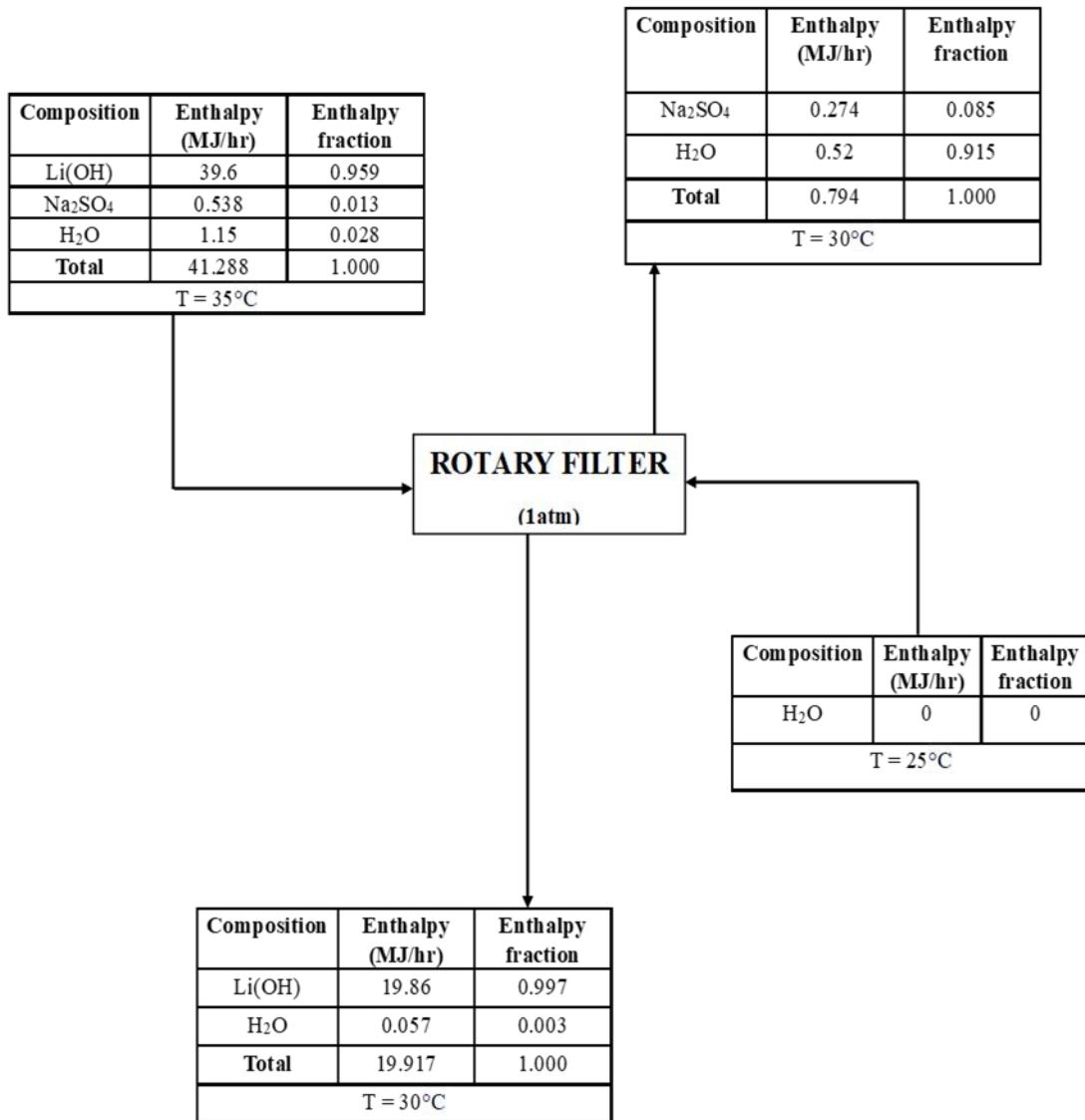
## Energy Balance on a Crystallizer



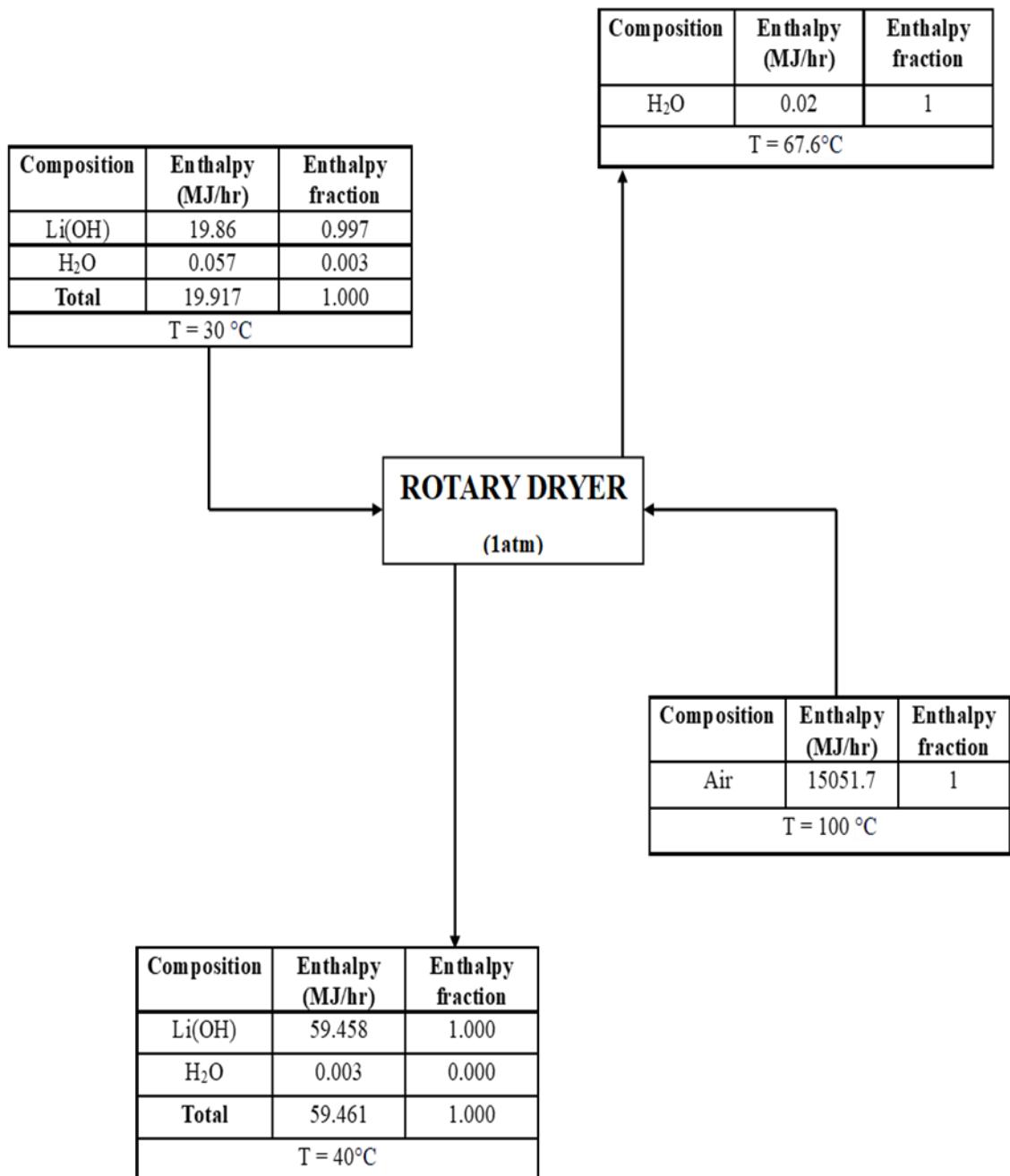
## Energy Balance on Clarifier 2



## Energy Balance on a Rotary Filter



## Energy Balance on a Rotary Dryer



## **CHAPTER FIVE**

### **5.0 EQUIPMENT, PIPELINE AND PUMP SPECIFICATION**

#### **5.1 INTRODUCTION**

In designing a plant, it's important to specify the major and all other ancillary equipment used including pipelines. The purpose of this is to help in estimating the total capital investment required for the project, and secondly, it ensures that the equipment and installation adhere to the applicable regulatory standards. The major considerations to be specified for all the equipment are;

- Function of equipment
- Basic design parameters
- Material of construction
- Power consumption

#### **5.2 PIPELINES SPECIFICATION**

Nominal diameter designates standard pipes which are made in a discrete number of sizes. Depending on the size, the same outside diameter is made with up to fourteen different wall thicknesses. They are identified by schedule numbers, of which the most common are Schedule 40, 80 and 160. The preferred schedule number for pipe specification is 40S.

Pipelines have been built to transport many other fluids (liquids and gases). Pipelines use pumps, valves, and other controls to regulate fluid and slurry flow. Pipes used in process industries might be made of wood, fiberglass, glass, or steel, aluminum, plastic, copper, concrete or others.

In design, the major parameters usually used to specify pipes are;

- Nominal pipe size: gives an indication of the internal diameter

- Schedule number: gives an indication of the thickness of the pipe wall

As defined in Sinnott (2005), the optimum diameter is given by the equations below:

$$\text{For carbon steel, } d_{\text{opt}} = 293 \times \dot{m}^{0.53} \times \rho^{-0.37}$$

$$\text{For stainless steel, } d_{\text{opt}} = 260 \times \dot{m}^{0.52} \times \rho^{-0.37}$$

where  $\dot{m}$  is mass flow rate in kg/s

$\rho$  is average density of the stream in kg/m<sup>3</sup>

$d_{\text{opt}}$  is the optimum diameter in mm

**Table 6: Pipeline Specification**

Description	Content	$\dot{m}$ (kg/s)	$d_{\text{opt}}$ , inch	Flow Area, inch <sup>2</sup> , SS	Schd. Nb	NPS	Wall thickness	OD
Between mixer and CSTR-1	Slurry	15.76	3.00	7.07	40	3	0.216	3.5
Between CSTR-1 and Clarifier-1	Slurry+ Lime	16.38	3.05	7.32	40	3	0.216	3.5
Between Clarifier-1 and CSTR-2	Impure sulfate solution	5.65	1.77	2.46	40	2.5	0.55	2.87

Between CSTR-2 and Filter belt-1	Impure Lithium sulphate solution	5.68	1.78	2.49	40	2.5	0.55	2.87
Between Filter belt-1 and CSTR-3	Impure Lithium sulphate solution	4.73	1.60	2.02	40	1.5	0.145	1.9
Between CSTR-3 and Filter Belt-2	Impure Lithium sulphate solution	4.77	1.48	1.71	40	2	0.43	2.37
Between Filter Belt-2 and CSTR-4	Lithium sulphate solution	3.98	1.46	1.66	40	2	0.43	2.37
Between Clarifier-2 to Rotary Filter	Lithium Hydroxide solution	0.55	0.52	0.21	80	0.5	0.147	0.84
Sulphuric acid storage tank to rotary kiln 2	Sulphuric acid	1.88	0.88	0.61	40	0.75	0.133	1.05
Between Water	Water	8.33	2.75	4.50	40	2.5	0.083	2.875

storage tank and mixer								
Between water storage tank and Rotary filter	Water	0.17	0.31	0.08	80	0.5	0.147	0.84
Between Lime storage tank and CSTR-1	Lime (Calcium hydroxide)	0.61	0.45	0.16	40	1	0.33	1.315
Between Lime storage tank and CSTR-2	Lime	0.03	0.09	0.01	40	0.5	0.294	0.84
Between Lime storage tank and CSTR-3	Lime	0.04	0.10	0.01	40	0.5	0.294	0.84
Between NaOH storage tank and CSTR-4	Sodium Hydroxide solution	0.87	0.56	0.25	80	0.5	0.147	0.84

Between Boiler and Evaporator	Steam	1.39	0.94	0.70	80	1	0.179	1.315
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### 5.3 PUMP

**Table 7: Pump Specification**

Pump ID	Type	Substance handled	Capacity ( $m^3/h$ )	Efficiency (η) %	Power (kW)	Material of Construction
P-1	Centrifugal	Sulphuric Acid	6752.269	0.75	3617.20	Carbon Steel
P-2	Centrifugal	Water	508.41	0.75	496.11	Carbon Steel
P-3	Positive Displacement	Product from mixer	56752.269	0.75	32958.96	Carbon Steel
P-4	Positive Displacement	Product from CSTR	58958.29	0.75	33201.10	Carbon Steel
P-5	Positive Displacement	Product from clarifier	38606.17	0.75	20785.95	Carbon Steel
P-6	Centrifugal	Lime	104.12	0.75	43.74	Carbon Steel
P-7	Positive	Product from	17033.8	0.75	12462.96	Carbon Steel

	Displacement	BF-1				
P-8	Centrifugal	Lime	128.39	0.75	53.94	Carbon steel
P-9	Positive Displacement	Product from BF-2	14341.23	0.75	10315.93	Carbon Steel
P-10	Positive Displacement	Product from CSTR-4	17471.14	0.75	12762.09	Carbon Steel
P-11	Positive Displacement	Sodium Hydroxide	3129.9	0.75	1463.31	Carbon Steel
P-12	Positive Displacement	Product from CR	1983.2	0.75	1237.36	Carbon Steel
P-13	Positive Displacement	Product from Rotary Filter(RF)	1918.96	0.75	1231.42	Carbon Steel

## 5.4 COMPRESSORS

When specifying compressors, three primary parameters are typically considered:

- Flowrate: This parameter quantifies the volume of fluid delivered by a compressor.
- Pressure: It indicates the level of pressure exerted on the fluid.
- Power: This parameter represents the energy generated by the motor to drive the compressor.

**Table 8: Compressor Specifications**

Compressor ID	Efficiency ( $\eta$ ), %	Head (m)	Capacity (Q), $m^3/s$	Useful Power (kW)	Power Consumption (kW)	Type	Material of construction
CP-1	78	2.5	6.49	12.9	30.62	Centrifugal pump, single stage radial flow	Carbon steel

## 5.6 SCREENS

**Table 9: Specification of Screens**

Screens					
Screen ID	Function	Mass(kg/h)	Oversize (microns)	Undersize (microns)	Material
SC-1	Milled spodumene to rotary kiln 1	20000	+76	-76	Manganese steel
SC-2	Milled spodumene to rotary kiln 2	20000	+38	-38	Manganese steel

## 5.7 STORAGE TANKS

**Table 10:** Specification of Storage tanks

Tank ID	Tank content	Capacity, (m <sup>3</sup> /h)	Diameter,m	Height, m
TT-2	Sulfuric Acid	6752.26	3.31	9.93
TT-3	Water	30000	6.61	19.82
TT-8	Slurry from Clarifier 1	38606.2	6.40	19.19
TT-4	Calcium Hydroxide	2438.5	2.18	6.54
TT-5	Slurry from Belt Filter 1	3422.47	3.12	9.36
TT-6	Slurry from Belt Filter 2	2820.9	2.66	7.98
TT-7	Sodium Hydroxide	3129.9	2.45	7.36
TT-9	Sodium Sulphate	561.17	1.29	3.86
TT-10	Lithium Hydroxide	1918.2	2.09	6.26

## 5.8 CONVEYOR BELT

**Table 11:** Specification of conveyor belt

Conve-yer Belt ID	Function	Capacity kg/h	Diame-ter, m	Speed, rpm	Length, m	Material
CB-1	Convey crushed spodumene from crusher	20000	2.6	50	7	Carbon steel

CB-2	Convey crushed alpha spodumene to rotary kiln 1	20000	2.6	50	7	Carbon steel
CB-3	Conveys beta spodumene to ball mill	20000	2.6	50	7	Carbon Steel
CB-4	Conveys milled beta spodumene to rotary kiln 2 for acid roasting	20000	2.6	50	7	Carbon Steel
CB-5	Conveys calcined beta spodumene to leach mixer	16770	2.2	65	9	Stainless Steel
CB-6	Conveys materials from rotary dryer	1918	1.8	63	5	Stainless Steel

## CHAPTER SIX

### 6.0 INDIVIDUAL DESIGN OF A CONTINUOUS STIRRED TANK REACTOR

#### 6.1 INTRODUCTION

Chemical reactor design and optimization are critical in a variety of industrial operations. Many reactions are too slow to be economically feasible, regardless of how much we want the products. Over the years, catalysts have been developed and added to such processes so that they can occur at commercially feasible rates. One such process is the neutralization of lithium sulfate and sodium hydroxide, which combines two reactants to produce the desired product. It is essential to select and construct a suitable reactor system to ensure effective neutralization.

A catalyst is a substance that accelerates a specific reaction towards its equilibrium but remains unchanged after the reaction achieves equilibrium. Catalytic reactions and reactors have widespread applications in the production of chemicals in the processing industries. There are a wide range of catalytic reactors available. Depending on the type of reaction occurring, a suitable reactor (and catalyst) is designed to carry out the reaction.

The CSTR is an effective kind of reactor that allows for continuous mixing of the reactants while maintaining a homogenous mixture throughout the reaction. This design seeks to achieve optimal reactant conversion while taking essential parameters such as reaction kinetics, heat transfer, and safety into account. The neutralization of lithium sulphate and sodium hydroxide is of particular importance due to the increasing demand for lithium-based products in various industries, including energy storage and electronics. Lithium sulphate, a by-product of lithium extraction processes, needs to be neutralized to minimize environmental impact and facilitate the recovery of valuable resources.

## **Problem Statement**

It is required to design a continuous stirred tank reactor suitable for producing 1918 kg/h of lithium hydroxide at 100 °C and 1 atm. The reaction produces lithium hydroxide by synthesizing lithium sulfate and sodium hydroxide. The overall conversion is to be set at 97%.

## **Main Objective**

To design a reactor suitable for the neutralization of Li<sub>2</sub>SO<sub>4</sub> and NaOH at 100 °C and 1 atm.

## **Specific Objectives**

- To justify the choice of reactor type to be designed.
- To perform chemical engineering design calculations on the selected equipment.
- To perform a detailed mechanical engineering design calculations on the selected equipment.
- To perform material and energy balance on the reactor

## **6.2 LITERATURE REVIEW**

A reactor is an equipment in which chemical reactions are carried out. Process engineers design reactors to maximize the net positive value for a given reaction. They ensure that the reaction proceeds with the highest efficiency (conversion) towards the desired product, while requiring the least capital and operational cost. Different reactor types are utilized depending on the specific reaction requirements, desired product, and operating conditions. Selecting the appropriate reactor type involves considering factors such as reaction requirements, safety, scale, product quality, process efficiency, economics, scalability, flexibility, and regulatory compliance. Process efficiency is optimized through proper heat and mass transfer, minimizing energy consumption, and maximizing

yield. Economic viability and cost-effectiveness are crucial considerations. The reactor type should be scalable, flexible, and compliant with regulations. The overall goal is to achieve optimal performance, sustainability, and successful implementation of chemical processes.

## **CLASSIFICATIONS OF CHEMICAL REACTIONS**

Chemical reactions are classified on different basis. They include:

### **1. ABSORPTION OR RELEASE OF HEAT**

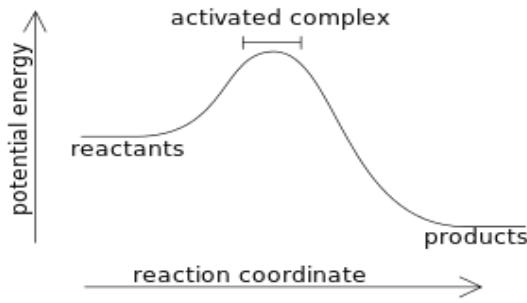
In chemical reactions, heat may be released or absorbed. These are exothermic and endothermic reactions respectively.

- **Exothermic Reaction:** An exothermic reaction is a reaction for which the overall standard enthalpy change  $\Delta H^\circ$  is negative (Laidler, 1996). Exothermic reactions usually release heat and entail the replacement of weak bonds with stronger ones (Galley, 2004). In an exothermic reaction, energy in the form of heat is released to the surroundings.

In an exothermic reaction, by definition, the enthalpy change has a negative value:

$$\Delta H = H_{\text{products}} - H_{\text{reactants}} < 0$$

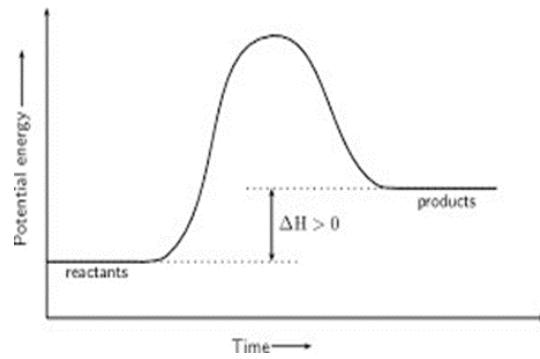
where a larger value (the higher energy of the reactants) is subtracted from a smaller value (the lower energy of the products).



**Figure 21:** An energy profile of an exothermic reaction

- **Endothermic reactions:** Endothermic reactions are reactions that require external energy, usually in the form of heat, for the reaction to proceed. Since endothermic reactions draw in heat from their surroundings, they tend to cause their environments to cool down (Oxtoby et al, 2015). They are also generally non-spontaneous, since endothermic reactions yield products that are higher in energy than the reactants. As such, the change in enthalpy for an endothermic reaction is always positive. In an endothermic reaction, by definition, the enthalpy change has a negative value:

$$\Delta H = H_{\text{products}} - H_{\text{reactants}} > 0$$



**Figure 22:** An energy profile of an endothermic

Whether a reaction is endothermic or exothermic depends on the direction that it is going; some reactions are reversible, and when you revert the products back to reactants, the change in enthalpy is opposite.

## THE GENERAL MOLE BALANCE EQUATION

$$\text{Accumulation} = \text{In} - \text{Out} + \text{Generation} - \text{Consumption}$$

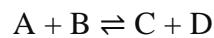
$$[\text{accumulation of species } j] = [\text{flow of species } j \text{ in}] - [\text{flow of species } j \text{ out}] + [\text{generation of species } j]$$

From this general mole balance equation, we can develop the design equations for the various types of industrial reactors: batch, semi-batch, and continuous-flow. Upon evaluation of these equations, we can determine the time (batch) or reactor volume (continuous-flow) necessary to convert a specified amount of the reactants to products (Fogler, 2020).

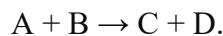
## REVERSIBILITY

A reaction may also be reversible or irreversible.

- **Reversible reactions:** In a reversible reaction, as products are formed, these same products can combine to form the reactants again and hence technically, there is no end to such a reaction. Equilibrium is however attained when the concentration of products and reactants no longer change with time for the given set of conditions. Generally, reversible reactions are represented by:



- **Irreversible reactions:** An irreversible reaction comes to completion when all the limiting reactant is exhausted in an ideal scenario. Generally, irreversible reactions are given by:



## RATE OF CHEMICAL REACTIONS

The rate of reaction is the speed at which a chemical reaction takes place, defined as proportional to the increase in the concentration of a product per unit time and to the decrease in the concentration of a reactant per unit time (McMurry et al, 2014). Reaction rates can vary dramatically depending on the type and conditions of the reaction. For most reactions, the rate decreases as the reaction proceeds. A reaction's rate can be determined by measuring the changes in concentration over time. Mathematically, the rate of reaction is expressed as;

$$r = \frac{dc}{dt}, \text{ mol/m}^3.\text{s}$$

Where r represents the rate of reaction and C is the concentration of reactants or reactant.

### Factors that Affect the Rate of a Chemical Reaction.

There are four main factors that can affect the reaction rate of a chemical reaction:

- 1. Reactant concentration.** Increasing the concentration of one or more reactants will often increase the rate of reaction. This occurs because a higher concentration of a reactant will lead to more collisions of that reactant in a specific time period.
- 2. Physical state of the reactants and surface area:** If reactant molecules exist in different phases, as in a heterogeneous mixture, the rate of reaction will be limited by the surface area of the phases that are in contact. For example, if a solid metal reactant and gas reactant are mixed, only the molecules present on the surface of the metal are able to collide with the gas molecules. Therefore, increasing the surface area of the metal by pounding it flat or cutting it into many pieces will increase its reaction rate.
- 3. Temperature:** An increase in temperature typically increases the rate of reaction. An

increase in temperature will raise the average kinetic energy of the reactant molecules.

Therefore, a greater proportion of molecules will have the minimum energy necessary for an effective collision.

**4. Presence of a catalyst:** A catalyst is a substance that accelerates a reaction by participating in it without being consumed. Catalysts provide an alternate reaction pathway to obtain products, and hence affects the rate of the reaction.

## **CLASSIFICATION OF REACTORS**

Different kinds of reactors can be obtained with different basis of classification. The table below summarizes the various classification of reactors and the basis of classification.

Classifications of Reactors. (Source: Sinnott, 2005)

**Table 12: Classification of reactors**

<b>Basis of classification</b>	<b>Type</b>
Mode of operation	Batch; continuous; semi-batch reactors
Reactor geometry	Stirred tank reactor; plug flow reactor; fixed bed reactor; fluidized bed reactor.

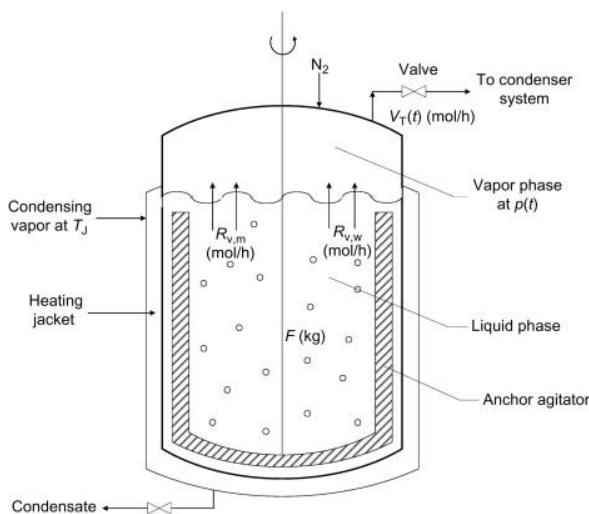
## **BASED ON MODE OF OPERATION**

- **Batch Reactor**

In a batch reactor, all reactants, catalysts and diluents necessary for the reaction are charged into the reactor. The necessary operating conditions are set and time is allowed for the reaction to occur to the desired conversion. When this time elapses, the product is removed and the reactor is prepared for the next batch. The time involved in preparing

the reactor, the reaction itself and withdrawing of product constitute the total batch time for a particular batch reactor.

This type of reactor is good for small-scale processes, experiments, seasonal products and situations where a varying range of products are to be obtained. It however, has the disadvantage of low conversion, high production cost, more difficulty in design and control and labour intensive in most cases. Food, pharmaceutical and some polymer industries use the batch reactor type for their processes.



A typical batch reactor

A typical batch reactor consists of a tank with an agitator (for stirring and mixing) and integral heating/cooling system. They are usually fabricated in steel, stainless steel, glass-lined steel, glass or exotic alloy. Reactants and catalysts, as well as diluents are usually charged via connections in the top cover of the reactor. Vapours and gases also discharge through connections in the top. Liquids are usually discharged out of the bottom. The main distinguishing factor is that no material is allowed to enter or move out once the reaction commences. The equation below is used to model batch reactors.

$$\frac{dN_i}{dt} = \int_O^V r_i dV$$

Where  $N$  is the number of moles of specie  $i$ ,  $t$  is time and  $V$  is the reactor volume  $r$  is the rate of reaction. This equation can be rewritten in the integral form and solved for the batch time and reactor volume. In most cases, the other variables are known.

- **Semi-Batch Reactors**

The semi-batch reactor is a hybrid reactor configuration that combines the features of both batch and continuous reactors. It operates with a varying volume of reactants, where one reactant is fed continuously while the other is added incrementally or intermittently (Varma and Mondal, 2017). This unique design offers advantages and disadvantages, making it suitable for specific industrial application.

- **Continuous Reactors**

For these types of reactors, raw materials are fed and products are withdrawn continuously from the reactor as the reaction is taking place. It is usually suited for very high production rates and reactions that do not result in a wide range of products. These types of reactors save cost. Occasionally, there is shut down for maintenance work to be done. Continuous reactors are further classified based on their geometry.

## **BASED ON REACTOR GEOMETRY**

- **Stirred Tank Reactors (STRs)**

STRs are usually used to carry out liquid phase reactions both in industries and in laboratories. It may also be used to carry out gaseous phase reactions with solid catalyst batch-wise or in laboratories for the purposes of studies. It may be used singly or in multiples for a desired conversion to be achieved. When more than one STR is used, it might be arranged in series or in parallel. The reaction occurring in an STR is assumed to be perfectly mixed so that the concentration in the reactor vessel at any time is the same

as the concentration of the products flowing out the reactor. The design equation for continuous STRs is

$$V = \frac{F_{AO} X}{-r_A}$$

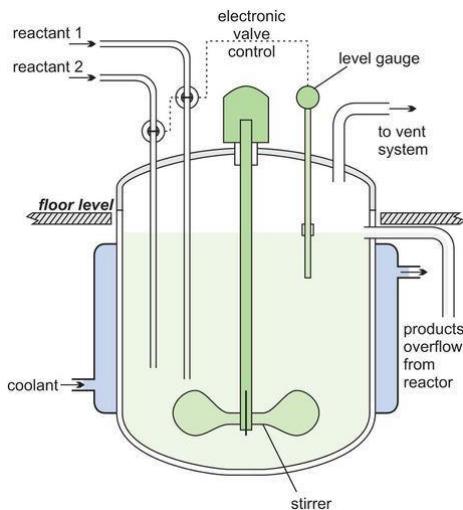
Where  $V$  is the reactor volume,  $F_{AO}$  is the flow of component A into the reactor vessel,  $X$  is the fractional conversion,  $-r_A$  is the rate of the reaction.

In as much as the reactor usually is operated continuously, the content is retained for a period of time to enable the desired conversion be attained. This amount of time required, called the residence time is calculated as  $\tau = V/q$

Where  $\tau$  is the residence time;  $V$  is the reactor volume;  $q$  is the volumetric flow rate of material. For steady state operations, where the conditions of the system do not vary with time,  $q$  is same for both the inlet feed and outlet product.

The CSTR can be used to carry out both endothermic and exothermic reactions. In such cases, heat transfer may be achieved in two main ways.

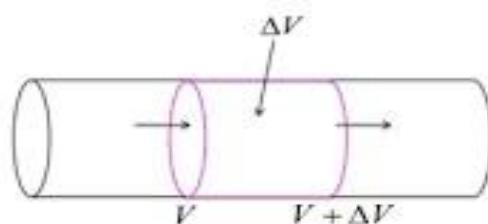
- The coolant or heating medium is passed through a jacket surrounding the reactor vessel. This configuration is mostly used when the amount of heat to be transferred is not tremendous.
- The coolant or heating medium is passed through coils that are embedded in the walls of the reactor vessel. This is a more efficient method of heat removal and it is employed when the amount of heat to be transferred is large.



**Figure 23: A typical CSTR**

- **PLUG FLOW REACTORS (PFRs)**

A PFR is usually tubular and therefore it is also called Tubular reactor. The reactants are assumed to flow in a plug so that there is no mixing along the entire length of the reactor. However, there is complete mixing at a given cross-section. As the reactants move along the length of the reactor, they react so that conversion increases from one cross-section to another. Due to this, the concentration of reactants and products varies constantly along the reactor length. This is unlike the CSTR. The PFR is also operated continuously most of the time.



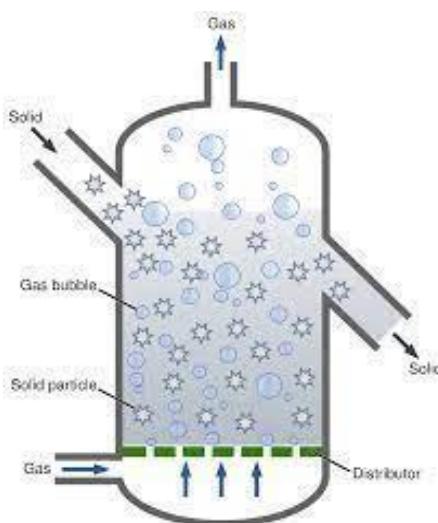
**Figure 24:Diagram of a typical PFR**

- **FLUIDIZED BED REACTORS**

This type of reactor is employed for solid-fluid reactions. Most of the time, the solid is the catalyst.

A reactor vessel is filled with this solid. A fluid is then ducted into the vessel through the bottom. This fluid, moving at a given velocity, affects the movement of the solid particles.

At low velocity values, the solids might not even move at all. However, when this velocity is increased, it gets to an instance where the solid particles begin to move. The higher the velocity, the more vigorous the movement of the solid particles. At appreciable velocities, the solid particles are suspended in the fluid as though they are also fluid. The particles are said to be fluidized.



**Figure 25: Fluidized bed reactor**

- **FIXED BED CATALYTIC REACTORS**

Fixed bed reactors (also known as Packed Bed reactors) are heterogeneous reactors that are used to carry out reactions that usually require solid catalyst. The solid catalyst is mostly present as a bed of relatively small grains or pellets, randomly arranged and fixed in position. The fluid then moves by convection through the void present in the bed of particles and the reaction occurs alongside. Practically, there is transport of the particles but to an insignificant degree. Most reactors commonly use transitional-metal-containing solid-supported oxide catalysts. Most catalysts can be regenerated after it is spent and the

method of regeneration depends on the type of catalyst and the process it is being used for. Typically, catalysts have a lifespan of about 5 years.

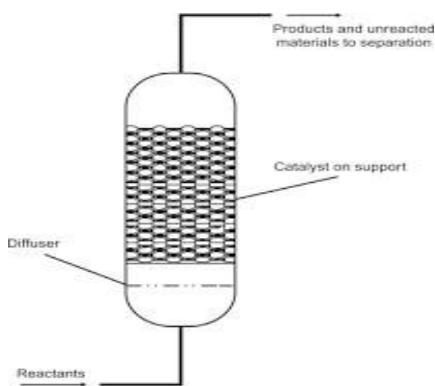
These reactors can be operated adiabatically, non-adiabatically or isothermally.

### i. Adiabatic Fixed Bed Reactors

In adiabatic operations, no attempt is made to cool or heat up the system as the reaction occurs, thus adjusting T. Hence for an exothermic reaction, temperature is allowed to increase and for an endothermic reaction, the temperature of the reaction decreases. This however, is usually done for those reactions that do not release or consume significant amount of heat.

In certain situations, the catalyst bed is divided into more than one bed so that there is some form of heat exchange from one bed to another. This may be done for two main reasons

- (a) To shift equilibrium in order to increase yield
- (b) To maintain the rate of reaction thereby decreasing the amount of catalyst and size of vessel



**Figure 26: Adiabatic fixed bed reactors**

### ii. Non-Adiabatic Fixed Bed Reactors

For non-adiabatic operations, there is heat exchange with the main aim of controlling temperature. For this, the shell-and-tube type of fixed bed reactor is essentially employed. The catalyst is either packed in the tube or shell or supported with sieves to allow the flow of fluid product while maintaining the catalyst in position.

### **iii. Isothermal Fixed Bed Reactors**

The isothermal reactor is a fixed bed reactor with indirect heat exchange suitable for endothermic and exothermic catalytic reactions. This reactor provides the benefits of a tube reactor while simultaneously avoiding the heat tension problems of a straight tube reactor. The palpable head of gases and liquids as well as the latent evaporation heat can be used for cooling or heating operations. The heating or cooling tube bundle embedded in the catalyst transfers the reaction heat in such a way that the catalyst can work at an optimum temperature. This results in higher outputs, a longer catalyst lifetime, fewer by-products as well as efficient recovery of the reaction heat and lower reaction costs. The reactor is based on the design of the coil-wound heat exchangers. The reactor is advantageously applicable in

- methanol synthesis
- carbon monoxide shift conversion
- hydrogenation
- ethylene oxide synthesis, etc.

## **7.3 EQUIPMENT SELECTION AND JUSTIFICATION**

In selecting a reactor for a particular process, several factors must be considered which will ultimately influence one's choice of selection. In chemical reaction engineering, different reactors are used based on their efficiency and flow capacity. Since our reaction is an exothermic reaction operating at temperature and pH dependent conditions, careful

analysis is made and a reactor suitable for this process is a continuous stirred tank reactor (CSTR).

Mainly, CSTRs run at steady state with continuous flow of reactants and products; the feed utilizes a homogeneous composition throughout the reactor, and the exit stream has the same composition as the tank. For such reactors, temperature regulation is easy to maintain. CSTR behavior is well established, including mixing (the capacity to handle solids and slurries), reaction calorimetry, dosing choices, chemical kinetics, less costly, easier to construct and allows for steady state conditions. If there is a need for flexibility in production volumes or if the reaction needs to be performed at different scales, a CSTR can offer versatility and adaptability. This enables convenient control of process variables like temperature, pH, and pressure, thereby simplifying the determination of chemical kinetics

Above all, CSTRs are used when there involves series of configurations for different concentration streams. Consequently, the output composition is identical to composition of the material inside the reactor, which is a function of residence time and reaction rate. The CSTR is the ideal limit of complete mixing in reactor design, which is the complete opposite of a plug flow reactor (PFR).

#### **7.4 EQUIPMENT DESCRIPTION.**

The CSTR, also known as a vat or back-mix reactor, is typically used for liquid phase processes.

It is typically operated in a steady state and presumed to be perfectly mixed. A CSTR is equipped with an impeller or stirrer, driven by a motor located at the top, to ensure efficient mixing of reactants while continuously removing the product. This steady state operation facilitates easy control and manipulation of process variables such as temperature and pressure. Baffles are strategically placed within the CSTR to promote

efficient mixing of reactants. Since the reaction is exothermic, a cooling jacket is employed, utilizing process water, to remove excessive heat.

Major components of Continuous Stirred Tank Reactor are;

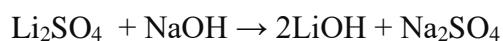
1. Jacket
2. Baffles
3. Impeller
4. Recycle stream outlet
5. Liquid inlet and outlet
6. Reactor supports
7. Drain
8. Liquid distributor
9. Steam coils

### **Mode of Operation of a CSTR**

A Continuous Stirred Tank Reactor (CSTR) operates by continuously feeding reactants into a tank reactor. This reactor is fitted with a stirring mechanism that ensures complete mixing of reactants, which is required for a homogenous reaction mixture. Reactants enter at a constant flow, conduct the appropriate chemical reaction within the reactor, and products are continually extracted to keep the volume consistent. The reaction can be exothermic or endothermic, necessitating precise regulation of temperature, pressure, and flow rates to get optimal results. With continuous mixing, the reactor achieves a steady state in which reactant and product concentrations stay constant throughout time. This mode of operation is appropriate for operations that need consistent reaction conditions.

## 7.5 CHEMICAL ENGINEERING DESIGN

In the design of a reactor, it is very essential to identify the design parameters associated with the engineering principles governing its operation. The basic principle in reactor design is the reaction kinetics of the reaction. The chemical engineering design of a continuous stirred tank reactor involves the material and energy balance on the reactor, design parameters and dimensions of the reactor. For the reactor, the reaction to form lithium hydroxide from lithium sulfate and sodium hydroxide occurs in the following reaction;



### Reactor Sizing

By sizing a chemical reactor, we are either deterring the reactor volume to achieve a given conversion or determine the conversion that can be achieved in a given reactor type and size. In sizing the reactor, the following assumptions were made;

- First-order elementary reaction
- Homogenous reaction
- Steady-state operation
- Perfect mixing
- The rate-determining step is the limiting reactant ( $\text{Li}_2\text{SO}_4$ )

**Table 13: Component Balance on CSTR 4**

Composition	Amount (Kg/h) In	Amount (kg/h) Out	Density (kg/m <sup>3</sup> )	Molecular weight (kg/kmol)	Concentration (Kmol/m <sup>3</sup> )	Enthalpy (MJ/h) In	Enthalpy (MJ/h) Out
-------------	------------------------	-------------------------	---------------------------------	-------------------------------	---	--------------------------	---------------------------

NaOH	3129.906	-	2130	40	53.35	0	-
Li <sub>2</sub> SO <sub>4</sub>	4405.96	-	2220	110	20.18	191.3	-
H <sub>2</sub> O	9935.27	9935.27	998	18	54.89	482.96	173.03
LiOH	-	1918.07	1460	23.96	60.93	-	298
Na <sub>2</sub> SO <sub>4</sub>	-	5617.796	2660	142.04	18.73	-	411.4

## Equipment Parameter Calculations

### 1. Volume of Reactor

The general mole balance for a reactor is,  $F_{j0} - F_j + \int_0^V r_j dV = \frac{dN_j}{dt}$  ;

But for a continuous flow reactor, there is no accumulation and conditions do not change with time (operated at steady state), so therefore;  $\frac{dN_j}{dt} = 0$

Since there are no special variations, that is since perfect mixing is assumed;

$$\int_0^V r_j dV = Vr_j$$

Thus, after series of calculation, the final design equation of a CSTR takes the form;

$$V = \frac{F_{j0} - F_j}{-r_j}$$

Representing the equations as; A + B → C + D then the equation can be re-written as;

$$V = \frac{F_{A0} - F_A}{-r_A}$$

But since  $F_A = F_{A0} - F_{A0}X$ , then after simplifying, we see the CSTR volume necessary to achieve a specified conversion X is;

$$V = \frac{F_{A0}X}{-r_A}$$

Where, V = volume of reactor

$F_{A0}$  = inlet molar flow rate of limiting reactant

X = conversion of limiting reactant

$-r_A$  = rate of consumption of reactant

Since the order of the rate of reaction is first order, the rate depends on the limiting reactant which is  $\text{Li}_2\text{SO}_4$ . This implies that, the rate of reaction can be written as;

$$-r_A = kC_A$$

The concentration of the final product can be found by subtracting the amount consumed from the initial concentration;  $C_A = C_{A0} - C_{A0}X$

Substituting and factorizing gives;  $-r_A = k C_{A0} (1-X)$

From the material balance, the inlet mass for the reactants were obtained to be;

$$m_{A0} = 4405.96 \text{ kg/h}$$

$$m_{B0} = 3129.9 \text{ kg/h}$$

To calculate the molar flow rate of the limiting reactant  $F_{A0}$  then;

$F_{A0} = \frac{m_{A0}}{M_A}$ ; where  $m_{A0}$  is the mass flow rate and  $M_A$  is the molecular mass of the lithium sulfate  $F_{A0} = \frac{4405.96 \text{ kg/h}}{110 \text{ kg/kmol}}$

$$= 40.05 \text{ kmol/h}$$

Total volumetric flow rate of the species is given as;

$v_0 = \sum v_{i0}$ , where i is the various components present. But  $\rho = \frac{\text{mass}}{\text{volume}}$

Therefore; volume =  $\frac{\text{mass}}{\text{density}}$  ;

$$V_{BO} = \frac{m_{BO}}{\rho}$$

$$= \frac{3129.9 \text{ kg/h}}{2130 \text{ kg/m}^3}$$

$$= 1.469 \text{ m}^3/\text{h}$$

$$\sum V_{i0} = \frac{4405.96}{2220} + \frac{3129.9}{2130} + \frac{9935.27}{998}$$

$$= 13.409 \text{ m}^3/\text{h}$$

Similarly, concentration =  $\frac{\text{mole}}{\text{volume}} = \frac{\text{density}}{\text{molar mass}}$

Therefore;  $C_{AO} = \frac{\rho_{AO}}{M_A}$

$$= \frac{2220}{110}$$

$$= 20.18 \text{ kmol/m}^3$$

The conversion used was 97% so to calculate the concentration of the limiting reactant then;

$$C_A = C_{A0} - C_{A0}X$$

$$= 20.18 (1 - 0.97)$$

$$= 0.6054 \text{ kmol/m}^3$$

For a first order reaction;  $n = 1$

$$-\ln\left(\frac{C_A}{C_{AO}}\right) = kt$$

Taking a reference and solving for k for the reactor;

For  $t = 1 \text{ hour}$ ;  $k = 3.9 \text{ h}^{-1} = 0.00109 \text{ s}^{-1}$ ;

$$R = 8.3144598 \text{ J/mol.K} ; T_1 = 323 \text{ K} ;$$

Then at  $T = 100^\circ\text{C} = 373\text{K}$

$$E_A = 78.104 \text{ KJ/mol}$$

$$k(373) = 0.00109 \exp \left[ \frac{-78.104 \times 10^3}{8.314} \times \left( \frac{1}{373} - \frac{1}{323} \right) \right]$$

$$k(373) = 0.01647 \text{ s}^{-1}$$

Thus, the volume of the CSTR can be calculated as;

$$V = \frac{\left(\frac{40.05}{3600}\right) \times 0.97}{0.01647 (1-0.97) \times 0.6054}$$

$$V = 36.076 \text{ m}^3$$

## 2. Residence Time

$$\text{Residence time of a CSTR can be calculated as; } \tau = \frac{V}{V_O}$$

Therefore, for our reactor, the residence time will be given as;

$$\tau = \frac{36.076}{13.409} = 2.69$$

## Reactor Dimension

The reactor is built in such a way that the reaction mass does not surpass 90% of the reactor volume.

Actual reactor volume can be calculated as;

$$V = \frac{\text{Volume of feed}}{\text{percentage of reactor volume occupied by feed}}$$

$$= \frac{36.076}{0.9} = 40.08 \text{ m}^3$$

For a height to diameter ratio (H/D ratio), standard vessel proportions range between 2 and 5.

H/D = 2 is used to achieve both effective mixing in the vessel. Where H is the height of the vessel and D is the internal diameter of the vessel.

### **3. Diameter of Reactor Cylinder**

The volume of a cylinder can be calculated as;  $V = \frac{\pi d^2}{4} h$ . Then making d the subject, we calculate the diameter as;

Taking H/D ratio of 2, then  $H = 2D$

Substituting into the volume equation gives;  $V = \frac{\pi d^2}{4} \times 2D$

$$= \frac{\pi d^3}{2}$$

Then, the diameter will be calculated as;  $D = \sqrt[3]{\frac{2V}{\pi}}$

$$= \sqrt[3]{\frac{2(40.08)}{\pi}}$$

$$= 2.944 \text{ m}$$

$$H = 2D$$

$$H = 2(2.944)$$

$$H = 5.888 \text{ m}$$

### **Number of Impellers**

Dickey's criteria are used to determine the position and number of impellers (Walas, 1990). A single impeller is usually sufficient for off-bottom suspension in vessels with dished heads for which  $\frac{h}{D} < 1.3$  while dual impellers are recommended for

vessels with  $1.3 < \frac{h}{D} < 2.5$ , where  $h$  is the liquid level in the tank  $D$  is the vessel diameter.

Finding the liquid height in vessel,  $h$

The volume of liquid in a vertical vessel with a torispherical bottom can be calculated using the relation below (Jones, 2001)

$$V_f = \frac{\pi}{4} \left( \frac{2a_1^3}{3} + \frac{a_1 D_1^2}{2} \right) + \frac{\pi t}{2} \left[ \left( \frac{D}{2} - kD \right)^2 + s \right] + \frac{\pi t^2}{12} \\ + \pi D (1 - 2k) \left[ \frac{t\sqrt{s}}{4} + \frac{k^2 D^2}{2} \sin^{-1}(\cos\alpha) \right] + \frac{\pi D^2}{4} [h - (a_1 - a_2)]$$

For a standard torispherical head,

$$k = 0.06$$

$$u = 1.0$$

Defining parameters and solving for them; then

$$\alpha = \sin^{-1} \left[ \frac{1 - 2k}{2(u - k)} \right] = \sin^{-1} \left[ \frac{1 - 2 \times 0.06}{2(1 - 0.06)} \right] = 27.910$$

$$a_1 = fD(1 - \cos\alpha) = 1 \times 2.944[1 - \cos(27.910)] = 0.34$$

$$a_2 = kD\cos\alpha = 0.06 \times 2.944 \times \cos(27.910) = 0.156$$

$$D_1 = 2fDs\sin\alpha = 2 \times 1 \times 2.944 \times \sin(27.910) = 2.756$$

$$s = (kDs\sin\alpha)^2 = [0.06 \times 2.944 \times \sin(27.910)]^2 = 0.0387$$

$$t = 2a_2 = 2 \times 0.156 = 0.312$$

To calculate volume of the feed,  $V_f$  :

$$V_f = 36.076 = \frac{\pi}{4} \left( \frac{2 \times (0.34)^3}{3} + \frac{0.34 \times (2.756)^2}{2} \right) + \frac{\pi \times 0.312}{2} \left[ \left( \frac{2.944}{2} - (0.06 \times 2.944) \right)^2 + 0.0387 \right] + \frac{\pi \times (0.312)^2}{12} + (\pi \times 2.944) \times (1 - (2 \times 0.06)) \left[ \frac{0.312 \times \sqrt{0.0387}}{4} + \frac{(0.06)^2 \times (2.944)^2}{2} \sin^{-1}(\cos(27.910)) \right] + \frac{\pi \times (2.944)^2}{4} [h - (0.34 - 0.156)]$$

$$36.076 = 1.0347 + 0.8413 + 8.0338 + 6.807(h - 0.184)$$

$$h = 4.028 \text{ m}$$

Therefore, the height of liquid in the tori spherical head vessel, h is 4.028  $\approx$  4.0m. Now,

$$\frac{h}{D} = \frac{4.0}{2.944} = 1.36$$

Since the  $\frac{h}{D} = 1.36$  and is in the range of  $1.3 < \frac{h}{D} < 2.5$ , 2 impellers are required for effective stirring.

Distance between upper and lower impeller =  $kD_i$

where k = factor for finding the interval = 0.75

### **Stirrer Design**

In order to accomplish both axial and radial movement, a turbine with four blades and a pitch of 45 degrees is utilized. It is essential to maintain an axial flow pattern to ensure complete suspension of solids. Turbines are suitable for fluids with viscosities up to 20 cP, as stated by Garvin (2018). By incorporating a mixed flow pattern, heat transfer at the walls of the vessel is enhanced, and it aids in maintaining a consistent temperature within the vessel. The recommended dimensions for a turbine agitator can be found in McCabe et al.'s study from 1993 as given below;

$$\frac{D_a}{D_t} = \frac{1}{3}$$

$$\frac{W}{D_a} = \frac{1}{5}$$

Where  $D_a$  = impeller diameter

$D_t$  = tank diameter

$W$  = Width of impeller

$E$  = Height impeller

$L$  = Length of impeller blade

Based on this analysis,

$$\frac{D_a}{D_t} = \frac{1}{3}$$

$$\text{Impeller diameter} = D_a = \frac{D_t}{3} = \frac{2.944}{3} = 0.981 \text{ m}$$

$$\text{Distance between upper and lower impeller} = kD_i = 0.75 (0.981) = 0.736 \text{ m}$$

$$\frac{W}{D_a} = \frac{1}{5}$$

$$\text{Width of impeller} = W = \frac{D_a}{5} = \frac{0.981}{5} = 0.196 \text{ m}$$

$$\frac{E}{D_t} = \frac{1}{3}$$

$$\text{Height of impeller} = E = \frac{D_t}{3} = \frac{2.944}{3} = 0.981 \text{ m}$$

$$\frac{L}{D_a} = \frac{1}{4}$$

$$\text{Length of impeller} = L = \frac{D_a}{4} = \frac{0.981}{4} = 0.245 \text{ m}$$

## Determination of Impeller Clearance

Mixing System	Maximum Liquid Height, Z/T	Number of Impellers	Impeller Elevation from Tank Bottom	
			Bottom	Top
Liquid blending	1.4	1	Z/3	—
	2.1	2	T/3	2 Z/3
Solids suspension	1.2	1	Z/4	—
	1.8	2	T/4	2 Z/3
Gas dispersion	1.0	1	T/6	—
	1.8	2	T/6	2 Z/3

Fig : Dickey's criteria for selection of number of impellers

Testing for  $\frac{Z}{T}$  for which

$$Z = \text{liquid height} = 4.0 \text{ m}$$

$$T = \text{diameter of vessel}, \frac{Z}{T}$$

$$\frac{Z}{T} = \frac{4.0}{2.944} = 1.36, \text{ hence two impellers must be used just as obtained earlier from } \frac{h}{D}.$$

$$\text{Bottom clearance} = \frac{T}{3} = \frac{2.944}{3} = 0.981 \text{ m}$$

$$\text{Top clearance} = \frac{2Z}{3} = 2 \times \frac{4}{3} = 2.67 \text{ m}$$

### Stirrer speed

Calculating stirrer speed as a percentage of critical stirrer speed; from (McCabe, 1993)

For a liquid mixture, critical speed occurs at  $Re = 10,000$ . Complete off-bottom suspension of solids is necessary for adequate mass transfer required for reaction. Zweitering's correlation gives the essential stirrer speed required for the condition (McCabe, 1993).

$$n_c D_a^{0.85} = S \nu^{0.1} D_p^{0.2} \left( g \frac{\Delta \rho}{\rho} \right)^{0.45} \times B^{0.13}$$

Where;

$n_c$  = Critical stirrer speed, r/s

$D_a$  = agitator diameter, m

$S$  = Shape factor, 7.5

$\nu$  = Kinematic viscosity of liquid phase, m/s<sup>2</sup>

$g$  = Gravitation acceleration, m<sup>2</sup>/s

$D_p$  = average particle size, m =  $250 \times 10^{-6}$

$\Delta\rho$  = Density difference, kg/m<sup>3</sup>

$\rho$  = Liquid density, kg/m<sup>3</sup>

$B = 100 \times (\text{weight of solid}) / \text{weight of liquid}$

$$B = 100 \times \frac{1017.72685}{313.11575} = 325.03$$

Taking dynamic viscosity;  $\mu$  to be = 1.5 cP

Then kinetic viscosity will be gotten as;  $\nu = \frac{1.5 \times 10^{-3}}{2137} = 7.019 \times 10^{-7} \text{ m}^3/\text{s}$

For a six-blade turbine with D/W = 3, S = 7.5 (McCabe et al., 1993).

Substituting, this gives

$$n_c \times 0.981^{0.85} = 7.5 \times (7.019 \times 10^{-7})^{0.1} \times (250 \times 10^{-6})^{0.2} \times (9.81 \times 0.04)^{0.45} \times 325.03^{0.13}$$

$$n_c = \frac{0.482}{0.981^{0.85}}$$

$$= 0.489 \text{ r/s}$$

The minimum speed to turn the impeller is 0.468 r/s. A typical impeller speed is 10% above the minimum value hence,

$$n = 1.1 \times n_c$$

$$n = 0.539 \text{ r/s}$$

$$= 32.33 \text{ rpm}$$

### **Impeller Tip Speed**

The impeller tip speed is given by the relation; Tip speed =  $\pi \times D \times n$

Where;

Where  $D_i$  = Impeller diameter = 0.981

$n$  = Speed of agitation = 0.539 r/s

Therefore, tip speed =  $\pi \times 0.981 \times 0.539 = 1.66 \text{ m/s}$

### **Density of feed**

To determine the density of the mixture;

$$\text{Density of feed } (\rho) = \frac{\text{total mass}}{\frac{\text{mass of } Li_2SO_4}{\text{density of } Li_2SO_4} + \frac{\text{mass of NaOH}}{\text{density of NaOH}} + \frac{\text{mass of } H_2O}{\text{mass of } H_2O}}$$

$$(\rho) = \frac{\text{total mass}}{\text{total volume}}$$

$$(\rho) = \frac{3129.906 + 4405.96 + 9935.27}{13.409}$$

$$(\rho) = 1302.941 \text{ kg/m}^3$$

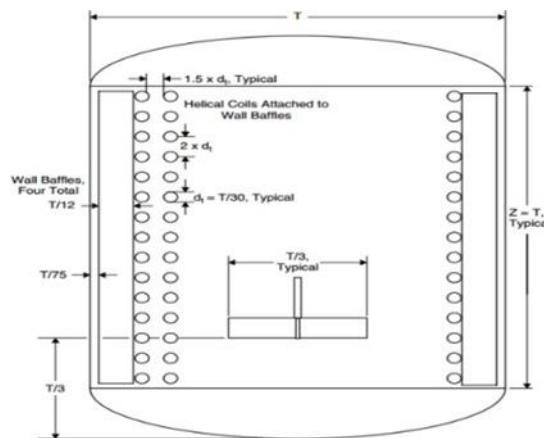
### **BAFFLES**

A vortex created by the spinning of an agitator is detrimental to mixing in a reactor vessel.

If the agitator is a rotating bed reactor, it also interrupts the liquid flow through the RBR.

Baffles are components of the reactor vessel that interrupt the circulating flow pattern, reducing vortex formation and enhancing overall mixing.

Baffles are components placed inside a tank to guide the fluid flow. They prevent the formation of a swirling vortex by breaking up the flow mixture. Their purpose is to enhance mixing effectiveness and promote turbulence. Baffles are typically metal strips that are radially attached to the CSTR's wall and have a width of about one-tenth of the vessel diameter. Increasing the width of the baffles has a limited impact on agitation, while reducing their size significantly decreases agitation. Usually, CSTRs are equipped with four to eight baffles to optimize mixing (Pukkella et al., 2019).



**Figure 27: Geometry of a Baffle**

### Number of Baffles

For most mixing situations, four baffles are used (Waals, 1990). As a result, four equally spaced baffles are used. Waals 1990 lists the features of baffles as;

- i. **Baffle width;**

$$W_b = \frac{D}{12} = \frac{2.944}{12} = 0.245\text{m}$$

D = diameter of vessel or reactor

ii. **Distance between baffles and wall of vessel;**

$$D_{wb} = \frac{D}{75} = \frac{2.944}{75} = 0.039\text{m}$$

iii. **Baffle height;**

$$H_b = H = \text{height of liquid} = 4.0\text{m}$$

### **Heat Removal System**

Cooling water is supplied at 50°C to cool and maintain the temperature of the reactor at the desired 100°C.

$$\Rightarrow Q = m_w \times c_{pw} \times (100 - 0)$$

Where;  $Q = 1355.56 \text{ MJ/h}$

$$c_{pw} = 4.182 \text{ kJ/kgK}$$

$$m_w = \frac{Q}{c_{pw} \times T} = \frac{1355.56}{4.182 \times 100} \times 10^3 = 3242.9 \text{ kg/h} = 0.9 \text{ kg/s}$$

Cooling Jacket Area available,  $A_j$

$$A_j = \pi D H + \frac{\pi D^2}{4}$$

$$A_j = \pi(2.944 \times 5.89) + \frac{\pi \times 2.944^2}{4} = 61.28\text{m}^2$$

Area of cooling jacket =  $61.28\text{m}^2$

Towler and Sinnott (2021) mentions that the spacing between jacket and vessel wall ranges between 50 mm and 300 mm.

**Table 14: Chemical Engineering Design Parameters**

Parameter	Specifications
<b>Reactor Dimensions and Reactor Conditions</b>	
Volume	40.08 m <sup>3</sup>
Height	5.89 m
Diameter	2.94 m
Maximum liquid height	4.02 m
Filling time	2.94 hr
Temperature	100°C
<b>Agitation System</b>	
Impeller type	45° Pitched six - blade turbine
Number of impellers	2
Interval between impellers	0.736m
Impeller diameter	0.981 m
Impeller width	0.196 m
Length of impeller blade	0.245 m
Height impeller	4.0 m
Top clearance	2.67 m
Bottom clearance	0.981 m
Impeller speed	0.489 rpm

Impeller tip speed	1.66 m/s
Number of baffles	4
Baffle width	0.245 m
Baffle height	4.0 m
Baffle offset from wall	0.039 m
<b>Heat Removal System</b>	
Heat transferred	1355 MJ/h
Temperature of inlet cooling water	50°C
Temperature of outlet cooling water	100°C
Mass flowrate of cooling water	0.9 kg/s
Jacket height	5.89 m
Jacket cooling area	61.28 m <sup>2</sup>

## 7.6 MECHANICAL DESIGN

Mechanical design is the process of designing mechanical parts, components, products, or systems. Mechanical design includes the design of numerous machine components such as shafts, bearings, clutches, gears, and fasteners. Many criteria have been presented in mechanical design processes; some of the fundamental design criteria include functionality, safety, dependability, manufacturability, weight, size, wear, and maintenance. material of construction. To evaluate and select a STR, the following factors are considered:

- i. mixing

- ii. heat transfer
- iii. jacket pressure drops
- iv. Cleaning

## **1. Material of Construction**

The main material, used in the construction of many industrial equipment, is stainless steel which is an alloy. Stainless steel (316L) is a corrosion-resistant material commonly used in chemical reactor construction, including CSTRs (Brunner, 2009). It offers excellent resistance to chemicals like lithium sulfate ( $\text{Li}_2\text{SO}_4$ ) and sodium hydroxide ( $\text{NaOH}$ ). Stainless steel's passive oxide layer protects it from corrosion, ensuring long-term durability. It can withstand high temperatures and pressures, making it suitable for various operating conditions. Stainless steel is strong and durable, capable of withstanding mechanical stresses. It is easily fabricated into different shapes and sizes, enabling customization. The smooth, non-porous surface of stainless steel simplifies cleaning and maintenance.

Because 316L has a low carbon content, welding-related harmful carbide precipitation is reduced to a minimum. So, when welding is required to achieve maximum corrosion resistance, 316L is employed.

**Table 15: Physical Properties of Stainless Steel (316L)**

Properties	Values
Density	0.799 g/cm <sup>3</sup>
Electrical Resistivity at 20°C	74 Ωm
Thermal Conductivity at 100°C	16.2 W/m.K

Melting Temperature	2500°F -2550 °F (1371°C-1399 °C)
Hardness, Brinell	149
Tensile Strength, Ultimate	515 MPa
Tensile Strength, Yield	205 MPa
Specific Heat Capacity	0.5 J/g. °C
Modulus of Elasticity	193 GPa

### Design Pressure

By the definition of maximum allowable pressure, using 10% of the total pressure as the maximum allowable pressure by Sinnott, 2005 and making it equivalent to design pressure.

$$P = \text{Design pressure, Pa}$$

$$P = 10\% \text{ more than } P_o$$

$$P_o = \text{Operating pressure in the column} = 1\text{atm} = 101325 \text{ Pa}$$

$$P = 1.1 \times 101325 = 111458 \text{ Pa}$$

### Design Temperature

It is usually accepted that the design temperature should be 10°C above the operational temperature (Sinnott, 2005)

$$T = 10^\circ\text{C} + T_o$$

Where;

$$T = \text{design temperature, } ^\circ\text{C}$$

$$T_o = \text{operating temperature} = 100^\circ\text{C}$$

$$T = 100^\circ\text{C} + 10^\circ\text{C} = 110^\circ\text{C}$$

Design stress of materials given at different temperatures

Material	Tensile strength (N/mm <sup>2</sup> )	Design stress at temperature °C (N/mm <sup>2</sup> )									
		0 to 50	100	150	200	250	300	350	400	450	500
Carbon steel (semi-killed or silicon killed)	360	135	125	115	105	95	85	80	70		
Carbon-manganese steel (semi-killed or silicon killed)	460	180	170	150	140	130	115	105	100		
Carbon-molybdenum steel, 0.5 per cent Mo	450	180	170	145	140	130	120	110	110		
Low alloy steel (Ni, Cr, Mo, V)	550	240	240	240	240	235	230	220	190	170	
Stainless steel 18Cr/8Ni unstabilised (304)	510	165	145	130	115	110	105	100	100	95	90
Stainless steel 18Cr/8Ni Ti stabilised (321)	540	165	150	140	135	130	130	125	120	120	115
Stainless steel 18Cr/8Ni Mo 2½ per cent (316)	520	175	150	135	120	115	110	105	105	100	95

Source: Coulson, 1996

Based on the table above, the tensile strength of stainless steel 316 obtained is 150 N/mm<sup>2</sup> at 100°C. Thus, the maximum allowable stress(S) at 110°C can be presumed as 150 N/mm<sup>2</sup>.

### Corrosion Allowance

The corrosion allowance is the additional thickness of metal added to allow for material lost by corrosion and erosion or scaling. Corrosion is a complex phenomenon, and it is not possible to give specific rules for the estimation of the corrosion allowance required for all circumstances. The allowance should be based on experience with the material of construction under similar service conditions to those for the proposed design. For stainless steel 316L, where severe corrosion is not expected, a minimum allowance of 2.0 mm should be used ( Sinnott, 2013). The corrosion allowance is chosen to be 2.0mm.

### Reactor Thickness

The minimum wall thickness of any vessel with a diameter ranging from 3 m to 3.5 m is 12 mm which includes a corrosion allowance of 2 mm (Sinnott, 2013). Hence, the chosen reactor thickness is 12 mm. Therefore, the minimum practical thickness of the inner reaction vessel,  $t_i = 14 \text{ mm}$ .

Table: minimum wall thickness of vessel allowed based on 2mm corrosion allowance

Vessel diameter (m)	Minimum thickness (mm)
1	5
1 to 2	7
2 to 2.5	9
2.5 to 3.0	10
3.0 to 3.5	12

Source: Coulson, 1966

### **Shaft Design**

A shaft is a mechanical part with a circular cross-section. It is used to transfer power via rotation. It serves as a pivot for a variety of mechanical components attached to it, including sprockets, gears, pulleys, flywheels, and cams. Finally, it is used to control the geometry of their rotational motion. The shaft is designed to accommodate hydraulic and mechanical loads and to avoid vibration near the natural frequency. Hydraulic loads on the shaft result from the torque required to turn the impellers and random or systematic lateral hydraulic loads on the impeller. The shaft design uses impeller power to calculate torque and hydraulic forces and thus size the shaft within allowable stress limits.

The parameters of shaft diameter as well as the length of shaft is calculated as follows;

Paul et al., (2015) gives the expression for the various parameters;

### **Shaft Length; Ls**

Shaft length,  $L_s = H - \text{clearance}$

$H$  = height of the reactor = 5.89

Shaft clearance = 1 impeller diameter = 0.981m

Shaft length,  $L_s$  = 5.89 – 0.981 = 4.91m

### **External diameter of the reactor**

The external diameter of the cylindrical part of the reactor will be a sum of the internal diameter and the thickness.

$$D_e = D + 2e$$

$D$  = reactor diameter = 2.94

$e$  = thickness = 14mm =  $14 \times 10^{-3}$  = 0.014

$$D_e = 2.94 + 2(0.014) = 2.968 \text{ m}$$

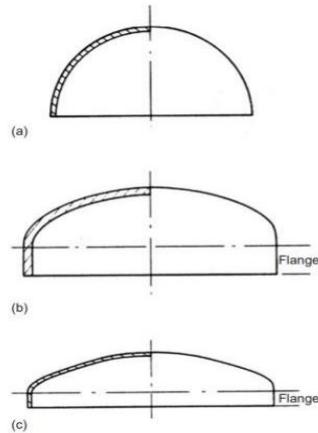
### **Heads and Closure Design**

The ends of a cylindrical vessel are closed by heads of various shapes. The principal types used are:

- i. Flat and formed heads: flat plates are used as covers for manways. Formed flat ends, known as “flange-only” ends, are the cheapest type of formed head to be manufactured, but their use is limited to low-pressure and small-diameter vessels. They are also ineffective for mixing operations where they require more than power than their domed counterparts.
- ii. Torispherical heads: For operating pressures below 15 bar, standard torispherical heads (dished heads) are the most common used type of heads.
- iii. Ellipsoidal heads: these heads are considered the most economical heads for pressures above 15 bar.

iv. Hemispherical heads: hemispherical head is the strongest shape; capable of resisting about twice the pressure of a torispherical head of the same thickness.

The cost of forming a hemispherical head will, however, be higher than that for a shallow torispherical head. Hemispherical heads are used for high pressures.



**Figure 28: Doomed Heads (a) Hemispherical (b) Ellipsoidal (c) Torispherical**

(Source: Coulson and Richardson, page 817)

### **Access Port design**

Access ports go by several different names; manholes, manways and hatches. They allow access to the internal parts of the equipment for periodic maintenance, inspection and servicing of the reactor vessel. Flat plates are used as covers for access ports. The typical diameter of manholes for vertical vessels is mentioned as 18 inches (450 mm) according to ICARU corporation (1998). Hence, the manhole diameter used on the reactor is 450 mm. this value also conforms to the rule of thumb that the manhole diameter should not be greater than half of the vessel diameter.

### **Weight of Vessel**

Weight of cylinder and torispherical head

For preliminary calculations, the approximate weight of a cylindrical vessel with domed end, and uniform wall thickness for a steel vessel can be estimated from the following equation: (Coulson, et al., 1996)

$$W_c = 240C_W D_M (H_v + 0.8D_M)t \quad (\text{Sinnott, 2013})$$

Where;

$W_c$  = total weight in kN of shell without internal fittings

$C_W$  = factor accounting for internal support, weight of nozzles

which is 1.08 for vessels with only a few internal fittings such as internal tubes

$D_M$  = mean diameter =  $D + e = 2.94 + 0.014 = 2.954\text{m}$

$H_v$  = height of column = 5.89 m

$t$  = column thickness = 14mm = 0.014m

Substituting gives;

$$W_c = 240 \times 1.08 \times 2.954 \times [5.89 + 0.8(2.954)] \times 0.014$$

$$W_c = 88.47 \text{ kN}$$

### Weight of the Reactor Content

$$W_{rc} = \rho_m \times V_l \times g$$

$\rho_m$  = density of mixture = ( $\rho$ ) = 1302.941 kg/m<sup>3</sup>

$V_l$  = volume of liquid = 13.409 m<sup>3</sup>

$$W_{rc} = 1302.941 \times 13.409 \times 9.81$$

$$W_{rc} = 171.39 \text{ kN}$$

$$\text{Total weight of vessel, } W = W_{rc} + W_c = 171.39 + 88.47 = 259.86 \text{ kN}$$

## Stress Analysis

### 1. Longitudinal stress $\sigma_L$

$$\sigma_L = \frac{P_d \times D}{4t}$$

where  $P_d$  = design pressure = 111457.5 Pa

$D$  = diameter of vessel = 2.94m

$t$  = thickness of vessel = 14mm = 0.014m

$$\sigma_L = \frac{111457.5 \times 2.94}{4(0.014)} = 5.85 \text{ N/mm}^2$$

### 2. Circumferential stress, ( $\sigma_h$ )

$$\sigma_h = \frac{P_d \times D}{2t}$$

where  $P_d$  = design pressure = 111457.5 Pa

$D$  = diameter of vessel = 2.94 m

$t$  = thickness of vessel = 14 mm

$$\sigma_h = \frac{111457.5 \times 2.94}{2(0.014)} = 11.7 \text{ N/mm}^2$$

### 3. Stress due to combined/ dead weight ( $\sigma_w$ )

$$\sigma_w = \frac{W}{\pi(D + t)t}$$

where  $W$  = total weight of vessel and content = 259.86 kN

$D$  = diameter of vessel = 2.94m

$t$  = thickness of vessel = 0.014m

$$\sigma_w = \frac{259.86 \text{ kN}}{\pi(2.94 + 0.014)(0.014)} = 2000 \text{ kPa} = 2.0 \text{ N/mm}^2$$

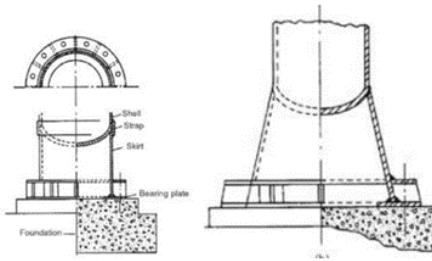
## Vessel Support Design

Type of supports to be provided to a vessel depend upon its configuration, height to diameter ratio of the vessel, convenience of location, operating temperature and materials of construction. The attachment of support at suitable location helps to transfer the load from the vessel to support safely. Cylindrical vessels can be installed either in vertical position or horizontal position and the types of supports for such installations are different. Similarly, for spherical vessels, the types of supports required vary considerably as compared to cylindrical vessels. It is necessary to analyze the stresses induced very carefully, so that adequately strong and cost-effective supports can be provided. Supports failure can lead to major hazards in a chemical process plant. Type of supports to be provided to a vessel depends upon its configuration, height to diameter ratio of the vessel, convenience of location, operating temperature and materials. There are four types of supports commonly used.

1. Skirt support
2. Bracket or Lug support
3. Saddle support
4. Leg support

## Skirts Supports

Skirts support is suitable for the tall, vertical column. In this plant, a straight skirt is chosen as a support. This is because as they do not impose concentrated loads on the vessel shell; they are particularly suitable for use with tall columns subject to wind loading.



**Figure 29: Image of a typical column skirt**

### Stress Analysis on Skirt

#### i. Dead Weight Stress on Skirt ( $\sigma_W$ )

$$\sigma_W = \frac{W}{\pi(D_s + t_s)t_s} \quad (\text{Sinnott, 2013})$$

$W$  = total weight of vessel and content = 259.86 kN

$D_s$  = Inside diameter of skirt = Diameter of column =  $D$  = 2.94 m

$t_s$  = 0.014m

$$\sigma_W = \frac{259.86 \text{ kN}}{\pi(2.94 + 0.014)(0.014)} = 2000 \text{ kPa} = 2.0 \text{ N/mm}^2$$

### Specifying Thickness Stress on Skirt Support

The thickness of the skirt,  $t_s$ , is taken to be the same as that of the vessel and this value is checked for suitability.

Thus using,  $t_s = 14 \text{ mm}$

### Summary of Mechanical Engineering Calculations

Parameter	Values
Operating Pressure, $P_o$	1 atm
Operating Temperature, $T_o$	100°C

Design Pressure, $P_i$	1.1 atm or 111458 Pa
Design Temperature, $T_i$	110 °C
Safety factor	10%
Material of construction	Stainless Steel 316 L
<b>Thickness of Cylindrical Shell</b>	
Design Stress, $S$ (from table)	150 N/mm <sup>2</sup>
Corrosion Allowance	2 mm
Thickness with corrosion allowance	14 mm
Shaft length	4.91 m
Shaft Material of Construction	Stainless Steel 316 L
External diameter of vessel	2.968 m
<b>Thickness of Spherical Head</b>	
Head type	Ellipsoidal
Corrosion Allowance	2 mm
Total height of reactor	5.89 m
<b>Support Type</b>	
Vessel Orientation	Vertical
Type of support	Straight Skirt
Material of construction	Stainless Steel 316 L

Skirt thickness	14 mm
Corrosion allowance for skirt	2 mm
Total weight of reactor	259.86 kN
Dead weight stress	2.0 N/mm <sup>2</sup>
Longitudinal stress	5.85 N/mm <sup>2</sup>
Circumferential stress	11.7 N/mm <sup>2</sup>

## Equipment Process Control and Instrumentation

The quality of lithium hydroxide produced by the Continuous Stirred Tank Reactor (CSTR) is significantly influenced by the operating temperature and pressure. The P-I (Proportional Integral) controllers are suitable for controlling both temperature and pressure in a reactor producing LiOH. These controllers compare setpoints with actual values (100°C for temperature) using sensors and adjust heating/cooling systems accordingly. P-I controllers can handle multiple variables simultaneously, allowing independent control of temperature and pressure.

### Temperature Control

In control system for the production of lithium hydroxide from lithium sulphate and sodium hydroxide, temperature is an important parameter to control. Higher inlet temperatures above the designed temperature would cause abrupt expansion and contraction of the system.

### Pressure Control

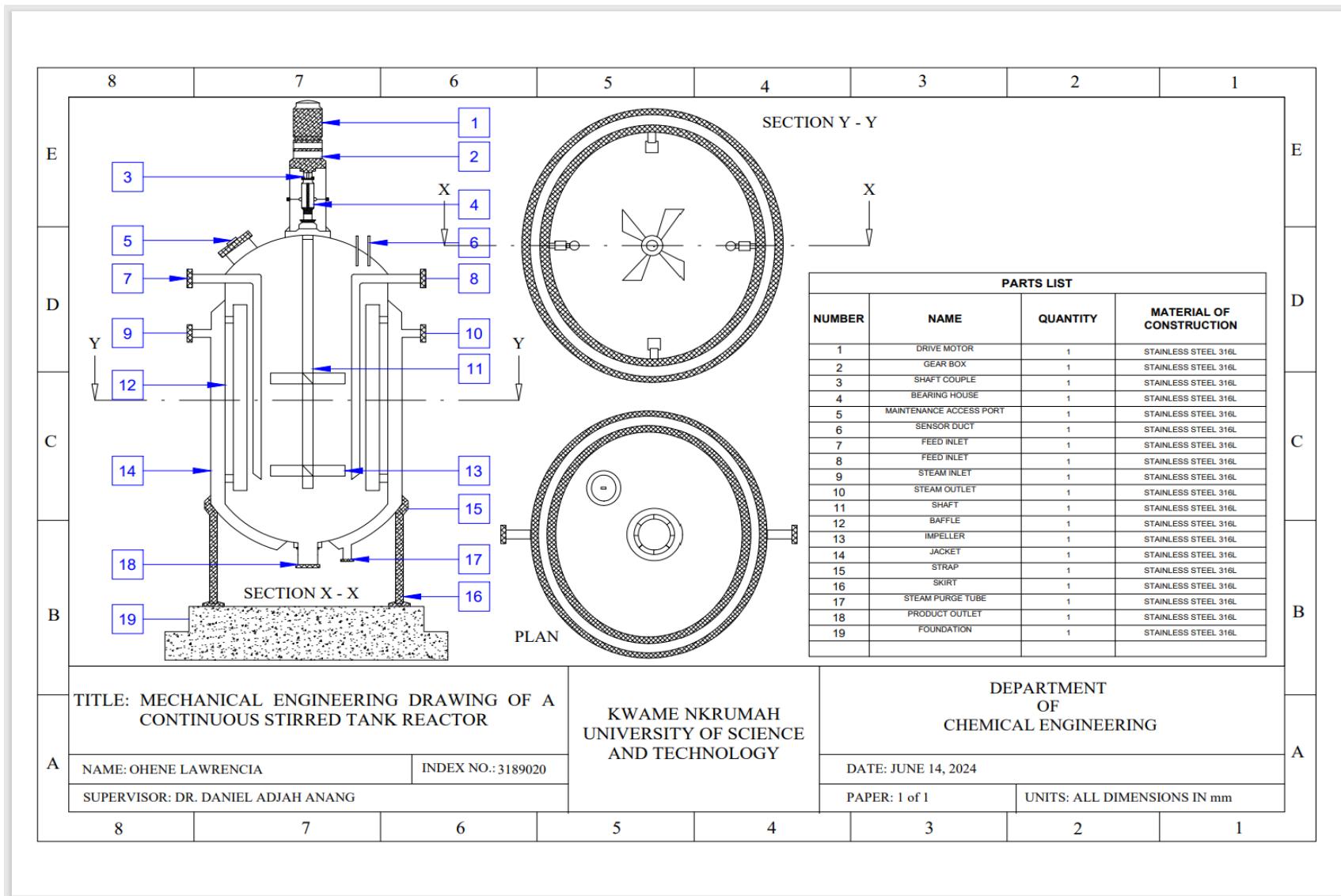
Pressure control is necessary for most systems handling vapor or gas which helps in maintaining the optimal pressure levels during operations and also to ensure safety. The

pressure control system and sensor are purposely for measuring the pressure of a unit, to generate a signal value which is readable to operators and to be able to maintain set point of operating pressures in order to avoid excessive build-up of pressures.

### **Equipment Safety and Pollution Control**

The unit includes a pumped condensate steam line that can generate temperatures hot enough to burn skin. Prior to unplugging any pipes or tubing:

- a) Turn off every pump.
- b) Give the water time to cool.
- c) Verify that the temperature is appropriate.
- d) While the equipment is in use, avoid touching any surfaces near the “Hot surfaces” warning labels or any of the interconnecting tubing.



**Figure 30: Mechanical drawing of a CSTR**

## **CHAPTER SEVEN**

### **7.0 DESIGN OF A DRYER FOR THE SEPARATION OF WATER FROM LITHIUM HYDROXIDE.**

#### **7.1 DESIGN PROBLEM**

A dryer is required to effectively dry wet Lithium hydroxide containing 0.5% moisture to a dry Lithium hydroxide containing 0.005 % moisture.

#### **7.2 INTRODUCTION**

Drying is a unit operation in which a liquid-solid separation is achieved through the use of heat, with separation occurring as a result of liquid evaporation (Patra et al., 2021). Drying constitutes a process that demands a considerable amount of energy. Generally, the application of heat and the subsequent evaporation necessitate significant energy inputs.

While water stands out as the most frequently eliminated liquid, the potential for solvent evaporation also exists. The act of drying can encompass various tasks such as dehydrating food, feed, and salts, as well as eliminating hydroxyl groups from organic compounds. To facilitate storage and transportation, the majority of solid materials undergo a drying phase at some stage of their production process (Daud, 2008). The essential mechanisms of convection, conduction, and radiation offer the necessary heat for the drying process:

- (i) Convection - By converting sensible heat into latent heat, a carrier gas (typically air) provides heat for the evaporation of the liquid. After that, the carrier gas entrains the volatile substance.
- (ii) Conduction - The heat is delivered in an indirect manner, and the carrier gas is solely used to evacuate the evaporated liquid. The airflow utilized in a

convective process is typically 10% of the airflow used in this process. The heat transmission process for contact drying is the conduction of heat.

- (iii) Radiation - In principle, this sort of drying can be nonpenetrating, like drying paint with infrared radiation, or penetrating, like drying food or pharmaceuticals with dielectric drying.

The only procedure in which heat is produced in the material being dried rather than being dispersed into the substance is dielectric drying (radio-frequency drying and microwave drying). To evacuate the evaporated liquid, another carrier gas is required. A combination of two mechanisms may be encountered in some dryer type;

Convective dryers are one of the most extensively utilized dryer systems in food industry.

The dried material is circulated around a stream of a hot drying agent - air, flue gases, etc. The way the procedure is organized differs in all convection dryers (periodic and continuous), in the direction of the dried material's and drying agent's movement (direct-flow and counter-current), and by pressure in the drying chamber (atmospheric and vacuum); by type of drying agent (air, gas, steam).

The liquid in the material to be dried can exist in two forms, either bound or unbound liquid. Bound moisture refers to the moisture that is retained in a weak chemical compound (appearing as a liquid solution within the solid material or possibly trapped within the microstructure of the solid) and possesses a lower vapor pressure compared to pure liquid. Unbound moisture, on the other hand, is the excess moisture present beyond the bound moisture.

Drying is carried out in the chemical industries for the following reasons:

1. Drying is one of the most common processes used to improve food stability since it decreases considerably the water activity of the material.

2. To make material more suitable for handling.
3. To reduce the moisture content of material to facilitate further processing. e. g.  
roasting
4. To preserve or improve the quality of the material to reduce the cost of transport.
5. Drying reduces microbiological activity and minimizes physical and chemical changes during its storage.

### **Main Objective**

To design a dryer to separate Lithium hydroxide in its aqueous form from water and also separate Lithium hydroxide from water.

### **Specific Objectives**

1. To select the appropriate unit type for the separation.
2. To conduct material and energy balance on the unit.
3. To carry out detailed chemical engineering calculations on the unit.
4. To carry out detailed mechanical engineering calculations on the unit.
5. To give a detailed mechanical engineering drawing of the selected unit.

## **7.3 LITERATURE REVIEW**

### **Classification of Dryers**

The categorization technique indicates various variations in dryer operation and design. There are numerous subcategories of drying techniques and procedures. It can be categorized as batch drying, where the material is placed into the drying apparatus and dried for a predetermined amount of time, or continuous drying, where the material is continuously fed to the dryer and dried material is continuously withdrawn. Drying processes can also be categorized according to the physical conditions used to add heat and remove water vapour.

The various categories are as follows;

- Method of heat transfer to the wet solids.
- Batch or continuous process used.
- Scale of production.
- Physical properties of the wet material and its handling characteristics.

(Sinnott, 2005).

### **Classification on basis of heat transfer method**

Dryers can be categorized based on the process by which the heat (energy) for drying is delivered. Conduction, convection, and radiation are the three primary methods of heat transport. Each of these systems operates in a peculiar manner and transfers heat to the things being dried in different ways. Methods of heat application and specific examples of driers in which these methods are used are as follows;

1. When water must be sublimed from frozen objects, infrared or radiant heat dryers and dielectric heat dryers are typically utilized. Microwaves and infrared dryers are two examples.

2. Direct contact with heated air at atmospheric pressure adds heat to direct continuous dryers, and the air being continuously taken from the product removes the water vapour that forms. Some examples are tunnel dryers, rotary dryers, spray dryers, through-circulation dryers, and pneumatic conveying dryers.
3. Direct contact with heated air at atmospheric pressure adds heat to direct batch dryers, and the air with the product obtained in batches removes the water vapour that forms throughout the drying process. Batch through-circulation dryers, batch tray and compartment dryers, and fluidized beds are a few examples.
4. Indirect continuous dryers continuously extract goods while evaporating water at low pressures and adding heat indirectly by radiation or contact with a metal wall. Low temperatures and vacuum are utilized when products that can deteriorate at high temperatures. Examples include screw conveyors, drum dryers, and cylinder dryers.
5. Although their product is extracted in batches, indirect batch dryers operate on the same principles as indirect continuous dryers. Dryers with agitated pans, freeze dryers, and vacuum rotary dryers are a few examples.

### **Dryer Selection based on physical form of the material.**

Based on the condition of the feed, or the state of the material to be dried, different types of dryers are chosen. Examples of dryers used in the various shapes that material might take are given below:

1. Wet feed which is in liquid form. Examples of dryers used include agitated batch, drum, and spray dryers.
2. Slurry (pumpable slurry or suspension). Examples of dryers used include, agitated batch, vacuum band, drum and spray dryer.

3. Paste (soft paste or slurry). Examples of dryers used include vacuum tray, agitated batch, convection tray, fluid bed dryer, vacuum band dryer, etc.
4. Reform (preformed paste). Examples of dryers used include vacuum trays, convection trays, batch through-circulation, fluid beds, etc.
5. Hard (hard-paste or matrix). Examples of dryers used include vacuum trays, convection trays, indirect rotary, continuous tray dryers, etc.
6. Granular (free-flowing granular or crystalline solid). Examples of dryers used include vacuum tray, agitated batch, pneumatic, batch through-circulation fluid bed, etc.
7. Fibrous (fibrous solid). Examples of dryers used include convection tray, vacuum tray, indirect rotary, batch-through circulation, pneumatic, etc.
8. Sheet. Examples of dryers used include vacuum tray, convection tray, drums, and continuous tray.

### **Dryer Selection Based on Scale of production**

Additionally, dryers are chosen based on the quantity of the material to be dried. While some dryers are utilized for a narrower range of tasks, others are employed for larger operations (Sinnott, 2005). Dryers are categorized using the following scales:

1. Feed rates for small-scale processes range from 20 to 50 kg/h. Vacuum tray, agitated, convection tray, through-circulation, and fluid bed dryers fall under this group of dryers, which are all batch processes.
2. Processes on a medium scale that use feed rates between 50 and 1000 kg/h. This can be batch- or continuous-based. Agitated, through-Circulation and fluid bed dryers are examples of medium-scale batch dryers. Fluid bed, vacuum band,

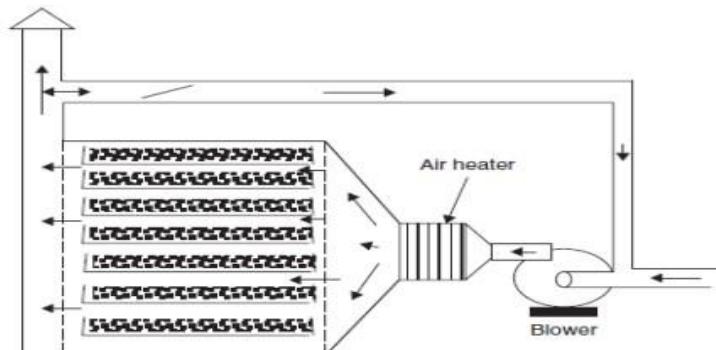
indirect rotary, spray, pneumatic dryers, and other medium-scale continuous dryers are examples.

3. Tons per hour are fed into a large-scale, continuously running process. Indirect rotary, spray, pneumatic, direct rotary, and fluid bed dryers are a few types of dryers that can be utilized in this procedure.

## Types of Dryers

The following types of dryers are used in the industry for different purposes of drying;

**Tray/Cabinet dryer** - Tray dryers are used for batch drying of solid foods at small to moderate scale (say, 2000 to 20 000 kg per day). They are affordable and easy to build. In a confined chamber, trays containing the food to be dried are placed in tray dryers.

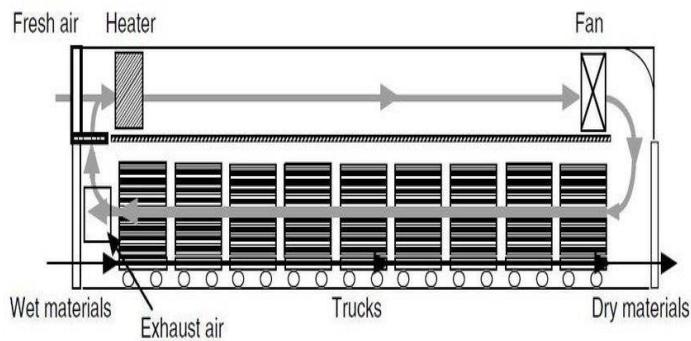


**Figure 31: typical tray dryer**

On shelves with enough room between them, the trays are placed. Between the shelves, heated, dry air is circulating. In order to allow for some air flow through the trays as well, tray bottoms are frequently slatted or perforated. The position of the material on the tray affects how quickly it dries and, consequently, how much moisture it contains (Sinnott, 2005).

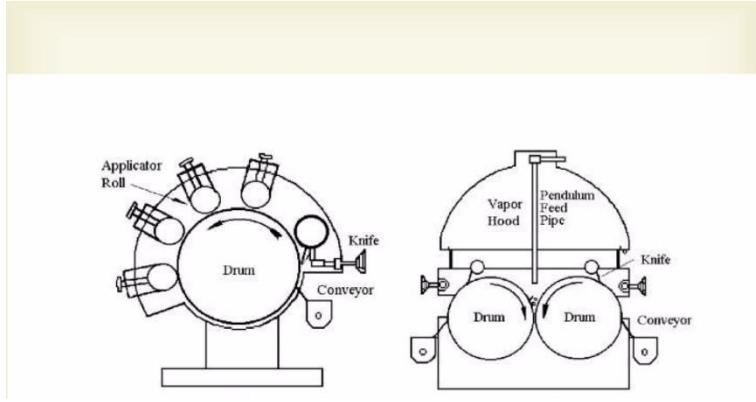
Tray dryers are typically seen in rural settings where they are used for drying fruits, vegetables, and herbs, such as grapes, dates, and apples (parsley, basil, mint, dill). Typically, air inlet temperatures vary from 60 to 80 °C. (Cabinet, 2000)

**Tunnel dryer** - Tunnel dryers consist of long tunnels through which trucks carrying stacks of trays travel with or against a stream of drying air. On the trays, the material to be dried is equally distributed (Sinnott, 2005). A vehicle carrying dehydrated goods exits the tunnel at the opposite end as one truck carrying wet goods enters the tunnel from one end. Depending on the size of the trucks and the tunnel, the trucks are moved manually or mechanically, e.g., with the help of chains



**Figure 32: A tunnel dryer**

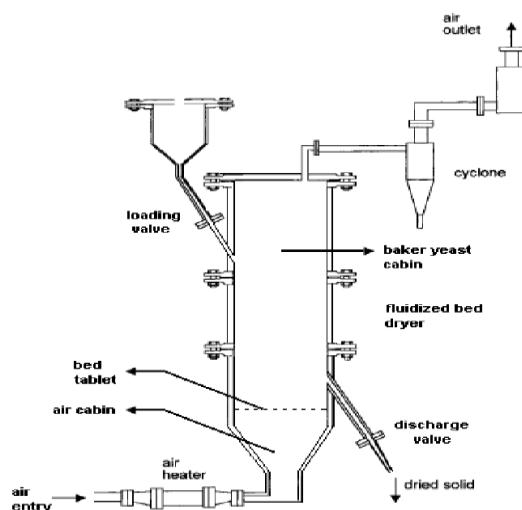
**Roller /Drum dryers** - The basic principle of operation of the heated surface is the envelope of a rotating horizontal metal cylinder. The cylinder is heated by steam condensing inside, at a pressure in the range of 200 to 500 kPa bringing the temperature of the cylinder wall to 120–155°C. By using a variety of different techniques that will be discussed later, the wet material is applied to the drum surface in a relatively thin layer (Couper et al., 2010). With the use of a blade, the dried product is removed from the drum.



**Figure 33: Single and double drum dryer**

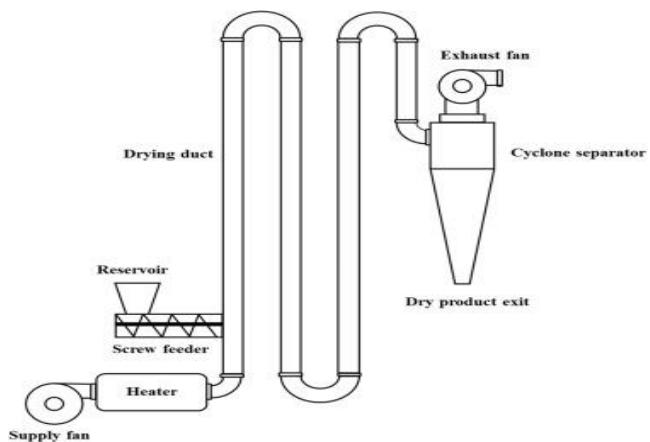
Single drum dryers and double drum dryers are the two categories into which drum dryers fall. Two drums that are revolving in opposite directions with a small, movable gap between them make up double-drum dryers. The so-called "twin" drum dryer really comprises of two separate, independently spinning single drums that share some auxiliary equipment. (Fox et al., 2014)

**Fluidized bed dryer** - In fluidized bed dryers, hot and dry air is used both for fluidization and for drying. Depending on the density, fluidized bed drying can be used to dry non-sticky, particulate meals with particle sizes between 0.05 and 10 mm. Fluidized bed drying can be in batches or continuous. (Daud, 2008)



**Figure 34: A fluidized bed dryer**

**Pneumatic dryers** - Also called flash dryers are similar in operation to spray dryers. The product to be dried is dispersed into an upward flowing stream of hot gas by a suitable feeder. The equipment acts as a pneumatic conveyor and dryer. Contact times are short. This limits the size of particles that can be dried. Pneumatic dryers are suitable for materials that are too fine to be dried in a fluidized bed dryer but which are heat sensitive and must be dried rapidly (Sinnott, 2005). The thermal efficiency of this type is low.



**Figure 35: A pneumatic dryer**

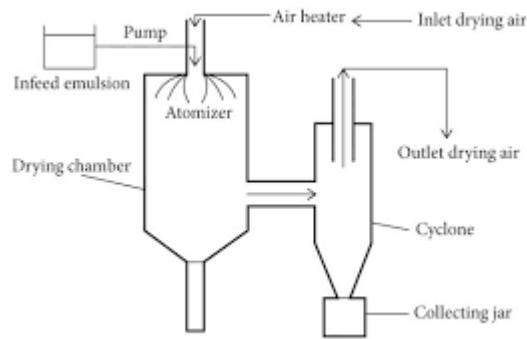
**Vacuum shelf dryer** - A vacuum chamber contains hollow shelves that make up a vacuum shelf drier. On flat metal trays with good shelf contact, food is arranged in thin layers. To dry the food, a chamber is partially vacuumed and steam or hot water is fed through the shelves. Both methods are excellent for heat-sensitive meals due to their quick drying times and minimal heat damage to the item. However, shrinkage lessens the contact between the food and heated surfaces in both types of equipment, and care must be taken to prevent the dried food from burning onto trays in vacuum shelf dryers.

**Rotary dryer** - The cascading rotary dryer is a continuously-operated direct contact dryer made up of a slowly rotating cylindrical shell that is usually slightly inclined to the horizontal to help transport the wet feedstock, which is fed into the drum at the top and

removed as the dried product at the bottom. Rotary dryers have different types as narrated below;

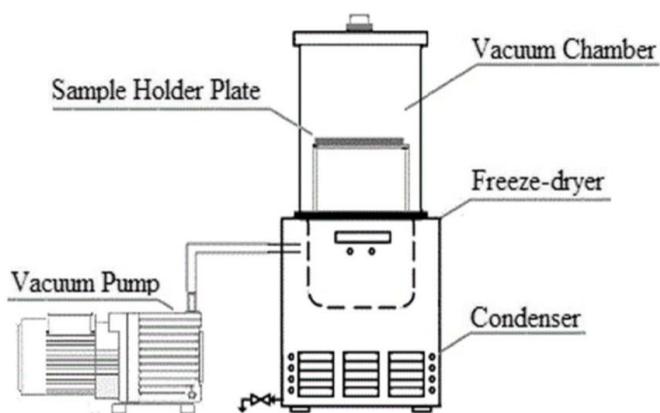
- Direct-heat rotary dryer - It is made up of a bare metal cylinder, either with or without flights, and it can only be used for low- and medium-temperature activities because to the metal's strength limitations. (Krokida et al., 2006)
- Direct rotary kiln - It is made up of a metal cylinder that is lined with insulation or refractory brick on the inside so that it can operate at high temperatures.
- Indirect steam-tube dryer - It consists of a cylindrical casing made entirely of bare metal with one or more rows of metal tubes inserted longitudinally inside. It can be used in operations that call for water cooling of the tubes or up to the available steam temperature.
- Indirect rotary calciner - It consists of a cylindrical casing made entirely of bare metal with one or more rows of metal tubes inserted longitudinally inside. It can be used in operations that call for water cooling of the tubes or up to the available steam temperature.
- Direct Roto-Louvre dryer - Due to the solids' crosscurrent motion toward the gas and its suitability for low- and medium-temperature applications, is most significant of the special types.

**Spray dryer** - In order to create light, porous powders, liquid solutions and suspensions are dried using spray dryers. Milk and whey powders, coffee creamers, cheese powder, dehydrated yeast extract, instant coffee and tea, isolated soybean protein, enzymes, maltodextrin, egg powder, and many other goods in powder form are produced using the established process of spray drying. Another technique for micro-encapsulation is spray drying (Arsenijevic et al., 2002).



**Figure 36: Spray dryer**

**Freeze dryer** - By lowering the product's temperature to a point where the majority of its moisture is in a solid state, freeze-drying can be completed. Ice can also be sublimated by lowering the pressure surrounding the product. When product quality is an important factor for consumer acceptance, freeze-drying provides an alternative approach for moisture removal. The heat- and mass-transfer processes during freeze-drying are unique. Depending on the configuration of the drying system, heat transfer can occur through a frozen product layer or through a dry product layer. Freeze-drying is used to dry high-value foods that have delicate aromas or textures. Examples include coffee, mushrooms, herbs and spices, strawberries and raspberries, fruit juices, meats, seafoods or vegetables.



**Figure 37: A freeze dryer**

## **Advantages of Drying**

1. Drying operations, such as cogeneration units, can make use of waste heat from other processes.
2. It is possible to contain and manage odors produced by the procedure.
3. The amount and weight of wet cake generated at the facility are decreased by drying. This lowers the cost of transportation for useful purposes.
4. Drying is a well-proven existing technology. (Babic et al., 2020)

## **Disadvantages of Drying**

1. The possibility of product overheating and fires, as well as the dust's explosive potential, are safety concerns with drying.
2. Any drying facility will emit air pollutants. It can be necessary to control air pollution and issue air permits.
3. Skilled operational personnel are necessary due to the intricacy of various drying systems. Both direct and indirect systems have significant maintenance needs.
4. The dried material's marketability is dependent on local circumstances. To establish the best uses and value of the dried product, a market analysis should be carried out

(Babic et al., 2020).

## **Typical design parameters of dryers are;**

1. Mode of operation
2. Retention or cycle time
3. Feed conditions
4. Suitability for heat sensitive materials
5. Operating pressure

6. Capacity
7. heat transfer method

#### **7.4 JUSTIFICATION**

The aim of the dryer section is to remove water from lithium hydroxide till the desired final moisture content is attained. That is 0.005% moisture content.

Direct or indirect methods of providing heat are both possible, with the former having better thermal efficiency.

A stream of hot gases in direct contact with the material being processed that's moist lithium hydroxide in a directly fired rotary dryer, which is the specific kind chosen. The hot gases are produced when a fuel is burned by a burner within the dryer.

The direct-heat dryers are the simplest, most economical and are used when the contact between the solids and gases or air is not harmful. The direct rotary dryer is highly suitable for low and medium-temperature operations, which are limited by the strength characteristics of the metal. Directly fired rotary dryers are further classified into co-current and countercurrent flow design. The counter-current is chosen since the process gives a more thorough dry, and it is more modern and established.

Therefore, direct-heat rotary counter-current dryer is the dryer of interest for this purpose.

#### **7.5 EQUIPMENT DESCRIPTION**

##### **Description of Rotary Dryer**

The rotary dryer, commonly known as the tumble dryer, is a device used to reduce a feed material's moisture content by delivering. It is made up of an inclined long drum or cylindrical shell that is frequently equipped with internal flights or lifters and rotates slowly on bearings. The material to be dried flows through it in a tumbling or cascading

motion either co-currently (for heat-sensitive materials) or counter-currently with the heating air or gases.

The internal tumbling action, also known as mechanical flip over, and the inclination of the shell to the horizontal together cause the material to move, giving the device the name "tumbling dryer."

The type of feed dictates the relative to the solid nature of the gas flow directions through the cylinder. The wet feed can be processed in batches or continuously using this drying machine, but for this purpose, continuous process will be used.

Rotary dryer is made up of a bare metal cylinder, either with or without flights, and it can only be used for low- and medium-temperature activities because of the metal's strength limitations.

It has a spinning drum furnace that is inclined and moves the material being treated downward. Direct or indirect methods of providing heat are both possible, with the former having a better thermal efficiency.

When heat is delivered to the particles by direct exchange between the gas and solids, a rotary dryer is considered to be of the direct type. Thus, the dryers that uses convection for transferring heat directly is termed as "direct" dryers. Hot air or gas passes through a process vessel in direct heat dryers, coming into direct contact with wet solids particles. Almost any heat source, but most frequently a gas or oil-fired furnace, can provide the hot air or gas. Direct-heated counter-current rotary dryers are mostly widely used in drying processing plant than any other type of dryer. This is understandable of the wide range of materials that can be dried in them and high efficiencies that are achieved in them.

In counter-current systems, the dryer's feed and drying medium move in opposite directions. Heat is needed to be supplied to the material in order to vaporize the water, and the resulting vapour is carried away by some means such as air streams (Keey et al., 2008). The capacity of a rotary dryer is determined to be related to the shape and arrangement of the lifters.

### **Mode of Operation**

A chute extending into the cylinder is used to feed materials into one end of the cylinder as it progresses through it by virtue of rotation, head effect, and slope of the cylinder to the discharge end of the dryer as finished (dried) product. In a direct heat rotary dryer, heat is being added to or removed from the solids by direct exchange between flowing gas and solids. Flow of hot air is opposite to the direction of flow of wet material in a direct heat countercurrent rotary dryer. Countercurrent flow of gas and solids gives greater heat transfer efficiency with a given inlet gas temperature. To be able to generate enough heat to dry the material, hot air is added by way of a blower. At the entry point of a direct heat countercurrent rotary dryer is also an exhaust hood which provides a controlled escape for fumes and water vapor that is generated in the drying process. Inside the dryer are flights that lift the wet mass of feed up to the top of the shells rotation and drop them through the hot air. The flights are usually offset every 0.6 – 2 m and their shape depend upon the characteristics of the solids (Couper et al., 2010). Radial flights with a 90° lip are used for free-flowing materials and flat radial flights without lip for sticky ones. It is a common practice to employ different flight designs along the dryer length to accommodate with the changing characteristics of the material during drying. In the first meter or so at the feed end, spiral flights are used for better distribution of the material under the feed chute (Mujumdar, 2006).

The movement of moisture is assumed to occur through various mechanisms including liquid diffusion, capillary flow and surface-activated diffusion. The two mechanisms mainly involved during drying are heat transfer and mass transfer.

When the feed is subjected to thermal drying, there is complicated interaction (Luikov et al.,2006) between heat- transfer and mass transfer both on the surface and in the structure of the solid. Heat transfer during occurs through the flow of heat as a result of convection, conduction or radiation in some cases as a result of a combination of any of these effects. In Mass transfer, drying of the wet solid depends on two, mechanisms: the internal movement of the moisture as a function of the internal structure and moisture content of the solids, and the external movement of the water vapour from the material surface as a result of temperature, airflow, and area of exposed surface.



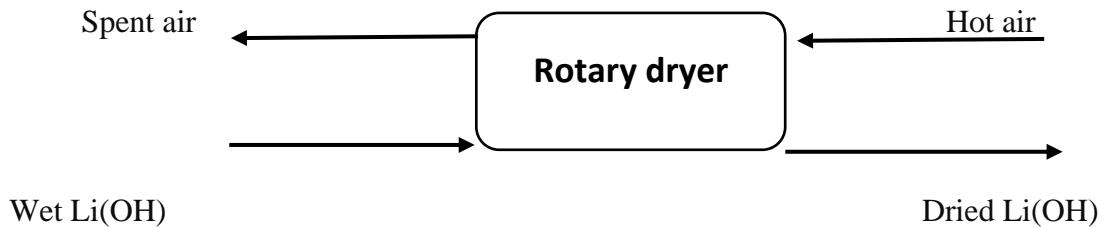
**Figure 38: Typical Rotary dryer**

## **7.6 CHEMICAL ENGINEERING DESIGN**

The typical design parameters of dryers are:

Heat transfer area, retention or cycle time, feed conditions, suitability of the heat, sensitivity of the drying materials, typical evaporation capacity (area in volume), capacity and operating pressure. The chemical engineering design of rotary dryer involves the

material and energy balance on the dryer, design parameters, dimensions of the equipment as well as nominal operations of the dryer.



### Material balance

Feed flowrate = 1918.96593 kg/h

Water content in product = 0.089593 kg/h

Amount of water in feed = 0.89593 kg/h

Water dried in the dryer = 0.806337 kg/h

Dry crystal in feed = 1918.07 kg/h

Dry crystal in the product = 1918.07 kg/h

Drying aims at taking the purified cake Li(OH) moisture content from 0.5% - 0.005%.

For air at 100°C and 3% relative humidity (RH) was assumed.

**Table 16: Properties of hot air at 100°C and 20% relative humidity (RH) assumed**

Property	Value
Relative humidity	3%
Wet-bulb temperature	38.752°C
Enthalpy	152.427 kJ/kg dry air
Specific heat	1.052 kJ/kg·K
density	0.935 kg/m <sup>3</sup>

Driers operate economically when the total number of transfer units (NTU) is between

1.5 – 2.5 (Perry and Green, 2008).

The number of heat transfer units (NTU) is given by

$$\text{NTU} = \ln\left(\frac{T_{ai} - T_w}{T_{ao} - T_w}\right) \quad (\text{perry and Green, 2008})$$

Where,  $T_{ai}$  (inlet air temperature) = 100°C.

$T_{ao}$  (outlet air temperature) = ?

$T_w$  (wet bulb temperature of inlet air) = 38.752°C, from the table above.

Assuming an average NTU = 2.

$$2 = \ln\left(\frac{100 - 38.752}{T_{ao} - 38.752}\right)$$

$T_{ao} = 47.04^\circ\text{C}$ .

Hence, the drying air leaves the drying section at 47.04°C.

### **Energy balance**

$C_p$  of lithium hydroxide = 3.980 kJ/kg· K.

$C_p$  of water = 4.187 kJ/kg· K

Inlet temperature of the feed = 30°C

Inlet air temperature = 100°C

Outlet temperature of the feed = 40°C

Outlet air temperature = 47.04°C

### **Calculating Heat Transferred by hot air to the lithium hydroxide**

The total heat transferred by the hot air in drying the lithium hydroxide is used for five different operations. That is:

- To vaporize the water that leaves the lithium hydroxide.
- To heat the water vapour from the initial wet-bulb temperature of the air to the exit air temperature.
- To heat the water that evaporates from its initial temperature as it enters the dryer to the inlet wet-bulb temperature of the air.
- To heat the lithium hydroxide from its inlet temperature to its exit temperature.
- To heat the water that remains in the final product from the inlet to the exit temperature of the material.

Calculating Heat Transferred by Air to the Lithium hydroxide

$$Q_T = (1 + y) \left[ \left( \frac{dQ}{dt} \right) + \left( \frac{dQl}{dt} \right) + \left( \frac{dQv}{dt} \right) + \left( \frac{dQh}{dt} \right) \right]$$

Where,

$Q$  = total heat transferred by air to feed (kJ/kg).

$(1 + y)$  = Account for heat loss from the dryer surface

$\frac{dQ}{dt}$  = heat received by the lithium hydroxide which is invested in raising temperature.

$\frac{dQl}{dt}$  = heat received by the water used in raising the temperature before vaporizing.

$\frac{dQv}{dt}$  = heat for evaporation

$\frac{dQh}{dt}$  = heat for reheating which is invested in raising the vapour temperature

**The quantity of heat received by the lithium hydroxide which is invested in raising the temperature,**

$$\frac{dQ}{dt} = M_s \times C_{ps} \times (\Delta T)$$

Where,

Mass flowrate of Li(OH),  $M_s = 1918.07 \text{ kg/h}$

Specific heat capacity of Li(OH),  $C_{ps} = 3.980 \text{ kJ/kg}\cdot\text{K}$

Change in temperature of the lithium hydroxide,  $\Delta T = 100 - 30$

$$= 70^\circ\text{C}.$$

Hence,

$$\frac{dQ}{dt} = 1918.07 \times 3.980 \times 70$$

$$\frac{dQ}{dt} = 534374.302 \text{ kJ/h}$$

### **Quantity of heat received by the water before evaporation.**

$$\frac{dQ_l}{dt} = M_w \times C_p \times (\Delta T)$$

$C_p = 4.187 \text{ kJ/kg}\cdot\text{K}$ , this is the mean specific heat of water ( $30^\circ\text{C} - 40^\circ\text{C}$ )

Change in temperature raised before evaporation,  $\Delta T = 40 - 30 = 10^\circ\text{C}$

$$\frac{dQ_l}{dt} = 0.89592 \times 4.187 \times 10$$

$$\frac{dQ_l}{dt} = 37.5122 \text{ kJ/h}$$

### **Quantity of heat transferred for Evaporation**

$$\frac{dQ_v}{dt} = M_{\text{water evaporated}} \times \lambda \quad (\text{perry et al., 2008})$$

Where,

$M_{\text{water evaporated}} = 0.806328 \text{ kg/h}$

Latent heat of vaporization of water,  $\lambda = 2260 \text{ kJ/kg}$  (geankoplis..1993)

$$\frac{dQ_v}{dt} = 0.806328 \times 2260$$

$$\frac{dQ_v}{dt} = 1822.3013 \text{ kJ/h.}$$

### Heat Transfer for Reheating of the Vapour

$$\frac{dQ_h}{dt} = M_{\text{water evaporated}} \times C_{p_v} \times (\Delta T)$$

$$M_{\text{water evaporated}} = 0.806328 \text{ kg/h}$$

$C_{p_v}$ , specific heat capacity at constant pressure of water vapour =  $1.996 \text{ kJ/kg}\cdot\text{K}$

$\Delta T$  = change in temperature in reheating of the vapour =  $47.04 - 38.752 = 8.288^\circ\text{C}$

$$\frac{dQ_h}{dt} = 0.806328 \times 1.996 \times 8.288$$

$$\frac{dQ_h}{dt} = 13.339 \text{ kJ/h}$$

### Total Heat Transfer

Therefore, the total heat transfer;

Heat loss from the dryer body is 5-10% (Kemp, 2012).

Assuming an average of 7.5% heat loss,  $y = 0.075$

$$Q_T = (1 + y) \left[ \left( \frac{dQ}{dt} \right) + \left( \frac{dQ_l}{dt} \right) + \left( \frac{dQ_v}{dt} \right) + \left( \frac{dQ_h}{dt} \right) \right]$$

$$Q_T = (1 + 0.075) (534374.302 + 37.5122 + 1822.3013 + 13.339)$$

$$Q_T = 1.075 \times 536247.4545 = 576460.4979 \text{ kJ/h}$$

### Mass flowrate of drying air

$C_{p_a}$ , specific heat capacity of air at  $100^\circ\text{C} = 1.052 \text{ kJ/kg}\cdot\text{K}$

$T_{a1}$  is the inlet air temperature = 100°C

$T_{a2}$  is the outlet air temperature = 47.04°C

$\Delta T$ , change in temperature of the air =  $(100 - 47.04) = 52.96^\circ\text{C}$

$M_{\text{air}}(1 + H_1) = QT / [Cp_a \times (100 - 47.04)]$ , where  $H_1$  = inlet humidity

Assuming an inlet humidity of 0.01,

$M_{\text{air}}(1 + H_1) = (576460.4979) / [1.052 \times (52.96)]$ ,

$M_{\text{air}}(1 + H_1) = 10346.795 \text{ kg/h}$

$M_{\text{air}} = 10244.35 \text{ kg/h}$

### Humidity of air

$$H_2 = H_1 + \frac{M \text{ water evaporated}}{M_{\text{air}}}$$

$$H_2 = 0.01 + \frac{0.806328}{10244.35}$$

$$H_2 = 0.0101$$

### The Volumetric Flowrate of Drying Air

Density of air at 100°C is approximately  $0.935 \text{ kg/m}^3$ . Thus, the volumetric flowrate of drying air is;

$$\frac{10244.35}{0.935} = 10956.52 \text{ m}^3/\text{h} \text{ since } \text{Volumetric flowrate} = \frac{\text{Mass flowrate}}{\text{Density}}$$

Thus, the volumetric flowrate of drying air is  $10956.52 \text{ m}^3/\text{h}$ .

### Cooling Section Calculations

The cooling section of the drier uses ambient air to cool the dried lithium hydroxide at

100°C leaving the drying section to 40°C.

The quantity of heat (Q) given off to cooling is given by

$$Q = M_s \times C_p \times \Delta T$$

where,  $M_s$ , mass of dried lithium hydroxide = 1918.1592 kg/h,

$C_p$ , specific heat of lithium hydroxide = 3.980 kJ/kg.K.

$\Delta T$ , temperature change in the lithium hydroxide = 60°C

$$Q = 1918.1592 \times 3.980 \times 60$$

$$Q = 458056.417 \text{ kJ/h}$$

The mass of cooling air required is given by;

$$m = \frac{Q}{(C_p \times \Delta T)}$$

where,  $Q = 458056.417 \text{ kJ/h}$

$C_p$ air, specific heat of air = 1.052 kJ/kg· K

$\Delta T$ , change in temperature of cooling air (100 - 40) = 60°C

$$\text{hence } m = \frac{458056.417}{1.052 \times 60} = 7256.914 \text{ kg/h}$$

### **The Efficiency of the Dryer**

The efficiency of the dryer is the ratio of the initial moisture and the final moisture content of the lithium hydroxide.

Mathematically;

$$\text{Efficiency} = \left[ \frac{\text{(initial moisture} - \text{final moisture)}}{\text{initial moisture}} \right] \times 100\%$$

Where, E = efficiency of the dryer.

$$E = \left[ \frac{0.89593 - 0.089593}{0.89593} \right] \times 100\%$$

$$E = 90\%$$

Hence the efficiency of the designed dryer is 90%.

### **Logarithmic Mean Temperature Difference**

The driving force for the heat transfer process is the log mean temperature difference.

Where,

$T_{a1}$  is the inlet air temperature of the dryer = 373K

$T_{a2}$  is the outlet air temperature of the dryer = 320.04K

$T_w$  is the wet-bulb temperature of the inlet air = 311.752K

$$\Delta T = \frac{(T_{a1}-T_w) - (T_{a2}-T_w)}{\ln(\frac{T_{a1}-T_w}{T_{a2}-T_w})}$$

$$\Delta T = \frac{(373-311.752) - (320.04-311.752)}{\ln(\frac{373-311.752}{320.04-311.752})}$$

$$\Delta T = 26.48^\circ\text{C}$$

### **Calculation of Inside Diameter & Length**

The allowable mass velocity of the air ranges from 2000 kg/h·m<sup>2</sup> up to 25000 kg/h·m<sup>2</sup> (McCabe, Harriot, & Smith, 1993).

According to Perry, it is important to select the proper air mass velocity to prevent serious dusting. Thus, assuming an air mass velocity of 13590 kg/h· m<sup>2</sup> based on the flowrate and the inlet air flowrate.

$$A = \frac{Q}{\text{mass velocity}}$$

$$A = \frac{24019.187}{13590}$$

$$A = 1.7671 \text{ m}^2$$

Hence the area for the dryer is  $1.7671 \text{ m}^2$

Also,

$$A = \pi D^2 / 4$$

$$D = \sqrt{[4A] / \pi}$$

$$D = \sqrt{[(4 \times 1.7671) / \pi]}$$

$$D = 1.4999 \text{ m} \approx 1.50 \text{ m}$$

Drying diameter ranges from 1 to 3 m (McCabe, Harriot, & Smith, 1993)

### Overall heat transfer coefficient

From Perry's handbook, the overall heat transfer coefficient;

$$U = \frac{KG^n}{D}$$

With an index  $n=0.67$  (McCormick, 1692). The K values fall in the ranges  $3.75 \leq K \geq 5.25$  as suggested by AIChE for SI unit (van't Land 2012). Assuming  $K= 4.75$

$$U = \frac{4.75(13590)^{0.67}}{1.4999}$$

$$U = 1861.61 \text{ kJ/h}\cdot\text{m}^3\cdot\text{K}$$

### The Length of the Dryer

The length to diameter ( $L/D$ ) ratio for the design of a dryer should be between 4 m - 10 m for the dryer to work efficiently and for safety (Hugot, 1986).

Taking the average length to diameter ratio to be 5,

$$L/D = 5$$

$$L = 5 \times 1.4999$$

$$L = 7.4995 \text{ m} = 7.5 \text{ m}$$

### The Volume of the Dryer

$$V = \pi D^2 l / 4$$

Where, d, internal diameter of the dryer = 1.4999 m

L = length of the dryer = 7.4995 m

$$V = [\pi \times (1.4999)^2 \times 7.4995] / 4$$

$$V = 13.25 \text{ m}^3$$

### Curved Surface Area

$$A = 2\pi r L$$

Where, A = heat transfer area

R, radius of the dryer = 0.75 m

L, length of the dryer = 7.4995 m

$$\text{Hence; } A = 2 \times \pi \times (0.75) \times (7.4995)$$

$$A = 35.3406 \text{ m}^2$$

### The Speed of Rotation of the Dryer

A single-rotary dryer rotates at a spherical speed that typically falls between 12 and 16 meters per second. Assume that the design dryer rotates at a speed of 12 m/min. (Hugot, 1986).

Revolution per minute (RPM) = Peripheral Speed / Diameter (Perry et al., 2008).

$$\text{RPM} = 12/D$$

RPM = 12/1.50

RPM = 8 rpm

### Retention Time

The average time of passage of feed in the rotary dryer can be estimated using the empirical equation below as recommended by AICHE (Van't Land, 2012).

$$\tau = (0.23 \times 4V) / (S \times N^{0.9} \times \pi \times D^3) \quad (\text{Perry et al, 2008}).$$

Where,

V, volume of the dryer =  $13.25 \text{ m}^3$

S, the slope of the dryer = 1.74

D, internal diameter of the dryer = 1.50 m

N, the speed of the dryer = 8 rpm

$$\tau = [\{0.23 \times 4 \times 13.25\} / \{1.74 \times 8^{0.9} \times \pi \times (1.50)^3\}]$$

$$\tau = 0.102 \text{ s}^{-1}$$

$$\tau = 6.10 \text{ mins} \approx 6 \text{ mins}$$

Residence time of the rotary dryer typically ranges at 5-90 minutes (Walas et al., 2005)

### Operating Pressure in the Dryer

Pressure within the dryer = pressure exerted by the lithium hydroxide + pressure exerted by the gas (air)

Pressure exerted by lithium hydroxide =  $m_s \times [g / \text{Area } (A)]$

Area = curved surface area x hold up (Walas, 1990).

The dryer hold up is assumed to be about 0.07 – 0.08 of the dryer volume, as values in this range give good performance in industrial dryers (Mujumdar, 2006).

Assuming hold up of 0.08

$$A = 35.3406 \times 0.08$$

$$A = 2.8272 \text{ m}^2$$

Where  $m_s$ , mass of the dried lithium hydroxide= 1918.07 kg/h.

$$\text{Acceleration due to gravity}(g) = 9.81 \text{ m/s}^2$$

$$A, \text{ surface area of the shell} = 2.8272 \text{ m}^2$$

$$\text{Pressure exerted by the lithium hydroxide} = [1918.07 \times (9.81/2.8272)]$$

$$= 1848.7 \text{ Pa}$$

### Pressure by Air

$$\text{Pressure by air, } P = RT / V \quad (\text{McCabe et al., 1993}).$$

$$\text{Where R, universal gas constant} = 8.314 \text{ m}^3\text{Pa} / \text{kmol}$$

$$T, \text{ temperature in kelvin} = (100 + 273) = 373 \text{ K.}$$

$$V, \text{ volume of the dryer} = 13.25 \text{ m}^3$$

Hence, pressure by vapour

$$P = (373 \times 8.314) / 13.25$$

$$P = 234047 \text{ Pa}$$

$$\text{working pressure} = 101.325 \text{ kPa}$$

$$\text{Design pressure} = 1.5 \times \text{working pressure} \quad (\text{Perry et al., 2008})$$

$$= 1.5 \times 101.325$$

$$= 151.988 \text{ kPa}$$

### Summary of the Chemical Engineering Design Calculations

<b>Parameter</b>	<b>value</b>
Inlet feed flowrate (kg/h)	1918.9656
Dry feed flowrate (kg/h)	1918.1592
Drying air flowrate (kg/h)	10244.35
Operating pressure (kPa)	101.325
Design Pressure (kPa)	151.988
Logarithmic mean temperature difference (°C)	26.48
Outlet air temperature (°C)	47.04
Number of transfer unit	2
Heat transfer area ( $m^2$ )	35.3406
Total energy required to dry feed (kJ/h)	576460.498
Overall heat transfer coefficient (kJ/h·m <sup>3</sup> ·K)	1861.61
Residence time (min)	6
Speed of rotation (rpm)	8
Evaporation efficiency (%)	90
Diameter of dryer (m)	1.50

Length of dryer (m)	7.50
Length to diameter ratio	5
Volume of the dryer ( $m^3$ )	13.25

## 7.7 MECHANICAL ENGINEERING DESIGN

### Material of Construction

- Material of construction is stainless steel (Type 316)
- Density of stainless steel 316L =  $8000 \text{ kg/m}^3$
- Yield strength of stainless steel = 205 MPa
- Tensile strength of stainless steel = 515 MPa
- Permissible stress of stainless steel =  $1.26 \times 108 \text{ N/m}^2 = 136 \text{ MPa}$  at ambient conditions.
- Percentage of dryer cross-section free(j) = 0.85

Stainless steel type 316L, and fibre glass are chosen as the main construction materials for the design of the cylindrical rotary dryer.

### Flight Design

The flight's goal is to elevate and sprinkle the sodium hydroxide in the gas stream, promoting close contact between the feed and the hot gas. The flights are usually offset every 0.6 to 2m and their shape depends on the characteristics of the feed. When materials change properties when drying, the flight design changes along the dryer length.

### Number of Flights

$$N_F = 3 \times D \text{ (Hugot, 1986).}$$

Where, NF = number of flights.

D = internal diameter in feet = 4.9212 ft

$$N_F = 3 \times 4.9212$$

$$N_F = 14.76 \text{ ft}$$

Hence approximately 15 flights are required.

### **Width of the Flight**

The width of the flight is taken as 1/12th of the internal diameter of the dryer.

$$W_d = \frac{1}{12} \times 1.50$$

$$W_d = 0.125 \text{ m}$$

### **Radial Height of the Flight**

The radial height of the flight taken as 1/8th of the internal diameter of the dryer.

$$R_H = \frac{1}{8} \times 1.50$$

$$R_H = 0.1875 \text{ m}$$

### **Shell dryer thickness and Outer diameter**

As a result of its capacity to tolerate greater temperatures and its resistance to corrosion and oxidation at excessive temperatures, the rotary dryer's shell is made of cylindrical stainless-steel types 316L.

The following equation calculates the shell thickness of a cylindrical direct-heat rotary dryer with a corrosion allowance of 3 mm.

$$ts = \frac{PD}{(2 \times Sw \times E) + P} + C$$

Where  $t_s$  is the shell thickness

D is the dryer internal diameter = 1.50 m

C is the corrosion allowance = 3 mm

E is the efficiency of the joint = 0.70

For double-welded butt joint without radiographing (Sinnott, 2005)

$S_w$  is the working stress of the material (type 316L, stainless steel) which is one-fourth the ultimate strength of the material (Couper et al., 2010).

The Ultimate Tensile Strength of the material ranges from 485MPa – 620MPa and for a conservative approach, 485MPa is chosen as the UTS for the material.

$$S_w = \frac{1}{4} \times 485 \text{ MPa}$$

$$S_w = 121.25 \text{ MPa}$$

Substituting,

$$t_s = \frac{10334.233 \times 1.5 \times 1000}{(2 \times 121.25 \times 10^6 \times 0.70) + 10334.233} + 3$$

$$= 3.091 \text{ mm}$$

Therefore, the shell thickness for the direct-heat rotary dryer is 3.091 mm.

### Fiber Outer diameter of the Shell

Outer shell diameter ( $D_s$ ) = internal dryer diameter +  $2 \times$  (shell thickness) (Perry et al., 2008).

$$\text{Outer shell diameter } (D_s) = 1.5 + 2 (3.091 \times 10^{-3})$$

$$\text{Outer shell diameter, } D_s = 1.506 \text{ m}$$

### Insulation thickness

It is required to insulate a direct-heat rotary dryer to save energy, keep process temperatures from varying with ambient conditions, and to safeguard workers who work near the machine. In order to prevent heat loss, the thickness of insulation is critical in the construction of a direct-heat rotary dryer. Fiberglass was chosen as the insulating material because it has a lower thermal conductivity (0.04 W/m·K) and thus a stronger heat resistance.

The thickness of insulation can be estimated using the heat transfer equation with insulation and is expressed as;

$$Q = \frac{2\pi L(T_{hot} - T_{am})}{\frac{\ln(\frac{R_{in}}{R_s})}{K_{ins}} + \frac{\ln(\frac{R_s}{R_i})}{K_s}}$$

Where,

$Q$  is the total heat transfer = 576460.4979 kg/h.

$L$  is the length of the dryer = 7.4995 m

$T_{hot}$  is the dryer inlet air temperature = 100°C

$T_{am}$  is the ambient air temperature = 40°C

$R_{ins}$  is the insulation radius = ?

$R_s$  is the shell radius = 0.753 m

$R_i$  is the internal radius of the dryer = 0.75 m

$K_s$  is the thermal conductivity of the shell = 14 W/m·K

$K_{ins}$  is the thermal conductivity of the insulator = 0.04 W/m·K

Substituting,

$$576460.4979 = \frac{2\pi \times (7.4995) \times (100 - 40)}{\frac{\ln(\frac{R_{ins}}{0.753})}{0.04} + \frac{\ln(\frac{0.753}{0.750})}{14}}$$

$$R_{ins} = 0.75312 \text{ m}$$

Insulation thickness ( $t_i$ ) = insulation radius ( $R_{ins}$ ) – shell radius ( $R_{shell}$ )

$$\text{Insulation thickness } (t_i) = 0.75312 - 0.753$$

$$= 0.00012 \text{ m}$$

### **Total Diameter of the Dryer**

$D_t$  = diameter of the shell + 2(insulation thickness)

$$D_t = 1.506 + 2(0.00012)$$

$$D_t = 1.50624 \text{ m} \approx 1.51 \text{ m}$$

### **Volume of Shell Material**

$$V_{SM} = \frac{\pi L (D_s^2 - D_i^2)}{4}$$

Where  $V_{SM}$  is the volume of the shell =?

$D_s$  is the shell diameter = 1.506 m

$D_i$  is the internal diameter = 1.50 m

L is the length of dryer = 7.4995 m

$$\text{Therefore, } V_{SM} = \frac{\pi \times 7.4995 ((1.506)^2 - (1.50)^2)}{4}$$

$$V_{SM} = 0.106 \text{ m}^3$$

### **Volume of Insulating Material**

$$V_{IM} = \frac{\pi \times 7.4995 ((1.51)^2 - (1.4999)^2)}{4}$$

$$V_{IM} = 0.179 \text{ m}^3$$

### Weight of Unloaded dryer

$W_{dryer}$  = Volume of shell material ( $V_{SM}$ ) x Density of shell material

Density of stainless steel (Grade 316L) = 8000 kg/m<sup>3</sup>

$$W_{dryer} = 0.106 \times 8000$$

$$W_{dryer} = 848 \text{ kg}$$

$$W_{dryer} = 848 \times 9.81$$

$$W_{dryer} = 8.32 \text{ kN}$$

### Weight of Insulating Material

$W_{insulation}$  = Volume of insulating material ( $V_{ln}$ ) x Density of insulation

Density of fiberglass = 1522 kg/m<sup>3</sup>

$$W_{insulation} = 0.179 \times 1522$$

$$W_{insulation} = 272.438 \text{ kg}$$

$$W_{insulation} = 272.438 \times 9.81$$

$$W_{insulation} = 2.67 \text{ kN}$$

### Weight of Material in the Dryer

$W_{material}$  = Inlet mass flowrate of feed x Residence time

Inlet mass flowrate of feed = 1918.9656 kg/h = 31.98 kg/mins

Residence time = 6 mins

$$W_{material} = 31.98 \times 6$$

$$W_{material} = 191.88 \text{ kg}$$

$$W_{material} = 191.88 \times 9.81 = 1.882 \text{ kN}$$

### Total Weight of Loaded Dryer

$$W_{TOTAL} = W_{dryer} + W_{insulation} + W_{material}$$

$$W_{TOTAL} = 8.32 \times 10^3 + 2.67 \times 10^3 + 1.882 \times 10^3$$

$$W_{TOTAL} = 12.872 \text{ kN}$$

### Stress Analysis

The principal stresses are pressure-induced longitudinal and circumferential stresses.

$$\text{Longitudinal stress} = \frac{PD}{4 \times t_s}$$

Where  $P = 7527.48538 \text{ Pa}$ ,

$D = 1.50 \text{ m}$  and

$t = \text{thickness} = 3.091 \text{ mm}$

$$\text{Longitudinal stress} = \frac{7527.48538 \times 1.50}{4 \times 3.091 \times 10^{-3}}$$

$$\text{Longitudinal stress} = 913234.2341 = 0.913 \text{ MPa}$$

$$\text{Circumferential stress } (\sigma) = \frac{PD_i}{2t_s}$$

$$\text{Circumferential stress } (\sigma) = \frac{7527.48538 \times 1.50}{2 \times 3.091 \times 10^{-3}}$$

$$\text{Circumferential stress } (\sigma) = 1826468.468 = 1.826 \text{ MPa}$$

Pressure vessels are called thin-walled if the internal radius of the dryer to wall thickness ratio is greater than 10.

$$\text{Ratio} = \frac{\text{internal radius of dryer}}{\text{dryer thickness}}$$

$$\text{Ratio} = \frac{0.75}{3.091 \times 10^{-3}}$$

$$= 242.6399 \approx 243$$

### **Design of the support**

The cylindrical drum is held at both ends of the dryer by two trunnion roll assemblies on a concrete support

Total weight of the dryer = 12.872 kN

$$\text{Uniform distributed load} = \frac{\text{total weight of the dryer}}{\text{length of the dryer}}$$

$$\text{Uniform distributed load} = \frac{12.872}{7.4995}$$

$$\text{Uniform distributed load} = 1.7164 \text{ kN/m}$$

### **Design of the Trunnion rolls assembly**

When sufficient nipping pressure is applied to establish tension separation, the metal roller deflects significantly.

$$\text{Maximum deflection } (\alpha) = \frac{5WL}{384EI}$$

Where,  $\alpha$  is the maximum deflection

W is the uniformly distributed load = 1.7164 kN/m

E is young's modulus of elasticity of steel = 200GPa

L is the distance between supports = length of dryer/ideal number of supports

$$L = \frac{7.4995}{4}$$

$$L = 1.875 \text{ m}$$

$i$  is the area moment of inertia of the metal roller,

$$i = \pi r^3 t$$

Where,  $t$  = shell thickness = 3.091 mm

$r$  = internal radius of dryer = 0.75 m

substituting,

$$\alpha = \frac{5 \times 17164 \times 7.4995}{384 \times 200 \times 10^9 \times 3.091 \times 10^{-3} \times 0.75}$$

$$\alpha = 3.615 \times 10^{-6} \text{ m}$$

### Bending Moments

$$M_B = \frac{WL^2}{8}$$

Where,

$M_B$  is the bending moment

$W$  is the uniformly distributed load = 17164 N

$L$  is the distance between support = 1.875 m

$$M_B = \frac{17164 \times 1.875}{8}$$

$$M_B = 4022.81 \text{ Nm}$$

### Bending Stress

$$S_B = \frac{4M}{\pi D t}$$

Where,

$S_B$  is the bending stress

$M_B$  is the bending moment = 4022.81 Nm

D is the internal diameter of dryer = 1.50 m

t is the shell thickness = 3.091 mm

$$\text{therefore, } S_B = \frac{4 \times 4022.81}{\pi \times 1.50 \times 3.091}$$

$$S_B = 1104.7 \text{ Pa}$$

### Torque

Torque = Rotating load  $\times$  Radius of the dryer

$$\text{Torque} = 12872 \times 0.75$$

$$\text{Torque} = 9654 \text{ Nm}$$

### Diametral Pitch

A gear's diametral pitch is the number of teeth per inch of pitch diameter.

The shorter the tooth of the gear, the stronger it is and the greater its load carrying capacity

(Mayuram, 2012).

Choosing module 1.5 of gear, Equivalent diametral pitch(P) = 16.8 teeth/inch

(Engineers edge, 2019)

### Pitch Diameter (PD)

Pitch diameter is the diameter of the pitch circle. In parallel shaft gears, the pitch diameter can be calculated directly from the center distance and the number of teeth. Assuming 68 teeth of gear.

$$\text{PD} = N/P = 68/16.8 = 4.048 \text{ inch} = 102.82 \text{ mm}$$

## **Pitch Circle**

The pitch circle is the circle formed by a specific number of teeth and diametral pitch. It is sometimes referred to as the circle on which spacing or tooth proportions are created and from which tooth proportions are constructed.

## **Outside Gear diameter (OD)**

$$OD = (N+2) / P$$

$$OD = (6 +2) / 16.8$$

$$OD = 0.48 \text{ inch} = 12.19 \times 10^{-3} \text{ m}$$

## **Power Requirement**

During dryer operation, power is necessary to turn the rollers, the blower, and the exhaust fan. As a result, the quantity of power required by the dryer for efficient operation must be calculated.

## **Motor Power**

This is the power required to drive the power and it can be estimated using the equation proposed by Van't Land.

$$P_m = \frac{0.3D_i^2L}{4}$$

Where  $P_m$  is the motor power

$D_i$  is the internal diameter of the dryer = 1.5 m

L is the length of the dryer = 7.4995 m

Putting it all together, yields

$$P_m = \frac{0.3 \times 1.5^2 \times 7.4995}{4}$$

$$P_m = 1.27 \text{ kW}$$

### **Blower power requirement**

The blower provides the necessary air to dry the moist feed. The power required to convey the required volume of air is calculated using the following equation.

$$P_B = 2.72 \times 10^{-5} \times Q_{\text{inlet air}} \times P$$

Where,  $P_B$  is the blower power

$P$  is the fan operating pressure = 20 cm water column

$Q_{\text{inlet air}}$  is the volumetric flowrate of hot air

$$Q_{\text{inlet air}} = 10956.52 \text{ m}^3/\text{h}$$

Substituting,

$$P_B = 2.72 \times 10^{-5} \times 10956.52 \times 0.2$$

$$P_B = 0.10 \text{ kW}$$

### **Power required by exhaust fan**

The equation below is used in estimating the power needed to drive air out of the rotary dryer.

$$P_E = 2.72 \times 10^{-5} \times Q_{\text{outlet air}} \times P$$

Where  $P_E$  is the power of the exhaust fan

$P$  is the fan operating pressure = 20 cm water column

$Q_{\text{outlet air}}$  is the volume of the outlet air

$T$  = temperature of outlet air = 320.04K

$R$  = Gas constant = 8.314 kPa.  $\text{m}^3/\text{kmol. K}$

P = Outlet air pressure = 101.325 kPa

$$Q_{outlet\ air} = \frac{10956.52 \times 8.314 \times 320.04}{101.325}$$

$$Q_{outlet\ air} = 287720.1681 \text{ m}^3/\text{h}$$

Substituting,

$$P_E = 2.72 \times 10^{-5} \times 287720.1681 \times 0.20$$

$$P_E = 1.57 \text{ kW}$$

### Power required to drive the dryer

The equation below is used to estimate the power needed to drive a rotary dryer with flights.

$$\text{Bhp} = \frac{N(4.75D_s W_{material} + 0.1925D_T W_{total} + 0.33W_{total})}{100000} \quad (\text{Perry et al., 2008})$$

Where,

bhp is brake horsepower required (1bhp = 0.75kW)

N is the speed of rotation of the dryer = 8 rpm

$D_s$  is the shell diameter of the dryer = 4.9409ft

$D_r$  is the riding ring diameter =  $D_s + 2 = 6.9409\text{ft}$

$W_{material}$  is the weight of the material = 423.090 lbs

$W_{total}$  is the total rotating load = 2893.741 lbs

$$\text{Bhp} = \frac{8(4.75 \times 4.9409 \times 423.090 + 0.1925 \times 3.091 \times 2893.741 + 0.33 \times 2893.741)}{100000}$$

$$\text{Bhp} = 1.101 \text{ bhp} = 0.821 \text{ kW}$$

## Calculating the diameter of feed chute

Inlet feed rate = 1918.9656 kg/h

Average density of empty lithium hydroxide = 1540 kg/m<sup>3</sup>

Velocity of inlet feed = 7 m/h

$$\text{Cross sectional area of feed chute}(A) = \frac{\text{inlet mass flowrate}}{\text{density} \times \text{velocity}}$$

$$\text{Cross sectional area of feed chute}(A) = \frac{1918.9656}{1540 \times 7} = 0.1780 \text{ m}^2$$

$$\text{Diameter} = \sqrt{\frac{4A}{\pi}}$$

$$\text{Diameter} = \sqrt{\frac{4 \times 0.1780}{\pi}} = 0.476 \text{ m}$$

Summary of mechanical engineering design calculation

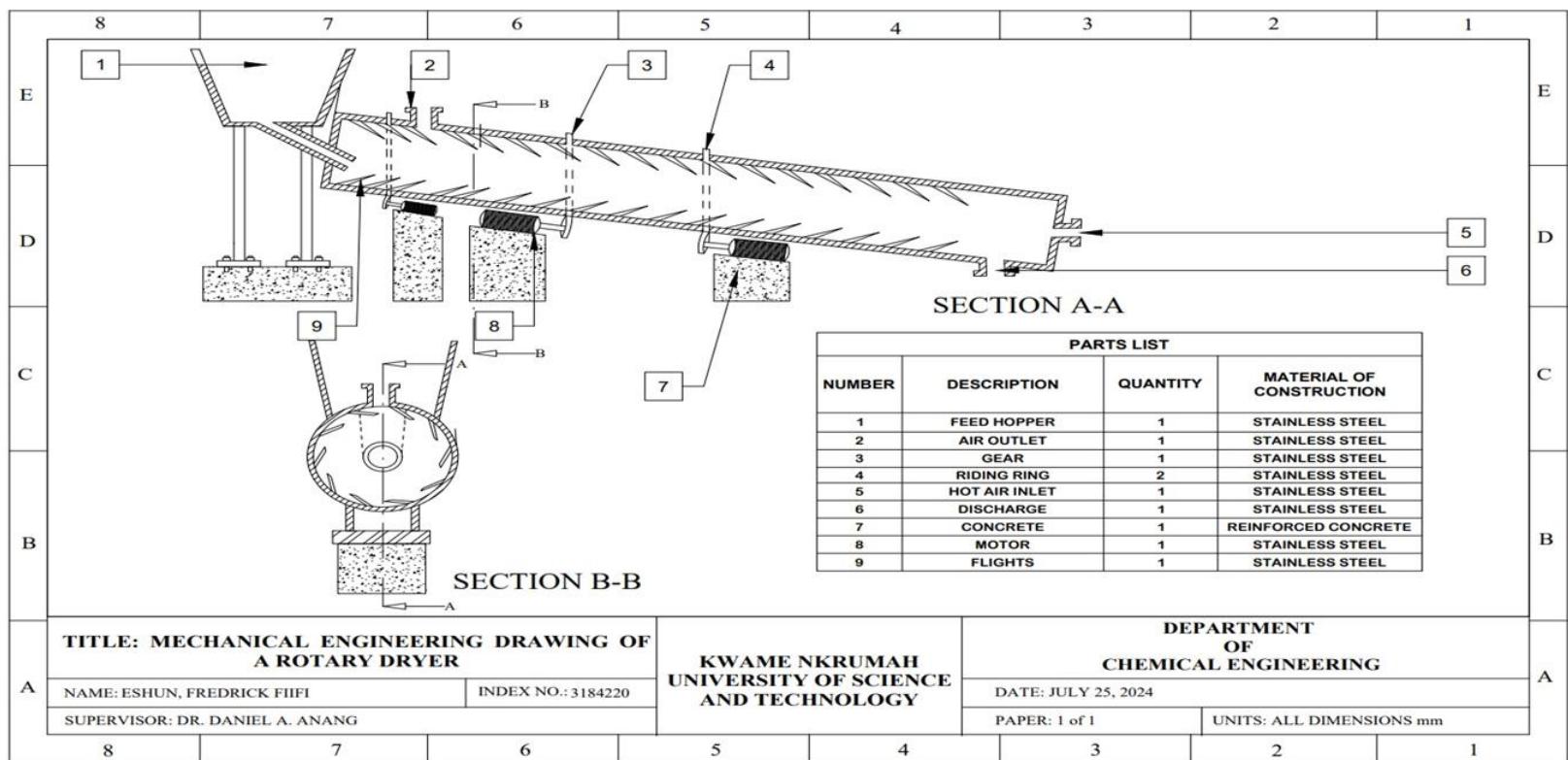
Parameters	Value
Outside diameter of dryer (m)	1.51
Shell thickness (mm)	3.091
Insulation thickness (mm)	0.12
Number of flights	15
Radial height of flight (m)	0.1875
Weight of shell material (kN)	8.32
Width of the flight (m)	0.125

Weight of insulating material (kN)	2.67
Total weight of loaded dryer (kN)	12.872
Circumferential stress (MPa)	1.826
Longitudinal stress (MPa)	0.913
Bending moment (Nm)	4022.81
Bending stress (Pa)	1104.7
Torque (Nm)	9654
Uniformly distributed load (kN)	17.164
Motor power (kW)	1.27
Blower power (kW)	0.10
Exhaust fan power (kW)	1.57
Drive power of dryer (kW)	0.821
Internal diameter of feed chute (m)	0.147

**Table 17: Rotary dryer Specifications**

Function	To remove water from the crystallized Lithium hydroxide
Capacity	1918.96593 kg/h
Number	1

Diameter	1.50 m
Length	7.50 m
Temperature	100°C
Material of construction	Stainless steel



**Figure 39: Mechanical engineering drawing of a rotary dryer**

## **CHAPTER EIGHT**

### **8.0 DESIGN OF A JAW CRUSHER**

#### **8.1 INTRODUCTION**

Mineral beneficiation is considered as the first step of mineral extraction from its natural resources. It is very rare to find mineral particles on the surface which is ready for the extraction process. When ore aggregates with gangue minerals, it becomes necessary to separate valuable minerals from the gangue. Mineral beneficiation is the process of converting raw materials into marketable products. This process is also known as mineral dressing or ore dressing (Clarence H. Lorig, March 2017). During this operation, the mineral doesn't lose its chemical and physical identity.

Mineral beneficiation is done by crushing and grinding with the help of different equipment. Large particles pass through the hard surfaces where it gets hammered and broken into small pieces. These pieces further move to the different machines with allowable size openings and again go under intense size reduction and turn into fine particles. In extractive metallurgy, such operations are technically known as comminution.

Comminution process creates not only the larger surface area, by breaking the particle it adds new surfaces which may be more active for any chemical and physical reaction as they are not weathered by any geological phenomena (Gross, 1938). There are very few processes where it requires large particle sizes. The degree of fineness of the particle depends upon the acceptance of the particle size for the upcoming process that needs to be operated. Comminution, in general, consists of two processes together namely crushing and grinding.

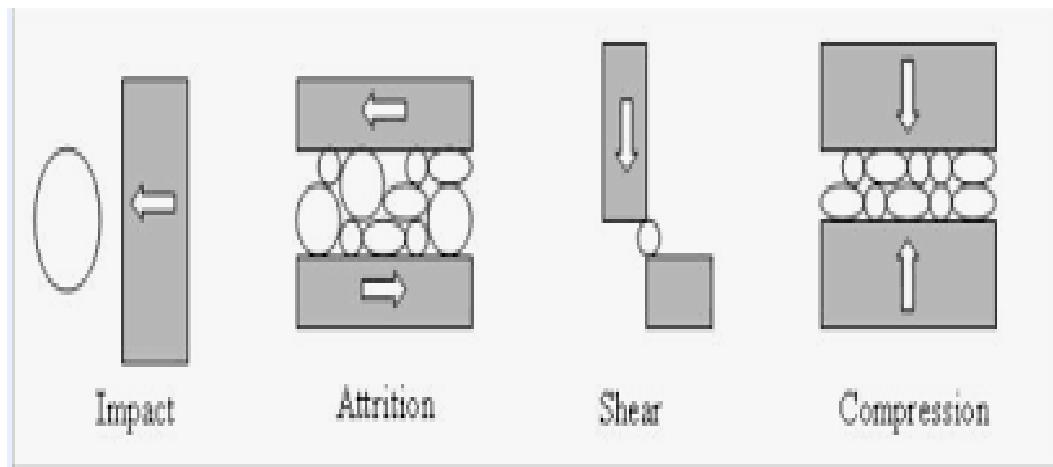
Crushing is the first stage in comminution where lumps of ore from natural reserves are being broken down into intermediate sizes from their originally huge size. The screening

process is followed by crushing at the individual stage. At every stage, oversize ore pieces which are not crushed into proper size, are again conveyed towards re-crushing by the process circuit and the under-size particles are sent to upcoming size reduction operations. After crushing, valuable mineral gets liberated up to a certain extent but the mineral particles which are still agglomerated with gangue minerals because of the particle size need to go further intense the crushing stage which is called grinding or milling. In the milling operation, the feed is under-size from the crushing process (Inoue).

The types of equipment employed in the crushing process are known as crushers. Examples include; jaw crusher, roll crusher, cone crusher, gyratory crusher, and impact crusher. Factors that affect crusher selection are; plant throughput, size of feed, desired product size, ore characteristics (hard rock, clay, gravel, etc), capital cost, operational cost (power consumption, maintenance), etc.

According to Coulson and Richardson, there are four patterns through which forces are applied to break down particles. These include;

- a) Impact ---- forcible contact with particle by a single force
- b) Compression---- disintegration of particle by two opposite rigid forces
- c) Shear---- particle reduction by particle-particle interaction
- d) Attrition---- particles scraping against one another or against a rigid surface.



**Figure 40: Forces of comminution**

### LAWS OF COMMINUTION

Different empirical rules have been presented, despite the fact that it is difficult to precisely determine the amount of energy needed to effect a size reduction of a given material.

**Von Rittinger's Law:** This law states that the energy required for size reduction is proportional to the change in surface area of the particles. This law requires that grinding rate function is proportional to particle size which would be the case for very hard materials over limited size ranges, that is, for  $d < 0.05\text{mm}$ .

**Kick's Law:** The Kick's law states that the energy required to reduce the size of particles is proportional to the ratio of the initial size of a typical dimension (for example the diameter of the particles) to the final size of that dimension. This relation is derived directly from the elasticity theory of ideal brittle solids.

**Bond's Law:** The work required to form particles of size from a very large particle size is proportional to the square root of the surface to volume ratio of the product.

## **PROBLEM STATEMENT**

Particle size is a critical component in mineral processing plants. Crushing ores with crushers can obtain the desired particle size.

## **MAIN OBJECTIVES**

- Design a crusher for the comminution process of spodumene ore in a lithium hydroxide production facility.

## **SPECIFIC OBJECTIVES**

- To specify the type of crusher to be used for the comminution
- To perform detailed chemical engineering design calculations on the selected crusher
- To carry out detailed mechanical engineering design calculations on the selected crusher
- To provide a mechanical engineering drawing of the selected crusher

## **8.2 LITERATURE REVIEW**

### **The Crushing Process**

Crushing is the first step of mineral processing where the ore/rocks from the mine site is fed into the mechanical equipment in order to reduce the size of masses for subsequent usage by liberating the valuable mineral from the gangue. Generally, it is a dry process and is usually performed in two or three stages. This method is frequently applied in the mining, building, and recycling sectors to get ready raw materials for additional processing or to make finished goods. Producing material particles appropriate for next processing stages, such grinding, screening, or conveying, is the main goal of crushing.

The type of material and the intended final product size determine which crushing equipment is used. The products in each case are generally similar and the difference between the equipment is associated mainly with the mechanical aspect of applying the

force to the various sizes of particles. When the particle is large, the energy required to break each particle down is high, even though the energy per unit mass is low. As the particle size decreases, the energy per unit mass rises more rapidly. Consequently, crushers have to be massive and structurally strong.

Primary crushing, which is usually the initial step in the crushing process, is accomplished by equipments including jaw crushers, cone crushers, impact crushers, and gyratory crushers. To meet the required product criteria, the particle size may be further reduced in secondary crushers. Secondary crushing involves recuperating the main crusher product, which is typically 0.5-2 cm in diameter, from ore storage to final disposal. Most metallic ores may be crushed and screened successfully. Secondary plants typically have one or two size-reduction stages with suitable crushers and screens and then to tertiary crushers. For slippery or rough ore, coarse grinding in rod mills can substitute tertiary crushing. In secondary crushing, extra-hard ore may require multiple size-reduction stages. In rare cases, penalty output must be decreased.

Depending on the product size, crushing can be done in an open or closed circuit.

In open circuit crushing, the material is sent through the crusher once without being screened or further treated to remove particles. The product size is defined by the crusher setting or the size of the primary crusher, and it is usually greater than the desired product size. After passing through the crusher, the oversize material (which is larger than the desired product size) is either returned to the crusher for further crushing or transported to another processing facility. Open circuit crushing is commonly employed with larger, coarser input materials where final product size control is less important.

Closed circuit crushing involves a screening process to separate the desired final product from oversized material that needs further crushing. The material that has been crushed to the desired size passes through the crusher once and then undergoes screening. The

screened oversize material is returned to the crusher for further reduction if needed, while the correctly sized product is discharged for use or further processing. Closed circuit crushing is commonly used to control product size and ensure uniformity, especially for finer product sizes required in applications such as aggregate production or finer mineral processing.

The qualities of the input material, the kind of crusher, the operating conditions, and the maintenance procedures all affect the final crushed material's quality and quantity, and they also affect the efficiency of the crushing process.

## **Classification Of Crushing**

### **1. Primary Crushing.**

The primary crushing stage involves breaking down raw materials mined from quarries or mines to manageable sizes. Typically, this process employs heavy-duty equipment such as jaw crushers or gyratory crushers. Jaw crushers use compressive force to break down materials by compressing them between a fixed and movable jaw plate (Smith, 2017). Gyratory crushers, on the other hand, employ a cone-shaped crushing head that rotates within an inverted, truncated cone. This motion compresses the material against the fixed concaves (Wills & Finch, 2016).

### **2. Secondary crushing.**

Secondary crushing refines the particle size of the material after it has been reduced to a desirable size in primary crushing. Cone crushers and impact crushers are frequently employed at this stage. Cone crushers compress material between an eccentrically rotating mantle and a concave bowl liner, therefore reducing material size (Mular et al., 2002). Impact crushers operate on the impact crushing concept, which involves striking

the material with fast moving hammers or blow bars, forcing it to break along its natural cleavage lines (Bengtsson and Evertsson, 2006).

### 3. Tertiary and Quaternary Crushing.

Tertiary and quaternary crushing stages are used to produce finer particle sizes and specified product forms. Finely crushed materials are produced using tertiary crushers such as vertical shaft impactors (VSI) and high-pressure grinding rollers (HPGR) (Saramak et al., 2019). VSI crushers work by throwing the material against a hard surface inside a crushing chamber, whereas HPGRs use compression and inter-particle breakage to reduce size (Napier-Munn et al., 1996).

### 4. Sorting and Screening

Following the crushing phases, the material may go through additional processing, like screening, to eliminate undesirable fines or separate different sized particles. For this reason, screening devices such as trommel or vibrating screens are frequently utilized (King, 2000). Furthermore, materials can be separated according to their density or magnetic characteristics using air classifiers or magnetic separators (Wills & Finch, 2016).

## **TYPES OF CRUSHING UNIT**

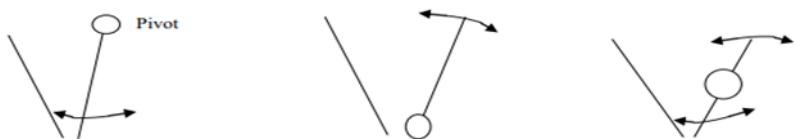
### **Jaw Crusher**

All jaw crushers feature two jaws: one of which is fixed while the other moves. The working principle of jaw crushers is based on the reciprocating movement of the movable jaw that compresses and crushes the rock or ore between itself and the fixed jaw, as the material enters the zone between the jaws.

The moving jaw moves back and forward against the fixed jaw, and material fed from the top of the machine is compressed between the two, breaking it into smaller pieces. As the

moving jaw moves away from the fixed jaw, the crushed material is discharged from the crusher at the bottom, with the size of the ejected material determined by the gap between the jaws.

Jaw crushers are generally defined by their opening size. For example, a 900x650 mm crusher refers to a crusher with an inlet measuring 900 mm by 650 mm. Jaw crushers are classified by the method of pivoting the swing jaw. In the Blake crusher the jaw is pivoted at the top and thus has a fixed receiving area and a variable discharge opening. In the Dodge crusher the jaw is pivoted at the bottom, giving it a variable feed area but fixed delivery area. The Dodge crusher is restricted to laboratory use, where close sizing is required, and is never used for heavy-duty crushing as it chokes very easily. The Universal crusher is pivoted in an intermediate position, and thus has a variable delivery and receiving area. (Wills and Napier-munn, 2006).



**Figure 41: Types of jaw crushers.**

Source: (Wills et al., 2006)

The Blake crusher was patented by W.E. Blake in 1858 and variations in detail on the basic form are found in most of the jaw crushers used today. (Wills and Napier-munn, 2006).

The blake crushers are further classified in single toggle and double toggle depending on the mechanism used for swinging the movable jaw.

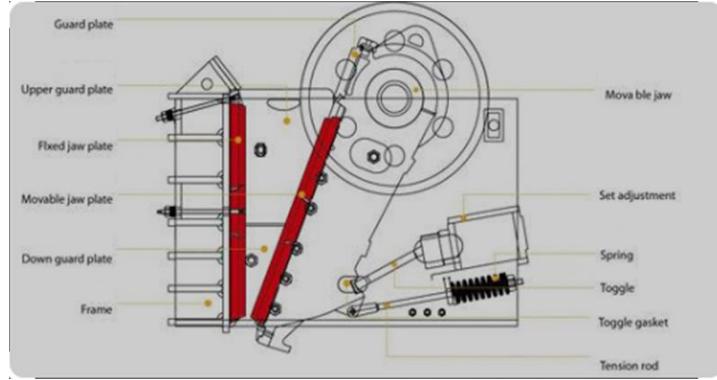
The single toggle swing jaw is suspended on the eccentric shaft, which allows a lighter, more compact design than with the double toggle crusher (Ciężkowski P, et al.). Moreover, the single toggle crusher is taking over most new applications due to lower

cost and higher capacity; hence the need to carry out a holistic review of the critical aspects of the machine that requires close attention during its design and operations stages; (C. Okechukwu, et al).

### **Overview of Crushing Process in A Single Toggle Jaw Crusher**

The feed supplied to a single toggle jaw crusher through its gape, is crushed by the compression of the feed by the movable jaw against the fixed or stationary jaw. Movement of the movable jaw is controlled at the top by an eccentric shaft, which is driven by a pulley whose weight is counter-balanced by a flywheel at the opposite end of the eccentric shaft. The flywheel stores energy on the idling half of the stroke and delivers it on the crushing half (Ciężkowski P, et al). Through alternate nipping and releasing of the feed, the jaws reduce the material in stages in the crushing chamber to sizes smaller than the discharge aperture or set, and the crushed product exits by gravity and with the aid of the movement of lower end of the movable jaw, which is controlled by toggle plate and drawback rod. The eccentric motion causes it to move in an elliptical route, which not only crushes the rock by moving horizontally towards the fixed jaw, but also helps to force the rock downhill by virtue of its vertical motion. As a result, it has a greater capacity than double toggle.

However, because the arrangement is pivoted on the shaft itself, it adds stress to the shaft and hence raises maintenance costs.



**Figure 42: A diagram of a single toggle jaw crusher**

### **Double toggle jaw crushers**

The double-toggle jaw crusher has a heavy flywheel attached to the drive, which is required to store energy on the idling half of the stroke and deliver it on the crushing half (Wills and

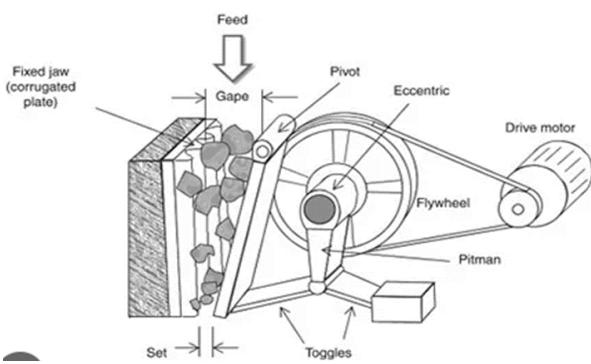
Napier, 2006). This crusher has a fixed jaw and a moving jaw pivoted at the top, with manganese steel crushing faces (Coulson and Richardson, 2002). The horizontal displacement of the swing jaw is largest at the bottom of the pitman cycle and gradually decreases over the rising half of the cycle as the angle between the pitman and the back toggle plate becomes less severe (Wills and Napier, 2006).

The crushing force is least at the beginning of the cycle, when the angle between the toggles is most acute, and greatest at the end, full power is supplied across a shortened travel of the jaw (Wills and Napier, 2006). Because the largest movement of the jaw is at the bottom, there is little chance of the machine clogging, however some uncrushed materials may slip through and have to be returned to the crusher (Coulson and Richardson, 2002). The double toggle is pushed along by the pitman arrangement.

#### Arrangements of swinging mechanism of double toggle jaw crusher

Moving on to the configuration of double toggle crushers in this model, the oscillating movement of the swinging jaw is caused by vertical movement of the pitman. This

fluctuates due to the eccentric's impact. As the pitman is pushed upward, the back-toggle plate causes it to move sideways. This motion is transmitted to the front toggle plate, causing the swing jaw to close on the fixed jaw. Similarly, downward movement of the pitman allows the swing jaw to open (Ayush Pandey, 2020).



**Figure 43: A diagram of a double toggle jaw crusher**

## GYRATORY CRUSHER

The gyratory crusher has a crushing head in the shape of a truncated cone positioned on a shaft, the upper end of which is retained in a flexible bearing while the lower end is pushed eccentrically to form a circle. The crushing action occurs throughout the cone, with the highest movement occurring at the bottom (J. F. RICHARDSON et al., 2002).

The process is comparable to that of a mortar and pestle used in chemistry laboratories.

A cone-shaped component (the pestle) revolves within a tapering bowl (the mortar). The motion of the cone-shaped component against the tapered part creates attrition because the gap between the two parts fluctuates, allowing the material to be trapped between the two sections. The gyratory crusher, like the jaw crusher, can accommodate boulders as large as 1.5 m (for the largest equipment). This crusher can handle items with Mohs hardness ranging from 3 to 5. As with jaw crushers, there is no mutual sliding on grinding surfaces, only rolling of the nut on concaves. The clothing is then uniform and minimal.

However, the gyratory crusher is sensitive to jamming if it is fed with a sticky or moist product loaded with fines. This inconvenience is less sensitive with a single-effect jaw crusher because the mutual sliding of grinding surfaces promotes the release of a product that adheres to surfaces. (Wills et al., 2006). Wear is maximum at the lower part because at this location compression forces are exerted on the entirety of grinding surfaces. The typical reduction ratio for gyratory crushers is bound to be between 6 and 8 and is similar to that of jaw equipment. Applying the crushing force by rolling the nut breaks flats and gives a product that is more “cubic” than the jaw crusher. The largest-scale equipment can treat a flow with a magnitude of 4,000 tons/h. With a production rate and flow that are equal, the power consumed by a gyratory crusher is similar to a jaw crusher.

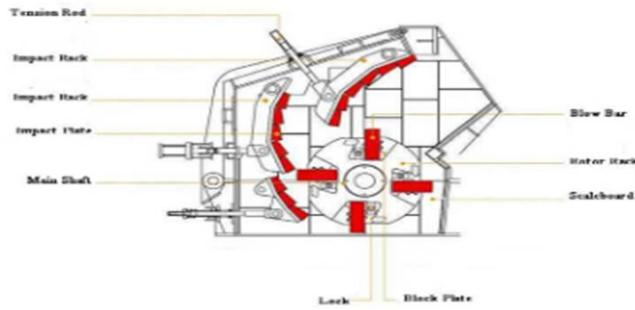


**Figure 44 : A diagram of a single toggle jaw crusher**

## **IMPACT CRUSHER**

In an impact crusher, rock is crushed using sharp strikes at high speeds, rather than compression. Beaters, which transfer kinetic energy to ore particles upon contact, are the moving parts. The internal stress caused by the particles is often large enough to shatter them. These forces are amplified by causing the particles to collide with an anvil or breaker plate. There is a significant difference between materials crushed by pressure and those crushed by impact. Materials shattered by pressure may have internal tensions that

induce cracking later, while an impact fracture leaves no residual tensions. The feed enters the crushing chamber via the hopper, where impact bars fastened to a spinning mechanism and impact plates attached to a dampening device work together. When a large rock enters the chamber, the plates retract and crush the rock into small particles.



**Figure 45: A diagram of an impact crusher**

### **8.3 EQUIPMENT SELECTION AND JUSTIFICATION**

A variety of factors determine which specific crusher is used for comminution. These variables provide you the freedom to select the kind of crusher that is required for that specific procedure. The parameters include the necessary product size, material to be crushed, amount of power used, production capacity, and more. The kind and composition of the material to be crushed affects the crusher that is being used. For example, a material's hardness influences the machine's wear and power consumption. When the material is abrasive and hard, a low-speed crusher must be used (Coulson and Richardson, 2002). 20,000 kg of feed must be handled per hour in this process.

## **Characteristics of the types of crushers**

<b>Crusher type</b>	<b>Size of feed (mm)</b>	<b>Size of product(mm)</b>	<b>Reduction ratio</b>	<b>Capacity (tons/h)</b>	<b>Power consumption(kW)</b>
Jaw	100 – 1000	25 – 100	8	10 -10000	5 -200
	200 – 2000	25 – 250	8	100 - 500	100 – 700
Impact	50 – 300	1 – 10	40	10 - 100	100 – 2000
Cone	50 – 300	5 – 50	8	10 – 1000	20 – 250

(Source: (Couper, 2012)

### **Advantages of a Jaw Crusher**

Many factors make laboratory jaw crushers the optimum selection for effective and efficient primary crushing of hard, brittle, rock-like materials.

Jaw plates are available in a variety of material types to optimize performance with different materials. Models have throughput capacities from 50lb to 4,000lb (22kg to 1,800kg) per hour for volumes from small samples to pilot-plant scale production.

A broad range between maximum size (top size) capacity and final particle size of the processed sample means fewer steps for complete reduction. Adjustable spacing between jaws regulates the final sample size and compensates for wear. Reversible jaw plates extend service life between replacements. The proven technology provides consistent, repeatable results. Jaw crushers feature safe, simple operation and maintenance.

### **Disadvantages of a Jaw Crusher**

Jaw crushers can also have some disadvantages compared to other types of size reduction.

Opposing jaw plates are less efficient at reducing materials that are tough, ductile, or soft instead of hard and brittle.

The fine sizes are generally coarser than those produced with pulverizers, hammer mills, and other types of crushers.

The final particle size range is not as tightly controlled as other crusher types.

### **Selection Of the Jaw Crusher**

According to the above table, the jaw and gyratory crushers can both produce products with comparable sizes and reduction ratios. The gyratory crusher's minimum capacity is 100 tons/h, which is more than the 10 tons/h that is anticipated of it. Consequently, a jaw crusher is advised.

The jaw crusher can handle feed materials that are sticky, hard, and abrasive. It has been found that gyratory crushers perform especially well with hard, abrasive material. The maximum size of ore that the crusher must process and the necessary capacity are the main determinants of whether a jaw or gyratory crusher should be used in this comminution process. Gyratory crushers are typically used when a large capacity is required.

Jaw crushers are commonly used when the crusher gape is more important than the capacity. If a large capacity is required, a gyratory is the best option. However, a jaw crusher is used in the process because of its low power consumption, which makes it more economical, if both the capacity and the gape are important factors.

The cost of buying and maintaining a jaw crusher is slightly lower than that of a gyratory crusher. In contrast to the more intricate and costly gyratory crusher, the jaw crusher has become more and more popular in the crushing industry because of its simple design,

ease of use, and convenient maintenance. The necessity for transportation to remote areas further suggests that installing jaw crushers might be beneficial.

It is clear from the discussion above that the jaw crusher would be an appropriate piece of crushing machinery to use for the spodumene ore. However, since it was used for this operation and has a relatively simple structural layout that makes it easy to manufacture and maintain, a single toggle jaw crusher is advised.

Additionally, it uses the least amount of labor and power when operating. Double-toggle crushers are typically used on hard, abrasive materials and are approximately 50% more expensive than single-toggle crushers of the same size. (Wills and Napier, 2006).

#### **8.4 EQUIPMENT DESCRIPTION**

When designing and operating jaw crushers, many parameters must be considered to ensure efficient performance. Important aspects to consider are feed material, angle of nip, jaws, pitman, eccentric shaft, toggle plate, drawback rod, cheek plates, bearing, crusher frame, pulley, and flywheel.

##### **Nature of feed material**

Compression-based crushers are ideal for hard, brittle, and abrasive rocks.

Rock fracture occurs when its tensile strength exceeds a certain threshold. Rock strength decreases due to imperfections and cracks acting as stress concentrators, as well as moisture influencing mineral grains.

Physical properties of materials such as: moisture content, structure, friability, density, hardness and crushing strength are important design criteria, as these affect both the life of the liners and power requirement. Also, the greater the strength of a rock, the higher the crushing time under the influence of a crusher; this implies more wear to the crusher jaws.

## **Crusher Jaw Plate**

In a jaw crusher, materials are compressed when the movable jaw presses against the stationary jaw. The jaws can be flat or corrugated. Crusher jaws were originally made of white cast iron and later replaced with high manganese austenitic steel, also known as Hadfield steel, which is the primary wear material for the jaws. The wear on these components increases as the feed lump reduces and moves towards the discharge. The lower edges of the jaws experience the most crushing and wear, as they receive the most movement from the movable jaw. These plates are reversible, allowing the worn end to become the upper end. This reduces the cost of replacing worn jaws. Variations in wear on the fixed jaw lead to varying surface hardness levels. Hardfacing techniques are more effective than casting for achieving this goal.

The fixed jaw is bolted to a support plate, while the movable jaw is bolted to the pitman. This allows for easy dismantling when worn out, replaced, or inverted.

## **The Pitman**

The pitman's upper end is designed to accommodate the eccentric shaft. This structure houses the eccentric lobe and supports the movable jaw. The lower end of the pitman is guided by the toggle plate and drawback rod attached to it. In comparison to a pitman without such support, a pitman with a cross-sectional support in the shape of a honeycomb structure lessens or eliminates pitman bending and wear. When using an open-structured pitman, there are additional benefits such as fewer holes caused by casting, less energy needed from the flywheel, less material needed to produce the pitman, and the ability to crush material with fewer strokes and shorter stroke lengths. The cross-sectional supports also prevent bending and distortion horizontally.

## **Toggle plates**

The toggle plate is used to hold the lower part of the jaw in position; this depends on the desired product size. Toggles are made to be easily adjusted in order to attain the correct discharge setting and to remove uncrushable objects like tramp irons.

Toggle adjustment mechanisms comprise of shims and hydraulic cylinders that facilitate effortless adjustment of the discharge setting by moving the toggle block to the desired setting; additionally, remote controlled electromechanical actuation mechanism is possible. The spring relief mechanism relieves strain on the jaws when tramp iron lodges between the jaws.

## **Retraction spring and screw rod**

The screw rod is fastened to the lower end of the movable jaw or the pitman bearing the movable jaw and has a spring on the other end. The rod-spring subassembly retrieves the moveable jaw from its maximum range of motion. The retraction screw rod system keeps the pitman securely attached to the toggle plate. It is intended to fail before any damage to the crusher body, pitman, or shaft happens. The toggle seats are the fixed points at which the toggle plate makes contact with the pitman and toggle beam.

## **Pulley and Flywheel**

The weights of these two machine elements need to be balanced as any deviation may lead to undesired twisting of the eccentric shaft and increased vibration. They are firmly keyed to the opposite ends of the eccentric shaft. Usually, they are made of gray cast iron because of its good vibration damping, machinability and resistance to sliding wear. The pulley is powered by belts that are fastened to the prime mover, which could be either an electric motor or a combustion engine. The pulley has two or more grooves. The flywheel acts as a reservoir, storing energy when the supply exceeds the requirement and releasing

it when the demand for energy exceeds the supply, which provides the system's moment of inertia. Therefore, the flywheel provides the inertia needed to crush a material in a jaw crusher. Gorman, D. G. and ennedy, W. "Applied solid dynamics", Butterworth and Co. (Publishers) Ltd, London, 1988.

## **Bearing**

The eccentric shaft normally has four bearings: two on each side of the jaw frame that supports the shaft and two at either end of the pitman. Typically, these bearings have rollers. On an overhead eccentric jaw crusher, spherical tapered roller bearings typically support the main shaft.

## **Cheek Plates**

The cheek plates are positioned on the left and right ends of the crushing chamber to prevent the material being crushed from reaching the frame of the crusher, which will lead to the wear of the frame. The cheek plates are also made of manganese steel; materials such as white cast iron and hardfaced steel can be used since the impact on the side plates is minimal compared to the stationary jaw.

## Eccentric Shaft

The eccentric shaft rotates when the pulley is operating, which results in an elliptical movement of the movable jaw. By increasing the eccentric shaft's stroke and decreasing the speed without increasing the crushing force through wider jaws, one can increase throughput capacity without expanding the jaw crusher's physical dimensions. Increased eccentricity of the shaft also results in increased throw. Increased throw also has the benefit of lowering machine loads while maintaining the crusher's structural design. There is a massive flywheel at each end of the revolving main shaft. Its movable jaw moves in and out due to its eccentric shape. Eccentric shafts are made of alloy steel.

The pitman is propelled into motion by the oscillation of an eccentricity on a shaft that runs the length of it. They are made of larger, stronger steel because this force is also applied to the shaft. There are anti-friction bearings fitted. The eccentric shaft is rotated by the flywheel, pulling the arm upward and moving the toggle plates. As a result, the materials are crushed as the swing jaw is forced outward and toward the stationary jaw.

### **Angle of Nip**

The jaws are set at an acute angle to each other. This angle commonly known as the “angle of nip”. This angle is responsible for slipping effect, when the angle is larger it reduces the capacity of the equipment.

Exceeding the maximum angle causes slipping from the machine, while operating below the desired range leads to the production of undesirable dust and fines.

### **Crusher Body**

This is the component of the crusher that is designed to exhibit resistance to bending stress even when the material being crushed is hard. It serves as the supporting structure and strong foundation during the crushing operation

## **OPERATION OF THE JAW CRUSHER**

The jaw crusher's operation is based on contemporary design. Two jaws—one fixed and the other moving—make up the crushing machine. The gaps between the jaws are wider at the top and smaller at the bottom. The moving jaw swings on the center pin while the pitman moves on an eccentric shaft on a bearing. The stone is crushed by mechanical pressure as it feeds between two jaws. The belt pulley is driven by the AC motor, which also drives the eccentric shaft. This causes the moving jaw to approach and periodically depart from the fixed jaw shaft rotation, crushing, rubbing, and grinding the material sand

as it gradually descends and eventually discharges from the discharge opening of the desired dimension.

The rock or ore is fed into the crusher at the maximum opening, or gape, where the jaws are farther apart. The ore is crushed and falls into the crushing chamber when the jaws make contact. Until the particles have a size smaller than the bottom opening passage, the process is repeated. For smaller crushers, springs are used to restore the jaw from its farthest point of travel; for larger crushers, a pitman is used. Both kinds of crushers use heavy flywheels in addition to the eccentric shaft. For the moving jaws to reciprocate smoothly

### **MATERIAL OF CONSTRUCTION FOR JAW CRUSHER**

Single toggle jaw crushers come in both heavy and light versions, with housings made of welded steel, moving jaws made of cast or welded steel, and eccentric shafts made of premium forged steel. Double toggle jaw crushers come in two different housing options: light or heavy, made of welded steel. They also have a moving jaw and a pulling rod that can be made of cast or welded steel, as well as an eccentric shaft and an axis made of premium forged steel. The highly wear-resistant austenitic manganese steel casting crushing plates are designed so that turning them is simple once the clamping components have been loosened. The wear-resistant material used to make the readily replaceable side wedges is sheet steel or steel casting.

### **MAINTENANCE OF A JAW CRUSHER**

Cleanliness is key when it comes to maintenance of every equipment. Keep the toggle area clean, particularly before any crusher adjustments. Also, remove dirt and debris from crusher frame surfaces and from areas around the machine. Monitor discharge chutes for any obstructions and check for any material buildup under the jaw to avoid excessive wear on the pitman.

Conducting of regular inspections helps avoid future problems with the equipment. Check jaw dies daily, making sure bolts are kept tight. Check cheek plates for any excessive wear that may affect the mainframe of the jaw. Monitor flywheels for cleanliness, tightened bolts and potential cracks; and make sure drive belts aren't worn or cracked. Monitor operations and logging of data. Check all indicators and switches daily for proper operation – as they are integral to proper shutdown in the event of issues such as high temperatures or low hydraulic pressures.

## **8.5 CHEMICAL ENGINEERING DESIGN CALCULATION**

The parameters to be considered to specify the design of this jaw crusher include the set, gape, height of jaw, width of jaw, throw, angle of nip, critical speed and many others.

### **The Gape**

The crusher gape is typically the distance between the stationary and movable jaw plates of the jaw crusher. The largest particle is considered when sizing the gape.

#### Assumptions

- Assuming a particle size of 35 cm – 55 cm
- Assuming 85% passing

The following relation applies:

$$\text{Largest particle size} = 0.85 \times \text{Gape} \quad (\text{Gupta and Yan, D, 2006})$$

With the assumed particle size, the largest particle size is 55 cm

$$55 \text{ cm} = 0.85 \times \text{gape}$$

$$\text{Gape} = 64.706 \text{ cm}$$

### **The Set**

The opening at the discharge end of the jaws is referred to as the set. The reduction ratio of jaw crushers are usually between 8:1 and 6:1.

### Assumptions

- Consider a reduction ratio of 6:1

The set, which is the minimum discharge opening of the jaw crusher is given by:

Where, G= the gape

R = reduction ratio = 4:1/9:1, and an average reduction ratio of 6:1 (Adam Orner, 2021)

$$S = 10.7843 \text{ cm}$$

### The Width of the Jaw

The width of the jaw crusher and the gape are related by the equation:

$$1.3G < W < 3G \text{ (Gupta and Yan, D, 2006)}$$

Where G = gape = 64.706 cm

Setting the width to be 1.4 x the gape,

$$W = 1.4 \times 64.706 \text{ cm}$$

$$W = 90.588 \text{ cm}$$

### Vertical Height Of The Crusher

Vertical height of crusher = 2 x Gape (Gupta and Yan, D., 2006)

Where gape =64.706 cm

$$H = 2 \times 64.706 \text{ cm}$$

$$H = 129.412 \text{ cm}$$

### Throw; Length of stroke

$$LT = 0.0502(G)0.85 \quad (\text{Gupta and Yan, D, 2006})$$

Where  $G = \text{gape} = 64.706 \text{ cm}$

$$\text{Throw} = 0.0502(0.64706)0.85$$

$$LT = 3.47 \text{ cm}$$

$$L_{\max} = LT + L_{\min}$$

$$= 3.47 \text{ cm} + 10.78 \text{ cm}$$

$$= 14.25 \text{ cm}$$

Area of Operation between plates

$$A_{op} = G \times W$$

$$= 5861.5871 \text{ cm}^2$$

Critical Velocity

$$V_c = 47 \frac{1}{\sqrt{L_t}} \left( \sqrt{\frac{R-1}{R}} \right)$$

$$V_c = 230.3257 \text{ rev/min}$$

### Angle of Nip

$$V_c = \frac{66.6}{(\sqrt{\frac{L_t}{\tan \alpha}})} \tan \alpha = 0.4227$$

$$\alpha = 22.9^\circ$$

Distance a particle falls through after a half cycle

$$h = \frac{L_t}{\tan \alpha}$$

$$h = 8.2146 \text{ cm}$$

Capacity of the jaw crusher = volumetric flow rate of the discharge

**Table 18: Mass and Density of components**

Component	Mass flow rate, kg/h	Density, kg/m <sup>3</sup>
$\alpha$ -spodumene	16000	3270
Quartz	2000	2650
Alumina	500	3970
Annite	500	3100
Phlogopite	1000	2700
Total	20000	

Average density = 3171.94 kg/m<sup>3</sup> = 3.17194 tons/m<sup>3</sup>

$$\text{Volumetric flow rate} = \frac{833.33 \text{ kg/h}}{3171.94 \text{ kg/m}^3} = 0.2627 \text{ m}^3/\text{h}$$

$$\text{Volumetric flow rate} = 0.2627 \text{ kg/hr} = 6.3053 \text{ kg/h}$$

Therefore the jaw crusher capacity (QF) = 0.2627 kg/hr

Plate Frequency

$$\text{Coefficient of fluctuation of speed, } Cs = \frac{V_c - V}{V_{avg}}$$

Coefficient of fluctuation of speed for crushing equipment = 0.2 (reference on phone)

Where V = plate frequency

$$L_{min} = set = 10.784 \text{ cm}$$

$$W = \text{width of jaw} = 90.5883 \text{ cm}$$

$$V = 188.4484 \text{ rev/ min}$$

Maximum Capacity of the Crusher

$$Q_m = 2820 \sqrt{L_t} W (2L_{min} + L_t) \left( \sqrt{\frac{R-1}{R}} \right) \rho_s f(P_k) f(\beta) S_c$$

Where;

$\rho_s$  = Density of the rock

$f(P_k)$  = Function of the packing characteristic

$f(\beta) = 1$

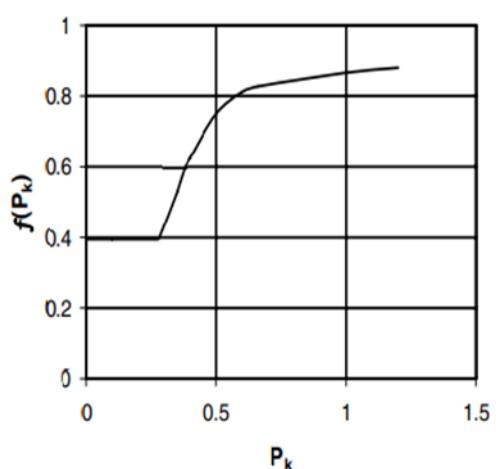
$S_c$  = Surface characteristic of the rock

$P_k = (d_{max} - d_{min}) / d_{mean}$

$$= 0.4444$$

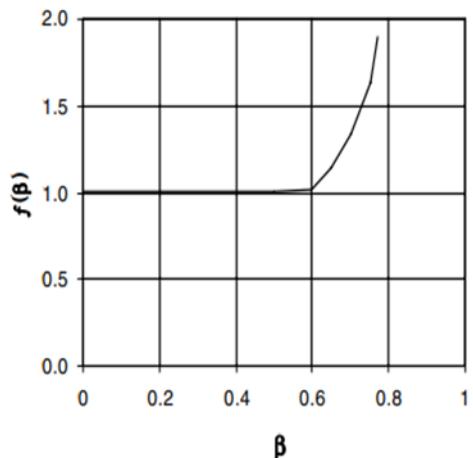
$\beta = (\text{Set} / \text{Mean feed size})$

$$= 0.2397$$



$$f(P_k) = 0.6$$

$$f(\beta) = 1$$



Hence the maximum capacity,

$$Q_{\max} = 90.6758 \text{ tonnes/hour}$$

Actual Capacity

$$Q_a = Q_m \cdot \frac{V}{V_c}$$

$$Q_A = 74.1893 \text{ tonnes/ hour}$$

### Maximum Power

Summary of chemical engineering calculations

PARAMETER	VALUE
Crusher type	Single toggle
Gape	64.71 cm
Set	10.78 cm
Width	90.59 cm
Height	129.41 cm
Throw	3.47 cm
Angle of nip	22.9°
Area of operation	66
Critical frequency	230.33 rpm
Plate frequency	188.45 rpm
Height particles fall after half a cycle	8.21 cm

Actual capacity	74.19 tonnes/ hour
Maximum capacity	90.68 tonnes/ hour
Maximum power	63.21 kW
Actual power	13.53 kW
Efficiency	21.4 %

## 8.6 MECHANICAL ENGINEERING DESIGN

Flywheel

Coefficient of fluctuation speed

$$C_s = \frac{N_1 - N_2}{N}$$

Where  $N_1$  = Maximum speed = 230.3257 rpm

$N_2$  = Minimum speed = 188.4484 rpm

$N$  = mean speed

$$\therefore C_s = 0.2$$

Coefficient of fluctuation of energy (CE)

$$CE = \frac{\text{Maximum fluctuation of energy}}{\text{Work done per cycle}}$$

$$\text{work done per cycle} = \frac{63.205 \times 60}{N}$$

where  $P$  = maximum power = 63.205 kW

work done per cycle = 18.1 kJ

CE = 0.4 for crushing machines (Khurmi and Gupta, 2005)

Maximum fluctuation of energy = 7.24 kJ

### Moment of Inertia of the Flywheel

$$I_g = \frac{(\Delta E) \times K}{\omega^2 C_s}$$

$$\omega_{ave} = \frac{2\pi \times N}{60}$$

where  $\omega$  = mean angular speed

N = average mean speed of the flywheel = 209.38705 rpm

$\omega = 21.93 \text{ rad/s}$

K = 0.9 (Bhandari, 1953)

Ig = moment of inertia of the flywheel

Cs = 0.2

$(\Delta E) = 7.24 \text{ KJ}$

$\therefore I_g = 67.7445 \text{ kgm}^2$

Total energy stored (E)

$\Delta E = 2E \cdot CS$

Where  $\Delta E$  = maximum fluctuation of energy

E = total energy stored = 18.1 kJ

Mean Diameter and Mean Radius of the Flywheel.

D = 33 cm

R = 16.5 cm

### Mass of the Flywheel

$$MF = \frac{I_g}{10 \times R^2}$$

Where,

$$I_g = \text{Moment of inertia} = 67.74 \text{ kgm}^2$$

$$R = \text{Mean radius} = 16.5 \text{ cm}$$

$$\therefore MF = \text{Mass of the flywheel} = 264.61 \text{ kg}$$

### Thickness and Width of the Rim

For flywheels

$$\text{Width (BR)} = 2 \times tr \text{ (thickness of the rim)} \text{ (V.B Bhandari, n.d.)}$$

$$MF = \pi R \left( \frac{B_r}{1000} \right) \times \left( \frac{t_r}{1000} \right) \times \varphi$$

Where,

$$R = \text{mean radius}$$

$$\varphi = \text{density of the material of construction (cast iron)} = 7250 \text{ kg/m}^3$$

$$\therefore tr = \text{thickness of the rim} = 13.27 \text{ cm}$$

$$Br = \text{Width of the rim} = 26.54 \text{ cm}$$

### Linear Velocity (peripheral velocity)

$$Z = \omega_{ave} \times R$$

$$\therefore Z = \text{linear velocity} = 3.62 \text{ m/s}$$

### Centrifugal Force Acting on the Flywheel

$$FC = \frac{2 \times \varphi \times B_R \times t_r \times R^2 \times \omega_{ave}^2}{g} \text{ (Lingaiah, 2003)}$$

Where,

$$g = \text{acceleration due to gravity} = 9.81 \text{ m/s}^2$$

$$\therefore FC = \text{centrifugal force} = 681.58 \text{ N}$$

### **Tangential Force Acting on the Flywheel**

$$FT = \frac{\varphi \times B_R \times t_r \times Z^2}{g}$$

$$\therefore FT = 341.08 \text{ N}$$

Centrifugal Tensile Stress  $\sigma_c$

$$\sigma_c = \frac{0.01095 \times \varphi \times R^2 \times N^2}{g} \quad (\text{Lingaiah, 2003})$$

$$= 9659.41 \text{ Pa}$$

### **Bending Stress**

$$\sigma_B = \frac{0.2146 \times \varphi \times R^3 \times N^2}{g \times t_r}$$

$$= 0.24 \text{ MP}$$

### **Mean torque (T mean) by the flywheel**

$$T_{\text{mean}} = \frac{P_{\text{max}} \times 60}{2\pi N} \quad (\text{Khurmi and Gupta, 2005})$$

$$T_{\text{mean}} = 2882.5244 \text{ Nm}$$

### **Maximum Torque of the Flywheel**

$$\Delta E = P_{\text{MAX}} \left( \frac{T_{\text{max}} - T_{\text{mean}}}{T_{\text{max}}} \right)^2$$

Where,

$P_{MAX}$  = Power transmitted = 63.205 kW

$\Delta E$  = Maximum Fluctuation of Energy = 7.24 kJ

$T_{MEAN}$  = Mean torque = 2882.52 Nm

$\therefore T_{max}$  = Maximum torque = 4356.86 Nm

## **DESIGN OF THE SHAFT AND KEY**

### **Length of the Shaft**

$L_s$  = 100cm

### **Torsional Moment under Fluctuating Load**

Mass of flywheel = 264.61 kg

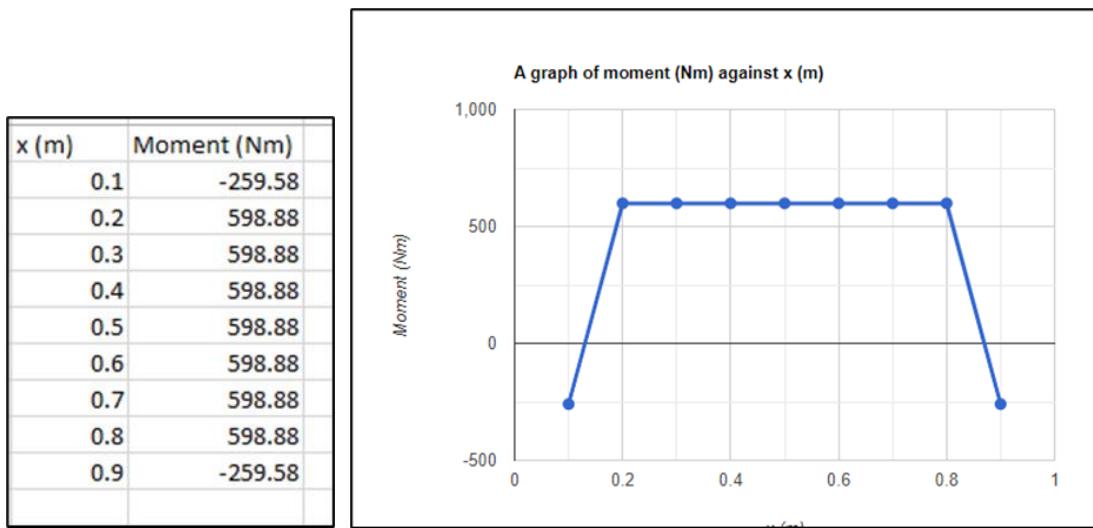
Mass of Jaw Plate = 875.09 kg

Reactions of the support from equality of action and reaction forces

$$R_s = 2595.82 + 8584.63 = 11180.45 \text{ N}$$

Using the singularity method;

$$M_x = -2595.82(x - 0) + R_s(x - 0.1) - 8584.63(x - 0.2) - 8584.63(x - 0.8) + R_s(x - 0.9) - 2595.82(x - 1)$$



**Figure 46: Moment Distribution**

$\therefore M_x = \text{maximum torsion in the shaft} = 598.88 \text{ Nm}$

### Thickness of the Shaft

$$d^3 = \frac{16}{\pi \tau_d} \sqrt{k_b T_w^2 + k_t M_x^2} \quad (\text{ASME}) \quad (\text{Lingaiah, 2003})$$

Where,

$T_w = \text{Torque} = 2882.52 \text{ Nm}$

$M_x = \text{maximum torsion in the shaft} = 598.88 \text{ Nm}$

$\tau_d = \text{Design stress} = 115.5 \text{ MPa}$

$K_b = \text{Combined shock and fatigue factor applied to the bending moment} = 2$

$K_t = \text{Combined shock and fatigue factor applied to the torsional moment} = 1.5$

$\therefore d = \text{thickness of the shaft} = 56.74 \text{ mm}$

### Design of the Key

Table 4 below, with a shaft diameter of 56mm

Width of the key (Wk) = 16mm

Thickness or height (Tk) = 10mm

### **Length of the key**

Lk = 1.5d (Lingaiah, 2003)

Where,

d = thickness of the shaft = 56.74 mm

∴ Lk = length of the key = 85.10 mm

### **Design of the Hub**

#### **Diameter and length of the Hub**

Dh = 2d              Lh = 2.5d        (Lingaiah, 2003)

∴ Dh = 113.48 mm

Lh = 141.85 mm

### **DESIGN OF THE SPRING**

#### **Spring Index (Ks)**

For industrial applications, KS lies between 4 and 16 (V.B Bhandari, n.d.)

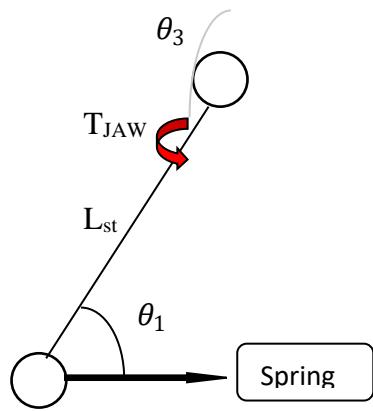
KS = 5 was chosen to minimize spring size

Spring Factor

$$K = \frac{2}{K_S^{0.25}}$$

∴ K = Spring factor = 1.34

Load on the Spring ( WS )



free body diagram illustration on the Spring

From Table 7,  $\theta_1$  was determined to be  $19.71^\circ$

$$W_s = T_{JAW} \cos(\theta_1)$$

$$\therefore W_s = 59495.30 \text{ N}$$

Wire Diameter

$$\epsilon = \frac{K \times 8 \times W_s \times K_s}{\pi \times d_w^2}$$

where,

$$\epsilon = \text{Design shear stress} = 238.3 \text{ MPa}$$

$$\therefore dW = 65.27 \text{ mm}$$

Mean Diameter of the Spring

$$DS = KS \times dW$$

$$\therefore DS = 326.35 \text{ mm}$$

Deflection of the Spring

$$\delta = \frac{8 \times W_s \times D_s^3 \times n}{G d_W}$$

$G$  = Modulus of Rigidity for steel wires

n= number of active coils= 6

∴  $\delta$  =Deflection of the spring = 60.36 mm

$N_t = (n+2) = 8$  coils

Total gap under Action of Maximum

$$\delta_g = 0.15 \times \delta$$

∴ $\delta_g$ =total gap under action of maximum load=10 mm

### Solid Length

$$L_s = N_t \times d_W$$

∴ LS = solid length = 522.16 mm

### Free Length

$$L_F = L_s + \delta_g + \delta \text{ (V.B Bhandari, n.d.)}$$

∴ LF = Free length= 592.52 mm

Pitch of the Spring Coil

$$P_{th} = \frac{L_F}{(N_t - 1)}$$

∴  $P_{th} = 84.65$  mm

### Energy Stored

$$U = \frac{1}{2} W_s \times \delta \text{ (V.B Bhandari, n.d.)}$$

∴ U= Energy stored = 1.7956 kJ

Mean Torque in the Swing Jaw (bending Moment of the Jaw)

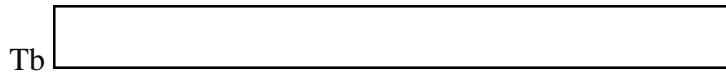
$$39 \times T_{fw} \leq T_{jaw} \leq 60 \times T_{fw} \text{ (ABUBAKAR MOHAMED, n.d.)}$$

$$T_{JAW} = 60 \times T_{FW}$$

$\therefore T_{JAW}$  = bending moment of the Swing Jaw = 172951.464 Nm

Thickness of the Jaw

$$0.9059\text{m}$$



Cross section of the jaw plate

$$\delta y = \frac{12 \times T_{JAW}}{W \times t_b^2} \quad (\text{ABUBAKAR MOHAMED, n. d.})$$

Where,

W = width of the swing jaw = 0.9059 m

$\delta y$  = yield strength of jaw material (manganese steel) = 250 MPa

$T_{JAW}$  = Bending moment of the jaw = 172951.464 Nm

$\therefore t_b$  = thickness of the jaw = 0.0957 m

Mass of the Jaw

Mass of the jaw ( $M_{jaw}$ ) = density(material of construction)  $\times$  volume of the jaw

$$M_{jaw} = \rho \times W \times t_b \times H_v$$

Where ;

$H_v$  = vertical height of the jaw = 1.2941 m

$t_b$  = thickness of the jaw = 0.0957 m

$\rho$  = density of cast steel = 7800 kg/m<sup>3</sup>

W = width of the swing jaw = 0.9059 m

$\therefore M_{jaw} = \text{Mass of the Jaw} = 875.09 \text{ kg/m}^3$

## DESIGN OF THE V-BELT

Diameter of the Larger Pulley

$$D_p = \frac{N_d}{N_D} \times d_p \quad (\text{V.B Bhandari, n.d.})$$

Where,

$N_D$  = velocity of the larger pulley = 230 RPM

$sN_d$  = velocity of the smaller pulley = 920 RPM

$d_p$  = Diameter of the smaller pulley = 75mm (Table 6, B section)

$\therefore D_p$  = Diameter of the larger pulley = 330 mm

$D_p$  of 330mm was chosen to meet standard.

The Center Length of Belt

$$CL = 2D_p$$

Where,

$D_p$  = Diameter of the larger pulley = 330 mm

$\therefore CL$  = Center length = 660 mm

Length of the Belt

$$L_B = 2 CL + \frac{\pi(D_p + d_p)}{2} + \left( \frac{(D_p - d_p)}{4CL} \right)^2$$

$\therefore L_B$  = Length of the belt = 1956.18 mm to meet standard

**Arc of Contact for the Smaller Pulley**

$$\theta_s = 180 - 2 \sin^{-1} \left( \frac{D_p - d_p}{2C_L} \right)$$

$$\therefore \theta_s = 134.54^\circ = 2.35 \text{ radians}$$

### Arc of Contact for the Larger Pulley

$$\theta_{LP} = 180 + 2 \sin^{-1} \left( \frac{D_p - d_p}{2C_L} \right)$$

$$\therefore \theta_{LP} = 225.46^\circ = 3.94 \text{ radians}$$

Mass of the Belts

$$M_B = A_B \times L_B \times \rho_B$$

$$L_B = \text{Length of the belt} = 1956.18 \text{ mm}$$

$$A_B = \text{Area of the belt} = 225 \text{ mm}^2$$

$$\rho_B = \text{Density of material of construction} = 1100 \text{ kg/m}^3$$

$$\therefore M_B = 0.48 \text{ kg}$$

Velocity of the Belt

$$V_B = \frac{\pi \times D_p \times N_D}{60}$$

$$\therefore V_B = \text{velocity of the belt} = 3.97 \text{ m/s}$$

Dimensions of standard key size with a given shaft diameter

<i>Shaft diameter</i>	<i>Key size</i>	<i>Keyway depth</i>	
<i>Above</i>	<i>Up to and including</i>	<i>b × h</i>	
6	8	2 × 2	1.2
8	10	3 × 3	1.8
10	12	4 × 4	2.5
12	17	5 × 5	3.0
17	22	6 × 6	3.5
22	30	8 × 7	4.0
30	38	10 × 8	5.0
38	44	12 × 8	5.0
44	50	14 × 9	5.5
50	58	16 × 10	6.0
58	65	18 × 11	7.0
65	75	20 × 12	7.5
75	85	22 × 14	9.0
—	—	—	—

Source: (V.B Bhandari, n.d.)

Dimensions of standard cross-sections for V- belts

<i>Belt section</i>	<i>Pitch width <math>W_p</math> (mm)</i>	<i>Nomi- nal top width <math>W</math>(mm)</i>	<i>Nomi- nal Height <math>T</math>(mm)</i>	<i>Recom- mended Mini- mum pitch diameter of pulley (mm)</i>	<i>Permis- sible Minimum pitch diameter of pulley (mm)</i>
Z	8.5	10	6	85	50
A	11	13	8	125	75
B	14	17	11	200	125
C	19	22	14	315	200
D	27	32	19	500	355
E	32	38	23	630	500

Source: (V.B Bhandari, n.d.)

Values of shock and fatigue factors  $k_b$  and  $k_t$  for shaft

<i>Application</i>	<i><math>k_b</math></i>	<i><math>k_t</math></i>
(i) Load gradually applied	1.5	1.0
(ii) Load suddenly applied (minor shock)	1.5–2.0	1.0–1.5
(iii) Load suddenly applied (heavy shock)	2.0–3.0	1.5–3.0

Source: (V.B Bhandari, n.d.)

## Crank angle, coupler angle and normalized torque relationship

$\theta_2$	$\theta_3$	$T_3/T_2$
0	159.405	-109.204
15	159.704	-411.192
18	159.763	-970.001
21	159.821	2636.133
24	159.88	556.854
27	159.938	311.191
30	159.995	216.07
45	160.265	86.875
60	160.493	56.547
75	160.669	44.128
90	160.779	38.406
105	160.817	36.301
120	160.824	36.849
135	160.777	40.276
150	160.475	48.309
165	160.228	65.117

$\theta_2$	$\theta_3$	$T_3/T_2$
195	159.622	406.292
198	159.559	896.376
201	159.496	-4421.31
204	159.432	-640.993
207	159.37	-346.713
210	159.308	-238.418
225	159.018	-95.825
240	158.772	-62.876
255	158.592	-49.258
270	158.488	-42.779
285	158.466	-40.092
300	158.528	-40.184
315	158.666	-43.163
330	158.86	-50.384
345	159.121	-66.283

Source: (V.B Bhandari, n.d.)

## Summary of mechanical engineering calculations

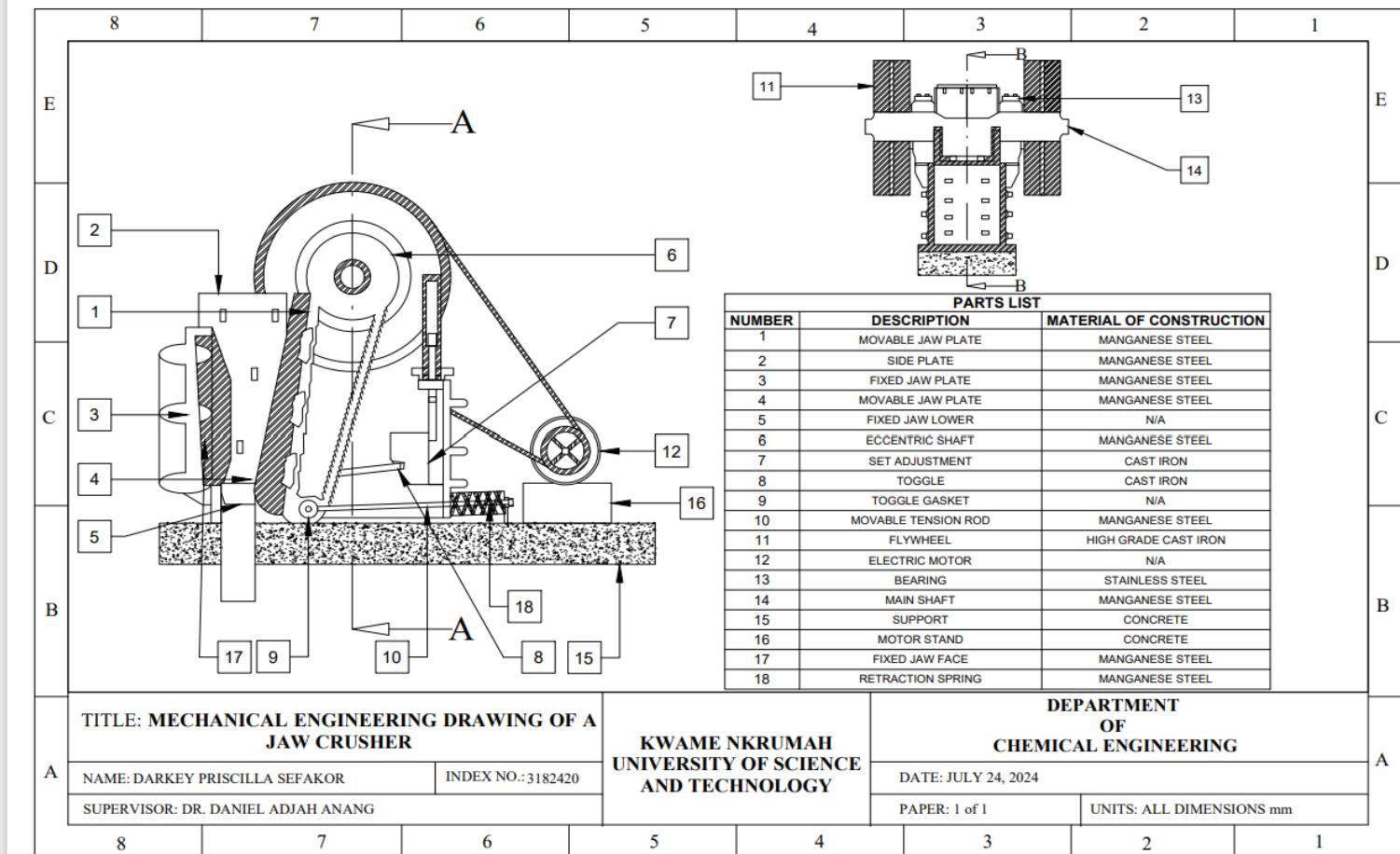
Parameter	Value
<b>Flywheel</b>	
Coefficient of fluctuation of speed	0.2
Coefficient of fluctuation of energy	0.4
Total energy stored in the flywheel	18.1 kJ
Energy of flywheel rim	3488.095 Nm
Mean torque transmitted by the flywheel	2882.52 Nm
Mass of the flywheel	6
Width of the rim	26.54 cm

Thickness of the rim	13.27 cm
Diameter of the flywheel	33 cm
Peripheral velocity of the flywheel	3.62 m/s
Centrifugal acting on the flywheel	681.58 N
Diameter of the hub	14.19 cm
Length of the hub	11.35 cm
Material of construction	Cast iron

<b>Spring</b>	
Spring index	5
Diameter of the wire	6.53 cm
Mean diameter of the spring coil	32.64 cm
Load on the spring	59495.30 N
Deflection of the spring	6.04 cm
Free length of the spring	59.25 cm
Pitch of the coil	8.47 cm
Energy stored by the spring	1.80 kJ
<b>V- belt and pulley</b>	
Diameter of large pulley	33 cm

Smaller pulley diameter	7.4 cm
Speed in the larger pulley ( $N_D$ )	230 rpm
Speed in the larger pulley ( $N_d$ )	920 rpm
Centre belt distance	66 cm
Length of the belt	195.62 cm
Velocity of belt	3.97 m/s
Mass of the belt	
Tension on the belt in the tight side	5583.42 N
Tension in the slack side	507.45 N
Maximum tension on the belt	5593 N
Angle of contact on the larger pulley	3.94 rad
Angle of contact on the smaller pulley	2.35 rad
Coefficient of friction	0.4
Thickness of the fixed jaw	9.57 cm
Width of the fixed jaw	917.64 mm
Length of the fixed jaw	941.18 mm
Torque acting about the swing jaw axis	172951 Nm
Width of movable jaw	90.6 cm
Mass of the movable jaw	875.09 kg

Length of movable jaw	1208 mm
<b>Shaft and Key</b>	
Length of key	8.51 cm
Width of the key	1.6 cm
Thickness of the shaft	5.6 cm
Length of Shaft	100 cm
Maximum torsion in the shaft	598.88 Nm
Diameter of Shaft	0.56 cm
Design stress	115.5 MPa
Material of construction	Carbon steel



## CHAPTER NINE

### 9.0 DESIGN OF A CRYSTALLIZER

#### 9.1 PROBLEM STATEMENT

Following the completion of chemical reactions in the fourth Continuous Stirred Tank Reactor (CSTR) of our production process, the resulting solution comprises a mixture of lithium hydroxide ( $\text{LiOH}$ ), sodium sulfate ( $\text{Na}_2\text{SO}_4$ ), and water. The primary challenge at hand is to devise an efficient crystallization process to extract high-purity lithium hydroxide crystals from this solution. The complexity arises from the coexistence of sodium sulfate, which could potentially lead to co-crystallization or contamination issues. The objective is to design and optimize a crystallization method that selectively precipitates lithium hydroxide while effectively separating sodium sulfate and other impurities.

#### 9.2 INTRODUCTION

Crystallization is a fundamental separation and purification technique widely utilized in various industries to obtain pure solid compounds from solutions. It plays a pivotal role in processes ranging from pharmaceutical manufacturing to chemical synthesis, offering an efficient means to isolate and refine desired products. The crystallization process involves the transformation of a supersaturated solution into solid crystalline particles through controlled nucleation and crystal growth. This transformation is driven by the reduction of solubility or changes in temperature, pressure, or solvent composition, leading to the formation of orderly crystal structures from dissolved solutes. (Tavare, N.S. and Springer, 1995). It enables the extraction of pure substances from complex mixtures, thereby enhancing product quality, yield, and market competitiveness.

### **9.3 MAIN OBJECTIVE**

To develop an efficient crystallization process for the selective extraction of high-purity lithium hydroxide (LiOH).

#### Specific Objectives

1. Select an appropriate crystallizer for the process
2. Determine the capacity of the equipment
3. Perform required chemical and mechanical calculations for the equipment

### **9.4 LITERATURE REVIEW**

#### **Working Principle of a Crystallizer**

Crystallization is the process by which solid particles form within a homogeneous phase. This can occur in several ways: as solid particles forming in a vapor, such as snow; as solidification from a liquid melt, like large single crystals; or as crystallization from a liquid solution. This explanation primarily focuses on the latter, crystallization from a liquid solution, which is highly relevant in various industrial applications. (Tavare, N.S. and Springer, 1995)

Crystallization from solution is a widely used industrial process due to its ability to produce pure crystalline materials from impure solutions. This method is preferred for obtaining pure chemical substances in a form suitable for packaging and storage. When a solute crystallizes from a saturated solution, the resulting crystals are typically pure, unless mixed crystals form. This purity makes crystallization an effective method for separating and purifying compounds.

The crystallization process begins with creating a supersaturated solution, where the concentration of the solute exceeds its solubility limit. Supersaturation can be achieved through cooling, evaporation of the solvent, or a chemical reaction. Once supersaturation

is attained, nucleation occurs, where small clusters of solute molecules (nuclei) form within the solution. These nuclei act as the foundation upon which larger crystals grow.

As nucleation progresses, crystal growth follows. Solute molecules continue to deposit onto the nuclei, forming larger crystals. This phase requires careful control of temperature, concentration, and agitation to ensure uniform crystal growth. In industrial crystallization, the two-phase mixture of mother liquor (the remaining solution) and crystals of various sizes is known as magma. This mixture occupies the crystallizer and is periodically withdrawn as the product.

## **COMPONENTS OF A CRYSTALLIZER**

### **1. The Crystallizer Vessel**

The crystallizer vessel is the central component of the crystallization system, providing most of the active volume necessary for the crystallization process. Its design and size are dictated by the residence time requirements, which ensure that the solution remains in the vessel long enough to achieve the desired level of supersaturation and crystal growth. The vessel must be designed to enable proper disengagement of process vapors, which are generated during the evaporation or cooling phases of crystallization. This disengagement is crucial to prevent vapor from interfering with the crystallization process and to allow for the efficient separation of crystals from the mother liquor.

### **2. The Circulation Pump**

The circulation pump is responsible for maintaining a sufficient circulation rate within the crystallizer. This continuous circulation ensures that the solution remains homogeneous, preventing localized supersaturation and promoting uniform crystal growth. Typically, axial-flow propeller pumps are used for this purpose due to their ability to handle high flow rates and provide efficient mixing with minimal shear forces, which

is important to prevent damage to the growing crystals. The pump operates under optimal supersaturation and superheating conditions, which are essential for controlling the rate of nucleation and crystal growth.

### **3. The Heat Exchanger**

The heat exchanger is a critical component that supplies the required thermal energy to the crystallizer to achieve the desired evaporation rate. In evaporative crystallization, the heat exchanger heats the solution, causing the solvent to evaporate and increase the concentration of the solute. This process creates the supersaturation necessary for crystal formation. The heat exchanger must be designed to provide precise temperature control, ensuring that the solution is heated uniformly and efficiently. This control is vital for maintaining the delicate balance needed for optimal crystal growth and preventing issues such as excessive nucleation or uneven crystal sizes.

### **4. Interconnecting Piping**

The interconnecting piping connects all the components of the crystallizer, including the crystallizer vessel, circulation pump, and heat exchanger. The design and layout of the piping system are crucial for ensuring efficient flow and minimizing pressure drops throughout the system. Proper piping design also helps prevent issues such as dead zones where the solution could stagnate, leading to uneven supersaturation and crystal growth. The piping must be constructed from materials that are compatible with the solution being crystallized to prevent corrosion or contamination. Additionally, the piping system should be designed to facilitate easy maintenance and cleaning to ensure long-term operational efficiency.

## **TYPES OF CRYSTALLIZERS**

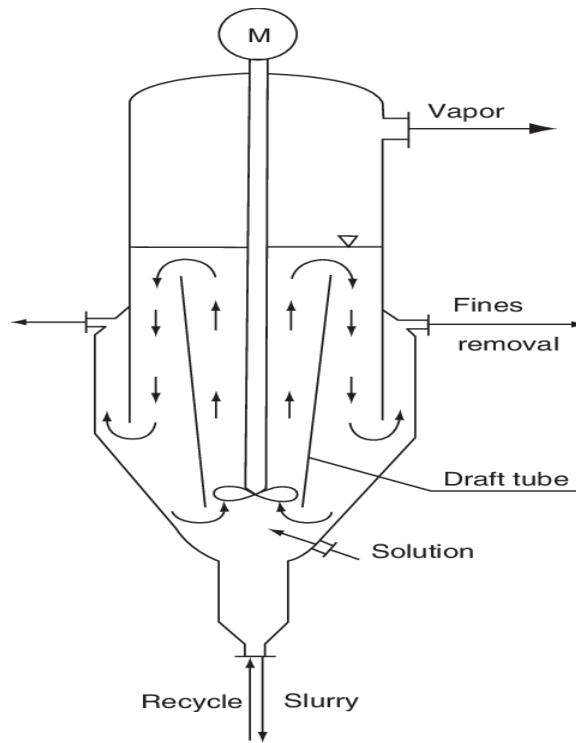
Continuous crystallizers are integral to the industrial production of chemical commodities and the recovery of valuable products from waste streams, with production capacities ranging from 12 to 1200 metric tons per day. Depending on specific process requirements or production economics, crystallizers can be configured in various ways—single or multiple stages, single or multiple effects, and with or without thermal or mechanical vapor recompression—leading to high levels of plant complexity. ( Paroli, F. ,2012)

The operation of a crystallization plant must ensure production capacity and yield, stable and trouble-free operation, required product purity, and optimal crystal size distribution (CSD). Achieving these goals requires a focused approach on four critical areas: process flow scheme, process conditions, mechanical design of the crystallization equipment, and the design of the control system and selection of proper instruments. Effective control systems are essential for maintaining constant growing conditions for the crystalline product, despite the challenges posed by the large dimensions of the equipment.

This chapter provides an overview of the most commonly used types of crystallizers—forced circulation (FC), draft-tube-baffle (DTB), and Oslo growth-type—along with the sensors and control devices employed in industrial practice.

## 1. DRAFT-TUBE AND BAFFLED CRYSTALLIZER

The Draft-Tube and Baffled (DTB) crystallizer is a specialized type of continuous crystallizer designed for producing large, high-purity crystals with precise control over crystal size distribution (CSD). It features a slow-speed propeller agitator within a draft-tube, which draws fine-crystal suspension into a wide cross-sectional boiling zone, enhancing heat transfer and evaporation. The suspension then passes through an annular zone with an additional baffle, promoting even distribution and efficient separation of crystals from the mother liquor. As the liquor flows upward at low velocity, crystals settle to the vessel's base, while an external pumped loop introduces liquor into an elutriating leg, allowing selective removal of crystals by size. This configuration ensures high-purity products, effective vapor disengagement, and reduced fouling, making DTB crystallizers ideal for industries such as pharmaceuticals and specialty chemicals, where high-quality, uniform crystals are essential. (Dunuwila et al, 1995)

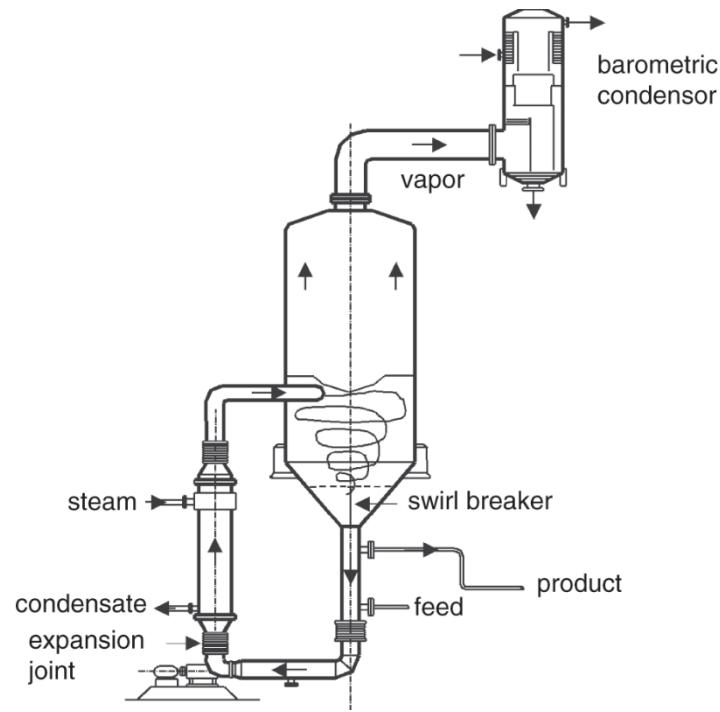


**Figure 47: Draft tube crystallizer**

## 2. FORCED CIRCULATION CRYSTALLIZER

The Forced Circulation (FC) crystallizer features an external circulation loop through a heat exchanger and may include an evaporation zone within the main vessel, with crystals removed at its base. In this design, both crystals and liquor circulate through the pump, leading to high rates of secondary nucleation and crystal breakage. Consequently, the product crystals are typically within the size range of 200-500 micrometers. This configuration is effective for processes that require efficient heat transfer and handling of large volumes, although it tends to produce smaller and more fragmented crystals due to the circulation dynamics.

(Dunuwila et al, 1995)

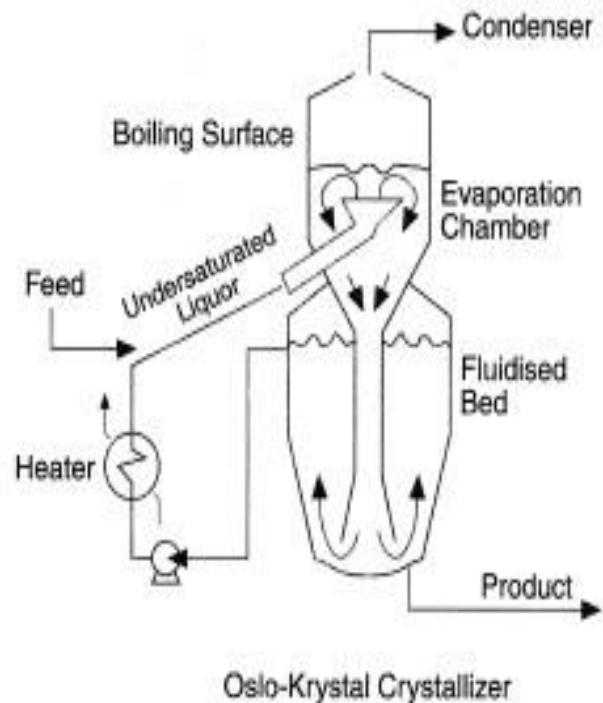


**Figure 48: Forced Circulating Crystallizer**

### 3. OSLO FLUIDIZED BED CRYSTALLIZER

The Oslo fluidized-bed crystallizer is characterized by separate sections for generating supersaturation and facilitating crystal suspension and growth. Supersaturation is induced externally through processes like cooling or evaporation, and the resulting supersaturated liquor is pumped into a central downcomer within the main vessel. Within the vessel, a classified bed of crystals is supported by an upward flowing stream of supersaturated liquor in the annular space surrounding the downcomer. Typically, relatively large crystals with a mean size exceeding 1000 micrometers are withdrawn from the base. However, in modern practice, Oslo units are often operated in the mixed-suspension mode to enhance efficiency, despite their original design intention. This mode allows for better control over crystal growth and product characteristics, making it a preferred option in contemporary crystallization processes.

(Dunuwila et al, 1995)



**Figure 49: Oslo Fluidized Bed Crystallizer**

### PARAMETERS TO CONTROL

In continuous industrial crystallizers, maintaining specific operating parameters such as supersaturation and residence time of crystals at steady-state operation is crucial. This is achieved by monitoring and controlling a series of process variables, including pressure, temperature, density, level, flow rate, and pH. Control of these variables ensures stable operation, fixed production capacity, high yield, low energy consumption, and low

fouling/encrustation levels. Additionally, variables specific to crystallization processes, such as crystal surface, nucleation rate, and mixing input power, need to be controlled to meet product quality and crystal size targets. (Paroli, F., 2012)

Despite the importance of these specific variables, their feedback control is not commonly practiced in industrial systems due to the lack of reliable and affordable instruments. Consequently, the control of industrial crystallizers often relies on traditional feedback control of single operating variables, similar to modern chemical plants.

## **9.5 EQUIPMENT SELECTION AND JUSTIFICATION**

**Equipment Selection:** Forced Circulation (FC) Crystallizer

For the crystallization of lithium hydroxide from a solution containing sodium sulfate and lithium hydroxide, we have selected the Forced Circulation (FC) crystallizer. This choice is based on the following considerations and justifications:

**Justification:**

**1. Versatility and Scalability**

The FC crystallizer is widely used in industrial applications for the bulk crystallization of inorganic salts, making it highly suitable for large-scale operations. Its capacity to handle production rates ranging from 12 to 1200 metric tons per day aligns well with both moderate and high production requirements.

**2. Design and Operational Simplicity**

The FC crystallizer's design is straightforward, consisting of a crystallization vessel and an external circulation loop that includes a propeller pump for mixing and a heat exchanger for thermal management. This simplicity translates to ease of operation, maintenance, and scalability, making it a robust and reliable choice for continuous production processes.

### **3. Effective Heat Transfer and Supersaturation Control**

The external heat exchanger in the FC crystallizer allows precise control of the temperature and supersaturation levels within the crystallizer. This is particularly important for processes like evaporative crystallization, where maintaining consistent supersaturation is critical for optimal crystal growth. The FC crystallizer is well-suited for substances with flat or inverse solubility trends with temperature, such as sodium sulfate.

### **4. Handling of Viscous and Scaling Liquors**

The FC crystallizer is adept at managing viscous process liquors and minimizing scaling issues, which are common in zero-liquid-discharge systems. This characteristic is crucial for maintaining operational efficiency and product purity, particularly in applications where scaling could pose significant challenges.

### **5. High Attrition Rates and Crystal Size Control**

The forced circulation loop, which includes a pump and a heat exchanger, facilitates high circulation rates, leading to increased attrition and secondary nucleation. This results in the production of smaller, uniform crystals typically in the size range of 200-500 micrometers. This capability is essential for achieving the desired crystal size distribution (CSD) and ensuring consistent product quality.

### **6. Operational Flexibility**

The FC crystallizer can be configured in single or multiple effect arrangements, with or without vapor recompression systems (thermo-compressors or mechanical compressors). This flexibility allows for optimization of energy usage and adaptation to varying production requirements. The ability to operate under vacuum and control the crystallizer level and slurry density further enhances its operational versatility.

## **7. Product Purity and Process Integration**

By maintaining constant suspension density and controlling the slurry extraction rate, the FC crystallizer ensures high product purity and consistent crystal size. The integration of a thickening hydrocyclone ahead of the centrifuge in certain configurations helps achieve the required solids content for efficient dewatering, contributing to the overall efficiency and effectiveness of the crystallization process.

In summary, the FC crystallizer is an ideal choice for the crystallization of lithium hydroxide from a solution containing sodium sulfate and lithium hydroxide. Its proven track record in industrial applications, coupled with its operational simplicity, effective heat transfer, handling of viscous and scaling liquors, and flexibility in configuration, make it a robust and reliable solution for achieving high-purity lithium hydroxide crystals with the desired size distribution.

### **9.6 CHEMICAL ENGINEERING CALCULATIONS**

#### **Crystallization of LiOH**

##### Data

Feed rate,  $Q_i = 20 \text{ m}^3/\text{h}$

Feed concentration,  $C_i = 95.95 \text{ kg/m}^3$

Feed temperature = 55 °C

Average density of the solution = 1310 kg/m<sup>3</sup>

Average specific heat = 2.075 kJ/kg.K

Operating pressure = 0.1316 atm

Boiling point elevation of the solution = 13 °C

Saturation concentration at the crystallization temperature = 250 kg/m<sup>3</sup>

Magma density allowed,  $M_t = 350 \text{ kg crystal/m}^3$  solution

Crystal Growth rate,  $G = 4.67 \times 10^{-8} \text{ m/s}$

Crystal density,  $\rho_c = 1475 \text{ kg/m}^3$

Desired dominant size  $L_D = 0.8\text{mm}$

Heat of crystallization,  $\Delta H_c = 75.36 \text{ kJ/mol}$

Latent heat of vapourization of water at crystallizer temperature = 2260 kJ/kg

Superheat allowed in the heat exchanger = 3 °C

Overall heat transfer coefficient in the heat exchanger,  $U = 116.3 \text{ kW/m}^2.\text{K}$

Latent heat of condensation = 200 kJ/kg

Low pressure steam available for the heat exchanger at 3kg/cm<sup>2</sup> gauge

Material Balance (1 hour basis operation)

Volume of solution entering = 34.25m<sup>3</sup>

Mass of solute entering =  $34.25\text{m}^3 \times 95.95 \text{ kg/m}^3$

$$= 3288.12 \text{ kg}$$

Mass of water entering =  $34.27 (1310 - 95.95)$

$$= 41605.50 \text{ kg}$$

Mass of crystals produced =  $\frac{350}{350 + 250} \times 3288.12 \text{ kg}$

$$= 1918.07 \text{ kg}$$

Mass of solutes leaving with mother liquor =  $3288.12 \text{ kg} - 1918.07 \text{ kg}$

$$= 1370.05 \text{ kg}$$

$$\text{Solubility (kg/m}^3) = \frac{\text{mass}}{\text{volume of solution}}$$

$$\text{Volume of solution} = \frac{1370.05}{250}$$

$$= 5.48 \text{ m}^3$$

$$\text{Mass of water leaving} = (5.48)(1310 - 250)$$

$$= 5808.8 \text{ kg}$$

$$\text{Mass of water evaporated} = (\text{water in the feed}) - (\text{water out with mother liquor})$$

$$= 41605.50 - 5808.8$$

$$= 35796.7 \text{ kg}$$

$$\text{Density of crystal} = \frac{\text{mass of crystal}}{\text{volume of crystal}}$$

$$\text{Volume of crystal} = \frac{1918.07}{1475}$$

$$= 1.3 \text{ m}^3$$

$$\text{Volume of slurry leaving per hour} = \text{vol of solution} + \text{volume of crystals leaving}$$

$$= 5.48 + 1.3$$

$$= 6.78$$

$$= 7.00 \text{ m}^3 \text{ per hour}$$

### Crystallizer Volume

$$L_D = 0.8 \text{ mm}$$

$$G = 4.67 \times 10^{-8} \text{ m/s}$$

$$L_D = 3G\tau$$

$$\tau = \frac{LD}{3G}$$

$$\tau = \frac{8 \times 10^{-4}}{3 \times 4.67 \times 10^{-8}}$$

$$\tau = 5710.21 \text{ s}$$

$$\text{residence time } (\tau) = 1.6 \text{ hr}$$

Volume of crystallizer at any time

$$= 7 \text{ m}^3/\text{h} \times 1.6 \text{ h}$$

$$= 11.2 \text{ m}^3 \text{ (known as the working volume)}$$

60% added to account for vapor bubbles and froth

Effective suspension volume in the crystallizer

$$= (11.2) \times (1 + 0.6)$$

$$= 17.92 \text{ m}^3$$

Boiling point of water = 52 °C

Boiling point elevation = 13 °C

Boiling point of the solution = 52 + 13 = 65 °C

$$PV = nRT \quad , \quad PV = \frac{mRT}{M}$$

$$\frac{PM}{RT} = \frac{m}{V} = \rho$$

$$\rho = \frac{0.1316 \times 18}{0.0821 \times 338}$$

$$\rho = 0.0854 \text{ kg/m}^3$$

$$\text{Volumetric rate of vapor generation} = \frac{35796.7}{0.0854}$$

$$= 41916.511 \text{ m}^3/\text{h}$$

$$= 4.2 \times 10^4 \text{ m}^3/\text{h}$$

$$\text{Allowable velocity } (U_v) = C_v \left( \frac{\text{density of solution}}{\text{density of vapor}} \right)^{\frac{1}{2}}$$

$$\text{Conservative value } (C_v) = 0.04 \text{ m/s}$$

$$(U_v) = 0.04 \left( \frac{1310}{0.0854} \right)^{\frac{1}{2}}$$

$$= 4.95$$

$$V = vA$$

$$\text{Area required for evaporation} = \frac{41916.511}{3600 \times 4.95}$$

$$= 2.35 \text{ m}^2$$

$$A = \frac{\pi}{4} d^2$$

$$\text{Diameter of the tank } (d) = \sqrt{\frac{4 \times 2.35}{\pi}}$$

$$= 1.73 \text{ m}$$

But a diameter of 1.8m will be used for better efficiency.

Cone angle 45°

Volume of the conical part of the tank

Radius = Height = 0.9m

$$\text{Volume} = \frac{1}{3} \times \pi \times 0.9^2 \times 0.9$$

$$= 0.76 \text{ m}^3$$

Volume of cylindrical part = 17.92 m<sup>3</sup> – 0.76 m<sup>3</sup>

$$= 17.16 \text{ m}^3$$

$$\text{Height of cylindrical part} = \frac{17.16}{\pi (0.9^2)}$$

$$= 6.74 \text{ m}$$

Add 0.35m space above boiling liquid for disengagement of the entrained droplets.

$$\text{Total height of the tank} = 6.74 \text{ m} + 0.35 \text{ m} + 0.9$$

$$= 7.99 \text{ m}$$

$$= 8.00 \text{ m}$$

### **Energy Balance and Heat Exchanger Area**

The feed liquor enters at 55 C

$$\text{Feed rate} = 34.27 \text{ m}^3/\text{h}$$

$$= (34.27 \text{ m}^3/\text{h}) (1310 \text{ kg/m}^3)$$

$$= 4.5 \times 10^4 \text{ kg/h}$$

Take the boiling point of the liquor = 65 C

Required heat input to raise the liquor temperature to 65 C

$$= (4.5 \times 10^4) (3.8) (65-5)$$

$$= 1.71 \text{ MJ}$$

Heat required for evaporation of water =  $(35796.7 \text{ kg/h})(2260 \text{ kJ/kg})$

$$= 80.9 \times 10^6 \text{ kJ/h}$$

Heat of crystallization =  $(1918.07 \text{ kg/h})(200 \text{ kJ/kg})$

$$= 3.84 \times 10^5 \text{ kJ/h}$$

The total heat input required,  $Q_h = 80.9 \times 10^6 + 1.71 \times 10^6 + 0.384 \times 10^6 \text{ kJ/h}$

$$= 82.994 \times 10^6 \text{ kJ/h}$$

$$= 83 \times 10^6 \text{ kJ/h}$$

Heating steam supplied at  $3 \text{ kg/cm}^2$  gauge (saturated)

Steam temperature =  $143 \text{ C}$

Latent heat =  $2260 \text{ kJ/kg}$

Steam rate =  $(113 \times 10^6 \text{ kJ/h}) / (2260 \text{ kJ/kg})$

$$= 50000 \text{ kg/h}$$

Heat Exchange Area

Temperature driving force,

$$(T)_1 = 143 - 65 = 78 \text{ C}$$

$$(T)_2 = 143 - 65 = 75 \text{ C}$$

$$(T)_m = 76.5 \text{ C}$$

Area of the heat exchanger =  $\frac{Q_h}{U(T)m}$

$$= \frac{83 \times 10^6}{116300 \times 76.5}$$

$$= 9.33 \text{ m}^2$$

## SUMMARY OF CHEMICAL CALCULATIONS

Design Parameter	Value	Unit
Crystallizer volume	17.9	$\text{m}^3$

Residence time	1.6	h
Volume of crystal	1.3	m
Total height of crystallizer	8.0	m
Total Heat Required	83.0	GJ/h
Heat exchanger area	9.3	m <sup>2</sup>

## 9.7 MECHANICAL ENGINEERING CALCULATIONS

The mechanical engineering calculations focus on the following topics:

- Material of construction
- Equipment wall thickness
- Equipment support structure
- Metal support positioning
- Crystallizer stress estimation

### Material of Construction

The selection of materials for constructing plant equipment is critical, ensuring that they can withstand operating conditions, maintain compatibility, and avoid failure, thereby ensuring durability and stability. For the crystallization operation of vanillin using a surface-cooled baffled crystallizer, stainless steel grade 316 is the optimal choice. The selection was based on several key properties:

1. Corrosion Resistance: Exhibits excellent resistance in various atmospheric environments and corrosive media.

2. Heat Resistance: Provides good oxidation resistance in both intermittent and continuous service up to 925°C.
3. Heat Treatment: Cannot be hardened by thermal treatment and has a high tolerance for thermal fluctuations.
4. Welding: Offers excellent weldability by all standard fusion methods, both with and without filler metals.
5. Machining: Features outstanding machinability, resulting in higher machining rates and reduced tool wear in many operations.

The composition of stainless steel type 316 includes chromium (18%), nickel (12%), molybdenum (2%), carbon (0.03%), and silicon (1%).

### **Equipment Wall Thickness**

For vessels under internal pressure, the design pressure (sometimes called maximum allowable working pressure or MAWP) is taken as the pressure at which the relief device is set. This will normally be 5 to 10% above the normal working pressure, to avoid spurious operation during minor process upsets (Towler and Sinnott, 2021).

Design pressure: 10 % above operating pressure of 1atm, 101.325kPa

$$= 1.1 \times 101.325 = 111.4575 \text{ kPa}$$

Minimum thickness required for pressure loading:  $e = \frac{P_i D_i}{2Jf - P_i}$ , (Towler and Sinnott, 2021)

Maximum allowable pressure,  $P_i = 111.4575 \text{ kPa} = 0.1114575 \text{ N/mm}^2$

Internal diameter,  $D_i = 1.8m = 1800 \text{ mm}$

Allowable stress,  $f$ , *design stress of stainless steel 316 from 0 – 50 °C =*

$$138 \frac{N}{mm^2}$$
 (Towler and Sinnott, 2021)

Welding joint efficiency,  $j = 1$ , (Towler and Sinnott, 2021)

Therefore, minimum thickness required for pressure loading,  $e = 0.5406\text{mm}$

The minimum thickness of the crystallizer can be estimated by the diameter of the vessel.

The table below contains the various minimum vessel thickness with their corresponding diameters (Towler and Sinnott, 2021).

**Table 19: Minimum thickness of vessel diameter**

Vessel Diameter (m)	Minimum Vessel Thickness(mm)
1.0	5
1.0-2.0	7
2-2.5	9
2.5-3.0	10
3.0-3.5	12

Thus, for a crystallizer vessel diameter of 1.5 m, corresponding minimum vessel thickness is 7mm  $\delta_s = 0.07m$

For knuckle of crystallizer

$$L_k = \sqrt{\frac{D_i e_k}{4 \cos \alpha}} \quad (\text{sinnnot, 2005})$$

$$L_k = \sqrt{\frac{1.8 \times 1000 \times 7}{4 \times \cos 45}} = 66.74\text{mm}$$

## EQUIPMENT SUPPORT POSITIONING

The support of equipment is vital to the stability of the equipment. Some equipment support include; saddle support, bracket and skirt support. The size and weight of the unit, internal and external fittings and vessel mount orientation are some factors that are considered when selecting an appropriate support. Skirt support are designed to be suitable for vertical vessels and the crystallizer mounted geometry thus making it the best choice of support for the crystallizer (Towler and Sinnott, 2021).

### Dead Weight of Vessels

$$W = c_v \pi \rho_m g D_o (H_v + 0.8D_m) t$$

For a steel vessel, the equation reduces to  $W = 240c_v D_o (H_v + 0.8D_o)t$  Towler and Sinnott, 2021)

Where  $c_v =$  for vessels with only a few internal fittings, 1.08 (Towler and Sinnott, 2021)

$H_v$  = height of the cylindrical part = 8.00 m

$D_o$  = outer diameter of the vessel =  $D_i + t = 1.807m$

$t$  = thickness of the vessel = 0.007 m

$\rho_m$  = density of stainless steel, 8000  $\frac{kg}{m^3}$ ,

Cylindrical part

$Hv$ , height of cylindrical part = 8.00 m

$D_o$ , outer diameter = 1.807m

$$\therefore Wv = 240 \times 1.08 \times 1.807 \times (8.00 + (0.8 \times 1.807)) \times 0.007 = 30.97kN$$

Conical part

$H_v$ , height of conical part = 0.9m

$$D_m = D_i + L_k = 1.8 + 0.06674 = 1.86093m$$

$$\therefore Wv = 240 \times 1.08 \times 1.86674 \times (0.9 + (0.8 \times 1.86674)) \times 0.007 = 8.11kN$$

$$\therefore \text{total weight of vessel} = 30.97 + 8.11 = 29.08kN$$

### Metal Support Positioning

An initial process of metal support mounting is done before the skirt concrete is put in place thus the position of the metal support is vital to the overall stability of the equipment.

Location of belt fastener=0.5m

Number of metal bar support = 4

Position of metal support bar around crystallizer circumference

$$\frac{C}{4} = \frac{2\pi \left(\frac{D_i}{2}\right)}{4} = \frac{2\pi \left(\frac{1.8}{2}\right)}{4} = 1.41 m$$

## CRYSTALLIZER STRESS ESTIMATION

### Primary stresses:

circumferential stress,  $\sigma_h = P D_i / 2t$ , (Towler and Sinnott, 2021).

$$\sigma_h = \frac{0.1114575 \times 1800}{2 \times 7} = 14.33 N/mm^2$$

longitudinal stress,  $\sigma_L = \sigma_h / 2$ , (Towler and Sinnott, 2021).

$$\sigma_L = \frac{14.33}{2} = 7.165 N/mm^2$$

## Direct Stress

*Direct stress,  $\sigma_w = Wv / \pi(D_i + t)t$ , (Towler and Sinnott, 2021).*

$$\sigma_w = \frac{29080}{\pi \times (1.8 + (7 \times 10^{-3})) \times (7 \times 10^{-3})} = \frac{73179.3057N}{m^2} = \frac{0.731793N}{mm^2}$$

## Bending Stress

$$bending stress, \sigma_b = \pm \frac{M}{I_v} \left( \frac{D_o}{2} + t \right)$$

$$I_v = \frac{\pi}{64} (D_o^4 - D_i^4), \text{ where}$$

$D_0$  is outer diameter in mm

$D_i$  is inner diameter in mm

$M = wx$ , where

w = total mass of the body

x = radius of the body, (Towler and Sinnott, 2021).

Cylindrical part

$$I_v = \frac{\pi}{64} (1807^4 - 1800^4)$$

$$= 8.06265 \times 10^9 N/mm^4$$

$$M = 30970 \times 900$$

$$= 2.7873 \times 10^6 Nmm$$

$$\sigma_b = \pm 3.1114 N/mm^2$$

Conical part

$$I_v = 8.06265 \times 10^9 N/mm^4$$

$$M = 8110 \times 900$$

$$= 7.299 \times 10^6 Nmm$$

$$\sigma_b = \pm 0.8148 N/mm^2$$

$$\sigma_{b\ total} = \pm(3.1114 + 0.8148) = 3.9262 N/mm^2$$

### **Resultant longitudinal stress**

$$\sigma_z = \sigma_l + \sigma_w \pm \sigma_b, \text{ (Towler and Sinnott, 2021)}$$

$\sigma_w$  is negative for compressive stresses

$$\sigma_{z1}(\text{upward}) = 7.165 - 0.731793 + 3.9262 = 10.3594 N/mm^2$$

$$\sigma_{z2}(\text{downward}) = 7.165 - 0.3431 - 3.9262 = 3.4257 N/mm^2$$

The greatest difference between the principal stresses would be on the downside,  $\sigma_{z2}(\text{downward})$ . (Towler and Sinnott, 2021)

$$\sigma_1 = \frac{1}{2}((\sigma_h + \sigma_z) + (\sigma_h - \sigma_z)) = 10.3594 N/mm^2$$

$$\sigma_2 = \frac{1}{2}((\sigma_h + \sigma_z) - (\sigma_h - \sigma_z)) = 3.4257 N/mm^2$$

$$\sigma_3 = 0.5P = 0.5(0.1114575) = -0.05572875 N/mm^2$$

$\sigma_3$  will be compressive (negative)

$$\text{maximum stress intensity} = \sigma_1 - \sigma_2 = 6.9337 \frac{N}{mm^2}$$

Failure theory: maximum principal stress theory which postulates that a member will fail when one of the principal stresses reaches the failure value in simple tension. The failure point is taken as the yield point stress.

*This stress is well below the design stress 195  $\frac{N}{mm^2}$ . Thus design is suitable*

(Towler and Sinnott, 2021)

### **Skirt support**

This consists of a cylindrical shell welded to the base of a support. A flange at the base of the skirt transmits the load to the foundations. Skirt supports are recommended for vertical vessel as they do not impose concentrated loads on the vessel shell. (Towler and Sinnott, 2021)

Straight cylindrical skirt ( $\theta_s = 90^\circ$ ). material of construction: carbon steel with design stress of 135 N/mm<sup>2</sup>, Young's modulus = 200000 N/mm<sup>2</sup>

$$total\ weight = vessel\ weight + water\ in\ feed + solute\ in\ feed$$

$$= 29080 + 41605.50 + 3288.12 = 73973.62\ N$$

Wind loading at wind pressure of 1280 N/m<sup>2</sup>,

$$F_w = \text{wind pressure} \times \text{diameter of the vessel}$$

$$= 1280 \times 1.8 = 2304 \frac{N}{m},$$

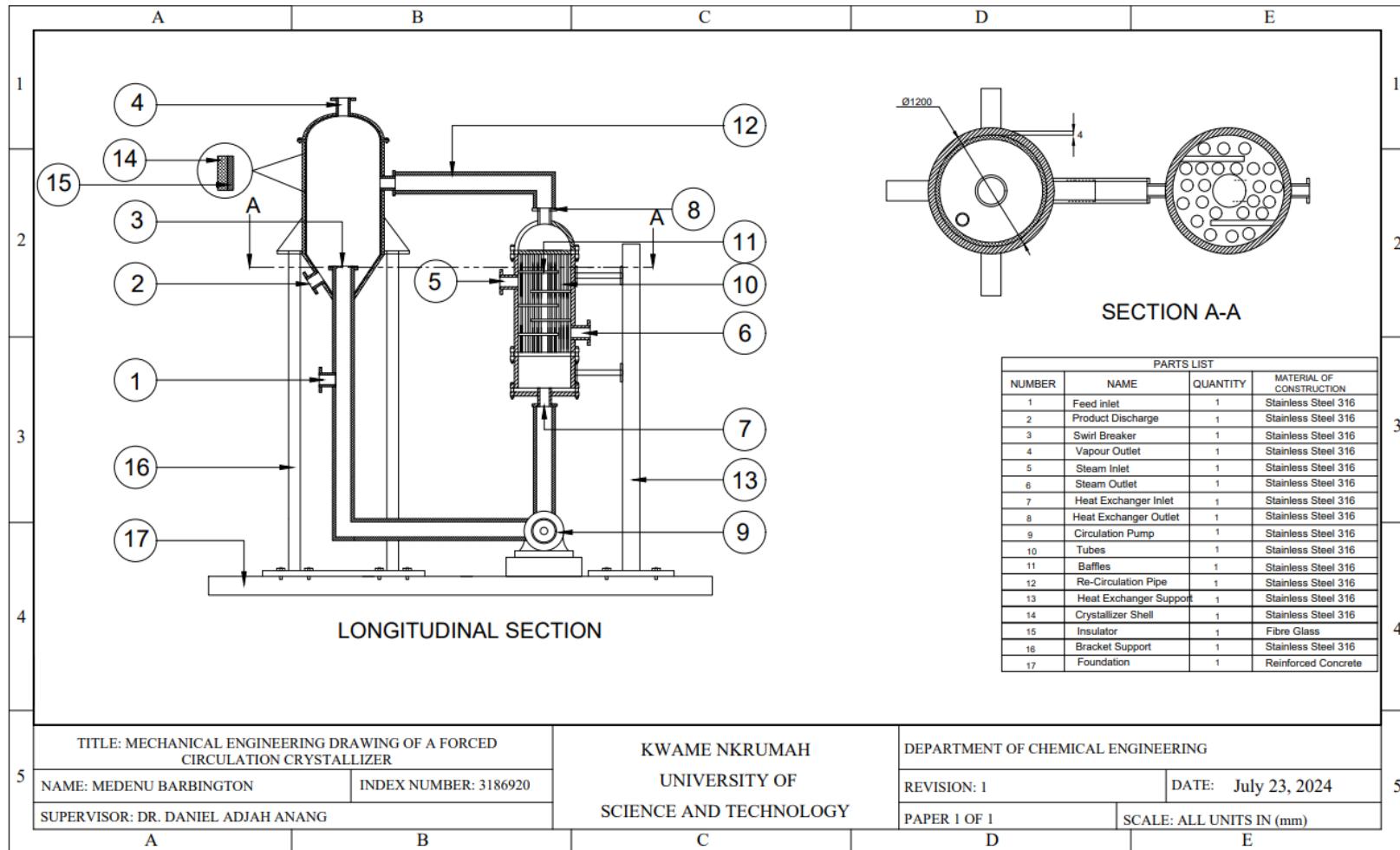
$$\text{Bending moment at base of skirt, } M_x = \frac{F_w \times H_{total}}{2}$$

$$= 2304/2 \times 3.3453^2 = 12892.069\ Nm$$

## **SUMMARY OF MECHANICAL ENGINEERING DESIGN**

Design Parameter	Value	Unit
Material Selection	Stainless 316	

Crystallizer thickness	7	mm
Dead weight of crystallizer	29	kN
Total weight of crystallizer	74	kN
Metal support positioning	1.41	m
Direct stress due to vessel weight	0.73	N/mm <sup>2</sup>
Knuckle of crystallizer	66.74	mm
Type of support	Straight cylindrical skirt	



**Figure 50: Mechanical drawing of a crystallizer**

## CHAPTER TEN

### 10.0 DESIGN OF A MIXER

#### Introduction

Mixing is an important part of majority of industrial processes. It is done for many several reasons but the primary reason is to create a uniform mixture from a number of streams (two or more). The requirement for a mixer cannot be overlooked in any process that requires its use. As such, there are different forms of mixer and each is used to meet a particular mixing requirement for a downstream process. Each of these mixers are used to meet a particular requirement of the downstream process. Even though, different process may require the same type of mixers for operation, each mixer is designed unique. That is, they have different design parameters even if they look the same. Therefore, even for two mixers that look alike, an operator may not be able to substitute one for the other for operation. As such, the design of a mixer is paramount in its specific operation. Since these kinds of equipment are unit operation specific, a detailed study is undertaken to identify and design a specific mixer best suited for the mixing requirement of the water and baked spodumene ore.

#### 10.1 Problem Statement

The acid roasted ore has lithium metals well activated. These activated lithium metals have to be leached from the ore. This is to be done by mixing with water in a mixing tank to produce a 50w/w% water content. This chapter is dedicated to discussing, identifying and designing the specific mixer required to carry out this process.

#### Main Objective

To design a mixing tank capable of mixing 26752.269kg/h of acid roasted ore with 30000kg/h of water.

#### Specific Objective

- i. To identify the various types of mixers available to choose from.
- ii. To select and justify the type of mixer most suitable for the required mixing process.
- iii. To perform a detailed chemical and mechanical calculations for the selected mixer.
- iv. To present a detailed mechanical

## **10.2 Literature review**

Mixing tanks are crucial components in various industrial processes, including chemical manufacturing, pharmaceuticals, mining, food and beverage production, and wastewater treatment. These tanks are designed to ensure the homogeneous mixing of different substances, which is essential for the consistency and quality of the final product (Baldyga and Bourne, 1999). This literature review aims to provide an overview of the design principles, types, and applications of mixing tanks, as well as recent advancements and challenges in their use.

### **Types of mixers**

#### **A. Agitator Type Mixers**

These types of mixers are constructed by mounting agitators to stir, mix, or blend liquids and low-viscosity materials within tanks or vessels. They are constructed as pairs together with containment vessels or tanks. These agitators typically consist of a shaft with blades or paddles that rotate to agitate the contents of the container. The design of agitators can vary significantly depending on the specific requirements of the mixing process, including factors such as viscosity of the materials, the size of the batch, and the desired mixing intensity (Shinnar and Turian, 1989).

## **Principles of Operation**

The fundamental principle behind agitator type mixers is the generation of fluid motion within the vessel to promote mixing (Sinnott and Towler, 2013). This is achieved through the rotation of the agitator blades, which create turbulence and shear forces within the liquid. The agitation helps to break up clumps, disperse gases, and ensure that all components are uniformly distributed throughout the mixture.

### **B. Planetary Mixers**

Planetary mixers derive their name from their distinctive mixing mechanism, which involves a stationary bowl and a mixing attachment that revolves around its own axis while also orbiting the bowl's perimeter. This dual motion ensures thorough mixing of containing components by covering the entire bowl surface, resulting in a homogeneous mixing.

### **Principles of Operation**

The operation of planetary mixers centers around a planetary gear system, where the mixing attachment (typically a whisk, paddle, or hook) rotates on its axis and simultaneously moves in a circular motion around the inside of the mixing bowl. This multi-directional movement facilitates efficient blending and kneading of ingredients, whether for dough preparation, cake batter, creams, or other viscous mixtures.

### **C. Drum mixers**

Drum mixers are mechanical devices used primarily in industrial settings to blend materials within a drum or container. These mixers are versatile and can handle a wide range of materials, from powders and granules to liquids and slurries. They come in various types and each has its own mechanism of operating.

## **Principles of Operation**

The operating principle of drum mixers solely depend on the particular type of drum mixer in operation.

- i.     Rotary drum mixers use a rotating drum or barrel to tumble materials together. As the drum rotates, the materials inside are lifted upwards by the drum's internal baffles or flights and then allowed to free-fall back into the bulk of the material. This tumbling action facilitates the blending of materials as they repeatedly mix and fold over each other.
- ii.    Paddle drum mixers feature internal paddles or blades that rotate inside the drum to agitate and mix materials. The paddles scoop up materials from the bottom of the drum and lift them upwards, then drop them back into the mixing zone. This action creates a gentle folding and kneading effect that ensures thorough blending of the materials.
- iii.   Static drum mixer unlike the rotating types, have stationary internal components that create turbulence and mixing as materials pass through the drum. The design of these mixers disrupts the flow of materials, causing them to mix thoroughly as they move through the drum. This turbulent flow ensures that all components of the mixture are well-mixed by the time they exit the mixer.
- iv.    Roller mixers use rollers inside the drum to knead and mix materials. As the rollers rotate, they knead and fold the materials within the drum. This continuous rolling and kneading action effectively blends viscous materials or pastes.

#### **D. High Shear Mixers**

High shear mixers are designed to produce intense mechanical forces that effectively break down particles and disperse them uniformly throughout a liquid medium. These

mixers are characterized by their high-speed rotation and use of specialized blades or rotor-stator assemblies to impart shear energy into the mixture.

### **Principles of Operation**

The operation of high shear mixers revolves around the principle of creating high velocity and turbulent flow within the mixing chamber. This is achieved through the rotation of blades or the high-speed movement of a rotor-stator configuration, which applies shear forces to the materials being processed. As a result, particles are reduced in size, and ingredients are thoroughly mixed and emulsified.

### **E. Static mixers**

These mixers are also known as inline mixers. They serve to put liquid in motion in order to achieve homogeneity of composition and avoid the sedimentation process. They are driven by auxiliary equipment, such as a shaft, speed reducer or electric motor, to sluice the product over the static mixer elements, to provide mixing action. Static mixers function by forcing sediment to flow in one direction and overcome the resistance during a liquid circulation flow in open reservoirs, ditches and canals (Anand and Sharma, 2016). They are also used to intensify physical and chemical processing in liquids, particularly the processes of gas and solid dissolution. Gas dissolution is usually used in sediment / waste water / anaerobic process. The intensified mixing operation is applied in order to lengthen the distance covered by gas bubbles and to prevent smaller bubbles from joining into bigger ones. Direct drive, fast rotating mixers may also be used to prevent surface scum from coming into existence and to destroy any surface scum that has already appeared.

### **10.3 Justification and equipment selection**

The many types of mixers discussed above all have their own advantages and limitations. However, for the purpose of the leaching process of the spodumene ore

in water, an agitator type mixer is opted for. This choice is accompanied by a number of reasons and some are as follows.

- i. Compared to the other types of mixers, agitator type mixers have intensive mixing capability especially when dealing less viscous mixtures. In leaching process such as this case for the spodumene, the agitator used is able to create turbulence and agitation which results in the particle in mixture being suspended to prevent a solid, unmixed buildup at the bottom of the tank and hence, enhancing mass transfer rates, promoting efficient dissolution of minerals into the leaching solution, that is, the water used.
- ii. Compared to other types, agitator type mixers offer precise control over mixing parameters such as speed, direction, and intensity of agitation. This level of control is critical in optimizing the leaching process to achieve desired extraction rates and minimize reagent consumption.
- iii. Due to the easy and simple construction of agitator type mixer, their maintenance is easy. They can easily be disassembled and reassembled. This cannot be said about the other types of mixers.
- iv. Agitator tanks can handle a wide range of materials and viscosities, from slurries and solids to liquids and high-viscosity pastes. This versatility is advantageous in mining applications where the composition and characteristics of ores can vary significantly. Agitator tanks can effectively mix these diverse materials to ensure uniformity and consistency in the leaching process.

### **Equipment description**

The agitator type mixer is employed in this process to maintain homogeneity in the mixing of the mixing of the crushed spodumene ore and the water so as to enhance

the leaching process. This equipment is basically made of two major parts: the vessel or tank and the agitator system.

**a. The vessel**

This serves as the containment chamber of the mixture. It houses the entire process and it is usually constructed to have a circular cross section. The size of the vessels of mixers are relative to the quantity of mixture fed to the mixture at a unit time. The greater the mass flow rate to the mixer, the bigger the size. The vessel may have some modification which has the potential to enhance upon the mixing of the content. Some of these modifications are;

i. **Baffles**

Baffles are connected to the inner walls of the vessel. They are bars whose height is slightly above the level of the mixture. They serve as an obstruction to the circular motion of the mixture. This results in the formation of turbulent flow in the vessel and tends to prevent the formation of vortex during mixing.

ii. **Floor and floor type**

The bottom of the vessel may be made flat (horizontal inclined to an angle), torispherical or ellipsoidal. Each configuration has its own reason of construction.

ii. **Top entry**

The top of the vessel may be covered (which may have the same configuration as the bottom) or opened (which will have a support upon which the agitator system is mounted). A flat bottom vessel is used as the tank for the mixer. This vessel's bottom is slightly inclined towards the exit nozzle.

**b. The agitator system**

The agitator system serves as the part of the mixer that deals with the flow of the mixture within the vessel (Sinnott and Towler, 2013). It is made up of three parts:

**i. Motor Component**

The motor drives the agitator assembly. It produces the required torque to induce flow and shear to the media at a controlled degree. The power requirement of an agitator depends on many factors such as:

- a) Viscosity, specific gravity, and solid content of the media
- b) Number of impellers
- c) Impeller diameter
- d) Speed or rpm of the impeller
- e) Power number of the impeller

**ii. The shaft component**

This is a long rod that connects the motor to the impeller. It does so in order to transmit the produced torque from the motor to the impeller.

**ii. The impeller component**

This is the component of the agitator system that does the actual mixing. It is considered the most critical component of agitators that determine the flow pattern, efficiency of the homogenizing process, and mixing parameters. They mainly consist of a hub and blades. The hub is directly connected to the shaft through a shaft key and a grub screw. The agitator blades are attached to the hub by welding or screwing. Impellers may be classified as an open- or disc-type. The open blade type has blades directly attached to the shaft whereas the disc-type impeller consists of a disc with attached blades.

## **Flow Patterns of Agitator Impellers**

The design of an impeller induces a specific kind of flow patterns in the mixture it drives. The flow patterns that can be developed within the mixture are axial, radial, and tangential flow patterns. There are some instances where more than one kind of flow pattern is developed during agitation.

### **i. Axial Flow**

During axial flow, the impellers cause the media to flow parallel to the impeller's axis of rotation. The rotation of the impeller causes a “top-to-bottom cyclic” flow pattern. The fluid at the upper portion of the tank is forced to flow down until it is deflected at the tank bottom. It then spreads over the tank bottom and subsequently flows up along the wall before being drawn by the impeller. This motion mixes the fluid at the top and bottom of the tank and prevents the solids and solutes from settling at the tank bottom.

### **ii. Radial Flow Impellers**

For this type of flow pattern, the responsible impellers cause the media to flow perpendicularly to the impeller's axis of rotation. As the fluid ejects from the impeller, it flows towards the tank's walls. It then moves either upward or downward until it is drawn again to the impeller's center. This motion is repeated to incorporate the contents of the tank thoroughly.

### **iii. Tangential Flow Impellers**

Tangential flow causes the media to flow in a circular path around the shaft. The fluid rotates around the vessel together with the impeller blades which causes a very low vertical flow to be produced as the fluid hits the tank wall.

## **Types of Impellers**

- i. Paddle agitators consist of two flat paddle-shaped impeller blades extending to reach the tank walls. They are used if no extensive axial and radial flow is required.
- ii. Anchor agitators have impellers that resemble the shape of an anchor. They typically have a U-shape that matches the contour of the tank. They generate a predominantly tangential flow pattern, but angled blades can be incorporated on their horizontal supports to create an axial flow.
- iii. Helical ribbon agitators have a helical impeller blade fixed in the shaft by rods. These impellers are an alternative to anchor impellers which can generate laminar flow. These agitators create an axial flow pattern. They generally have a higher fluid contact area to mix fluids with higher viscosities.
- iv. Double helical ribbon agitators are designed to have two helical blade flights running through the shaft in opposite directions. The additional flight enhances the mixing of more viscous fluids.
- v. Screw impellers are close clearance impellers with a helical flight directly attached to the impeller shaft. They provide an excellent top-to-bottom turnover. They are used in blending high viscosity and shear-sensitive media.
- vi. Propeller agitators mainly produce an axial flow pattern, though tangential flow can also be produced. The fluid is displaced and accelerated longitudinally after the impeller blades draw it. The deflection of the fluid depends on the inclination of the impeller blades. The impeller blades are tapered towards the shaft to minimize centrifugal force and maximize axial flow.
- vii. Turbine agitators are an intermediate between propeller and paddle agitators. They usually have larger diameters than propeller agitators. These agitators combine centrifugal and rotational motion. They are used in solid suspension

systems, emulsification and dispersion processes in which the media is required to flow at high speeds. They offer a good balance between flow and shear. They are typically operated at high speeds. They can handle a wide range of material viscosities and still provide a high mixing efficiency.

There are a number of different types of turbine agitators and they are as follows:

- Straight blade turbine impellers: They consist of two to eight flat vertical blades. These blades generate a radial flow pattern and high shear which are very efficient for solid suspension systems.
- Pitched blade turbine impellers: They have flat blades that make an angle less than 90 degrees with the vertical. They generate high shear and have good mixing efficiency. They are used in gas dispersions and solid suspensions.
- Rushton turbine impellers: Consist of flat vertical impeller blades arranged symmetrically around the circumference of a horizontal disc. These impellers generate a radial flow pattern. Rushton turbine impellers are effective for gas dispersion applications.
- Smith turbine impeller: A variation of a Rushton turbine impeller, in which semicircular or curved blades are attached instead of flat blades designed for gas-gas and gas-liquid dispersions and emulsions.

Among the many types of impellers available to choose from, the straight blade turbine impeller is opted for as it is the most efficient in keeping a less viscous mixture in constant suspension and at variable operating speed (Sinnott and Towler, 2013).

#### **10.4 Chemical Engineering calculation**

In this engineering calculation, there are some few assumptions being made:

- i. There is no loss of material during mixing.
- ii. There pressure drop associated with this mixing process is negligible and can be assumed to be zero. (Turton, Bailie, Whiting, Shaeiwitz, and Bhattacharyya, 2012).

The two streams entering into the mixing tank are water and roasted crushed spodumene

Mass flow rate of roasted spodumene,  $m_s = 26752.269 \text{ kg/hr} = 7.43 \text{ kg/s}$

Mass flowrate of water,  $m_w = 30000 \text{ kg/hr} = 8.33 \text{ kg/s}$

Total mass flow rate,  $m_t = m_s + m_w = 7.43 + 8.33 = 15.76 \text{ kg/s}$

Since we are mixing two different streams, finding the common density,  $\rho_m$

$$\rho_m = \frac{1}{\frac{x_s}{\rho_s} + \frac{x_m}{\rho_m}}$$

$$x_s = \frac{m_s}{m_s + m_w} = \frac{7.43}{7.43 + 8.33} = 0.47$$

$$x_m = 1 - x_s = 1 - 0.47 = 0.53$$

$\rho_s$  = density of spodumene =  $2450 \text{ kg/m}^3$

$\rho_w$  = density of water =  $1000 \text{ kg/m}^3$

$$\rho_m = \frac{1}{\frac{0.47}{2450} + \frac{0.53}{1000}} = 1386.9 \text{ kg/m}^3$$

For water content,  $x_w \geq 0.5$ , mixture is considered to be dilute and hence,

Viscosity of mixture,  $\mu_m = \mu_w(1 + 2.5\phi) \dots$  (Sinnott and Towler, 2013)

Where  $\mu_w$  = viscosity of water =  $1 \text{ cP} = 1 \times 10^{-3} \text{ Pa} \cdot \text{s}$

$$\phi = \text{volume fraction of solids} = x_s \cdot \frac{\rho_m}{\rho_s} = 0.47 \times \frac{1386.9}{2450} = 0.27$$

$$\therefore \mu_m = 1 \times 10^{-3} (1 + 2.5(0.27)) = 0.0017 \text{ Pa} \cdot \text{s} = 1.7 \text{ cP}$$

$$\text{Total volume flow rate of mixture, } \dot{v} = \frac{m_t}{\rho_m} = \frac{15.76}{1386.9} = 0.0114 \text{ m}^3/\text{s}$$

Since the mixing is being performed for leaching, the residence time of the content of the mixing tank should be sufficient to leach out the active components of the ore.

Hence using a residence time,  $t_r = 45 \text{ min} = 2700 \text{ s}$

$$\text{working volume of tank, } v_w = \dot{v} t_r = 0.0114 \times 2700 = 30.69 \text{ m}^3$$

Assuming working volume is 75% of actual volume,  $v$ ,

$$\text{then, } v = \frac{v_w}{0.75} = \frac{30.69}{0.75} = 40.92 \text{ m}^3$$

### Sizing of vessel

For efficient mixing, increasing the height of the tank will be ideal.

Taking  $\frac{h}{d} = 2$  as the minimum ratio of vessel height to diameter of the vessel, from

$$v = \frac{\pi d^2 h}{4} = \frac{\pi d^3}{2}$$

$$d = \sqrt[3]{\frac{2v}{\pi}} = \sqrt[3]{\frac{2 \times 40.92}{\pi}} = 2.964 \text{ m}$$

$$\therefore h = 2d = 2 \times 2.964 = 5.929 \text{ m}$$

Liquid height in vessel,  $h_L$

$$h_L = \frac{4v_w}{\pi d^2} = \frac{4 \times 30.69}{\pi (1.03^2)} = 4.447 \text{ m}$$

### Impeller calculation

The Dickey's criteria are used for the calculation for the impeller (Sinnott and Towler,

2013).

$$\frac{h_L}{d} = \frac{4.447}{2.964} = 1.5$$

For  $1.3 < \frac{h_L}{d} < 1.6$  and  $\mu_m = 1.7 \text{cP} < 25000 \text{cP}$ , 2 impellers are required for effective stirring.

Impeller clearance from the bottom of mixing tank

$$\text{Upper clearance, } c_u = \frac{2h_L}{3} = \frac{2(4.447)}{3} = 2.96 \text{m}$$

$$\text{Bottom clearance, } c_b = \frac{d}{3} = \frac{2.964}{3} = 0.99 \text{m}$$

$$\begin{aligned} \text{Length of shaft from vessel opening, } L_s &= h - 0.8c_b = 5.929 - 0.8(0.99) \\ &= 5.138 \text{m} \end{aligned}$$

$$\text{Diameter of shaft, } d_s = 0.1d = 1.4(2.964) = 0.296 \text{m}$$

Using a turbine impeller with six flat vertical blades.

$$\text{impeller diameter, } d_i = 0.4d = 0.4(2.964) = 1.186 \text{m}$$

$$\text{length of impeller blade, } l_i = 0.25d_i = 0.25(1.186) = 0.296 \text{m}$$

$$\text{width of impeller blade, } w_i = 0.2d_i = 0.2(1.186) = 0.237 \text{m}$$

Impeller speed

$$N_s = \frac{S \left( \frac{\mu_w}{\rho_w} \right)^{0.1} D_p^{0.2} \left[ \frac{g(\rho_s - \rho_w)}{\rho_w} \right]^{0.45} (100x_s)^{0.13}}{d_i^{0.85}}$$

where

$N_s$  = minimum stirring speed required for mixing  $x_s$  = solid fraction in mixture = 0.47

$S$  = Zwietering factor = 5.7 (For flat bottom mixing tanks where 6 blade turbine impeller is used with  $\frac{d_i}{d} < 0.5$ )

$$g = 9.81 \text{ m/s}^2$$

$D_p$  = diameter of particles of milled roasted spodumene ore = 50 micron

$N_s$

$$= 5.7 \frac{\left(1 \times \frac{10^{-3}}{1000}\right)^{0.1} (50 \times 10^{-6})^{0.2} \left[\frac{9.81(2450 - 1000)}{1000}\right]^{0.45} (100 \times 0.47)^{0.13}}{1.185^{0.85}}$$

$$N_s = 2.2921 \text{ rps} = \frac{2.2921 \text{ revolution} \times 60 \text{ s}}{1 \text{ s} \times 1 \text{ min}} = 137.5 \text{ rpm}$$

Actual stirring speed required, N

$$N = 1.50N_s = 1.5(2.2921) = 3.44 \text{ rps}$$

### Power requirement

The power required to drive the shaft and the impeller is calculated as follows

$$Re = \frac{d_i^2 N \rho_m}{\mu_m}$$

where

$d_i$  = impeller diameter = 1.186m

$N$  = actual stirring speed = 3.44rps

$\rho_m$  = density of mixture = 1386.9kg/m<sup>3</sup>

$\mu_m$  = viscosity of mixture = 1.7cP = 0.0017Pa · s

$$Re = \frac{1.186^2 \times 3.44 \times 1386.9}{0.0017} = 4021649 = 4.0 \times 10^6$$

From literature, (line 1) using a six-blade turbine impeller, a  $Re = 4.0 \times 10^6$  gives a

Power Number,  $P_N = 5$

$$\therefore \text{from } P_N = \frac{P}{\rho_m N^3 d_i^5}$$

$$\Rightarrow P = P_N \rho_m N^3 d_i^5 = 5(1386.9)(3.44)^3(1.186)^5 = 660662W = 660.7kW$$

### Baffles

Baffle are installed around the inside walls of the vessel. This is to provide an obstruction to the circular movement of the mixture so as to prevent the formation of vortex. This also enhances the formation of turbulent regime in the vessel. This ensures adequate mixing of the milled ore and the water which further optimizes the leaching process. Since the diameter of the vessel is less than 2m, installing four evenly spaced baffles will be sufficient enough to promote the required mixing.

$$\text{baffle width, } b_w = \frac{d}{12} = \frac{2.964}{12} = 0.247\text{m}$$

$$\text{baffle height, } b_h = 1.1h_L = 1.1(4.447) = 4.891\text{m}$$

### Summary of Chemical Engineering Design

Parameter	Value
<b>Mixer dimension</b>	
Actual volume	40.92m <sup>3</sup>
Diameter	2.964m
Height	5.929m

Maximum liquid height	4.447m
Residence time	45min
<b>Agitation system</b>	
Impeller type	Turbine impeller with six flat vertical blades
Number of impellers	2
Clearance between bottom impeller and bottom of tank	0.988m
Clearance between upper impeller and bottom of tank	2.964m
Impeller diameter	1.186m
Impeller blade width	0.237m
Impeller blade length	0.296m
Impeller speed	3.44rps
Length of impeller shaft	5.138m
Diameter of impeller shaft	0.296m
Total power consumption	660.71kW
Baffle height	4.891m
Baffle width	0.24m

## **10.5 Mechanical Engineering Calculations**

### **Materials of Construction**

The vessel should be able to withstand the wear and tear caused by the material being mixed. It should also be corrosion resistant and more. The design of the shaft should be able to withstand the torque and bending moment of the impeller. A strong material should be used for it to handle such forces. Stainless steel type 302 has a composition of 17% Cr, 8% Ni, and 0.15% C where its general purpose is similar to austenitic type with good corrosion resistance and mechanical process.

### **Corrosion allowance**

Corrosion is a complex phenomenon, and it is not possible to give specific rules for the estimation of the corrosion allowance required for all circumstances. The allowance is based on experience with the material of construction under similar service conditions to those for the proposed design. For stainless steel type 302, where mild corrosion is not expected, a minimum allowance of 2.0 mm should be used; where more severe conditions are anticipated this should be increased to 4.0 mm (Sinnott and Towler, 2013).

### **Design Pressure**

By the definition of maximum allowable pressure, using 10% of the total pressure as the maximum allowable pressure and making it equivalent to design pressure (Sinnot, 2005).

$$P = \text{Design pressure, Pa}$$

$$P = 10\% \text{ more than } P_o$$

$$P_o = \text{Operating pressure in the column} = 1\text{atm} = 101325 \text{ Pa}$$

$$P = 1.1 \times 101325 = 111458 \text{ Pa}$$

## **Design Temperature**

It is usually accepted that the design temperature should be 10°C above the operational temperature (Sinnot, 2005)

$$T = 10^\circ\text{C} + T_o \quad \text{where,}$$

$$T_o = 80^\circ\text{C}$$

$$T = \text{design temperature, } ^\circ\text{C}$$

$$T = \text{operating temperature} = 10^\circ\text{C} + T_o = 10^\circ\text{C} + 80^\circ\text{C} = 90^\circ\text{C}$$

## **Thickness required by the cylindrical body**

For a cylindrical shell, the additional minimum thickness required to resist internal pressure can be determined from equation below as given by British Standard PD 5500

as ;

$$e = \frac{P_i d_i}{2f - P_i}$$

where

$$P_i = \text{system pressure} \quad d_i = d = \text{internal diameter of vessel} = 1.03\text{m} = 1.03 \times 10^3\text{mm}$$

$$e = \text{design stress}$$

$$P_i = \text{hydrostatic pressure} + \text{atmospheric pressure}$$

$$\text{But hydrostatic pressure}$$

$$\begin{aligned} &= \text{density of mixture} (\rho_m) \times \text{gravity} \times \text{maximum height of liquid} (h_L) \text{ hydrostatic pressure} \\ &= 1386.9 \times 9.81 \times 1.54 = 20974\text{Pa} \end{aligned}$$

$$\text{atmospheric pressure} = 1\text{atm} = 101235\text{Pa}$$

$$\therefore P_i = 20974 + 101325 = 122299\text{Pa} = 0.122298989 \text{ N/mm}^2$$

Stainless steel type 302 has a tensile strength of  $895\text{N/mm}^2$  and a design stress of  $185 \text{ N/mm}^2$  at  $0 - 110^\circ\text{C}$  temperatures. (Coulson and Richardson, 2023)

Hence  $f = 185\text{N/mm}^2$

$$e = \frac{0.122298989 \times 2.964 \times 10^3}{2(185) - 0.122298989} = 1.297\text{mm}$$

Adding the minimum practical wall thickness of 7 mm (including corrosion allowance) yields  $\therefore$  true minimum thickness  $= 1.297 + 7 = 8.287\text{mm} = 0.008297\text{m}$

### **External diameter of the reactor**

The external diameter of the cylindrical part of the reactor will be a sum of the internal diameter and the thickness of calculated above.

$$d_e = d + 2e \quad d_e = 2.964 + 2(0.008297) = 2.981 \text{ m}$$

### **Heads and Closure Design**

The tank will have a closed head so as to prevent the contamination of the ore water mixture. The ends of a cylindrical vessel are closed by heads of various shapes. The principal types used are:

- i. Flat and formed heads: flat plates are used as covers for manways. Formed flat ends, known as “flange-only” ends, are the cheapest type of formed head to be manufactured, but their use is limited to low-pressure and small diameter vessels. They are also ineffective for mixing operations where they require more power than their domed counterparts.
- ii. Torispherical heads: For operating pressures below 15 bar, standard torispherical heads (dished heads) are the most common used type of heads.

- iii. Ellipsoidal heads: these heads are considered the most economical heads for pressures above 15 bar.
- iv. Hemispherical heads: hemispherical head is the strongest shape; capable of resisting about twice the pressure of a torispherical head of the same thickness. The cost of forming a hemispherical head will, however, be higher than that for a shallow torispherical head. Hemispherical heads are used for high pressures.

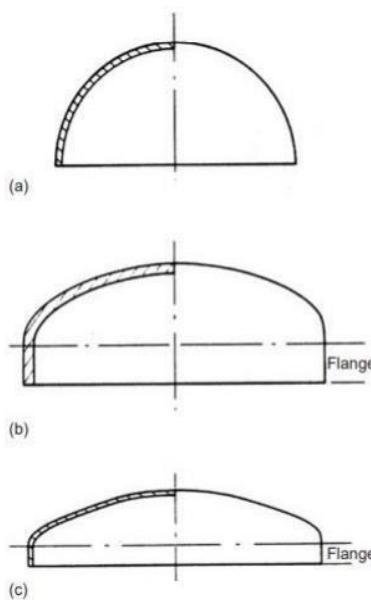


Figure 9.1: Doomed Heads (a) Hemispherical (b) Ellipsoidal (c) Torispherical (Source: Sinnott, 2005)

Considering a flat head and bottom

The thickness and diameters of the head and bottom are same as that of the vessel shell. That is, the thickness of head and bottom =  $e = 8.297\text{mm} = 8.297\text{m}$

### **Access Port design**

Access ports go by several different names; manholes, manways and hatches. They allow access to the internal parts of the equipment for periodic maintenance, inspection and

servicing of the reactor vessel. Flat plates are used as covers for access ports. The typical diameter of manholes for vertical vessels is mentioned as 18 inches (450 mm) according to ICARU corporation (1998). Hence, the manhole diameter used on the reactor is 450 mm. this value also conforms to the rule of thumb that the manhole diameter should not be greater than half of the vessel diameter.

### Nozzles

The inlet and outlet nozzles of the tank are designed to match pipes specified to transport material to and from the tank.

### Weight analysis

#### Weight of hollow, $W_v$

Height of shell = height of vessel,  $h = 5.929\text{m}$ ,

$t_s$  = shell thickness =  $0.00829\text{m}$

vessel internal diameter,  $d = 2.964\text{m}$

vessel outer diameter,  $d_e = 2.981\text{m}$

Density of stainless steel 302,  $\rho_{ss} = 7900\text{kg/m}^3$

Volume of stainless-steel material used in constructing vessel =  $V_v$

$$V_v = \frac{\pi}{4} (d^2 - d_i^2) h = \frac{\pi}{4} (2.981^2 - 2.964^2) \times 5.929 = 0.4594\text{m}^3$$

Weight of hollow vessel,  $W_v = \rho_{ss} V_v g = 7900 \times 0.4594 \times 9.81 = 35603\text{N}$

#### Weight of impeller blades and shaft rod, $W_i$

length of shaft,  $l_s = 5.138\text{m}$

diameter of shaft,  $d_s = 0.296\text{m}$

impeller diameter,  $d_i = 1.186\text{m}$

length of impeller blade,  $l_i = 0.296\text{m}$

width of impeller blade,  $w_i = 0.237\text{m}$

$$\text{volume of shaft rod, } V_s = \frac{\pi d_s^2 l_s}{4} = \frac{\pi(0.296^2)(5.138)}{4} = 0.355\text{m}^3$$

volume of all 6 impeller blades,  $V_b = 6L_i w_i e = 6(0.296 \times 0.237 \times 0.00829)$

volume of all 6 impeller blades,  $V_b = 0.0035\text{m}^3$

$$W_i = (V_b + V_s)\rho_{ss}g = (0.355 + 0.0035)(7900)(9.81) = 27754\text{N}$$

### **Weight of bottom, $W_b$**

$$W_b = \frac{\pi d^2 e}{4} \rho_{ss} g = \frac{\pi(2.964^2)(0.008297)}{4} (7900)(9.81) = 4437\text{N}$$

### **Weight of top closure, $W_u$**

$$W_u = W_b = 4437\text{N}$$

### **Weight of complete empty tank, $W_t$**

$$W_t = W_u + W_b + W_i + W_v = 4437 + 4437 + 27754 + 35603 = 72234\text{N}$$

### **Weight of the mixing tank content, $W_c$**

$$W_c = \rho_m \times v_w \times g$$

$$\rho_m = \text{density of mixture} = 1386.9\text{kg/m}^3$$

$$v = \text{volume of mixture} = \text{working volume of tank} = 1.28\text{m}^3$$

$$W_c = 1386.9 \times 1.28 \times 9.81 = 417554\text{N}$$

$$\text{Dead weight} = \text{Total weight of vessel during usage}, W = W_c + W_t = 17398 + 72234$$

$$= 489788\text{N} = 489.7888\text{kN}$$

### **Stress analysis on column vessel**

#### a) Longitudinal stress ( $\sigma_L$ )

$$\sigma_L = \frac{P_d \times d}{4e}$$

where  $P_d$  = design pressure,  $P = 111458\text{ Pa}$

$d$  = diameter of tank =  $2.964\text{ m}$

$e$  = thickness of tank =  $0.008297\text{m}$

$$\sigma_L = \frac{111458 \times 2.964}{4(0.008297)} = 9955245 \text{ Pa} = 9.955 \text{ N/mm}^2$$

b) Circumferential stress, ( $\sigma_h$ )

$$\sigma_h = \frac{P_d \times d}{2e}$$

where  $P_d$  = design pressure,  $P = 111458 \text{ Pa}$

$d$  = diameter of tank = 2.964 m

$e$  = thickness of tank = 0.008297 m

$$\sigma_h = \frac{111458 \times 2.964}{2(0.008297)} = 19910851 \text{ Pa} = 19.911 \text{ N/mm}^2$$

c) Stress due to dead weight ( $\sigma_w$ )

$$\sigma_w = \frac{W}{\pi(d + e)e}$$

where  $W$  = total weight of vessel and content = 489.7888 kN

$d$  = diameter of vessel = 2.964 m

$e$  = thickness of vessel = 0.008297 m

$$\sigma_w = \frac{489.7888}{\pi(2.964 + 0.008297)(0.008297)} = 6320 \text{ kPa} = 6.32 \text{ N/mm}^2$$

## Vessel Support Design

Type of supports to be provided to a vessel depend upon its configuration, height to diameter ratio of the vessel, convenience of location, operating temperature and materials of construction. The attachment of support at suitable location helps to transfer the load from the vessel to support safely. Cylindrical vessels can be installed either in vertical position or horizontal position and the types of supports for such installations are different. Similarly, for spherical vessels, the types of supports required vary considerably as compared to cylindrical vessels. It is necessary to analyze the stresses induced very carefully, so that adequately strong and cost-effective supports can be provided. Supports failure can lead to major hazards in a chemical process plant. Type of supports to be provided to a vessel depends upon its configuration, height to diameter ratio of the vessel,

convenience of location, operating temperature and materials. There are four types of supports commonly used.

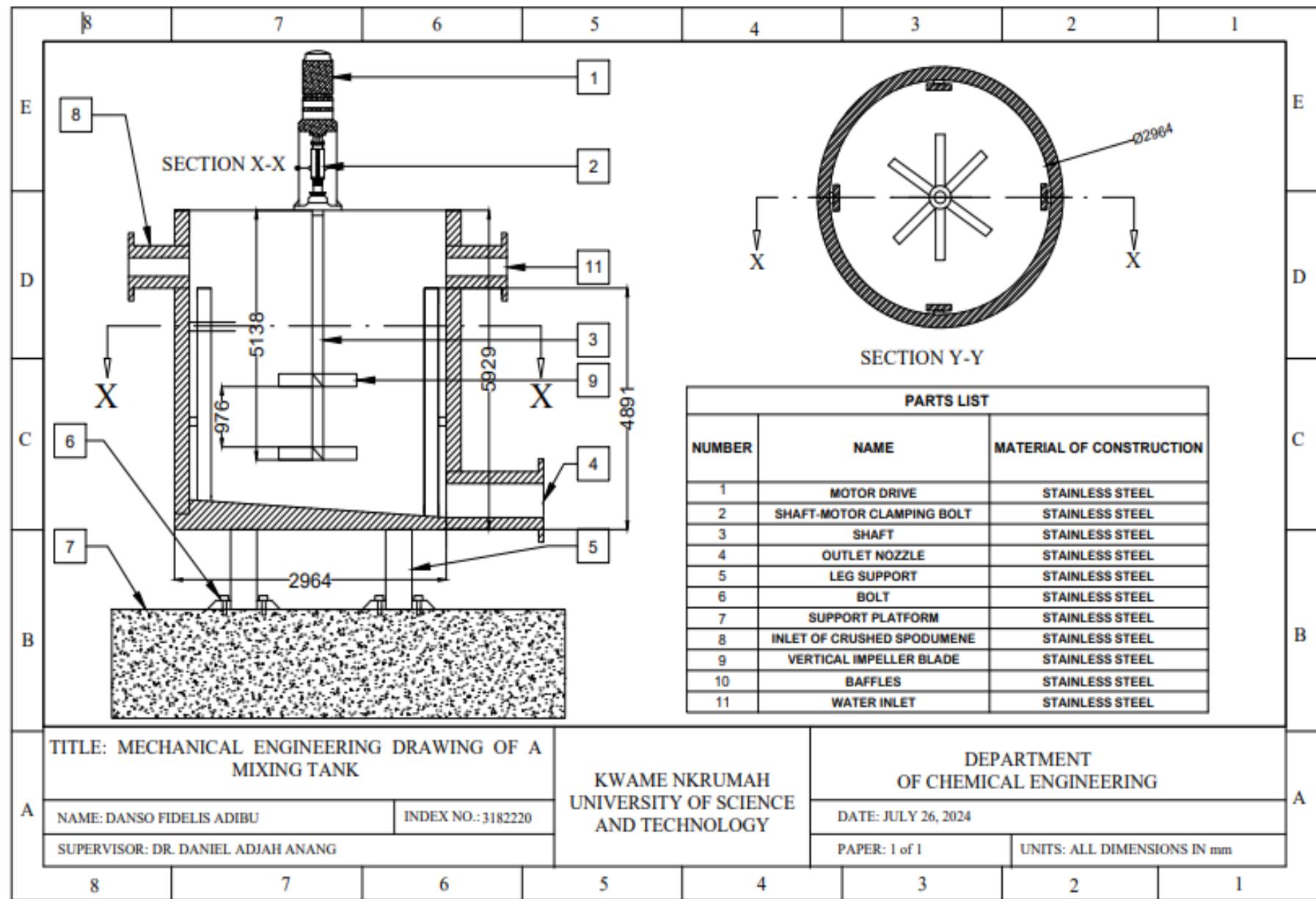
1. Skirt support: This is typically used for very tall vertical columns with torispherical bottom.
2. Bracket or Lug support: This is typically used for short vertical columns with torispherical bottom.
3. Saddle support: This is typically used for very horizontal columns.
4. Leg support: this is usually used for small tanks with light loads. However, the tank will be mounted on a well levelled concrete. This is because, the tank is small and its content is heavy. There will be a 1mm thick rubber insulator covering the concrete so as to cushion the mixing tank. The tank will be anchored to ground with bolts and nuts.

#### Summary of mechanical calculations

Parameter	Value	Units
Material of Construction	Stainless Steel 302	-
Design Pressure	111458	Pa
Design Temperature	90	°C
Thickness	8.30	mm
External diameter of vessel	2.981	m
Column Head and bottom	Flat	-
Weight of empty mixing tank	72234	N

Weight of mixture	17398	N
Dead Weight of Column	23318	kN
Dead Weight Stress	489	kN
Longitudinal Stress	9.95	MPa
Circumferential Stress	19.91	MPa
Support Type	Leg support on Concrete covered with rubber	





## **CHAPTER ELEVEN**

### **11.0 DESIGN OF A BALL MILL**

#### **11.1 INTRODUCTION**

##### **Comminution/Size reduction**

Comminution is the process of progressively reducing the particle sizes of materials until a desired particle size is obtained. Size reduction is done in majority of process industries including the food, pharmaceutical and mining industries. In the mining industries, comminution is done mainly to liberate the valuable mineral(s) locked up in the ore. These are the stages of comminution in mining.

1. Blasting: This involves the use of explosives to break up rocks into smaller fragments that can be handled and transported easily by scrapers/excavators, conveyors and haul and dump trucks for further size reduction.
2. Crushing: In the second stage, primary and secondary crushers are employed to reduce the size of the rocks to a few millimeters.
3. Grinding: The final stage of comminution involves the use of mills to grind rock material to obtain fine and super fine particles (below  $200\mu\text{m}$ ).

#### **11.2 Problem Statement**

Designing of a mill to grind spodumene ore in a dry state from a feed size of 22 mm to a product size of  $380\mu\text{m}$  at a feed rate of 20000 kg/h.

#### **Objectives**

##### **Main Objective**

To design a mill for the size reduction of spodumene ore.

##### **Specific objectives**

1. To compute the chemical and mechanical engineering design calculations of a ball mill.
2. To provide a mechanical engineering drawing of a ball mill.

## **11.2 Literature Review**

### **Milling**

Milling is a mechanistically-driven process that breaks solid materials into smaller pieces.

In milling, materials are broken down due to a few basic breakage mechanisms. Some particles tend to split, creating smaller particles that are one-half to one-third the size of the original particles. This type of breakage is termed ‘fracture’. In some cases, particles break as corners are knocked off or subsurface cracks form particles that are considerably smaller than the original particles. This type of breakage is termed ‘abrasion’. Some materials require many impacts or stress-strain events to induce breakage. This is termed ‘fatigue’ and can result in the formation of both small and large particles. The rate of breakage always changes during a fatigue-driven event. Some particles require large strains before breakage occurs. These large strains may be due to plastic deformations, such as in the case of plastic, rubber and ductile metals, separation of fibers and internal structures as in biomass, or due to non-linear elastic effects, such as with tissues and complex polymers(Kerry Johanson,2013,Chemical Engineering 120 (11))

### **INDUSTRIAL MILL COMPONENTS**

Mills are complex pieces of equipment that have a wide assortment of parts and components that have to work together to achieve a desired outcome. Although there are several different designs of mills, all types have certain components and pieces that are common. The most basic and beginning component of a mill is its shell, which has to be strong, durable, sturdy, and resilient to be able to withstand the demands of the milling process.

## **Shell**

Shells are made of a thick steel plate that is welded and sealed into a cylinder such that it will not distort or be susceptible to failure. For the best possible results, shells are electronically welded using an automated process. A basic feature of all mills is their ability to withstand impact and heavy loads. Holes are drilled in the shell such that bolts can be used to hold the liner of the mill, which is not necessary with wedge type liners.

## **Flanges**

Flanges are designed to meet the dimensions and material that will be processed by a mill. They must be uniform in diameter and may have recesses and undercuts near the arbor sleeve. It is essential that flanges be properly maintained in regard to flatness, finish, balance, and wear. The securing of flanges must be completed with care since over tightening can warp a flange and cause other damage.

## **Shell Liner**

The shell liner is a protective covering that lines the inner part of the shell and protects the mill shell from forces created by the milling process. They are made of various materials depending on the type of mill and material to be milled. Rubber liners are used in ball mills while metal liners are used in autogenous (AG) and semi-autogenous (SAG) mills. The design of the liner of a mill determines the movement of the media.

**AC Lorain Shell Liners -** A C shell liners are used in Allis-Chalmers mills and are made of carbon rolled steel with lifters.

**Wave Ball Mill Shell -** Wave liners are made with lifters and liner sections manufactured as one piece and are bolted in place by bolts placed through the liner and shell.

**Wedge Shell Mill Liners** - Wedge shell mill liners are used where the liner is connected without the use of bolts. They have a wave pattern and are made of highly resilient materials.

**Mill Spiral Shell Liners** - Mill spiral shell liners have a spiral shape that shoves the material along the length of the mill that causes the material to advance rapidly (911 Metallurgist, 2017).

### **Mill Heads**

Mill heads are made of iron or steel. They are bolted to the flange of the shell and are conical shaped or have a starfish design. Ball, rod, preliminary, Ballpeb and Compeb mills have conical heads while longer Ballpeb and Compeb have starfish heads, which offer extra support and strength due to the braces. An essential part of a mill head is its trunnion bearings that provide support for the revolving material in the mill.

### **Pinion Shaft**

The pinion shaft is the drive mechanism for a mill that moves the gear train that turns the mill assembly. It is mounted on roller bearings that have a decreased friction coefficient, which requires minimum lubrication and has excellent seal. Lubrication is provided from a reservoir located at the bottom of the gear housing where a lubricating pinion mates with the main gear.

### **Mill Drive**

Smaller mills of less than 250 hp have a V belt drive while larger mills of over 250 hp have direct drive. The motors for direct drive mills have wound rotor motors that are connected directly to the pinion shaft. Several types of motors are available to drive a mill with varying torques that are determined by a mill's design and its manufacturer.

## **Mill Feed Chutes**

Mill feed chutes come in a wide array of shapes, sizes, and configurations. The use of a chute depends on the industry and type of material to be milled with mining, cement, and rock production having the choice of mills with or without a chute. The types of feeders include combination, spout, scoop, and drum types.

**Spout Feeder** - A spout feeder has a very simple design. They have a steel or rubber lined chute that feeds directly into the trunnion opening of the mill.

**Scoop Feeder** - Scoop feeders return media to the mill, have a single or double scoop design, and may be used to take in the initial feed. They are used when the size of the feed is very fine. The internal structure of a scoop feeder uses a spiral to pass the feed to the trunnion.

**Drum Feeder** - Drum feeders are used in place of spout feeders. They have a drum and are used for single pass grinding. Drum feeders receive the mill feed via a chute and send the feed to the trunnion using a spiral configuration.

**Combination Feeder** - A combination feeder consists of a drum and scoop lift that returns media to the mill. The feed enters the drum like a drum feeder (911 Metallurgist, 2017)

## **Discharge Unit**

Once the material has been ground and processed, it is removed from a mill through the discharge unit. Discharging completed material from a mill can be completed in different ways depending on the design of the mill. In many cases, it resembles pouring liquid from a pitcher or barrel. The types of discharge methods or units are overflow, peripheral, and diaphragm.

**Overflow Discharge** - The process used by an overflow discharge unit is referred to as displacement where new material enters displacing an equal volume of processed material.

**Peripheral Discharge** - A peripheral discharge mill has an opening in the center of the mill or at one end through which the processed material is discharged and released.

**Diaphragm Discharge** - A diaphragm discharge mill has a diaphragm made of perforated steel placed near the discharge head. Lifters raise the ground material and drop it into a discharge cone that directs the material to the trunnion and out of the mill. A diaphragm discharge design keeps the grinding media in the mill as it releases the processed material.

### **Industrial Mill Grinding Media**

Grinding media is used to refine and reduce particle sizes to produce a wide variety of shapes and sizes to meet application needs. There are several varieties of grinding media that include metallic and non-metal types, each of which is designed to process various kinds of materials. The selection of the type of grinding media determines the success of the grinding process.

#### *Metallic Grinding Media*

Metallic grinding media includes carbon steel, forged steel, stainless steel, and chrome steel, which come in the form of balls, beads, bars, and tiny cylinders.

- Steel balls are the most common form of grinding media and come in diameter sizes of 20 mm up to 120 mm to meet different grinding requirements.
- Short tiny cylinders are used with open circuit and closed circuit grinding systems and used with fine grinding chambers.
- Steel bars are cut to the size of the mill shell and are used with ball mills. Bars can be 50 mm to 100 mm and are shorter than the milling shell.

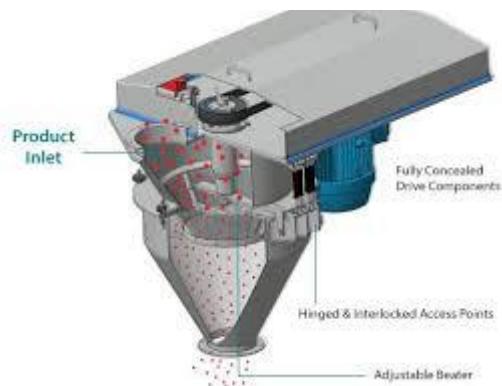
## Non-Metallic Grinding Media

Non-metallic grinding media includes alumina balls, glass beads, silicon carbide balls, silicon nitride balls, and ceramic balls. These media are used where contamination is a concern. The choice of non-metallic media is highly dependent on the type of material to be processed since the wrong choice can damage the interior lining of a mill (R.I. King et al.,

## TYPES OF MILLS

### Conical Mills

A conical mill is a type of mill that is used to reduce the size of materials uniformly. It is also referred to as a conical screen machine. It is a hammer mill alternative. As the name implies, the mill varies in its diameters between its opening, in which the feed enters, and where the product leaves. The conical mill functions by having the required product fed into the mill either by the force of gravity or a vacuum. The rotating impeller will force the material outward to the conical screen surface. The materials are resized on the conical screen surface, and, once finished, the product drops through the chamber where it is collected below.



**Figure 51: Figure of labelled conical mill**

The conical mill has exceptionally marked advantages when compared to other mills. The conical mill has low noise pollution, minimum heat and dust releasing, and a more-

uniform particle size, flexibility in design, and a higher carrying capacity. The principal concept of the conical mill originated back in 1976 and has been developed little by little where, today, they are made with a variety of sieves and impellers. The mill head is mounted on a mobile hoist which allows for an easy up-or-down movement for container docking, tablet presses, and other processes. The mill head of a conical mill can be dismounted for easy cleaning access.

### **Hammer Mills**

Modern hammer mills operate on a rather straightforward principle. To crush the intended material, one needs a suitable engine, strong crushing hammers, and knives. A quick-moving rotor's hammer impacts the stationary powder bed. The hammer mills break down the materials between the hammers and the casing and then sieve them until they are refined enough to fall through the sieve at the bottom. The hammer mill handles both the fibrous and brittle materials. The fibrous materials will require projecting parts on the casing to facilitate a cutting motion. Hammer mills are categorized into one of two types, namely the horizontal shaft or the vertical shaft. Abrasion-resistant materials can be used to coat and protect impact surfaces made of materials like steel or stainless steel.



**Figure 52: a hammer mill**

The particles are fed into the crushing chamber through the feeding mechanism. The hammer mill makes use of gravity or a metering system for feeding and this greatly

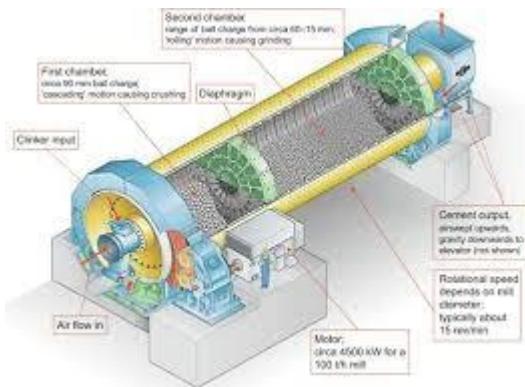
depends on the design of the hammer. In some applications where the product's uniformity is important, the metering systems are recommended since they eliminate any variables that may hinder the quality of the resultant product. In this case, a pneumatic rotating valve can be found between the crushing chambers; a feeding hopper is a good example of this type of mechanism.

The insertion of particles into the crushing chambers is facilitated entirely by the force of gravity. A control box can be used to turn the machine on or off. The electric motor and feeding system can be controlled depending on user requirements. Some pharmaceutical milling machines have a display panel that allows users to monitor every process that is done. The mill can run at high speeds between 2,500 to 60,000 revolutions per minute. The hammers will typically rotate either clockwise or anticlockwise on the horizontal shafts. This can be affected by the rotation of the rotor.

### **Ball Mills**

The phrase “ball mill” is used to define a grinding mill that has spherical balls inside. The length covered by the mill is equal to the diameter of the mill body. They are widely used in metallurgy, electric power, and mining activities. The ball mills are very adaptable and can be used across a wide variety of materials and have an outstanding capacity. Their main characteristic is that ball mills are very effective at processing both wet and dry materials.

Balls are used in ball mills to grind materials. The ball mill is switched on and allowed to begin rotating, either on its vertical or horizontal axis. The material is put into the chamber of the ball mill, where it is subjected to the spinning balls. The balls strike the contained substance as they bounce about in the ball mill while it spins. These blows have enough force to reduce the material to a finer, less coarse medium as the material gets ground into a finer, powder-like material by the rotation's force.



**Figure 53: a labelled ball mill**

A ball mill has to reach critical speed in order to function. The critical speed of a ball mill is the speed at which the contained balls start to revolve along the inside walls. The balls inside a ball mill will remain motionless at the bottom where they will have little to no effect on the material if the threshold speed is not reached.

Ball mills have some key differences from conventional milling equipment. First off, a cutting tool is absent from ball mills. Ball mills use the force of moving balls instead of the rotary cutting tool that conventional milling machines use. They don't require a cutting tool as a consequence.

Ball mills and conventional milling machines differ from one another in their distinct functions. Ball mills and conventional milling equipment are both used to grind materials into smaller pieces. Ball mills can handle materials like ore, ceramics, and paint, whereas conventional milling machines can handle huge workpieces.

### Tube Mills

In a tube mill, the grinding bodies are spherical but the length of the mill body is longer in proportion to the diameter. They consist of rotating cylinders with a perfectly plain interior and hard round balls. The lining of the mills is made of hardwood or iron alloys like steel or manganese steel.



**Figure 54: a ball mill**

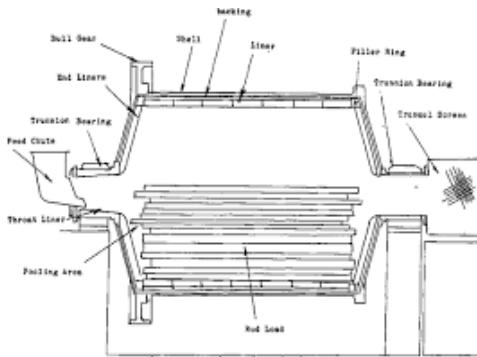
The revolving cylinders that make up a tube mill typically have a simple interior. On their interior walls, there are a lot of flat balls. They are different from roller mills in that they don't have a separate cooling system for the welded arc, which means a third cooling source is needed. Additionally, they frequently employ more advanced welding tools, such as cutting-edge laser welders. To further enhance the quality of the end product, the tubes are occasionally hand-woven as well. Perhaps the best-known use of these mills is their creation of plumbing tubes or pipes in sizes that are typically available in plumbing material warehouses and construction supply stores. Tube mills typically consist of a long, cylindrical drum with a number of balls on each side of it that can spin at very high rates and provide the hardness required for drainage systems.

Similar to belt grinders, tube mills operate by the operator loading the material (whether wire or sheet steel) into the drum, which then activates the "grinders." When cutting wire or sheet steel to the required diameter, the "grinder" will oscillate back and forth within the cylinder. For this technique, some will use a manual crank, while others will utilize an electric motor. A common version of this fundamental technique involves either inserting wire from the inside or completely removing the object by grinding it from outside. This is done to get a constant diameter and constant profile that will make fitting and cleanup simple.

## Rod Mills

In contrast to ball mills, rod mills employ long rods as the grinding medium. Similar to grinding balls in a ball mill, the rods tumble inside the mill to grind materials like limestone or various ores. The length-to-diameter ratio is kept between 1.4 and 1.6 to avoid circumstances that might result in rod charge tangling. Rod mills may process feed as small as 50 mm and deliver products with sizes ranging from 3000 mm to 270 mm. Line contact between the rods that stretch the length of the mill produces the grinding action. In order to simulate a row of roll crushers, rods spin and tumble in a parallel orientation. As a result, the generation of slime is reduced and coarse material is ground more effectively.

Only the overflow mill—not the end peripheral discharge or the center peripheral discharge—is one of the three primary types of rod mills widely used. Rod mills for wet grinding are often employed in the mineral processing sector. Although it is utilized in some places, dry grinding has drawbacks and should only be used when absolutely required. Because the rods are rolled rather than cascaded, rod mills run at a slower pace than ball mills. Rod mills require less steel than ball mills for a comparable grind because the media and ore make better contact at a reduced speed. Broken and damaged rods should be removed in order to keep the rod charge in excellent operating order. Rod mills often demand more operator focus. The rods must remain almost parallel to each other at all times. Rod tangles and reduced grinding action are the two effects of misaligned rods. The highest rod length is around 6.1 meters. As a result, rod mills' length, diameter, and capacity are constrained. Bulkier rods exerting force on the loaders and liners cause the mill liners to wear out more quickly.



**Figure 55: a labelled rod mill**

### **11.3 SELECTION AND JUSTIFICATION**

The selection of an appropriate mill for a process industry is a difficult task that leaves engineers and plant owners relying on vendors to get the right machine. Factors that must be considered in selecting a mill for a process plant include, whether the material will be milled in a dry or wet state, the hardness of the material to be milled, the type of grinding media you are able to use, whether the process will be batch or continuous and of course the cost of the mill, its installation and operation.

A ball mill was chosen for this process because:

1. It is suitable for milling toxic materials since it can be used in a completely enclosed form.
2. It can produce superfine particles (particle size less than 10 microns).
3. It has a wide range of applications.
4. It can be used for both continuous and batch operations.
5. It can be used for both dry and wet milling.
6. It can be used for processes with high throughput.

## **11.4 CHEMICAL ENGINEERING DESIGN**

Certain parameters must be specified in order for the proposed ball mill to be designed for the process.

### **Motor Power**

The power required for a ball mill to operate is determined by using Bond's law. The law states that the energy input required to mill a material is proportional to the new crack tip length produced in particle breakage (Wills & Napier-mum, 2006). Mathematically

Bond's law is written as  $W = 10 \times W_i \times \frac{1}{\sqrt{P_{80}}} \times \frac{1}{\sqrt{F_{80}}}$

Where,  $W$  = work input, kWh/ton

$W_i$  = work index of material (spodumene) = 11.43 kWh/ton (Table of materials reported by Fred Bond)

$P_{80}$  = diameter of sieve aperture that allows 80% of the product to pass through

$F_{80}$  = diameter of sieve aperture that allows 80% of the feed to pass through

$$P_{80} = [(Span \text{ of product size}) \times \% \text{ passing through sieve} + \text{minimum product size}] \times 10^3$$

Product size span = maximum product size - minimum product size

$$= 0.425 \text{ mm} - 0.212 \text{ mm} = 0.213 \text{ mm}$$

$$= [(0.213 \times 0.8) + 0.212] \times 10^3 = 382.4 \mu\text{m}$$

$$F_{80} = [(Span \text{ of feed size}) \times \% \text{ passing through sieve} + \text{minimum feed size}] \times 10^3$$

$$F_{80} = [(25-10) \times 0.8 + 10] \times 10^3 = 22000 \mu\text{m}$$

$$W = 10 \times 11.43 \times \left[ \frac{1}{\sqrt{382.4}} - \frac{1}{\sqrt{22000}} \right] = 5.0744 \text{ kWh/ton}$$

$$m = 20000 \frac{\text{kg}}{\text{h}} \times \frac{1 \text{ ton}}{1000 \text{ kg}} = 20 \text{ ton/h}$$

$$\text{Power} = m \times W = 5.0744 \times 20 = 101.5 \text{ kW}$$

About only 1% of the energy input into ball mill and rod mills are used in effective comminution (Wills & Napier-munn, 2006)

$$\text{Therefore actual power needed} = 101.5 \times 100 = 10.15 \text{ MW}$$

### **Reduction Ratio**

The reduction ratio of a material is the ratio of the average feed size to the average product size. It indicates the proportion by which the feed size has been reduced. The higher the reduction ratio, the more the energy needed for milling.

$$R = \frac{F}{P} = \frac{22000}{382.4} = 57.5$$

### **Design of grinding media**

The grinding media in this case are steel balls. Larger balls are needed for grinding coarser particles while smaller balls are used for grinding finer particles. The expression used in determining the diameter of the grinding media balls is:

$$D_m = 6(\log d_k) \times d^{0.5}$$

Where  $D_m$  = diameter of the single-sized ball, mm

$d$  = diameter of the largest chunk of particle in the mill feed, mm

$d_k$  = the  $P_{90}$  or size of the finished product,  $\mu\text{m}$

$$P_{90} = [(\text{span of product size}) \times \% \text{ passing through sieve} + \text{minimum product size}] \times 10^3$$

$$P_{90} = [(0.425 - 0.212) \times 0.9 + 0.212] \times 10^3 = 403.7 \text{ mm}$$

$$D_m = 6 \times (\log 403.7) \times 25^{0.5} = 60.56 \text{ mm} = 0.06056 \text{ m}$$

$$R = \frac{0.06056}{2} = 0.03028 \text{ m}$$

Mass of balls = bulk density of steel × volume occupied by balls =  $\rho_b \times V_b$

$$\text{Volume of feed materials, } V_f = \frac{m}{\rho}$$

Where  $m$  = mass of feed = 20000 kg/h

$$\rho = 2790 \text{ kg/m}^3 \text{ (SI metric, 2016)}$$

$$V_f = \frac{20000}{2790} = 7.1685 \text{ m}^3$$

The ratio of the volume occupied by the balls to that of the feed should be = 0.4:0.6  
(Richardson et al, 2002)

$$\text{The volume of balls, } V_b = \frac{0.4}{0.6} \times 7.1685 = 4.78 \text{ m}^3$$

Density of steel is 7850 kg/m<sup>3</sup> (SI metric, 2016)

$$\text{Mass of balls} = 7850 \text{ kg/m}^3 \times 4.78 \text{ m}^3 = 37523 \text{ kg}$$

### Number of Balls

$$N = \frac{V_b}{V_i}$$

$V_b$  = bulk volume of all balls, m<sup>3</sup>

$V_i$  = volume of one ball, m<sup>3</sup>

$$\text{Volume occupied by one grinding ball, } V_i = \frac{4}{3}\pi r^3$$

$$V_i = \frac{4}{3}\pi \times (0.03028)^3 = 1.163 \times 10^{-4} \text{ m}^3$$

$$N = \frac{4.78}{0.0001163} = 41101 \text{ balls}$$

## Diameter of Mill

The section of the ball mill where milling occurs is cylindrical. Due to this, the volume of the ball mill,  $V = \pi R^2 L$

Where,  $V$  = volume of the mill,  $m^3$

$R$  = radius of the mill,  $m$

$L$  = length of the mill,  $m$

For effective milling, the charge is to occupy 40% of the volume of ball mill (Wills & Napier-munn, 2006)

If 40% of volume of mill = volume of balls + volume of spodumene ore =  $V_b + V_f$

$$0.4Vm = 4.78 + 7.1685 = 11.95 m^3$$

$$Vm = \frac{11.95}{0.4} = 29.88 m^3$$

For a closed circuit ball mill with two compartments, the length:diameter ratio is 1.5:1 (Richardson et al,2002)

$$\frac{L}{2R} = \frac{1.5}{1}, L = 3R$$

By substitution,  $V = \pi R^2 (3R) = 3\pi R^3$

$$R = \sqrt[3]{\frac{V}{3\pi}} = \sqrt[3]{\frac{29.875}{3\pi}} = 1.47 m$$

Length of mill,  $L = 3 \times 1.47 = 4.41 m$

Diameter of mill =  $2 \times 1.47 = 2.94 m$

The height of charge also known as the height inside liners of the ball mill can be determined by using the percentage loading graph (D.J. McKee, 2012). The percentage loading of the mill is related to the ratio of height inside mill liners to the radius of the

mill. The percentage loading of the balls is assumed to be 40% (Richardson et al., 2002).

$H = 0.86$ , where  $H$  is the height of inside lining and  $R$  is the radius of the ball mill.

$$H = 0.86R$$

$$H = 0.86 \times 1.47 = 1.264 = 1.26 \text{ m}$$

Assuming a liner thickness of 40 mm, the effective diameter of the ball mill,

$$D_m = \text{Mill diameter} - (\text{Liner thickness} \times 2)$$

$$= 2.94 - (0.04 \times 2) = 2.86 \text{ m}$$

### Critical Speed

The critical speed of a mill is the speed at which the centrifugal forces equal the gravitational forces at the mill shell's inside surface and no balls will fall from their positions. The theoretical critical speed of rotation is the speed at which an infinitely small particle will cling to the inside of the liners for a complete revolution. Mathematically,

$$N_c = \frac{76.63}{\sqrt{D_m}} \quad (\text{D. J. McKee, 2012})$$

Where  $N_c$  = critical speed of ball mill, rev/min

$$D_m = \text{effective mill diameter} = 2.86 \text{ m} = 9.46 \text{ ft}$$

$$\text{Therefore the critical speed, } N_c = \frac{76.63}{9.46} = 24.9 \text{ rev/min}$$

Most ball mills operate at speeds between 65 and 79 percent of critical speed.

$$\text{Actual speed of ball mill, } N_A = 0.75 \times N_c \quad (\text{D. J. McKee, 2012})$$

$$N_A = 18.675 \text{ rev/min}$$

## **Design of Liners and Lifters**

In a ball mill both the trunnion and shell have liners and lifters. In milling this feed, magnetic liners will be used. The magnetic liner is a steel-encased liner. The magnet holds the metal magnetic liner to the shell and retains ball chips and magnetic minerals to form a protection layer which serves as the wear liner. The advantages of using the magnetic liner include:

1. They are lighter in weight than traditional steel liners, easier to handle and install.
2. Maintenance cost is greatly reduced since maintenance is not needed frequently
3. It lowers medium consumption
4. It prevents backwashing of mill.
5. It generates less noise compared to other mill liners. (Letwin Mubonesi, 2023)

The thickness of the liner is 40 mm and that of the lifter is 50 mm.

### **Summary of chemical engineering design**

Parameter	Value
Feed rate, kg/h	20000
Feed size, mm	22
Product size, mm	0.38
Reduction ratio	57.50
Work index of feed, kWh/ton	11.43
Motor power, MW	10.15
Theoretical critical speed, rev/min	24.90

Actual speed, rev/min	18.68
Diameter of ball mill, m	2.94
Effective diameter of ball, m	2.86
Volume of ball mill, m <sup>3</sup>	29.88
Length of ball mill, m	4.41
Volume of feed, m <sup>3</sup>	7.17
Height of charge inside of ball mill, m	1.26
Loading	40%
Diameter of grinding balls, mm	60.56
Mass of grinding balls, kg	37523
Volume occupied by grinding balls, m <sup>3</sup>	4.78
Number of grinding balls	41101
Liner thickness, m	40
Lifter thickness, m	50
Material of construction of shell	Mild steel
Material of construction of grinding balls	steel
Material of construction of lining	Steel-encased magnet

## **11.5 Mechanical Engineering Design**

### **Dead weight of ball mill**

The dead weight of the ball mill,  $W = \text{mass of the ball mill} \times \text{acceleration due to gravity}$

$$\text{Mass of ball mill} = V_m \times \rho$$

Where  $V_m$  = volume of ball mill =  $29.875 \text{ m}^3$  and  $\rho$  = density of mill =  $7850 \text{ kg/m}^3$

$$\text{Mass of the ball mill} = 29.875 \times 7850 = 234518.75 \text{ kg}$$

$$\text{Therefore, } W = 2300629 \text{ N}$$

The maximum dead weight compressive stress,  $\sigma_w = \frac{\text{dead weight}}{\text{area}}$  (the engineering toolbox, 2017)

$$\text{Area of shell of ball mill} = 2\pi r(r+L)$$

Where  $r$  = radius of shell and  $L$  = length of shell

$$\text{Area} = 2\pi \times 1.47(1.47+4.41) = 54.31 \text{ m}^2$$

$$\sigma_w = \frac{2300629}{54.31} = 42361 \text{ N/m}^2$$

For ball mills with diameter 2.5-3.5, shell thickness is normally equal to about 38 mm  
(cementequipment.org, 2023)

### **Design Pressure**

$$P = \frac{T \times t \times E}{R \times f} \text{ (Sinnott, 2012)}$$

$T$  = tensile strength of material =  $400 \text{ MPa} = 58015$  (Perry et al., 2018)

$t$  = Shell thickness, inch

$E$  = longitudinal seam efficiency, assumed to be 0.85 (PV Elite, 2017)

R = interior radius, inch

S = safety factor, assumed to be 3 (the engineering toolbox, 2017)

$$P = \frac{58015 \times 1.496 \times 0.85}{57.874 \times 3} = 424.9 \text{ psi} = 2.93 \text{ MPa}$$

Longitudinal stress of ball mill,  $\sigma = \frac{P \times r}{2t}$

r = radius of mill, P = design pressure and t = shell thickness (Sinnott et al., 2012)

$$\sigma = \frac{2929582 \times 1.47}{2 \times 0.038} = 5666428 \text{ N/m}^2$$

Circumferential stress of the ball mill,  $\sigma_2 = \frac{P \times r}{t \cos \alpha}$  where  $\alpha$  is the angle of inclination of shell (assumed to be 0°), P, r and t are the same as in the longitudinal stress formula (Sinnott et al., 2012)

$$\sigma_2 = \frac{2929582 \times 1.47}{0.038 \cos 0} = 113328566.8 \text{ Pa}$$

## Torque

$$\text{Torque} = \frac{P}{\omega}$$

Where T = torque, P = motor power,  $\omega$  = actual speed of rotation = 19.53 rev/min = 0.1 × 19.53 = 1.953 rad/s

$$T = \frac{422.85}{1.953} = 216.513 \text{ kNm}$$

Tangential force on gear,  $F_t = \frac{2T}{PCD}$  (Engineers Edge, 2017)

Where  $F_t$  = tangential force on gear, N

T = torque of the motor, kNm

PCD = pitch centre diameter, m

$$PCD = \frac{Nt}{D_p} \text{ (Engineers Edge, 2017)}$$

Where, Nt = number of teeth on gear

$$D_p = \text{diametric pitch} = \frac{25.4}{\text{module}}, \text{ inch}$$

Module is assumed to be 3 inch (Engineers Edge, 2017)

$$D_p = \frac{25.4}{3} = 8.47 \text{ inch} = 0.22 \text{ m}$$

The number of teeth id assumed to be 40

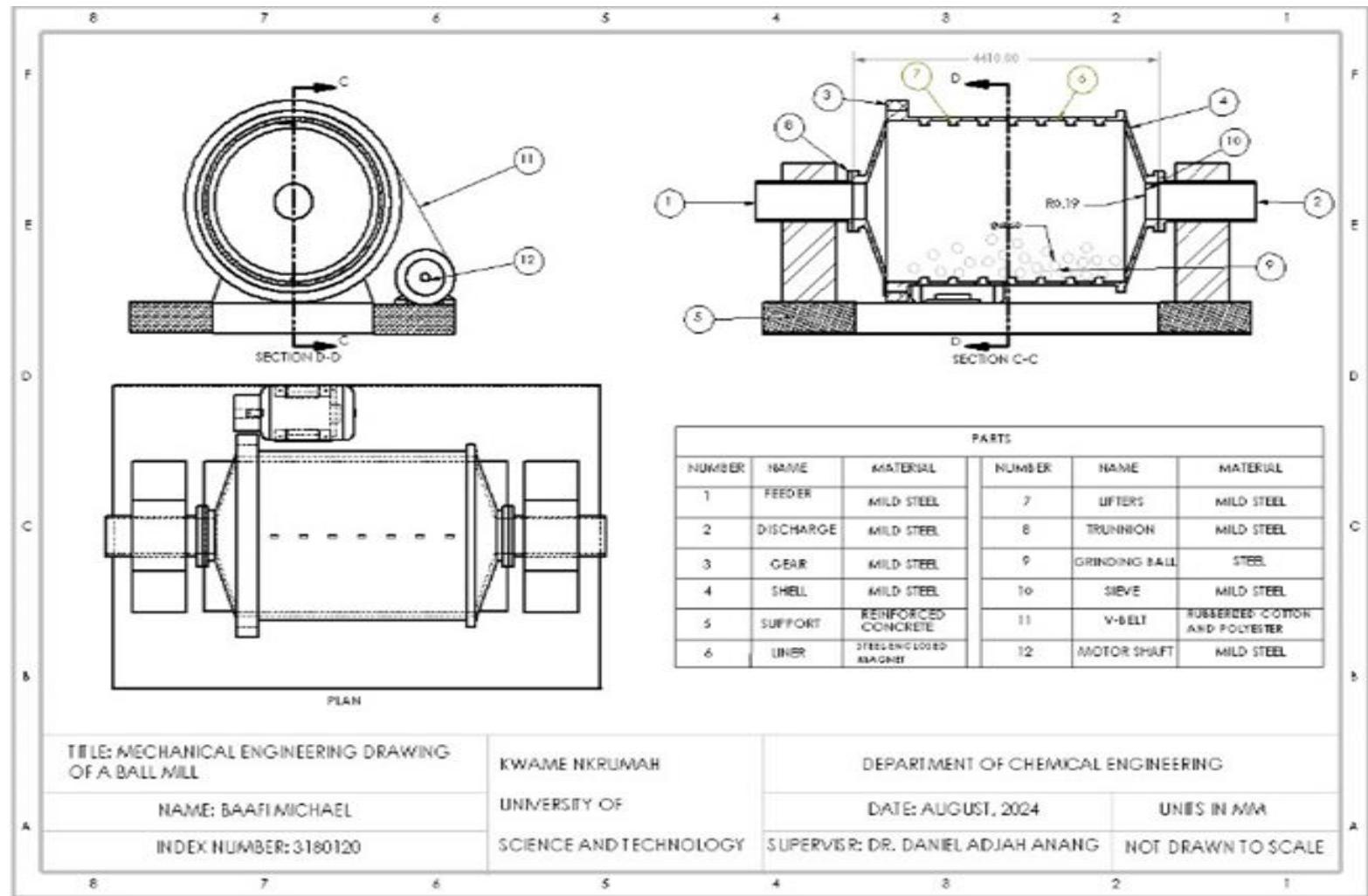
$$PCD = \frac{40}{8.47} = 4.72 \text{ inch} = 0.12 \text{ m}$$

$$F_t = \frac{2 \times 216.513}{0.12} = 3608.55 \text{ kN}$$

Table 2: Summary of Mechanical engineering design

Parameter	Value
Mass of ball mill, kg	234518.75
Dead weight of ball mill, N	2300629
Dead weight compressive stress, N/m <sup>2</sup>	42361
Shell thickness, mm	38
Tensile strength, Pa	400 × 10 <sup>6</sup>
Design Pressure, Pa	2929582
Angle of inclination of shell, °	0
Longitudinal stress, Pa	56664280
Circumferential stress, Pa	113328566.8

Torque, Nm	216513
Tangential force on gear	3608550
Mill support	Two saddles
Mill feeder	Drum feeder



## CHAPTER TWELVE

### 12.0 DESIGN OF A WET SCRUBBER

#### 12.1 PROBLEM STATEMENT

In the lithium extraction industry, processing plants emit pollutants like acidic gases, dust, and fine particulates, which pose environmental and health risks. Effective emission control is crucial to comply with stringent environmental regulations and ensure operational safety. Wet scrubbers are essential for capturing and neutralizing these pollutants through a scrubbing liquid. The design of a wet scrubber for a lithium extraction plant must consider the specific characteristics of the emissions, such as the concentration and composition of acidic gases and particulates. It must also address operational efficiency, reliability, maintenance needs, and regulatory compliance while balancing technological capabilities with economic feasibility. A well-designed wet scrubber system will reduce environmental impact, protect public health, and enhance the sustainability of lithium extraction operations.

#### 12.2 INTRODUCTION

The rapid growth of the lithium industry, driven by the increasing demand for lithium-ion batteries in electric vehicles and renewable energy storage, has highlighted the need for effective environmental management in lithium extraction processes. Lithium extraction, whether from hard rock mining or brine sources, involves several stages that can generate significant emissions of pollutants. These emissions, including acidic gases, dust, and fine particulates, pose serious risks to both environmental and human health.

To mitigate these risks, implementing efficient air pollution control technologies is essential. One of the most effective solutions for controlling gaseous and particulate emissions in industrial settings is the wet scrubber. Wet scrubbers work by introducing a scrubbing liquid

to capture and neutralize pollutants from the gas stream. This process not only removes harmful substances from emissions but also ensures compliance with stringent environmental regulations.

Designing a wet scrubber for a lithium extraction processing plant requires a thorough understanding of the specific pollutants involved, their concentrations, and the operational dynamics of the plant. Key considerations include selecting an appropriate scrubbing liquid, optimizing the system's configuration to handle variable flow rates, and ensuring the materials used can withstand corrosive environments. Additionally, the design must balance operational efficiency, maintenance requirements, and cost-effectiveness to ensure sustainable and long-term operation.

This introduction outlines the critical role of wet scrubbers in reducing the environmental footprint of lithium extraction plants and sets the stage for discussing the detailed design considerations necessary to develop an effective scrubbing system. Through careful design and implementation, wet scrubbers can significantly enhance the environmental performance of lithium extraction facilities, contributing to cleaner production processes and improved community health.

## **MAIN OBJECTIVE**

To develop an efficient, reliable, and cost-effective system capable of capturing and neutralizing a wide range of gaseous and particulate pollutants emitted during the extraction process.

## **SPECIFIC OBJECTIVES**

4. Select an appropriate wet scrubber for the process
5. Determine the capacity of the equipment
6. Perform required chemical and mechanical calculations for the equipment

## **12.3 LITERATURE REVIEW**

### **Working Principle of a Wet Scrubber**

A wet scrubber is a sophisticated air pollution control device designed to remove pollutants from industrial exhaust streams through the interaction with a scrubbing liquid. The essence of its operation lies in the processes of gas absorption, particulate collection, and chemical neutralization, all of which occur within a meticulously engineered system. Understanding the working principle of a wet scrubber involves exploring the various stages that the polluted gas undergoes to emerge as a cleaner, less harmful emission.

When polluted gas enters the wet scrubber, it typically passes through an inlet duct. At this initial stage, pre-treatment may be employed to remove larger particles or to cool the gas stream. Cooling is particularly important as it enhances the efficiency of subsequent scrubbing processes. The gas, now prepared for treatment, is introduced to the scrubbing liquid. This liquid can be distributed in various ways, such as through spray nozzles, a packed bed, or other liquid distribution mechanisms, ensuring thorough contact between the gas and the liquid.

The interaction between the gas and the scrubbing liquid can occur in different configurations, including countercurrent, cocurrent, or crossflow setups. In a countercurrent design, the gas and liquid flow in opposite directions, maximizing contact time and absorption efficiency. In cocurrent and crossflow designs, the gas and liquid flow in the same direction or perpendicular to each other, respectively, each offering unique benefits based on the specific application and pollutant characteristics.

As the polluted gas comes into contact with the scrubbing liquid, gaseous pollutants such as sulfur dioxide ( $\text{SO}_2$ ), hydrogen chloride (HCl), and ammonia ( $\text{NH}_3$ ) are absorbed. The effectiveness of this absorption process hinges on several factors, including the solubility of the gases in the liquid, the temperature of the gas and liquid, and the duration of their

interaction. Highly soluble gases are more readily absorbed, and cooler temperatures generally enhance the absorption capacity of the liquid.

In addition to absorbing gaseous pollutants, the scrubbing liquid also captures fine particulates and dust. This is achieved through mechanisms like impaction, interception, and diffusion, where the liquid droplets act as collectors. As the gas stream flows through the scrubber, particles collide with the droplets, become entrained, and are effectively removed from the gas stream.

Chemical neutralization plays a critical role in the functioning of a wet scrubber, particularly when dealing with acidic gases. The scrubbing liquid often contains chemical reagents, such as alkaline solutions, that react with and neutralize these gases. For instance, sulfur dioxide ( $\text{SO}_2$ ) can be neutralized by an alkaline scrubbing liquid like sodium hydroxide ( $\text{NaOH}$ ), resulting in the formation of sodium sulfite ( $\text{Na}_2\text{SO}_3$ ), a far less harmful compound. This chemical transformation not only neutralizes the pollutants but also facilitates their removal from the gas stream.

After the gas has been thoroughly treated, it passes through a mist eliminator or demister. This component is essential for removing any entrained liquid droplets from the cleaned gas, ensuring that only the purified gas is discharged through the outlet stack. The effectiveness of this stage is crucial for meeting environmental emission standards and preventing liquid carryover.

The scrubbing liquid, now containing absorbed pollutants and collected particulates, accumulates at the bottom of the scrubber. The handling of this spent liquid is a critical aspect of the wet scrubber's operation. It can be recirculated for further use, provided it retains sufficient capacity for pollutant absorption, or it can be treated to remove pollutants before being discharged or disposed of according to environmental regulations.

In summary, the working principle of a wet scrubber involves a series of well-coordinated processes that collectively ensure the effective removal of pollutants from industrial exhaust streams. Through gas absorption, particulate collection, and chemical neutralization, wet scrubbers play a vital role in mitigating the environmental impact of industrial emissions, contributing to cleaner air and a healthier environment.

## COMPONENTS OF A WET SCRUBBER

A wet scrubber is composed of several key components, each playing a crucial role in its overall functionality and efficiency in removing pollutants from industrial exhaust streams. These components work together to ensure effective gas absorption, particulate collection, and chemical neutralization.

### 1. Inlet Duct

The inlet duct is the entry point for the polluted gas stream into the wet scrubber. It is designed to evenly distribute the gas flow and direct it into the scrubber for treatment. In some

configurations, the inlet duct may include a pre-treatment section to remove large particulates or to cool the gas stream, enhancing the scrubbing efficiency.

### 2. Spray Nozzles or Liquid Distributors

Spray nozzles or liquid distributors are responsible for introducing the scrubbing liquid into the scrubber. They atomize the liquid into fine droplets, creating a large surface area for effective contact with the gas stream. The distribution pattern and droplet size can be adjusted to optimize the absorption of gaseous pollutants and the capture of particulates.

### 3. Scrubbing Chamber

The scrubbing chamber is the core component where the interaction between the gas and liquid phases occurs. Depending on the design, this chamber can be configured in several ways:

Packed Bed Scrubber Contains a packing material that provides a large surface area for gas-liquid contact. The packing material can be made of various materials, such as plastic, metal, or ceramics, and is designed to maximize contact time and enhance absorption efficiency.

Spray Tower Scrubber: Uses spray nozzles to create a fine mist of scrubbing liquid, which the gas stream passes through. This type is effective for gases with high solubility in the scrubbing liquid.

Venturi Scrubber: Features a constricted section that increases the velocity of the gas stream, creating a high-energy interaction with the scrubbing liquid. This design is particularly effective for removing fine particulates and droplets.

#### 4. Mist Eliminator (Demister)

The mist eliminator, or demister, is positioned after the scrubbing chamber and is responsible for removing any entrained liquid droplets from the cleaned gas stream. It typically consists of baffles, mesh pads, or other structures that coalesce the liquid droplets, allowing them to fall out of the gas stream. This step ensures that the discharged gas is free of excess moisture and liquid carryover.

#### 5. Outlet Duct

The outlet duct directs the cleaned gas from the scrubber to the stack or further treatment stages. It ensures that the gas stream is properly vented to the atmosphere or to additional air pollution control devices if necessary. The design of the outlet duct is critical for

maintaining the proper flow dynamics and ensuring the efficient discharge of the treated gas.

#### 6.Scrubbing Liquid Recirculation System

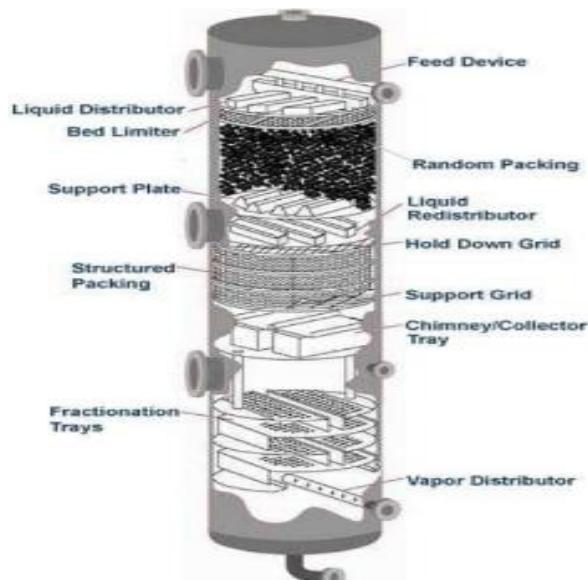
This system includes pumps, pipes, and storage tanks to manage the scrubbing liquid. It recirculates the liquid through the scrubber, maintaining a continuous supply of fresh liquid for absorption and neutralization processes. The system also include filtration or treatment units to remove contaminants from the spent liquid before it is reused or disposed of.

#### 7.Chemical Addition System

In cases where chemical neutralization is required, a chemical addition system is used to introduce reagents, such as alkaline solutions, into the scrubbing liquid. This system ensures the proper concentration of chemicals to effectively neutralize acidic gases like sulfur dioxide ( $\text{SO}_2$ ) and hydrogen chloride (HCl).

#### 8.Drainage and Sludge Handling System

After the scrubbing liquid has absorbed pollutants and captured particulates, it must be managed appropriately. The drainage and sludge handling system collects the spent liquid and any resultant sludge, which contains the captured pollutants. This system includes drains, collection tanks, and equipment for treating or disposing of the sludge according to environmental regulations.



**Figure 56: the parts of the wet scrubber**

In summary, a wet scrubber comprises several interconnected components that work together to remove pollutants from industrial exhaust streams effectively. From the initial entry of polluted gas through the inlet duct to the final discharge of cleaned gas via the outlet duct, each component plays a vital role in ensuring the scrubber's efficiency and reliability.

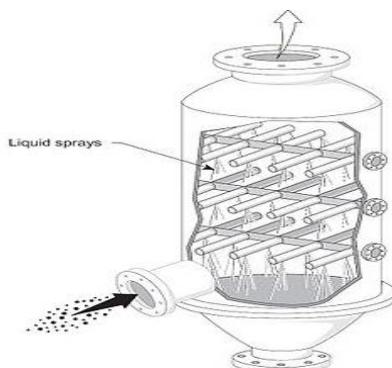
## **TYPES OF WET SCRUBBERS**

Wet scrubbers are essential air pollution control devices that come in various designs, each tailored to specific industrial applications and types of pollutants. The primary types of wet scrubbers include spray towers, packed bed scrubbers, venturi scrubbers, and orifice scrubbers, among others. Each type employs distinct mechanisms to enhance gas-liquid contact, absorption, and particulate capture, making them suitable for different operational needs and pollution control requirements.

### **Spray Tower Scrubbers**

Spray tower scrubbers utilize nozzles to disperse the scrubbing liquid into a fine mist or spray within a chamber. The polluted gas stream flows through this mist, allowing for thorough contact between the gas and liquid droplets. This intimate interaction facilitates

the absorption of gaseous pollutants and the removal of particulates. Spray tower scrubbers are particularly effective for gases with high solubility in the scrubbing liquid, such as hydrogen chloride (HCl) and ammonia (NH<sub>3</sub>). Their simple design and low pressure drop make them a popular choice for various industrial applications. However, they require substantial space for installation and are less effective in capturing fine particulates compared to other scrubber types.

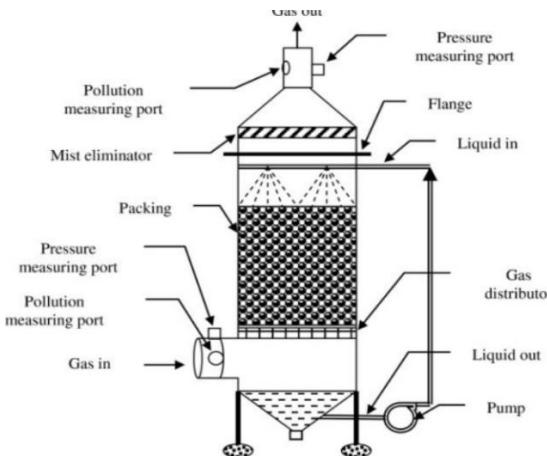


**Figure 57: a typical countercurrent flow spray tower**

### Packed Bed Scrubbers

Packed bed scrubbers feature a bed of packing material, which provides an extensive surface area for gas-liquid contact. The scrubbing liquid flows over the packing material, creating a thin film, while the gas stream passes through the bed. This configuration maximizes the contact time between the gas and liquid, enhancing absorption efficiency.

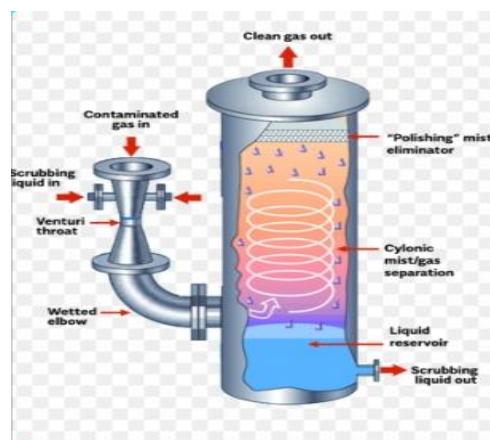
Packed bed scrubbers are versatile and can handle both gas absorption and particulate removal, making them suitable for a wide range of industries, including chemical processing and air purification. Despite their effectiveness, packed bed scrubbers can be prone to clogging and fouling, necessitating regular maintenance to ensure optimal performance.



**Figure 58: A countercurrent single staged packed tower scrubber**

### Venturi Scrubbers

Venturi scrubbers employ a high-velocity gas stream that passes through a constricted throat, creating a low-pressure zone where the scrubbing liquid is introduced. The resulting high-energy interaction between the gas and liquid leads to efficient particulate capture and gas absorption. Venturi scrubbers are particularly adept at removing fine particulates and mist droplets, making them ideal for industries such as metallurgy, mining, and power generation. Their compact design and high mass transfer rates are significant advantages. However, they require substantial energy due to the high-pressure drop, and efficient liquid handling and separation systems are necessary to manage the resulting slurry.



**Figure 59: a venturi scrubber**

## Orifice Scrubbers

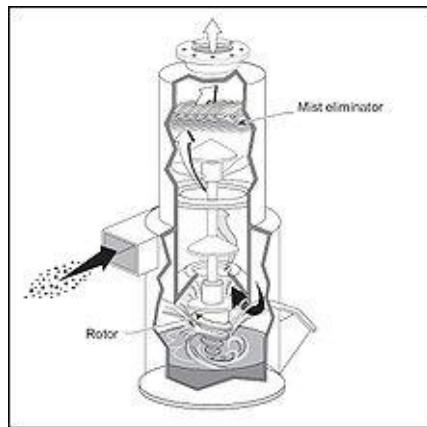
Orifice scrubbers force the gas stream through a series of orifices, generating turbulence that enhances gas-liquid contact. The scrubbing liquid is introduced either before or at the orifices, facilitating rapid mixing and interaction. This design is effective for applications requiring efficient gas-liquid contact and moderate particulate removal, such as chemical manufacturing and food processing. Orifice scrubbers offer a simple design with moderate pressure drop, making them a cost-effective solution for many industrial needs. However, they are less efficient at capturing fine particulates compared to venturi scrubbers and may experience erosion and wear at the orifices.



**Figure 60: an orifice scrubber**

## Mechanically Assisted Scrubbers

Mechanically assisted scrubbers use mechanical means, such as fans or impellers, to enhance gas-liquid contact. The scrubbing liquid is dispersed by the mechanical action, increasing the interaction surface area and improving pollutant removal efficiency. These scrubbers are suitable for applications requiring enhanced mass transfer rates, such as in the textile and pharmaceutical industries. The main advantages of mechanically assisted scrubbers include their enhanced gas-liquid contact and flexible design, which can handle variable gas flow rates. However, the presence of moving parts increases maintenance requirements and operational complexity.



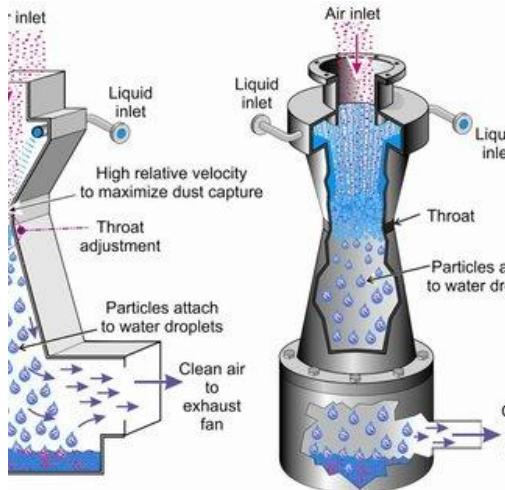
**Figure 61: a mechanically assisted scrubber**

### **Impingement Plate Scrubbers**

Impingement plate scrubbers utilize a series of plates with perforations or slots where the gas stream impinges on the liquid film. This creates turbulence and enhances gas-liquid contact, making these scrubbers effective for both gas absorption and particulate removal.

They are commonly used in industries like petroleum refining and petrochemical processing. Impingement plate scrubbers offer a moderate pressure drop and simple construction, which are significant advantages. However, they can be prone to clogging and fouling, requiring regular maintenance to prevent plate blockages and ensure consistent performance.

In summary, wet scrubbers are diverse and adaptable pollution control devices, each with unique advantages tailored to specific industrial applications and pollutant types. From the simplicity of spray towers to the high efficiency of venturi scrubbers, the selection of a wet scrubber depends on the nature of the pollutants, required removal efficiency, and operational conditions. Understanding the strengths and limitations of each type allows for informed decision-making in designing effective air pollution control systems.



**Figure 62: an impingement plate scrubber**

### PARAMETERS TO CONTROL

In the quest for efficient and environmentally responsible lithium extraction, the design of a wet scrubber system plays a pivotal role. Wet scrubbers are integral in controlling emissions and maintaining air quality standards by removing pollutants generated during the extraction process. This essay explores the key parameters to be considered when designing a wet scrubber for a lithium extraction plant, focusing on the removal of sulfur oxides (SO<sub>x</sub>), nitrogen oxides (NO<sub>x</sub>), and carbon dioxide (CO<sub>2</sub>).

A fundamental aspect of wet scrubber design is understanding the gas flow rate and composition. The gas flow rate, typically measured in cubic meters per hour (m<sup>3</sup>/h) or cubic feet per minute (cfm), determines the size and capacity of the scrubber. The temperature and pollutant concentration within the gas stream are equally important, as they influence the solubility and reactivity of the pollutants, thereby guiding the selection of the appropriate scrubbing liquid and design configuration.

The choice and management of the scrubbing liquid are crucial for effective pollutant removal. While water is commonly used, specific chemicals such as sodium hydroxide for SO<sub>x</sub>, ammonia for NO<sub>x</sub>, and alkaline solutions for CO<sub>2</sub> may be required depending on the nature of the pollutants. The liquid flow rate, measured in liters per minute (L/min)

or gallons per minute (gpm), must ensure sufficient contact with the gas stream to maximize absorption efficiency. Additionally, maintaining the pH and concentration of the scrubbing liquid is essential for optimizing the neutralization of pollutants. The temperature of the scrubbing liquid also plays a significant role, as it affects the absorption capacity and solubility of contaminants.

The scrubber's design and configuration are tailored to facilitate optimal gas-liquid contact. Different types of scrubbers, such as venturi scrubbers, packed bed scrubbers, spray towers, and tray scrubbers, offer varying mechanisms for achieving this interaction. Factors such as the residence time of the gas and liquid within the scrubber and the pressure drop across the system must be optimized to balance removal efficiency with energy consumption.

Understanding the characteristics of the pollutants is another critical design parameter. The type of pollutants—whether SO<sub>x</sub>, NO<sub>x</sub>, or CO<sub>2</sub>—affects the selection of the scrubbing liquid and the overall design strategy. For instance, in a lithium extraction plant, the scrubber must efficiently capture SO<sub>x</sub> and NO<sub>x</sub> gases, which are typically generated from the combustion processes involved in extracting lithium from spodumene ore. CO<sub>2</sub>, though less reactive, also requires efficient removal due to its greenhouse gas implications.

Operational parameters such as flow distribution, nozzle design for spray scrubbers, and maintenance requirements must be meticulously planned. Uniform distribution of gas and liquid phases ensures consistent scrubbing performance, while well-designed nozzles enhance the effectiveness of spray scrubbers. Maintenance considerations, including ease of access for cleaning and prevention of clogging or fouling, are vital for sustaining long-term operation.

Compliance with environmental regulations is a non-negotiable aspect of scrubber design. The system must meet local and international emission limits, necessitating rigorous monitoring and control mechanisms. Additionally, the handling of scrubber wastewater, which may contain absorbed pollutants, requires careful management to prevent secondary environmental contamination.

Safety considerations, such as material compatibility and leak detection systems, are essential to protect both the plant and the environment. The materials used in constructing the scrubber must resist corrosion and wear from the scrubbing liquid and pollutants, ensuring longevity and safety.

Instrumentation and control systems play a pivotal role in the efficient operation of a wet scrubber. Real-time monitoring of flow rates, pressure, temperature, and pollutant concentrations, coupled with automated control systems, allows for dynamic adjustments to maintain optimal performance.

Energy consumption is another critical factor. The design should aim to minimize energy use by optimizing the efficiency of pumps and fans, and where possible, incorporate heat recovery systems to utilize waste heat.

In conclusion, the design of a wet scrubber for a lithium extraction plant involves a multifaceted approach, balancing technical, environmental, and economic considerations. By carefully controlling parameters such as gas flow rate, scrubbing liquid characteristics, scrubber design, and pollutant properties, a highly efficient and compliant system can be developed. This not only ensures the sustainable operation of the lithium extraction process but also contributes to the broader goal of environmental protection and resource efficiency.

## **SELECTING AN OPTIMAL SCRUBBING LIQUID**

Wet scrubbers typically employ a range of scrubbing liquids such as water, sulphuric acid(H<sub>2</sub>SO<sub>4</sub>), hydrochloric acid (HCl), sodium hydroxide (NaOH), etc, each selected for its ability to target specific contaminants, however considering the specific requirements of lithium extraction, a basic solution such as sodium hydroxide (NaOH) emerges as the most suitable scrubbing liquid. NaOH is highly effective in neutralizing acidic pollutants, particularly acidic gases such as SO<sub>x</sub>, CO<sub>2</sub> and NO<sub>x</sub>, which are common by-products in lithium extraction processes. The use of NaOH in the scrubber will result in the formation of sodium sulphite, sodium carbonate, sodium nitrite and water, b which are less harmful and easier to manage.

Additionally, sodium hydroxide solutions offer several advantages:

1. **High Neutralization Efficiency:** NaOH provides a high degree of neutralization for acidic gases, ensuring compliance with emission standards.
2. **Cost-Effectiveness:** NaOH is relatively inexpensive and widely available, making it a cost-effective choice for large-scale industrial applications.
3. **Ease of Handling and Disposal:** Compared to more hazardous or reactive scrubbing liquids, NaOH solutions are easier to handle, store, and dispose of safely, reducing operational risks.

However, the implementation of NaOH as a scrubbing liquid necessitates the use of corrosion-resistant materials in the construction of the scrubber to withstand the basic environment. This consideration is essential to ensure the longevity and reliability of the scrubber system.

## **12.4 EQUIPMENT SELECTION AND JUSTIFICATION**

Among the various types of wet scrubbers, the Packed Bed Scrubber is the best option for a lithium extraction plant. Here's a detailed justification for this choice: Packed Bed Scrubber

Justification:

1. Effective Gas-Liquid Contact:

- Surface Area: Packed bed scrubbers provide a large surface area for gas-liquid interaction. The packing material (e.g., Raschig rings, Pall rings, or structured packing) creates a vast interface for the scrubbing liquid to absorb gaseous pollutants effectively.
- Efficiency: This extensive surface area ensures high removal efficiencies for gaseous pollutants such as sulfur oxides (SO<sub>x</sub>), nitrogen oxides (NO<sub>x</sub>), and carbon dioxide (CO<sub>2</sub>).

2. Versatility in Handling Pollutants:

- SO<sub>x</sub> Removal: The alkaline scrubbing liquid (e.g., sodium hydroxide) can be easily distributed over the packing material, enhancing the neutralization and absorption of SO<sub>x</sub>.
- NO<sub>x</sub> Control: For NO<sub>x</sub>, a packed bed scrubber can be designed with multi-stage packing and varied chemical treatments to handle different oxidation states of nitrogen oxides.
- CO<sub>2</sub> Capture: While CO<sub>2</sub> capture might typically require specialized solvents, the packed bed configuration can accommodate amine-based solutions or other advanced absorbents, making it versatile for multiple pollutants.

3. Customizable Design:

- Packing Material: The type and size of the packing material can be customized based on the specific pollutants and their concentrations. This customization allows for optimization of the scrubber's performance.
- Liquid Distribution: Advanced liquid distribution systems ensure uniform wetting of the packing material, which is critical for maintaining consistent scrubbing efficiency.

#### 4. Operational Reliability:

- Low Pressure Drop: Packed bed scrubbers generally have a lower pressure drop compared to other types of scrubbers like venturi scrubbers. This reduces the energy required for gas movement, enhancing operational efficiency.
- Scalability: These scrubbers can be easily scaled to accommodate different gas flow rates and pollutant loads, making them suitable for varying production capacities in a lithium extraction plant.

#### 5. Maintenance and Longevity:

- Durability: The materials used in packed bed scrubbers (both for the packing and the scrubber body) can be selected to resist corrosion from the scrubbing liquids and the pollutants.
- Ease of Maintenance: Packed bed scrubbers are relatively easy to maintain. The packing material can be accessed and replaced if fouling occurs, and the liquid distribution system can be cleaned to prevent blockages.

#### 6. Cost-Effectiveness:

- Initial Investment and Operating Costs: While the initial investment might be higher compared to simpler scrubbers, the operational savings due to lower energy consumption and high efficiency often justify the cost. The ability to handle multiple pollutants efficiently also reduces the need for additional treatment stages.

## Specific Advantages for Lithium Extraction:

- Handling Complex Emissions: Lithium extraction processes can produce a variety of gaseous pollutants, including acidic gases like HCl along with SO<sub>x</sub>, NO<sub>x</sub>, and CO<sub>2</sub>. The packed bed scrubber's ability to handle multiple types of pollutants effectively makes it ideal for this application.
- Customization for Process Variability: The flexibility in design allows the scrubber to be tailored specifically to the emission profile of the lithium extraction plant, ensuring optimal performance even with variations in the process conditions.

## Conclusion

The packed bed scrubber is the optimal choice for a lithium extraction plant due to its high efficiency in removing a wide range of gaseous pollutants, its customizable and scalable design, and its operational reliability and cost-effectiveness. This type of scrubber ensures that the plant can meet stringent environmental regulations while maintaining efficient and economical operations.

## Packing Materials

Packing materials used in packed columns are a variety of shapes and structures used to increase the surface area for contact between a gas and liquid. They essentially act as internal components within the column that enhance the efficiency of mass transfer processes. Random packings and structured packings are the two different kinds of packing. A random arrangement is created when packings are thrown into the column randomly. Compared to structured packings, this results in a less expensive installation. They also have a greater specific surface area of contact. Examples of random packings include Raschig rings, pall rings and saddles (Cheremisinoff, 2003).

Conversely, structured packings, such as stacked rings and grids, have a regular configuration. Their configuration has the benefit of less pressure drop and increased potential liquid throughput. However, they are generally expensive to install as well as construct (Towler & Sinnott, 2013).

The random packings will be chosen for this design since they are more affordable and effective throughout the absorption process.

A packing's primary characteristics should be that it:

- I. Offers a large interfacial area between the gas and liquid.
- II. Encourage uniform gas flow across the column cross section.
- III. Have an open structure so as to have low resistance to gas flow.
- IV. Facilitate uniform liquid distribution on packing surface.



**Figure 63: different types of packing materials**

### **CHOICE OF PACKING MATERIAL**

Raschig rings stand out as a preferred choice in many applications due to their unique combination of properties and benefits.

Firstly, one of the most compelling advantages of Raschig rings is their high surface area per unit volume. This characteristic stems from their cylindrical shape, typically having a diameter equal to their height. This geometry maximizes the available surface area for contact between different phases within a packed bed, such as gases and liquids. In processes involving mass transfer or heat exchange, such as in chemical reactors or distillation columns, this increased surface area facilitates more efficient interaction and improved process outcomes. By promoting better contact between reactants or separating components, Raschig rings contribute significantly to optimizing yields and reducing operational costs.

Moreover, the uniform shape of Raschig rings ensures uniform distribution and flow of fluids through the packed bed. This uniformity minimizes the risk of channeling, where fluids preferentially flow through localized pathways within the packing material, compromising overall process efficiency. The cylindrical design promotes even distribution of flow, ensuring consistent performance over the operational lifetime of the packing material.

## **12.5 CHEMICAL ENGINEERING DESIGN**

The parameters needed to fully specify the design of this packed column include the column diameter, column height, height of packed bed and pressure drop in the column.

### **ASSUMPTIONS**

These are the assumption made in the design:

- The waste gas compromises a two component waste gas mixture i.e the clean air and pollutant
- The pollutant consist of a single compound present which is CO<sub>2</sub>
- The waste gas is an ideal gas

- The scrubbing liquid or solvent is an ideal liquid

## MOLAR MASS FLUE GAS

Packing type = 2 inch ceramic rasching rings

$$M_w = \sum y_i \times M_i$$

Where  $y_i$  = mole fraction of the individual gases

$M_i$  = molar mass of the individual gases

$$M_w = (0.166 \times 44) + (0.626 \times 46) + (0.207 \times 18)$$

$$M_w = 39.826 \text{ Kg/Kmol}$$

## DENSITY OF FLUE GAS

Assuming ideal gas conditions, density of the flue gas is calculated using the formula below

$$\rho = \frac{PM_w}{RT}$$

Where P = pressure of the gas = 101.325 kPa

$M_w$  = molar mass of the gas = 39.826 kg/kmol

R = gas constant = 8.314 kPa.m<sup>3</sup> /kmol. K (Benítez, 2011)

T = inlet gas temperature = 107°C = 380K

$$\rho = \frac{101.325 \times 39.826}{8.314 \times 380}$$

$$\rho = 0.3535 \text{ Kg/m}^3$$

total moles entering the scrubber = 39008.3mol/h

$$y_{CO_2} = y_i = 0.166$$

$$\text{Inlet pollution concentration}(Y_i) = \frac{y_i}{1-y_i}$$

$$Y_i = 0.199$$

$$\text{Exit pollution concentration}(Y_o) = Y_i \left(1 - \frac{n}{100}\right)$$

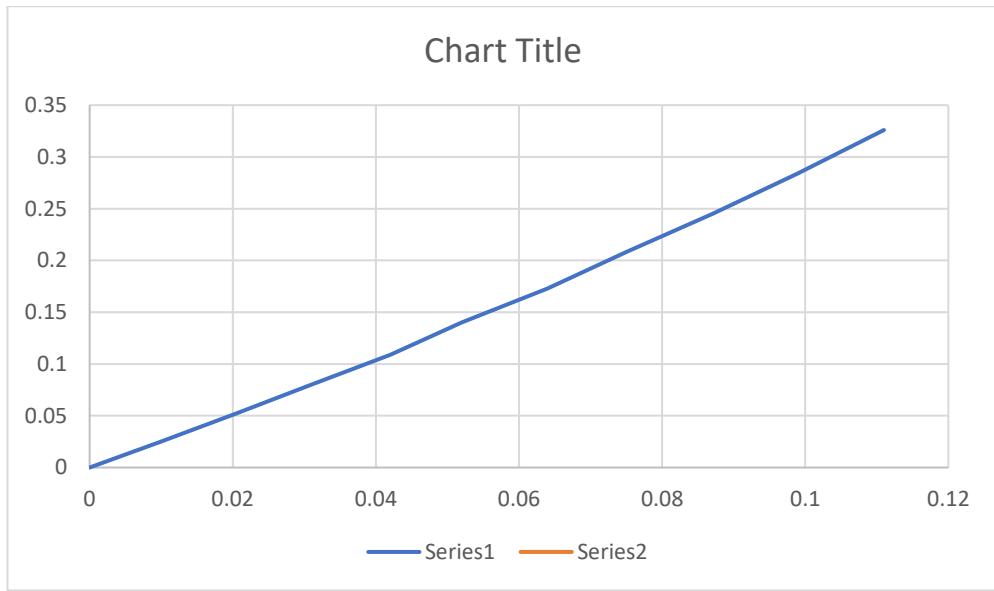
$$Y_o = 0.008$$

$$\text{Equilibrium equation , } y = 2.467x$$

**Table 20: Data Values**

X	Y	X	Y
0	0	0	0
0.01	0.0246	0.01	0.0252
0.02	0.0492	0.0204	0.052
0.03	0.0738	0.031	0.08
0.04	0.0984	0.042	0.109
0.05	0.123	0.052	0.14
0.06	0.1476	0.064	0.173
0.07	0.1722	0.075	0.208
0.08	0.1968	0.087	0.245
0.09	0.2214	0.0989	0.284
0.10	0.2416	0.111	0.326

## Plotting X against Y on a graph



**Figure 64: graph of x against y**

From the graph the value of  $X_o$  which is the minimum concentration of the pollutant if it were allowed to come to equilibrium with the pollutant entering the column in the gas phase,  $Y_i$  is 0.07.

The slope of the line  $(\frac{LS}{GS})_{min}$  is calculated from the equation:

$$(\frac{LS}{GS})_{min} = \frac{Y_i - Y_o}{X_o - X_i}$$

$$(\frac{LS}{GS})_{min} = \frac{0.199 - 0.008}{0.007 - 0.00}$$

$$(\frac{LS}{GS})_{min} = 2.743$$

We assume adjustment factor to be 1.5

$$(\frac{LS}{GS})_{act} = 2.743 \times 1.5$$

$$(\frac{LS}{GS})_{act} = 4.114$$

$$G_S = \frac{60 \times \rho g \times G_i}{M_W g (1 + Y_i)}, \quad \rho g = 0.3535 \text{ kg/m}^3$$

$$G_S = \frac{60 \times 0.3535 \times 39008.3}{39.826 \times (1+0.199)}$$

$$G_S = 17312.1 \text{ mol/h}$$

$$L_S = \left(\frac{L_S}{G_S}\right)_{\text{act}} \times G_S$$

$$L_S = 4.114 \times 17312.1$$

$$L_S = 71,221.98 \text{ mol/h}$$

Total molar flowrates Lmol,i and Gmol,i

$$G_{\text{mol},i} = G_S(1 + Y_i)$$

$$G_{\text{mol},i} = 17312.1(1 + 0.199)$$

$$G_{\text{mol},i} = 20,774.52 \text{ mol/h}$$

$$L_{\text{mol},i} = L_S = 71,221.98 \text{ mol/h}$$

$$\text{Volume flowrate of solvent}(L_i) = \frac{L_{\text{mol},i} \times \text{MW}_i}{60 \times \rho_l}$$

$$= \frac{71221.98 \times 39.826}{60 \times 2.13}$$

$$= 22,194.73 \text{ mol/h}$$

$$X_o = \frac{Y_i - Y_o}{\left(\frac{L_S}{G_S}\right)_{\text{act}}} + X_i$$

$$X_o = \frac{0.199 - 0.008}{4.114} + 0$$

$$X_o = 0.047$$

From the graph Xo corresponds to Yo\* = 0.135

Converting Xo and Yo\* to x\_o and y\_o\*

$$x_o = \frac{x_o}{1+x_o}$$

$$x_0 = \frac{0.047}{1+0.047} = 0.045$$

$$y_0 = \frac{Y_0}{1+Y_0}$$

$$y_0 = \frac{0.135}{1+0.135} = 0.119$$

the slope of the line, m is given by:

$$m = \frac{0.119 - 0}{0.045 - 0} = 2.643$$

$$\text{Absorption factor (A.F)} = \frac{L_{\text{mol},i}}{m \times G_{\text{mol},i}}$$

$$A.F = \frac{71221.98}{2.643 \times 20774.52}$$

$$A.F = 11.3$$

Estimating column diameter

From Eckert's modification

The abscissa X is given by;

$$X = \left( \frac{L_{\text{mol},i}}{G_{\text{mol},i}} \times \frac{MWl}{MWg} \times \sqrt{\frac{\rho g}{\rho l}} \right)$$

$$X = \left( \frac{71221.98}{20774.52} \times \frac{39.997}{39.826} \times \sqrt{\frac{0.3535}{2.13}} \right)$$

$$X = 0.044$$

Tracing the x coordinate we find the y – coordinate to be 0.181

Finding the superficial gas flowrate,  $G_{\text{sfr},i}$

$$G_{\text{sfr},i} = \frac{\rho g \times \rho l \times g \times y}{F_p \times \Psi \times \left( \frac{\mu}{2.42} \right)}$$

$\Psi$  is the ratio of the density of the scrubbing liquid to water.

$$\Psi = \frac{2.13}{1.0} = 2.13$$

Fp is the packing factor which is obtained from packing vendors

$$F_p = 40$$

$$G_{sfr,i} = \sqrt{\frac{0.0147 \times 2.13 \times 9.81 \times 0.3535}{40 \times 2.13 \times (0.078)}}$$

$$G_{sfr,i} = 4.0126 \text{ m}^3/\text{h}$$

Cross sectional area of the tower

$$A = \frac{G_{mol,i} \times MWg}{3600 \times G_{sfr,i} \times f}$$

Where f is the flooding factor which range from 0.60 to 0.75

We assume f to be 0.70

$$A = \frac{20774.52 \times 39.826}{4.0126 \times 3600 \times 1000}$$

$$A = 2.872 \text{ m}^2$$

Diameter of the column

$$D = \sqrt{\frac{4 \times A}{\pi}}$$

$$D = \sqrt{\frac{4 \times 2.872}{\pi}}$$

$$D = 1.912 \text{ m}$$

Superficial liquid flowrate entering the scrubber

$$L_{sfr,i} = \frac{L_{mol,i} \times MW_l}{A}$$

$$L_{sfr,i} = \frac{71221.98 \times 39.997}{2.872 \times 1000}$$

$$L_{sfr,i} = 991.875 \text{ kg/m}^2\text{h}$$

$$(L_{sfr,i})_{min} = MWR \times \rho_l \times a$$

MWR = minimum wetting rate

a = surface area to volume ratio

recommended MWR and a are  $0.121 \text{ m}^2/\text{h}$  and  $36 \text{ m}^2/\text{m}^3$  respectively

$$(L_{sfr,i})_{min} = 0.121 \times 2.13 \times 36$$

$$= 9.261 \text{ kg/m}^2\text{h}$$

Since  $L_{sfr,i}$  is greater than  $(L_{sfr,i})_{min}$ , there would be sufficient liquid flow to wet the packing.

Determining the tower height and surface area

$$H_{pack} = N_{tu} \times H_{tu}$$

Where  $N_{tu}$  is the number of transfer units and  $H_{tu}$  is the height of transfer unit

For negligible amount of pollutant that enters the liquid stream i.e  $x = 0$ ,

$$N_{tu} = \ln\left(\frac{y_i}{y_o}\right)$$

$$N_{tu} = \ln\left(\frac{0.166}{0.00664}\right)$$

$$N_{tu} = 3.22$$

$$H_{tu} = H_G + \frac{1}{A.F} \times H_L$$

$H_G$  = height of gas film transfer unit

$H_L$  = height of liquid film transfer unit

$$H_G = \left[ \frac{\alpha (3600 f \times G_{sfr,i})^\beta}{(L_{sfr,i})^r} \right] \times \sqrt{\frac{\mu g}{\rho g \times Dg}}$$

$$H_L = \Phi \times \left( \frac{L_{Sfr,i}}{\mu L} \right)^b \times \sqrt{\frac{\mu L}{\rho L \times D L}}$$

From the packing vendor's parameters, the following are the values of the above variables

$$\alpha = 3.82, \beta = 0.41, \gamma = 0.45, \Phi = 0.0125, b = 0.22, c = 0.24, j = 0.17 \text{ } \mu\text{g} = 1.619 \times 10^{-5} \text{ Pa.s}$$

$$H_G = \left[ \frac{3.82(3600 \times 0.7 \times 0.1143)^{0.41}}{(991.875)^{0.45}} \right] \times \sqrt{\frac{1.619 \times 10^{-5}}{0.3535 \times 1.6 \times 10^{-9}}}$$

$$H_G = 1.203 \text{ m}$$

$$H_L = 0.0125 \times \left( \frac{991.875}{0.078} \right)^{0.22} \times \sqrt{\frac{0.078}{2.13 \times 1.309 \times 10^{-4}}}$$

$$H_L = 0.762 \text{ m}$$

Since  $AF \rightarrow \infty$

$$H_{tu} = 1.203 + \frac{1}{\infty} (0.762)$$

$$H_{tu} = 1.203 \text{ m}$$

$$H_{pack} = 3.22 \times 1.203$$

$$H_{pack} = 3.874 \text{ m}$$

$$H_{tower} = 1.4 H_{pack} + 1.02D + 2.81$$

$$= 1.4(3.874) + 1.02(1.912) + 2.81$$

$$H_{tower} = 10.18 \text{ m}$$

Calculating the surface area(S)

$$S = \Pi D \times [H_{tower} + \frac{D}{2}]$$

$$S = \Pi(1.912) \times [10.18 + \frac{1.912}{2}]$$

$$S = 66.9m^2$$

Calculating the pressure drop

$$\Delta P = c \times 10 \times \left( \frac{j \times Lsfr}{3600} \right) \times \left( \frac{(f \times Gsfr, i)^2}{\rho g} \right)$$

$$\Delta P = 0.24 \times 10 \times \left( \frac{0.17 \times 991.875}{3600} \right) \times \left( \frac{(0.7 \times 0.1143)^2}{0.3535} \right)$$

$$\Delta P = 0.00484 \text{ Pa}$$

## SUMMARY OF CHEMICAL ENGINEERING DESIGN FOR SCRUBBER

Summary of Chemical Engineering Design for scrubber

PARAMETER	VALUE
Gas flow rate	39008.3mol/h
Liquid flow rate	22194.73mol/h
Superficial gas flowrate	4.0126m <sup>3</sup> /sec
Superficial liquid flowrate	991.875m <sup>3</sup> /sec
Diameter of column	1.912 m
Absorption factor	11.3
Area of column	2.872m <sup>2</sup>
Surface area of column	66.9m <sup>2</sup>
Wetting rate	0.121m <sup>2</sup> /h
Number of transfer units	3.22

Effective wetted area	36 m <sup>2</sup> /m <sup>3</sup>
Height of liquid film transfer unit	0.762m
Height of gas film transfer unit	1.203m
Height of overall gas phase transfer unit	1.203m
Height of packed bed	4m
Total height of column	10.18m
Type of packing	2 inch rasching rings
Packing installation	Random

## 12.6 MECHANICAL ENGINEERING DESIGN

The parameters required to fully specify the design of this packed column include the material of construction, vessel support, column stresses and the weight of the column.

### Material of construction

Given the corrosive nature of the flue gas stream, stainless steel is strongly recommended for the column design. The three types of stainless steel are:

1. Ferritic stainless steel: 13–20% Cr
2. Austenitic stainless steel: 18–20% Cr, >7% Ni
3. Martensitic stainless steel: 10–12% Cr, 0.2 to 0.4% C, up to 2% Ni

The most commonly used type is austenitic steel, as the uniform structure of austenite offers the desired corrosion resistance. Type 304 (18/8) stainless steel is suitable for the design because it contains the minimum Cr and Ni content necessary for a stable

austenitic structure. Considering material cost, type 304 stainless steel is the least expensive compared to other types (Towler & Sinnott, 2013).

## **DESIGN PRESSURE**

$P = 10\% \text{ more than } P_o$  (Towler & Sinnott, 2013)

Where  $P$  = design pressure, Pa

$P_o$  = operating pressure in the column = 101325 Pa Substituting,

$$P = 1.1 \times (101325) = 111458 \text{ Pa}$$

## **DESIGN TEMPERATURE**

For the design temperature, it is usually taken as 10°C above the operating temperature.

$$T = 10 + T_o$$

Where  $T$  = design temperature, °C

$T_o$  = operating temperature = temperature of the gas = 1100°C

$$T = 1100 + 10 = 1110^\circ\text{C}$$

## **COLUMN SHELL THICKNESS**

$$t = \frac{PD}{(2SE - P)} + C$$

Where  $t$  = column shell thickness, mm

$P$  = design pressure = 111458 Pa = 0.111458 N/mm<sup>2</sup>

$D$  = diameter of column = 1.912 m =  $1.912 \times 10^3$  mm

$S$  = maximum allowable design stress of type 304 stainless steel at 1110°C = 161.34 N/mm<sup>2</sup>

$E$  = welded joint efficiency = 1

C = corrosion allowance = 2 mm

Substituting,

$$t = \frac{0.111458 \times 1.912 \times 10^3}{(2 \times 161.34 - 0.111458)} + 2 = 2.66 \text{ mm}$$

In practice, the minimum wall thickness for any vessel with a diameter of 2 to 2.5 meters should be at least 9 mm, including the corrosion allowance (Towler & Sinnott, 2013). Consequently, the column shell thickness is set at 12 mm and will be used for subsequent calculations.

## COLUMN HEAD DESIGN

The column head can be selected from the following types: flat, hemispherical, ellipsoidal, and torispherical. Standard torispherical heads are commonly used as end closures for vessels operating up to 15 bars. Flat heads are the cheapest to manufacture, but their use is limited to small diameter vessels and as channel covers for heat exchangers. Ellipsoidal and hemispherical heads are used for operating pressures above 15 bars, though their high manufacturing costs make them more expensive compared to other types. A torispherical head will be used for the column because they are easier and cheaper to manufacture.

$$t_h = \frac{0.885 P R_c}{S_E - 0.1 P}$$

Where  $t_h$  = head thickness, mm

P = design pressure = 101325 Pa = 0.101325 N/mm<sup>2</sup>

Rc = crown radius = diameter of column =  $1.912 \times 10^3$  mm

S = maximum allowable design stress of type 304 stainless steel at 1110°C = 161.34 N/mm<sup>2</sup>

E = welded joint efficiency = 1

$$t_h = \frac{0.885 \times 0.111458 \times 1.912 \times 10^3}{(161.34 \times 1) - (0.1 \times 0.101325)} = 1.169 \text{ mm}$$

To ensure uniformity of the design, the shell thickness of the column should be the same as that of the head and so a head thickness of 12 mm will be used as well.

## **WEIGHT LOAD**

$$W_v = 240C_wD_M(H_v + 0.8D_M)t$$

Where  $W_v$  = total weight of shell without internal fittings, N

$C_w$  = factor accounting for internal support, weight of nozzles = 1.15

$$D_M = \text{mean diameter} = 1.912 + (12 \times 10^{-3}) = 1.924 \text{ m}$$

$$H_v = \text{height of column} = 10.18 \text{ m}$$

$$t = \text{column thickness} = 12 \text{ mm}$$

$$W_v = 240 \times 1.15 \times 1.924 (10.18 + 0.8 \times 1.924) \times 12 = 74678.12 \text{ N} = 74.7 \text{ kN}$$

## **WEIGHT OF PACKINGS**

$$\text{Height of packed bed} = 4\text{m}$$

$$\text{Area of column} = 2.82 \text{ m}^2$$

$$\text{Density of 50 mm rasching rings} = 650 \text{ kg/m}^3$$

$$\text{Mass of packings} = \text{Density of packings} \times \text{height of packed} \times \text{area of column}$$

$$\text{Mass of packings} = 650 \times 4 \times 2.82 = 7332 \text{ kg}$$

$$\text{Weight of packings, WP} = 7332 \times 9.81 = 71926 \text{ N} = 71.9 \text{ kN}$$

## **WEIGHT OF INSULATION**

Mineral wool with a thickness of 25 mm is generally used for insulation (Towler & Sinnott, 2013). Density of mineral wool =  $130\text{kg/m}^3$

$$\text{Volume of insulation} = V_{\text{ins}} = \pi \times D \times H \times t_{\text{ins}}$$

$$\text{Where } D = \text{diameter of column} = 1.912 \text{ m}$$

$$H_v = \text{height of column} = 10.18 \text{ m}$$

$$t_{\text{ins}} = \text{thickness of insulation} = 25 \times 10^{-3} \text{ m}$$

$$\text{Volume of insulation} = \pi \times 1.912 \times 10.18 \times 25 \times 10^{-3} = 1.5287$$

$$\text{Weight of insulation} = \text{density} \times \text{volume} \times \text{acceleration of gravity}$$

$$W_{\text{ins}} = 130 \times 1.5287 \times 9.81 = 1949.57 \text{ N} = 1.95 \text{ kN}$$

To allow for attachment fittings, sealing and moisture absorption, weight of insulation is doubled. Therefore,  $W_{\text{ins}} = 2 \times 1.95 = 3.9 \text{ kN}$

$$\text{TOTAL WEIGHT OF COLUMN WT} = W_V + W_P + W_{\text{ins}}$$

$$\text{Where WT} = \text{total weight of column, kN}$$

$$W_V = \text{weight of shell} = 74.7 \text{ kN}$$

$$W_P = \text{weight of packings} = 71.9 \text{ kN}$$

$$W_{\text{ins}} = \text{weight of insulation} = 1.95 \text{ kN}$$

$$\text{Substituting, WT} = 74.7 + 71.9 + 1.95 = 148.55 \text{ kN}$$

## **WIND LOADING**

This considers the amount of stress exerted on the column. Wind velocity is typically highest at the top, potentially causing the column to bend. For this design, a wind speed

of 100 mph will be used as basis, corresponding to a wind pressure of 1280 N/m<sup>2</sup> (Towler & Sinnott, 2013).

### I. Loading

$$L = Dm \times Pw$$

Where  $Dm = \text{mean diameter including insulation} = D + 2(t_{ins} + t)$

$$Dm = 1.912 + 2[(25 + 12) \times 10^{-3}] = 1.986 \text{ m}$$

$$Pw = \text{wind pressure} = 1280 \text{ N/m}^2$$

Substituting,

$$L = 1.986 \times 1280 = 2542.08 \text{ N/m}$$

### II. Bending moment at any plane on the column is given as:

Where  $MX = \text{bending moment at any plane, Nm}$

$$L = \text{loading per linear meter} = 2542.08 \text{ N/m}$$

$$H = \text{height of column} = 10.18 \text{ m}$$

Substituting,

$$MX = \frac{2433.28 \times 6.8^2}{2} = 131720.93 = 131.72 \text{ kNm}$$

## **ANALYSIS OF STRESS**

### I. Longitudinal stress

$$\sigma_L = \frac{PD}{4t}$$

Where  $\sigma_L = \text{longitudinal stress, N/mm}^2$

$$P = \text{design pressure} = 0.111458 \text{ N/mm}^2$$

D = diameter of column =  $1.912 \times 10^3$  mm

t = column thickness = 12mm

Substituting,

$$\sigma_L = \frac{0.111458 \times 1.827 \times 10^3}{4 \times 12} = 4.439 \text{ N/mm}^2$$

## II. Circumferential stress

$$\sigma_H = \frac{PD}{2t}$$

Where  $\sigma_H$  = circumferential stress, N/mm<sup>2</sup>

P = design pressure = 0.111458N/mm<sup>2</sup>

D = diameter of column =  $1.912 \times 10^3$  mm

t = column thickness = 12mm

Substituting,

$$\sigma_H = \frac{0.111458 \times 1.912 \times 10^3}{2 \times 12} = 8.88 \text{ N/mm}^2$$

## III. Dead weight stress

$$\sigma_{WS} = \frac{W_T}{\pi(D+t)t}$$

Where  $\sigma_{WS}$  = dead weight stress in the skirt, N/mm<sup>2</sup>

WT = total weight of vessel and contents = 148.55 kN

D = diameter of column =  $1.912 \times 10^3$  mm

t = column thickness = 12 mm

$$\sigma_{WS} = \frac{174 \times 10^3}{\pi(18270+12) \times 12} = 2.06 \text{ N/mm}^2$$

#### IV. Bending stress

Bending stress can be compressive or tensile and is calculated using the formula below:

$$\sigma_B = \pm \frac{M_X}{I_v} \left( \frac{D}{2} + t \right)$$

Where  $\sigma_B$  = bending stress, N mm<sup>2</sup>

$M_X$  = bending moment at any plane = 131720.93Nm

D = diameter of column =  $1.912 \times 10^3$  mm

t = column thickness = 12 mm

$I_v$  = second moment of area of the vessel about the plane of bending

$$I_v = \frac{\pi}{64} (D_0^4 - D_1^4)$$

Where  $D_o$  = outer diameter of column =  $D + 2(t) = 1912 + 2(12) = 1936$  mm

$D_i$  = internal diameter =  $1.912 \times 10^3$  mm

Substituting,

$$I_v = \frac{\pi}{64} (1936^4 - 1912^4) = 3.36 \times 10^{10} \text{ mm}^4$$

$$\sigma_B = \frac{131720.93 \times 10^3}{3.36 \times 10^{10}} \left( \frac{1912}{2} + 12 \right) = 3.79 \text{ N/mm}^2$$

#### V. Critical buckling stress

This factor considers the maximum allowable compressive stress exerted by the column on its support to prevent buckling failure. It is calculated using the following formula:

$$\sigma_c = 2 \times 10^4 \frac{t}{D_o}$$

Where  $\sigma_c$  = critical buckling stress, N/mm<sup>2</sup>

t = column thickness = 12 mm

$D_o = 1936 \text{ mm}$

Substituting,

$$\sigma_c = 2 \times 10^4 \frac{12}{1936} = 123.97 \text{ N/mm}^2$$

Maximum compressive stress = bending stress + dead weight stress =  $5.85 \text{ N/mm}^2$ .

Maximum compressive stress is well below the critical buckling stress making the design of the column safe and satisfactory.

## VESSEL SUPPORT

Vessel supports are classified into three types: saddle, skirt, and bracket support. The support must be designed to carry the weight of the vessel and its contents, as well as any additional loads such as wind loads. Skirt supports are primarily used for vertical vessels or columns and will be used for this packed column design. A skirt support consists of a cylindrical or conical shell welded to the base of the column. A flange at the bottom of the support transfers the load to the foundations. Skirt supports are recommended for vertical columns as they do not impose concentrated loads on the column shell and are suitable for columns subject to wind loading, providing equal strength in all directions (Towler & Sinnott, 2013)

## WEIGHT ON SKIRT

Weight imposed on skirt = total weight of column + weight of liquid feed

Weight of liquid =  $\rho L \times V \times g$

Where  $\rho L$  = density of liquid at  $25^\circ\text{C} = 997 \text{ kg/m}^3$  (Engineering Toolbox, 2019)

$V$  = volume of vessel = area  $\times$  height of column

$g$  = acceleration due to gravity =  $9.81 \text{ m/s}^2$

Weight of liquid =  $775 \times 2.82 \times 10.18 \times 9.81 = 218256.71\text{N}$

Weight imposed on skirt, WS =  $148550 + 218256.71 = 366806.7059\text{ N}$

## STRESS ANALYSIS ON SKIRT

### I. Bending moment at the base of skirt

$$M_s = \frac{LH^2}{2}$$

Where  $M_s$  = bending moment at the base of skirt, Nm

$L$  = loading per linear meter =  $2433.28\text{ N/m}$

$HC$  = height of column =  $10.18\text{ m}$

$HS$  = height of skirt =  $1\text{ m}$

$H$  = total height =  $11.18\text{ m}$

Substituting

$$M_s = \frac{2433.28 \times 10.18^2}{2} = 126083.32\text{ Nm} = 126.1\text{ kNm}$$

### II. Bending stress on skirt

$$\sigma_{SB} = \frac{4M_s}{\pi(D_s + t_{SK})t_{SK}D_s}$$

Where  $\sigma_{BS}$  = bending stress in the skirt,  $\text{N/mm}^2$

$M_s$  = bending moment at the base of skirt =  $126083.32\text{ Nm}$

$D_s$  = inside diameter of skirt at the base = diameter of column =  $1.912 \times 10^3\text{ mm}$

$t_{SK}$  = thickness of skirt = column thickness =  $12\text{ mm}$

Substituting,

$$\sigma_{SB} = \frac{4 \times 126083.321 \times 10^3}{\pi(1912+12)12 \times 1912} = 3.67 \text{ N/mm}^2$$

### III. Dead weight stress on skirt

$$\sigma_{WS \text{ (operating)}} = \frac{W_T}{\pi(D_S + t_{SK})t_{SK}}$$

Where  $\sigma_{WS}$  = dead weight stress in the skirt,  $\text{N/mm}^2$

$W_T$  = total weight of vessel and contents = 148.55kN

$D_S$  = inside diameter of skirt at the base = diameter of column =  $1.827 \times 10^3$  mm

$t_{SK}$  = thickness of skirt = column thickness = 12 mm

Substituting,

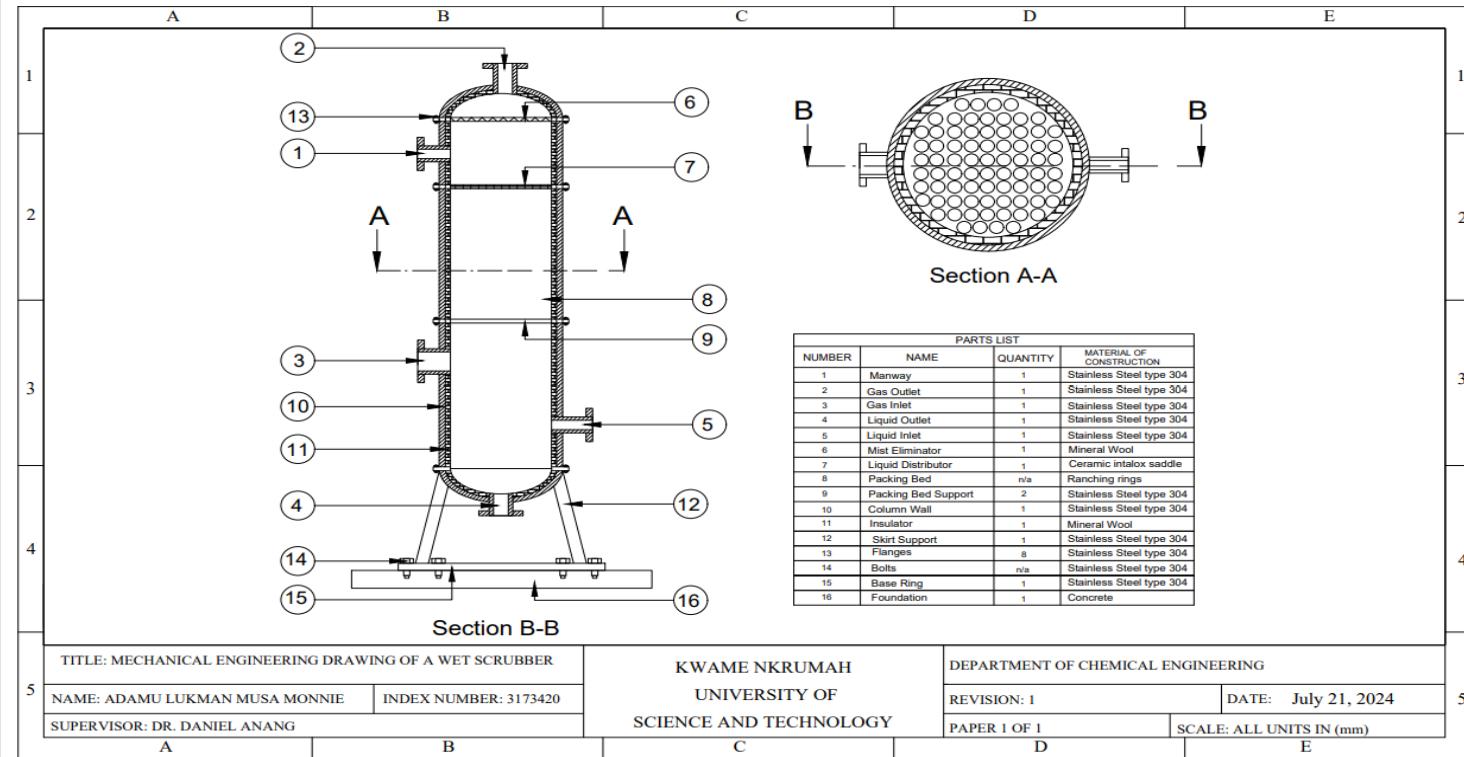
$$\sigma_{WS \text{ (operating)}} = \frac{148.55 \times 10^3}{\pi(1912+12) \times 12}$$

$$\sigma_{WS \text{ (operating)}} = 2.05 \text{ N/mm}^2$$

## SUMMARY OF MECHANICAL ENGINEERING DESIGN FOR SCRUBBER

Design Parameter	Value
Material of construction	Stainless steel Type 304 (18Cr/8Ni)
Material for insulation	Mineral wool
Column head	Tori spherical
Column shell thickness	12 mm
Column head thickness	12 mm
Corrosion allowance	2 mm
Weight of shell	74.7 kN

Weight of packings	71.9 kN
Weight of insulation	3.9 kN
Total weight of column	148.55 kN
Loading per linear meter	2433.28 N/m
Bending moment	131.72 kNm
Longitudinal stress	4.439 N/mm <sup>2</sup>
Circumferential stress	8.88 N/mm <sup>2</sup>
Dead weight stress	2.06 N/mm <sup>2</sup>
Bending stress	3.79 N/mm <sup>2</sup>
Buckling stress	5.85 N/mm <sup>2</sup>
Vessel support	123.97 N/mm <sup>2</sup>
Weight imposed on skirt, W <sub>S</sub>	36.7 kN
Thickness of support	12 mm



## CHAPTER THIRTEEN

### 13.0 PLANT LOCATION AND LAYOUT

#### 13.1 PLANT LOCATION

Plant location refers to the chosen region and place for establishing a business or factory.

However, the decision is taken only after assessing the costs and benefits of other alternative sites. It is a strategic decision that cannot be reversed once made. To avoid significant financial loss, choose a site based on its unique needs and circumstances.

Plant location comprises the strategic choice of a region where labor, resources, capital, machinery, and equipment meet to build a business or factory (Gothwal and Saha, 2015).

Choosing the right location is crucial for maximizing benefits and avoiding costs. When choosing a plant location, firms must consider labor, resource, and capital availability.

Thorough investigation and diligent study can increase the possibility of locating good places, yet there is no proven approach for determining the ideal one.

Organizations need to consider various factors in selecting plant location; availability of men, materials, money, etc. There is no exact method of analysis or assurance for the selection of an optimal location. But an extent of analysis and study can help in maximizing the probability of finding the right locations.

#### Plant Location Analysis

In selecting a plant location, it is imperative to consider certain factors. Some of the factors which can influence the selection of a plant location can be grouped under two main factors. That is, primary and secondary factors.

## **Primary Factors**

### **1. Availability of Raw Materials**

Material availability is the most important factor in selecting a plant location. When the conversion of raw materials into finished goods is the main task then such organization should be located in a place where the raw materials availability is maximum and cheap.

### **2. Availability of Labour**

The availability of labor is another crucial factor to consider when determining a plant's location. Labor can generally be grouped into skilled and unskilled, which are important for the construction, maintenance and operation of the proposed plant. The category of skilled labor which includes engineers and operators will be employed, trained to operate and maintain the plant. The unskilled labor which comprises people from the vicinity will be trained to carry out specific tasks. The rate of unemployment among the youth in the central region is relatively high ranking fourth out of sixteen regions across the country, therefore this would generally serve as a source of employment for the youth.



**Figure 65: Unemployment rate in Ghana**

### **3. Availability Of Utilities**

At any plant site, there should be easy access to utilities which will be needed in the operation of the plant. Portable water, fire station, emergency services and other offices for general administration should be made available for the smooth running of the plant. Insufficient access to fuel and power can be a major drawback for any manufacturing unit. Since fuel and power are essential for most manufacturing processes, it is advantageous to locate plants near power industries.

### **4. Nearness to the Market Proximity**

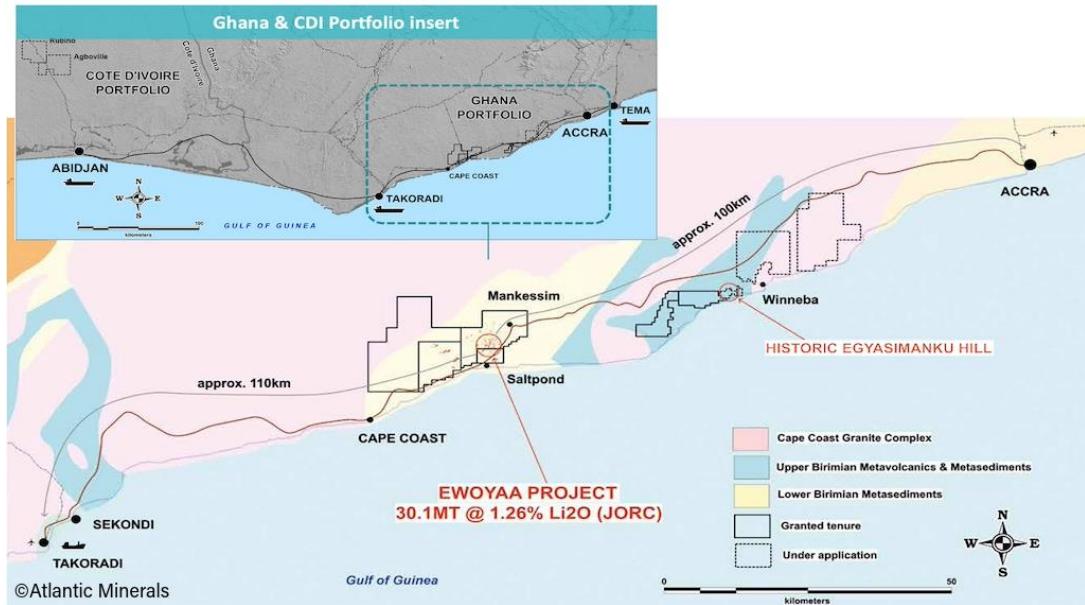
To the market for the finished goods not only minimizes transportation costs but also enables prompt customer service. When a plant is situated in close proximity to the markets, the risks of spoilage and breakage during transportation are significantly reduced.

### **5. Transportation Facilities**

The conveyance of raw materials to the plant site and products from the site considered in choosing the location of the plant. In Ghana, the most applicable means of transportation is by road and all the possible plant site are favoured by this factor. Depending on the size of the finished goods or raw materials suitable transportation is necessary such as roads, water, rail, and air. Here the point considered is transportation costs must be kept low.

### **6. Sufficient suitable land**

The land must be available for the proposed plant and for future expansion. The land should ideally be flat, well drained and have suitable load-bearing characteristics. According to the ministry of lands and natural resources, the Ewoyaa land is about 509km<sup>2</sup> land package secured located about 110km of Takoradi port and 100km of Accra.



**Figure 66: Geographical location of Ewoyaa**

## 7. Risk Assessment

It's critical to evaluate any threats a location might pose. To reduce possible hazards to the plant operations, personnel, and assets, variables like natural disasters, political unrest, security worries, and proximity to dangerous locations should be taken into account.

### Secondary Factors

Some secondary factors to consider when considering plant location includes;

#### 1. Government Policies

When selecting an appropriate plant location, it is crucial to consider the prevailing government policies in the local area. This includes factors such as licensing policies, access to institutional finance, government subsidies, and benefits associated with establishing a unit in urban or rural areas. Government influences both positive and negative incentives to motivate an entrepreneur to choose a particular location are made available. Positive includes cheap overhead facilities like electricity, banking, transport, tax relief, subsidies and liberalization. Negative incentives are in form of restrictions for

setting up industries in urban areas for reasons of pollution control and decentralization of industries.

## **2. Climate Suitability**

The suitability of the climate is an important factor to evaluate when deciding on a plant location. The climate can have a significant impact on labor efficiency and productivity.

## **3. Waste Disposal Industries**

Dealing with waste-intensive processes like chemical, sugar, and leather production face significant challenges in waste disposal. Therefore, a plant location should have proper provisions for the efficient disposal of waste.

## **4. Availability of Finance**

While finance is a critical factor for the smooth functioning of any business, it is of secondary importance when it comes to plant location decisions. Financial needs can typically be fulfilled if the plant is operating smoothly. However, it is still advantageous to locate the plant in areas where working capital and other financial requirements can be easily accessed (Trivikram, 2017).

### **Plant Location Justification**

Based on the conditions, Ewoyaa remains a suitable location for establishing the processing plant in Ghana. Ewoyaa's advantageous position less than a 2-hour drive from the Takoradi seaport and 100 kilometers from the capital city, Accra, ensures efficient product distribution and convenient transportation. Additionally, Ewoyaa's proximity to the lithium mining site allows for easy transportation of raw materials. Moreover, Ewoyaa is a developed community with readily available utilities such as electricity and water, thanks to its proximity to a High Voltage (HV) power source and the presence of nearby

rivers like the River Pra. The location also provides the benefit of access to skilled personnel for construction and plant maintenance.

## **13.2 PLANT LAYOUT**

In order to optimize the efficient movement of materials and personnel within a site, careful planning is required for the layout of process units and ancillary buildings. It is essential to ensure that hazardous processes are positioned at a safe distance from other structures. Additionally, consideration must be given to future expansion of the site (Sinnott, 1999).

The effective and cost-efficient construction and operation of a process unit rely heavily on the layout of the specified plant and equipment as outlined in the process flow-sheet. Several key factors should be considered:

**1. Economic Considerations:** Minimizing construction and operating costs is essential. This can be achieved by adopting a layout that reduces the length of connecting pipes between equipment and minimizes the amount of structural steelwork required.

**2. Process Requirements:** Specific process requirements must be taken into account. For instance, elevating the base of columns may be necessary to ensure the required net positive suction head for a pump.

**3. Convenience of Operation:** Equipment requiring frequent operator attention should be positioned conveniently close to the control room. Valves, sample 443 points, and instruments should be placed at easily accessible positions and heights. Sufficient working space and headroom should be provided to allow for easy equipment access and to minimize damage in the event of an explosion.

**4. Convenience of Maintenance:** Equipment necessitating dismantling for maintenance, like compressors and large pumps, should be positioned under cover.

**5. Safety:** Safety measures should be implemented, including the use of blast walls to isolate potentially hazardous equipment and confine the impact of an explosion. Process buildings should have at least two escape routes per level for operators.

**6. Plant Expansion:** Equipment should be placed in a manner that facilitates future process expansions. Sufficient space should be allocated on pipe alleys for future needs, and service pipes should be oversized to accommodate future requirements.

Therefore, the LiOH plant is recommended to follow a layout with the following considerations; efficient control and minimized transportation and handling of raw materials and products. Enough raw materials are to be obtained to feed process for at least two months and products remaining in storage for a maximum of three weeks. Restrict movement of workers especially at the plant area to ensure safety. Enough space to allow for maintenance works without ignoring the economic implications. Well ventilated rooms, especially those close to process area.

### **13.3 Plant Layout of Lithium Processing Plant**

For a typical ore processing plant, it is imperative to have a well layed out site to ensure safe and easy access and production.

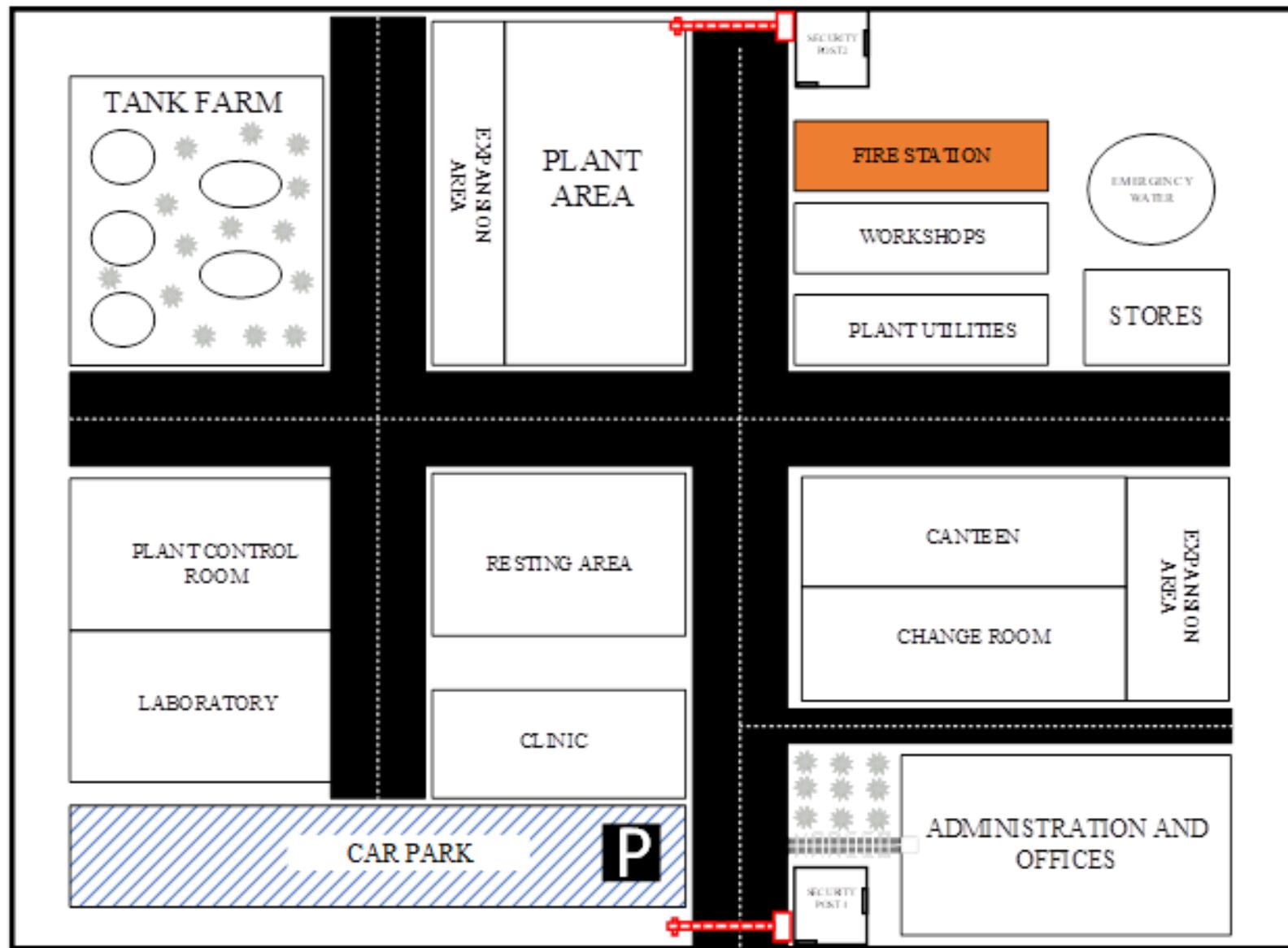


Figure 67: Plant layout of a lithium production site

## **CHAPTER FOURTEEN**

### **14.0 PLANT SAFETY AND POLLUTION CONTROL**

Safety is always an important factor when dealing with machinery and chemicals. It is vital to identify all potential hazards to protect employees and prevent accidents from occurring. Exposure to hazardous chemicals, extremely high pressures and temperatures should always be the main priority when dealing with an industrial process. There should always be a safety protocol in case of accidents, such as explosions or exposure to toxic chemicals. A well-written protocol will enhance the effectiveness with which people respond to accidents, reducing the number of mistakes made in dealing with the accident. Although the activated carbon produced at the plant is relatively harmless, the process itself has some inherent dangers.

#### **14.1 Hazards Associated with Lithium Production**

Hazard is any physical situation with a potential for human injury, damage to property, damage to the environment or a combination of these. The potential hazards associated with the production of activated carbon are; mechanical hazards, chemical hazards, electrical hazards and fire hazards.

These dangers are roughly categorized into four categories:

- i. Chemical hazards
- ii. Electrical hazards
- iii. Fire hazards
- iv. Mechanical hazards

##### **1. Chemical Hazards**

The plant is exposed to harmful compounds such as sulfuric acid, solvents, and other chemicals. The health consequences of these chemicals vary according to the level and duration of exposure. Solvents can cause skin irritation and respiratory problems, while acids can cause burns. Risks are recognized and mitigation steps are implemented to reduce exposure. This includes ventilation systems, PPE, and training programs. The Occupational Safety and Health Administration (OSHA) establishes rules for hazardous material handling and chemical storage to protect workers. Mitigating chemical dangers is vital for staff safety and well-being at the facility.

## **2. Mechanical Hazards**

Mechanical hazards are those associated with power-driven machines, whether automated or manually operated. Mechanical injuries are mostly caused either by contact or entanglement with machinery. Part of the machinery that could be hazardous to workers include sharp edges, hot surfaces, moving parts, belt and others. Mechanical hazards occur majorly in three areas:

- i. The point of operation: Point where work is performed on the material such as shredding, shaping etc.
- ii. Power transmission apparatus: Components of the mechanical system that transmit energy to the part of the machine performing the work. Examples include flywheels, pulleys, belts, gears etc.
- iii. Machine moving parts: Parts of the machine that move while the machine is working.

Mechanical hazards can be controlled by;

- i. Providing and applying appropriate safeguards; for example, guarding the point of operation, exposure of blades safeguards.

- ii. Conducting regular checks for any defects in machines as well as regular maintenance.
- iii. Anchoring or fixing machinery: machines must be anchored or fixed in one location to stop their movement.

### **3. Electrical Hazards**

Electrical hazards come in a variety of forms, but all have the potential to cause serious injury. Common types of electrical hazards include: contact with live wires resulting in electric shock and burns, fires due to faulty wiring, exposed electrical parts, ignition of 400 fires or explosions due to electrical contact with potentially flammable or explosive materials, improper wiring, improper grounding, damaged wire insulation and overloaded circuits. Electrical safety focuses on removing any potential electrical hazard and educating employees on the dangers of improperly working with electricity and electrical equipment. General electrical safety tips include:

- i. Inspecting electrical cords and plugs for damaged insulation and broken ground pins before use.
- ii. Reporting all exposed electrical parts, including wires, terminals, and missing circuit breakers immediately.
- iii. Regularly inspecting electrical tools and equipment for damage.
- iv. Replacing extension cords and wires with damaged insulation.
- v. Checking extension cords ratings to ensure they can handle the load required by electrical equipment before use.

#### **4. Fire Hazards**

Fire hazards are workplace hazards that involve the presence of flame or the risk of an uncontrolled fire. Fire hazards may arise because of accidental release and high-pressure release of flammable substance such as diesel and petrol in the presence of oxygen leading to an explosion.

Possible sources of fire outbreak are; electrical faults, smoking workers, flammable materials used in contact with hot surfaces, leaking valves or flanges which allow seepage of flammable liquids or gases, sparks from welding of equipment.

To avoid this, smoking would be prohibited in and around the plant, clearly labelling all tanks containing flammable liquids, storage tanks containing fuels should be coated with a heat-reflecting agent to minimize external heating, premises must have effective systems to detect fire and raise an alarm, fire extinguishers, blanket, water hydrants and other forms of fire prevention equipment would be situated at vantage points for easy access when the need arises.

#### **General Plant Safety Rules**

Measures to manage occupational risks and hazards are outlined in safety guidelines. Safety is a state in which hazards and conditions that can cause bodily, psychological, or material harm are avoided or regulated in order to protect people's and the environment's health and well-being. Some of the rules that will be enforced in the Lithium Hydroxide processing plant are listed below. These rules ensure that workers work in a safe environment.

- i. Training workers on the operation of the plant. This training will equip workers to know various hazards associated with the processing plant and ways to avoid them. The first responsibility of the worker is to ensure that he or she works in a

safe environment, knowing this will help workers ensure safety in the processing plant.

- ii. Provision of the necessary personal protective equipment and tools to workers. This will ensure that workers are protected before commencement of any activity in the processing plant. This regulation will be enforced by management as a prerequisite before any work is started.
- iii. Training employees on Material Safety Data Sheet (MSDS) and label to all chemicals. The MSDS contains information about chemicals. Knowing this will help workers know the type of chemical to be used, the safety requirements and storage conditions.
- iv. Placing warning signs at vantage points. These warning signs include fire alarm systems, gas leakage sensors and pressure sensors. In the case of fire accident, the fire alarm system will help workers respond quickly with the necessary action, and the pressure and gas leakage sensors will signal workers in case of high pressure build-up or gas leakage.
- v. Fire extinguishers, water hydrants and other firefighting equipment be placed at vantage points to help fight fire when there is an outbreak. In case of fire and/or explosion do not breathe fumes. Fight fire with normal precautions from a reasonable distance. Wear self-contained breathing apparatus. Wear full chemical protective clothing.
- vi. Working surfaces must be kept neat and tidy to prevent accidents. This helps workers to safeguard themselves in accident like fall

## **Environmental and Product Safety**

Gas emissions are the major environmental issue factories have to address to ensure their operation. With the burning of natural gas, gases like carbon dioxide, SO<sub>x</sub> and NO<sub>x</sub> are released. Gases like SO<sub>x</sub> and NO<sub>x</sub> have direct and indirect effects on human health and the environment. At high concentrations, SO<sub>x</sub> can harm trees and plants by damaging foliage and decreasing growth. SO<sub>2</sub> and other Sulphur oxides can contribute to acid rain which can harm sensitive ecosystems. NO<sub>x</sub> can cause breathing problems, headaches, chronically reduced lung function. High levels of NO<sub>x</sub> can harm vegetation, including leaf damage and reduced growth. It can make vegetation more susceptible to disease and frost damage. These gases have adverse effects on the ozone layer which results in the greenhouse effect and global warming.

Due to the harmful effects these gases pose to the environment, scrubbers are installed to trap and treat the gases before they are released into the atmosphere.

Product safety is the ability of a product to be safe for the intended use. A safe product provides either no risk or a minimum acceptable level risk, taking into account the normal or reasonably foreseeable use of the product and the need to maintain a high level of protection for consumers. The lithium hydroxide produced would be stored in a tank tightly closed to prevent it from escaping into the environment.

### **14.2 POLLUTION AND POLLUTION CONTROL**

Pollution is the introduction of harmful substances or products into the environment. These harmful materials are called pollutants. Pollutants damage the quality of air, water and land. Industrial pollution continues to cause significant damage to the earth and all inhabitants due to chemical wastes, radioactive materials and the emission of harmful gases. Pollution prevention comprises source reduction and in-process recycling leading

to waste minimization or waste reduction. It includes changes in technology, materials, processes, operations, and procedures and in-process recycling.

### **Pollutants Produced on the Plants and Ways it Can be Mitigated**

#### **i. Air Pollutants**

Particulate matter such as fine dust particles generated during crushing, grinding and handling of lithium spodumene ore can contribute to air pollution if not properly controlled. Gaseous emissions through combustion processes can release pollutants such as sulfur dioxide ( $\text{SO}_2$ ), nitrogen oxides ( $\text{NOx}$ ), and carbon monoxide (CO) into the air. Certain chemical solvents or reagents used in the extraction or purification processes may release VOCs, which can contribute to air pollution and potentially lead to the formation of smog or respiratory issues.

#### **Mitigation steps**

1. Particulate Matter: Dust collection systems such as cyclone separators, electrostatic separators, wet scrubbers and others can be installed to capture and remove particulate matter from air emissions. Regular maintenance and cleaning of equipment and work areas can also help minimize dust generation.
2. Gaseous Emissions: To mitigate the impact of  $\text{NOx}$  and  $\text{SOx}$  on the environment, strict adherence to EPA standards is followed, ensuring that only permissible emission limits are released into the surroundings. Several methods can be employed to reduce  $\text{NOx}$  and  $\text{SOx}$  pollution, including Selective Catalytic Reduction, Selective Non-Catalytic Reduction, wet or dry absorption, and adsorption. Among these methods, adsorption is often favored due to its cost effectiveness.

3. Volatile Organic Compounds (VOCs): The use of volatile solvents or reagents can be substituted and replaced with less volatile alternatives. Proper ventilation and containment measures are utilized to capture and control VOC emissions. With this, control technologies like carbon adsorption is used to remove VOCs from air streams due to its efficiency and cost.

## **ii. Solid Waste**

Tailings and Residues: After the ore is processed, solid waste materials like tailings and residues containing low-grade minerals and other by-products are generated. Packaging waste, such as cardboard, plastic or metal containers can contribute to solid waste generation.

To mitigate solid waste, a comprehensive waste management plan that includes containment and proper disposal of tailings is developed. Techniques like thickening, dewatering, and proper storage to reduce the volume and environmental impact of these waste materials are also implemented. Recycling and reusing are also employed for certain by-products.

## **iii. Water pollutants**

Dissolved Minerals and Chemicals: The process of leaching and purification of lithium spodumene ore involves the use of water, which can become contaminated with dissolved minerals, chemicals, and trace elements from the ore. Various stages of the production process generate wastewater containing suspended solids, residual chemicals and potentially high pH levels.

To prevent or control water pollution, water treatment processes including sedimentation, filtration and chemical precipitation to remove dissolved minerals and chemicals from process water is taken into consideration. Monitoring programs to ensure compliance with

water quality standards and regulations are also employed. The treated water can also be reused for non-potable purposes within the plant.

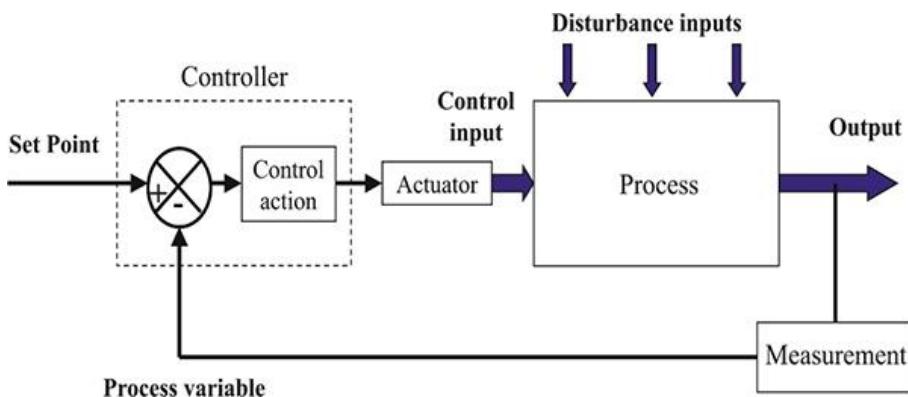
### **iii. Noise Pollution**

Noise is an unwanted loud sound that appears to be unpleasant to the ears. Long exposure to excessive noise levels can cause permanent damage to hearing, heart disease, sleep disturbances and stress. The World Health Organization defines noise above 65 403 decibels(dB) as noise pollution. Noise from the plant will result from the operations of equipment like the crusher, cyclones and screens. The plant will be sited away from residential areas to avoid disturbing residents. Employees will be given earplugs to protect them from any noise pollution.

## CHAPTER FIFTEEN

### 15.0 PLANT INSTRUMENTATION AND PROCESS CONTROL

In general terms, control is concerned with the manipulation of the inputs to a system (a machine, process or plant) so that the outputs meet certain specifications. Control is thus a broad concept which encompasses long-term process operating strategy based on process evaluation, as well as manual control and the various forms of real-time automatic control such as logic or sequencing control, single-variable or multivariable continuous control and supervisory control. These various forms of real-time control are collectively known as process control.



A typical control system

A process denotes an operation or series of operations on fluid or solid materials during which the materials are placed in a more useful state. The objective of a process is to transform raw materials into desired product using available feedstock sources of energy in the most cost-effective way.

process variables are factors that affect the process and can change the process. The main process variables involved in this project include; temperature, pressure, flow and level. Small changes in a process can have a greater impact on the output of the process.

Changes in process variables must be carefully and consistently controlled to produce the desired output with minimum raw materials and energy.

### **The primary objectives of instrumentation and control schemes for this plant**

- i. To keep the process variables within a safe operating mode
- ii. To achieve the desired product output (production rate).
- iii. To maintain the product composition within the specified quality standards
- iv. To operate at the lowest production cost.
- v. To detect dangerous situations as they develop and to provide alarms and automatic shut-down systems
- vi. To provide interlocks and alarms to prevent dangerous operating procedures

## **PRINCIPLES OF PROCESS CONTROL**

Instruments and devices used in control loops include transmitters, sensors, controllers, indicators, recorders, valves and pumps. All control loops in the process control industries require three tasks to occur:

### **1. Measurement or Collection**

This involves measuring devices, also known as sensors. Some of these devices are thermocouples, resistant temperature detectors, orifice plates, etc. A change in their properties indicates a corresponding change in the process variable of the system that is under study.

### **2. Comparison**

This task is taken by a device known as the controller. Controllers always have an ability to receive input, to perform a mathematical function or algorithm with the input, and to

produce an output signal. Common examples of controllers include: Programmable logic controllers (PLCs) and Distributed control systems (DCSs).

### **3. Action or Correction**

The action is taken by the final control element that acts to physically change the manipulated variable. In most cases, the final control element is a valve used to restrict or cut off fluid flow, but pump motors, louvers are typically used to regulate air flow, solenoids, and other devices can also be used as final control elements.

#### **15.1 Control Parameters and Sensors**

All units in the production of the hydroxides are monitored in the control room. By means of a signal, personnel would be alerted if a process variable is approaching a hazardous limit. Process variables to be measured are:

1. Temperature
2. Pressure
3. Level
4. Flow
5. pH

#### **Temperature Control System**

In control system for the production of lithium hydroxides from lithium containing ore, temperature is an important parameter to control. Higher inlet temperatures above the designed temperature would cause abrupt expansion and contraction of the system. Therefore, temperature control system will be used to monitor and regulate rotary kilns, reactors, boiler and crystallizer.

#### **Pressure Control System**

Pressure is an expression of the force required to stop a fluid from expanding and is usually stated in terms of force per unit area. A pressure sensor is a device for pressure measurement of gases or liquids. Pressure control is necessary for most systems handling vapor or gas which helps in maintaining the optimal pressure levels during operations in order to ensure safety. The pressure control system and sensor are purposely for measuring the pressure of a unit, generate a signal value which is readable to operators and to be able to maintain set point of operating pressures in order to avoid excessive build-up of pressures. This control system would be used specifically to check the pressure of specifically boiler and compressor in the plant.

### **Level Control System**

Level or liquid level is the height of the water column, liquid and powder etc., at the desired measurement of height between minimum level point to maximum level point. The measurement principle is, head pressure method. The units are m, mm, cm, percentage. The method used will vary widely depending on the nature of the industry, the process, and the application. The measured medium can be liquid, gas or solid and stored in vessels (open/closed tanks).

When determining the type of level sensor that should be used for a given application, there are a series of factors that must be considered such as the;

1. Type of tank (Open tank or closed tank)
2. Level sensor (inserted into the tank or completely external)
3. Contact (contact or non-contact)
4. Type of measurement (continuous or point)
5. Type of material
6. State of material (Liquid or Solid, Clean or Slurry).

## **Flow Control System**

Flow measurements are normally indirect measurements using differential pressures to measure the flow rate. Differential pressure measurements can be made for flow rate determination when a fluid flows through a restriction. The restriction produces an increase in pressure which can be directly related to flow rate. Flow control is one of the crucial process variables controlled in industry. It is usually associated with inventory control in a storage tank or other equipment or with feed to the process. The process material balance is set by flow controllers on feed streams as flow is related to the density and volume of contents to the process. The flow rate of materials is measured with valves, flow meters and weighing conveyors; depending on the material to be transported (gas, liquid or solid). Type of flow meters include: orifice plate, venturi tube, elbow flow meter, rotameter, pitot tube, etc.

**Table 21: Equipments and their control parameters**

<b>Equipment</b>	<b>Control parameter</b>	<b>Sensors</b>	<b>Working range</b>	<b>Properties</b>
Rotary kiln	Temperature	K-type thermocouple	-250°C to 1260°C	Good accuracy Fast response time
Storage tank	Level	Radar sensors	0 – 10 m	Non- invasive Withstand harsh environment conditions

				Good accuracy
Reactor	Temperature pH	Resistance-Type detector (RTD)  Differential pH sensor	-50°C to 250°C  0 - 14	Good accuracy  High stability and repeatability  Industrial applications  Variable span
Compressor	Pressure	Piezoresistive pressure sensor	3 – 14000 psi	High sensitivity  Faster response time  Wider pressure range
Boiler	Temperature Pressure	Resistance-Type detector	-200 - 600 °C	Good accuracy  More stable and less prone to drift over time

		Bourdon tube pressure sensor	Up to 100 MPa	Mechanical, no need for power supply  Robust to vibration and shock
Crystallizer	Temperature	T-type thermocouple	-160 to 400 °C	Good accuracy  Good reproducibility
Mixer	Flow	Magnetic flow sensor	0.0024 – 22.68 m <sup>3</sup> /h	Low maintenance  Good accuracy  For the flow of conductive liquids, including slurries and corrosive  Moderate cost as compared to other sensors

## CHAPTER SIXTEEN

### 16.0 PLANT ECONOMICS ANALYSIS

Plants are built to make products for sale where profit is expected in return. An acceptable plant design must therefore present a process that is capable of operating under conditions that will yield profit. An estimate of the investment required is needed before the profitability of a plant can be assessed. The economic analysis involves the estimation of the total capital investment, total production cost and the rate of return on the investment (Sinnott and Towler, 2013).

**Table 22: Basis of the Economics Analysis**

Construction Period	January 2025 – December 2025
Commencement Period	January 2026
Plant Operation	3 shifts per day; 8 hours per shift.
Type of Process	Continuous
Plant Life	25 years
Plant Attainment	335 years
Dollar to Cedi Exchange Rate	1 USD = GHS 15.41 (5th July, 2024); Bank of Ghana Exchange Rate
Tax Rate	25 %

## **TOTAL CAPITAL INVESTMENT**

Total capital investment is the amount of money needed to physically build, start the plant up and operate it to the point when income is earned. It comprises the fixed capital investment and the working capital. Fixed capital is the capital needed to supply the necessary manufacturing and plant facilities. It includes direct costs (such as equipment costs, equipment installation costs, service facilities, land costs etc.) and indirect costs (such as engineering and supervision costs, contractor fees and contingency fees) (Peters and Timmerhaus, 1991). Working capital is the capital needed for handling tax payment, labour service, and raw materials. Working capital also provides funds for the startup of the newly built plant and is typically 15% of the fixed capital investment (Sinnott, 2003).

There are several methods of estimating the capital cost of a plant. Among them include the cost escalation (inflation), rapid capital cost methods and the factorial method. There are two ways of going with the factorial method; the use of Lang factors or detailed factorial estimates. The Lang factors gives a function relating the entire plant cost to the cost of equipment.

In summary;  $TCI = FCI + WC$  (Working Capital)

$$FCI = DC + IC$$

$$DC = PCE + \text{fractions of PCE}$$

$$IC = \text{fractions of DC}$$

$$WC = 0.15 FCI$$

## Estimation of Equipment Costs (PCE)

EQUIPMENT	COST	QUANTITY	PURCHASE COST
Air compressor	\$35,000.00	2	\$70,000.00
Air cooler	\$70,000.00	2	\$140,000.00
Ball Mill	\$80,000.00	1	\$80,000.00
Belt filter	\$60,000.00	2	\$120,000.00
Boiler	\$54,000.00	1	\$54,000.00
Conveyor belt	\$20,000.00	6	\$120,000.00
Crystallizer	\$70,000.00	1	\$70,000.00
Evaporator	\$45,000.00	1	\$45,000.00
Gas compressor	\$20,000.00	1	\$20,000.00
Gas holder	\$29,000.00	1	\$29,000.00
Jaw Crusher	\$87,000.00	1	\$87,000.00
Mixer	\$28,000.00	1	\$28,000.00
Raw Material Storage	\$17,000.00	1	\$17,000.00
Reactor	\$79,000.00	4	\$316,000.00
Rotary Dryer	\$57,000.00	1	\$57,000.00
Rotary Filter	\$13,000.00	1	\$13,000.00
Rotary kiln	\$340,000.00	2	\$680,000.00

Screen	\$27,000.00	2	\$54,000.00
Scrubber	\$54,000.00	1	\$54,000.00
Tank	\$10,000.00	10	\$100,000.00
Three way Valve	\$80.00	3	\$240.00
Truck	\$45,000.00	3	\$135,000.00
Pump	7000	15	\$105,000.00
Total Purchase Cost of Equipment			\$3,306,445.44

### **Direct Plant Cost Estimation**

This is the investment needed to purchase and installed process equipment and its auxiliaries example cost of piping, process control systems, building and structure, purchase of equipment, etc (Sinnott, 2005).

Equipment Erection	0.5	\$1,653,222.72
Piping	0.7	\$2,314,511.81
Instrumentation	0.5	\$1,653,222.72
Electrical	0.3	\$991,933.63
Building	0.3	\$991,933.63
Utilities	0.5	\$1,653,222.72
Storages	0.2	\$661,289.09

Site Development	0.1	\$330,644.54
Ancillary Buildings	0.3	\$991,933.63
<b>TOTAL</b>		<b>\$14,548,359.94</b>

Physical Plant Cost = PCE + DC

$$= \$17,854,805.38$$

### INDIRECT COST ESTIMATION

INDIRECT COST	FACTOR	cost (\$)
Design and Engineering	0.65	\$9,456,433.96
Contractor's Fee	0.35	\$5,091,925.98
Contingency	0.75	\$10,911,269.95
INDIRECT COST	1.75	\$25,459,629.89

### Fixed Capital Investment

FCI = Direct Cost + Indirect Cost

$$= \$40,007,989.82$$

Working Capital = 15% of FCI

$$= \$6,001,198.47$$

Total Capital Investment = FCI + WC

$$= \$46,009,188.30$$

## **TOTAL PRODUCT COST ESTIMATION**

Total production cost involves the estimation of all expenses that go into the production process. It is the sum of manufacturing costs and general expenses (Peters and Timmerhaus, 1991). In total product cost estimation, the best source of information is data from similar or identical projects where quick and reliable estimates of manufacturing costs and general expenses are made from existing records. Where specific information is not available, standard factors used in the estimation of chemical engineering project costs are applied to make an estimate of the total production cost per annum (Peters and Timmerhaus, 1991).

Table:

<b>Raw material Cost Estimation</b>			
Raw material Cost Estimation	Cost(\$/tonnes)	qty needed(tonnage/year)	Total cost(\$)
Mined Alpha Spodumene	\$1,029.00	160800	\$165,463,200.00
Natural Gas	\$2.45	40200	\$98,490.00
Sulphuric Acid	\$160.00	54286.08	\$8,685,772.80
Calcium Hydroxide	\$300.00	19601.52	\$5,880,456.00
Sodium Hydroxide	\$570.00	25165.2	\$14,344,164.00

Total			\$194,472,082.80
<b>Labour Cost Estimation (Direct Labour Cost)</b>			
Staff	Annual cost/head(\$)	Number of Heads	Annual cost(\$)
Plant Manager	\$180,000.00	2	\$360,000.00
Production Manager	\$150,000.00	2	\$300,000.00
Chemical Engineer	\$83,000.00	5	\$415,000.00
Electrical Engineer	\$83,000.00	3	\$249,000.00
Mechanical Engineer	\$80,000.00	3	\$240,000.00
Quality Control	\$60,000.00	2	\$120,000.00
Technicians	\$50,000.00	10	\$500,000.00
Lab Technicians	\$44,000.00	5	\$220,000.00
Supervisor	\$57,000.00	5	\$285,000.00
Total			\$2,689,000.00
<b>Indirect Labour Cost</b>			
Staff	Annual cost/head(\$)	Number of Heads	Annual cost(\$)
Managing Director	\$210,000.00	1	\$210,000.00

Human resource manager	\$97,000.00	2	\$194,000.00
Sales manager	\$75,000.00	2	\$150,000.00
Accountant	\$50,000.00	2	\$100,000.00
Account clerk	\$46,000.00	2	\$92,000.00
Secretary	\$40,000.00	3	\$120,000.00
Storekeeper	\$40,000.00	2	\$80,000.00
Receptionist	\$30,000.00	2	\$60,000.00
Nurses	\$40,000.00	2	\$80,000.00
Cleaners	\$20,000.00	8	\$160,000.00
Caterer	\$25,000.00	6	\$150,000.00
Security personnel	\$25,000.00	6	\$150,000.00
Drivers	\$25,000.00	3	\$75,000.00
Total			\$1,621,000.00
<b>Total direct and indirect labor cost(\$)</b>			\$4,310,000.00
SSNIT	0.05 of total labour	\$215,500.00	
<b>Total labor cost(\$)</b>		\$4,525,500.00	

Total Labor Cost = Total direct and indirect cost + SSNIT cost

## **Manufacturing Costs**

Manufacturing costs are the expenses directly connected with the manufacturing operation or the physical equipment of a process plant. It is divided into 3 categories; direct production costs, fixed charges and plant overhead costs (Sinnott, 2003; Peters and Timmerhaus, 1991).

<b>Direct Production Cost</b>	
<b>Division</b>	<b>Cost(\$)</b>
Raw Materials	\$194,472,082.80
Operating Labor	\$4,525,500.00
Direct Supervisory and Clerical	\$791,962.50
Utilities	\$1,653,222.72
Maintenance and Repairs	\$2,000,399.49
Operating Supplies	\$200,039.95
Laboratory Charges	\$678,825.00
Patents and Royalties	\$400,079.90
Actual Total	\$204,722,112.36
<b>Fixed Charges</b>	
<b>Division</b>	<b>Cost(\$)</b>
Depreciation	\$1,600,319.59
Local taxes	\$400,079.90

Insurance	\$280,055.93
Rent	\$-
<b>Plant Overhead Cost</b>	50% of operating labour (\$2,262,750.00)
<b>Total Manufacturing Cost</b>	\$209,265,317.78

## **DEPRECIATION.**

Depreciation is the reduction in the value of a plant over time due to wear and technical obsolescence (Silla, 2003). To account for this reduction in value, it is assumed that the decrease occurs gradually over the typical lifespan of the plant. This reduction in value 473 is referred to as depreciation. The straight-line method is used to estimate depreciation, and the salvage value (SV) of the plant is projected to be 10% of the fixed capital investment (Sinnott, 2005).

$$\text{Annual depreciation} = \frac{V - V_s}{n} = \$1,600,319.59$$

d = Annual Depreciation

V<sub>s</sub>=SV= Salvage Value = 0.15 of FCI

V = Fixed Capital Investment (FCI)

<b>General Expenses</b>	
Sales expenses	\$12,555,919.07
Research and development	\$8,370,612.71
Total	\$20,926,531.78

<b>Total Product Cost</b>	\$230,191,849.56
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## TOTAL ANNUAL SALES

<b>Product</b>	<b>Unit Selling/tonnage</b>	<b>Ton/annum</b>	<b>Selling Price</b>
LiOH	\$15,500.00	15420.72	\$239,021,160.00
Na <sub>2</sub> SO <sub>4</sub>	\$300.00	451.6872	\$135,506.16
<b>Total(\$)</b>		<b>15872.4072</b>	<b>\$239,156,666.16</b>

## Gross Annual Profit (GAP)

GAP = Total Annual Sales – Total Product Cost

$$= \$8,964,816.60$$

## Net Annual Profit (NAP)

Net Annual Profit = Gross Annual Profit – Income Tax

Taking Income Tax Rate = 25% of Gross Annual Profit

$$\text{NAP} = \$6,723,612.45$$

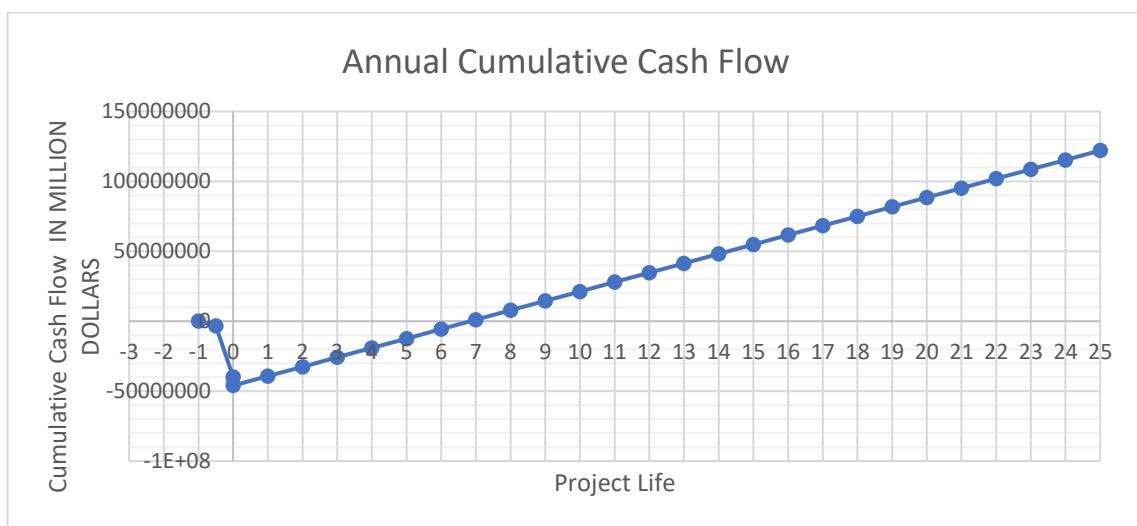
## PROFITABILITY ANALYSIS

Profitability analysis determines the economic valuation of a project. Profitability analysis of the project will be based on the Break-even point (BEP), Turnover ratio (TOR), Rate of Return (ROR), Discounted Cash Flow Rate of Return (DCFR) and Payback Period.

## Cumulative Cash Flow Analysis

<b>Year</b>	<b>Annual Production rate(ton)</b>	<b>Annual Product cost(\$)</b>	<b>Depreciation(\$)</b>	<b>Cumulative cash flow(\$)</b>
0	15872.4072	230191849.6	1600319.593	-46009188.3
1	15872.4072	230191849.6	1600319.593	-39285575.84
2	15872.4072	230191849.6	1600319.593	-32561963.39
3	15872.4072	230191849.6	1600319.593	-25838350.94
4	15872.4072	230191849.6	1600319.593	-19114738.49
5	15872.4072	230191849.6	1600319.593	-12391126.03
6	15872.4072	230191849.6	1600319.593	-5667513.581
7	15872.4072	230191849.6	1600319.593	1056098.871
8	15872.4072	230191849.6	1600319.593	7779711.324
9	15872.4072	230191849.6	1600319.593	14503323.78
10	15872.4072	230191849.6	1600319.593	21226936.23
11	15872.4072	230191849.6	1600319.593	27950548.68
12	15872.4072	230191849.6	1600319.593	34674161.13
13	15872.4072	230191849.6	1600319.593	41397773.59
14	15872.4072	230191849.6	1600319.593	48121386.04
15	15872.4072	230191849.6	1600319.593	54844998.49

16	15872.4072	230191849.6	1600319.593	61568610.95
17	15872.4072	230191849.6	1600319.593	68292223.4
18	15872.4072	230191849.6	1600319.593	75015835.85
19	15872.4072	230191849.6	1600319.593	81739448.3
20	15872.4072	230191849.6	1600319.593	88463060.76
21	15872.4072	230191849.6	1600319.593	95186673.21
22	15872.4072	230191849.6	1600319.593	101910285.7
23	15872.4072	230191849.6	1600319.593	108633898.1
24	15872.4072	230191849.6	1600319.593	115357510.6
25	15872.4072	230191849.6	1600319.593	122081123



Payback time = 6.8 years

Break-even point(tonne) = 11740.12 tonnes

$$\text{Turn Over Ratio} = \frac{\text{Total Annual Sales}}{\text{Fixed Capital}} = 5.98$$

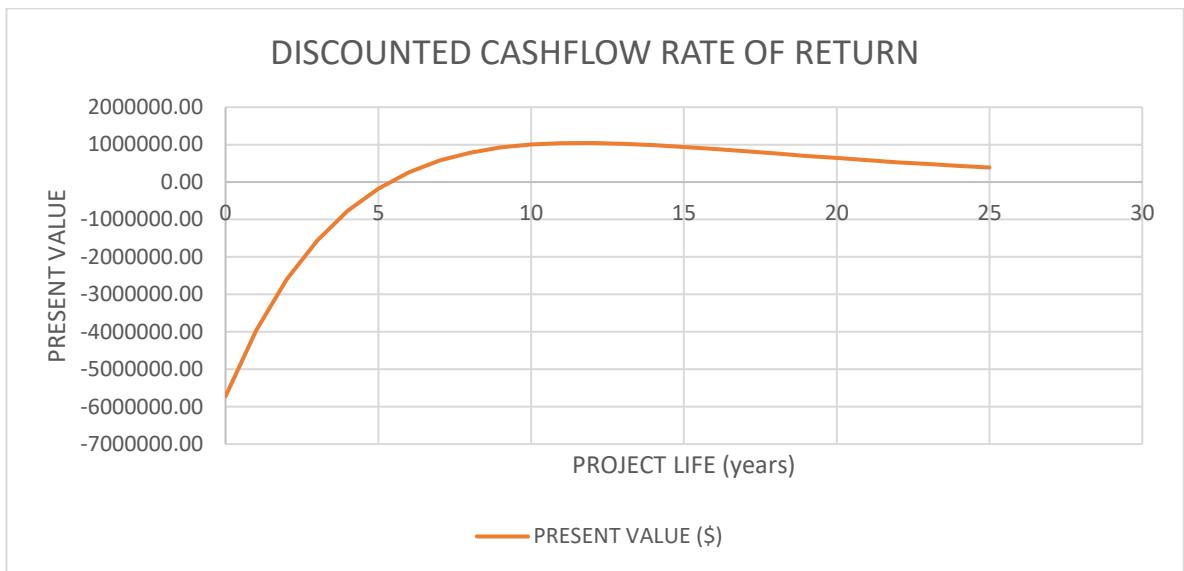
$$\text{Rate of Return} = \frac{1}{\text{Turn over ratio}} = 0.17$$

### Discounted Cash Flow of Return

Table:

Year	Cumulative cash flow(\$)	PRESENT VALUE (\$)		
		0.1214	0.14	0.1
0	-46009188.30	-46009188.30	-46009188.30	-46009188.30
1	-39285575.84	-35031142.75	-34461031.44	-35714159.86
2	-32561963.39	-25891245.72	-25055373.49	-26910713.55
3	-25838350.94	-18320120.95	-17440150.91	-19412735.49
4	-19114738.49	-12085178.96	-11317459.67	-13055623.58
5	-12391126.03	-6985808.58	-6435562.58	-7693914.37
6	-5667513.58	-2849178.49	-2582042.95	-3199163.66
7	1056098.87	473426.81	422056.53	541945.71
8	7779711.32	3109803.39	2727248.25	3629292.75
9	14503323.78	5169614.68	4459887.26	6150825.07
10	21226936.23	6746820.07	5725834.64	8183902.82
11	27950548.68	7921790.12	6613585.51	9796496.80

12	34674161.13	8763147.85	7196937.76	11048256.32
13	41397773.59	9329370.53	7537267.31	11991460.41
14	48121386.04	9670182.41	7685466.06	12671864.94
15	54844998.49	9827764.67	7683591.35	13129456.59
16	61568610.95	9837806.06	7566268.25	13399123.59
17	68292223.40	9730414.36	7361880.77	13511252.33
18	75015835.85	9530906.66	7093582.57	13492257.46
19	81739448.30	9260493.84	6780153.51	13365052.96
20	88463060.76	8936873.03	6436724.70	13149470.30
21	95186673.21	8574739.92	6075391.52	12862629.50
22	101910285.7	8186231.24	5705730.96	12519268.26
23	108633898.1	7781306.71	5335237.82	12132033.62
24	115357510.6	7368078.21	4969691.65	11711740.24
25	122081123	6953093.17	4613465.02	11267599.11
	TOTAL	0.0000004	-31310807.90	52558429.95

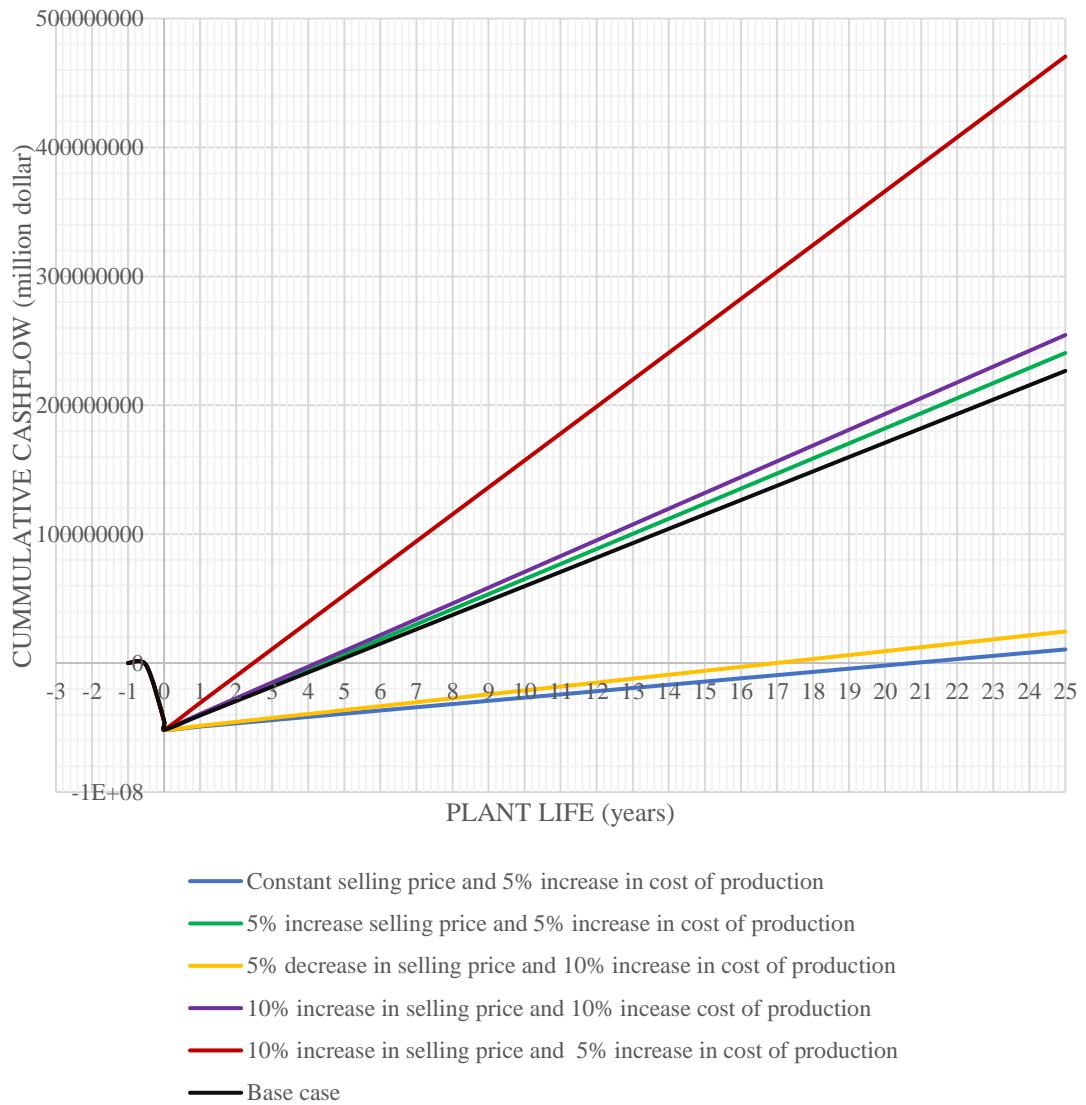


**Graph 1: Discounted Cashflow Rate of Return**

### SENSITIVITY ANALYSIS

Case	Percentage change in Selling price	Percentage change in Cost price	In years	approximate months
Base	constant	constant	6.84	64
1	0	0.05	8.95	107
2	0.05	0.05	5.10	61
3	0.05	0.1	8.26	99
4	0.1	0.1	4.87	58
5	0.1	0.05	3.56	43

## GRAPH OF SENSIVITY ANALYSIS



**Graph 2: Summary of sensitivity analysis**

## CHAPTER SEVENTEEN

### 17.0 CONCLUSION AND RECOMMENDATION

#### 17.1 Conclusion

In conclusion, spodumene, the raw material needed for the production of lithium hydroxide is readily available in Ghana and the processing plant is less detrimental to the environment.

The product is priced at \$ 20,000/ ton with ready market. However, this price is relatively less to already existing market. With a net annual profit of \$ 33,897,808.89/yr , a rate of return of 28% and a payback time of 6 year and 8 months from the economic analysis, the project for the processing plant for lithium hydroxide production from spodumene ore is economically feasible under the current technological and economic state of Ghana and the world at large.

#### 17.2 Recommendations

- Considering an estimated amount of 30.1 million tons of spodumene deposit in Ghana, it is recommended to increase the capacity of the processing plant to minimize exporting spodumene concentrate.
- Lithium hydroxide production from spodumene should be pursued by Government and investors as a viable venture in Ghana rather than ending production at primary stage.
- Roasting spodumene ore with sodium salts should be explored in the production of lithium hydroxide in order to minimize acid mist and SOx pollutants which comes with acid roasting with sulphuric acid.
- Alternative approaches in utilizing heat generated from the roasting processes should be looked at.

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## APPENDIX

### Appendix A (Material Balance)

Feed forward calculation

Initial amount of spodumene ore obtained or to be used =20,000kg/h =20t/day

COMPOSITION	% by mass	Amount (kg/h)
Alpha spodumene	80	16000
Quartz	10 (impurity)	2000
Alumina	5 (impurity)	500
Annite	2.5(solid solution)	500
Phlogopite	2.5 (solid solution)	1000

$$\alpha\text{-S} = \frac{80}{100} \times 20000 = 16000 \text{ kg/h}$$

$$\text{Alumina} = \frac{5}{100} \times 20000 = 1000 \text{ kg/h}$$

$$\text{Quartz} = \frac{10}{100} \times 20000 = 2000 \text{ kg/h}$$

$$\text{Annite} = \frac{2.5}{100} \times 20000 = 500 \text{ kg/h}$$

$$\text{Phlogopite} = \frac{2.5}{100} \times 20000 = 500 \text{ kg/h}$$

### Balance on Rotary-Kiln 1

Reaction :  $[\alpha - \text{S}] \rightarrow [\beta - \text{S}]$  Conversion= 96% (conversion based on molar basis)



$$m [\alpha - \text{S}] = 16000 \text{ kg/h}$$

$$Mr [\alpha - \text{S}] = 2[6.9 + 26.98 + 28(2) + 16(6)] = 371.76 \text{ g/mol}$$

$$n [\alpha - \text{S}] = \frac{m}{mr} = \frac{16000}{371.76} = 43.038 \text{ kmol}$$

$$n [\alpha - S] \text{ converted/reacted} = 0.96 \times 43.038 = 41.316 \text{ kmol}$$

$$n [\alpha - S] \text{ left unconverted} = 1.72 \text{ kmol}$$

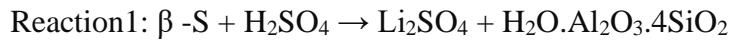
$$[\beta - S] \text{ obtained} = 41.316 \text{ kmol}$$

$$m = n \times Mr [\beta - S] = 41.316 \times 371.76 = 15359.636 \text{ kg}$$

$$\text{Mass of unconverted } [\alpha - S] = n \times Mr [\alpha - S] = 1.72 \times 371.76 = 640.36 \text{ kg}$$

## RK-2

Quartz ( $\text{SiO}_2$ ), from literature, does not react or dissolve in any of the common acids [i.e.  $\text{HCl}$ ,  $\text{H}_2\text{SO}_4$  or  $\text{HNO}_3$ ]. They only dissolve in HF (hydrofluoric acid) [source; echimi.com]



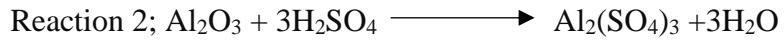
$$m [\beta - S] = 15359.636 \text{ kg} \quad n [\beta - S] = \frac{m}{mr} = 41.316 \text{ kmol}$$

$$n [\beta - S] \text{ converted} = 0.97 \times 1.316 = 40.076 = n (\text{Li}_2\text{SO}_4)$$

$$n [\beta - S] \text{ unconverted} = 41.316 - 40.076 = 1.239 \text{ kmol}$$

$$n [\text{H}_2\text{O} \cdot \text{Al}_2\text{O}_3 \cdot 4\text{SiO}_2] \text{ formed} = 40.076 \text{ kmol}$$

$$n [\text{H}_2\text{SO}_4] \text{ consumed/used} = 40.076 \text{ kmol}$$



From literature: conversion = 96%

$$\text{But } n = \frac{m}{Mr} = \frac{1000}{26.98(2) + 16(3)}$$

$$= 9.808 \text{ Kmol}$$

$$n (\text{Al}_2\text{O}_3 \text{ converted}) = 0.96 \times 9.808$$

$$= 9.415 \text{ Kmol}$$

$$n (\text{Al}_2\text{O}_3 \text{ left unconverted}) = 9.808 - 9.415$$

$$= 0.393 \text{ Kmol}$$

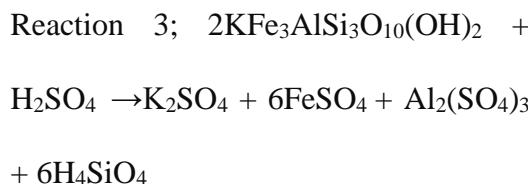
Also,  $n (\text{Al}_2\text{O}_3 \text{ converted}) = n [\text{Al}_2(\text{SO}_4)_3 \text{ formed}] = 9.415 \text{ Kmol}$

$$\text{From the mole ratio, } \frac{n[\text{Al}_2(\text{SO}_4)_3]}{n[\text{H}_2\text{O}]} = \frac{1}{3},$$

$$\text{thus } n (\text{H}_2\text{O formed}) = 9.415 \times 3$$

$$= 28.245 \text{ Kmol}$$

Therefore,  $n (\text{H}_2\text{SO}_4 \text{ consumed}) = 9.415 \times 3 = 28.245 \text{ Kmol}$



From literature: conversion = 96%

$$m (\text{Annite fed}) = 500 \text{ Kg}$$

$$\begin{aligned} \text{Mr (Annite)} &= 2[39 + 55.8(3) + 26.98 + \\ &28(3) + 16(10) + 16(2) + 1(2)] = 1022.76 \text{ Kg/mol} \end{aligned}$$

$$n (\text{Annite fed}) = \frac{500}{1022.76}$$

$$= 0.489 \text{ Kmol}$$

$$\begin{aligned} n (\text{Annite converted}) &= 0.96 \times 0.489 \\ &= 0.469 \text{ Kmol} \end{aligned}$$

$$\begin{aligned} n (\text{Annite unconverted}) &= 0.489 - 0.469 \\ &= 0.0197 \text{ Kmol} \end{aligned}$$

from reaction 4;

From literature; conversion = 96%

$$m (\text{phlogopite fed}) = 500 \text{ kg}$$

$$\begin{aligned} \text{Mr} &= 2(39 + 24(3) + 26.98 + 28(3) + \\ &16(10) + 16(2) + 1(2)) = 831.96 \end{aligned}$$

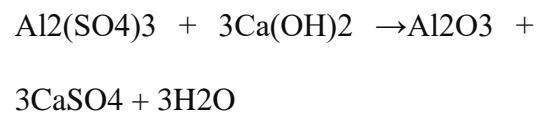
amount converted = 100% of 9.937 =  
9.937 moles

n(Al<sub>2</sub>O<sub>3</sub>) formed = 9.937

n[Ca(OH)<sub>2</sub>] consumed = 3 × 9.937 =  
29.811

n[CaSO<sub>4</sub>] formed = 29.811

n[H<sub>2</sub>O] produced = 29.81



From literature, conversion = 100%

m[Al<sub>2</sub>(SO<sub>4</sub>)<sub>3</sub> fed] = 3400.46

mr = 342.15

n = 9.937 mole

$$n(\text{phlogopite}) = \frac{m}{Mr} = \frac{500}{831.96} = 0.6 \text{ kmol}$$

$$n(\text{phlogopite converted}) = 0.96 \times 0.6$$

$$= 0.578$$

$$n(\text{phlogopite unconverted}) = 0.0239 \text{ kmol}$$

from the reaction:

$$n(\text{H}_2\text{SO}_4 \text{ consumed}) = \frac{1}{2} \times n(\text{annite}$$

$$\text{consumed} = \frac{1}{2} \times 0.469$$

$$= 0.2345 \text{ kmol}$$

$$n(\text{H}_2\text{SO}_4 \text{ consumed}) = \frac{1}{2} \times 0.578$$

$$= 0.289 \text{ kmol}$$

In the CSTR 1;

**Appendix B (Energy Balance)**

$$\Delta H_T = 469.73 \text{ MJ/h}$$

**Crusher Unit****Outlet**

$$\Delta H (\alpha\text{-spodumene}) = 114 \text{ MJ/h}$$

$$\Delta H (\text{annite}) = 2.93 \text{ MJ/h}$$

$$\Delta H (\text{alumina}) = 101.9 \text{ MJ/h}$$

$$\Delta H (\text{phlogopite}) = 3.6 \text{ MJ/h}$$

$$\Delta H (\text{quartz}) = 26.2 \text{ MJ/h}$$

$$\Delta H_T = 248.63 \text{ MJ/h}$$

**Outlet**

$$\Delta H (\beta\text{-S}) = 23537$$

$$\Delta H (\text{annite}) = 314.68$$

$$\Delta H (\text{phlogopite}) = 386.4$$

$$\Delta H (\text{quartz}) = 2812.03$$

$$\Delta H (\text{alumina}) = 1439.5$$

$$\Delta H (\text{unreacted S}) = 490.66$$

$$\Delta H_T = 28980.9 \text{ MJ/h}$$

**Inlet**

$$T = 25^\circ\text{C}$$

$$\Delta H = 0$$

$$\Delta H = \sum \Delta H_{\text{out}} - \sum \Delta H_{\text{in}}$$

$$= 248.63 \text{ MJ/h}$$

**Exhaust gases**

$$\Delta H (\text{NO}_2) = 55409.3$$

$$\Delta H (\text{CO}_2) = 9255.9$$

$$\Delta H (\text{H}_2\text{O}) = 7307.6$$

**RK-2**

$$\Delta H_T = 71972.79$$

**Inlet**

$$\Delta H (\text{reactants}) = 248.63 \text{ MJ/h}$$

$$\Delta H = \Delta H_{\text{out}} - \Delta H_{\text{in}} + \Delta H_{\text{rxn}}$$

$$\Delta H (\text{air}) = 36.4 \text{ MJ/h}$$

$$\Delta H_{\text{rxn}} = -100483.96 \text{ MJ/h}$$

$$\Delta H (\text{NG}) = 184.7 \text{ MJ/h}$$

$$= -100.4 \text{ GJ/h}$$

	$\Delta H (H_2SO_4) = 0$
<b>Air Cooler 1</b>	$\Delta H (\text{reactants}) = 2022$
$\Delta H_{\text{in}} = 28980.902 \text{ MJ/h}$	$\Delta H (\text{products}) = 9345.43$
$\Delta H_{\text{out}} = 2022$	$\Delta H_{\text{out}} - \Delta H_{\text{in}} + \Delta H_{\text{rxn}} = 0$
$\text{Duty} = 2022 - 28981$	$\Delta H_{\text{rxn}} = -7323.43 \text{ MJ/h}$
$= -26959 \text{ MJ/h}$	
	<b>AIR COOLER- 2</b>
<b>Mass flow rate required for cooling</b>	$\Delta H(\text{Reactants}) = 9345.43 \text{ MJ/h}$
$Q = mC_p\Delta T$	$\Delta H(Li_2SO_4)$
$Q = 26959$	$= \frac{4405.96 \times 190.98 \times (120-25) \times 1000}{109.95}$
$C_p = 29.5$	$= 727 \text{ MJ/h}$
$\Delta T = 1100 - 100$	$\Delta H(H_2O) = \frac{508.41 \times 4.22 \times (120-25) \times 1000}{18}$
$= 1000^\circ C$	$= 11.32 \text{ MJ/ day}$
$m = 0.9138 \times 10^3 \text{ kg/h}$	$\Delta H(FeSO_4) = \frac{213.7 \times 22 \times (120-25) \times 1000}{151.92}$
$m = 913.8 \text{ kg/h}$	$= 2.94 \text{ MJ/ day}$
	$\Delta H(Al_2(SO_4)_3)$
	$= \frac{3400.5 \times 63.5 \times (120-25) \times 1000}{342.17}$
	$= 60 \text{ MJ/ day}$
<b>RK-2</b>	$\Delta H(MgSO_4) = \frac{208.7 \times 26.7 \times (120-25) \times 1000}{120.375}$
Inlet	$= 4.4 \text{ MJ/ day}$

$$\begin{array}{l} \Delta H(\text{Al}_2\text{O}_3 \cdot 4\text{SO}_4) = \frac{14877.123 \times 147.29 \times (120-25) \times 1000}{342.32} \\ = 608.2 \text{ MJ/day} \end{array}$$

$$\begin{array}{l} \Delta H(\text{Unreacted}) = \frac{3135.9 \times 13.826 \times (120-25) \times 1000}{60.08} \\ = 68.6 \text{ MJ/day} \end{array}$$

$$\Delta H(\text{Products}) = 1482.46 \text{ MJ/day}$$

$$\begin{array}{l} \Delta H = \Delta H_{\text{out}} - \Delta H_{\text{in}} \\ = 1482.46 \text{ MJ/h} - 9345.43 \text{ MJ/day} \\ = 7862.97 \text{ MJ/day} \end{array}$$

$$Q = mC \Delta T$$

$$\begin{array}{l} m = \frac{Q}{c \Delta T} \\ = \frac{7862.97}{29.5 \times (250-120)} \\ = 2050 \text{ kg/h} \end{array}$$

$$\begin{array}{l} \Delta H = \Delta H_{\text{out}} - \Delta H_{\text{in}} + \Delta H_{\text{rxn}} = 0 \\ \Delta H_{\text{rxn}} = \Delta H_{\text{in}} - \Delta H_{\text{out}} \\ = 1240.05 - 7711.76 \\ = -6471.71 \text{ MJ/day} \end{array}$$

## MIXER

Inlet

$$\Delta H = 0$$

$$\Delta H(\text{Reactants}) = 1482.46 \text{ MJ/day}$$

Outlet

$$\Delta H(\text{Products}) = 1240.05 \text{ MJ/day}$$

$$\Delta H = \Delta H_{\text{out}} - \Delta H_{\text{in}}$$

$$\Delta H = 1240.05 - 1482.46$$

$$= -242.41$$

## CSTR-1

$$\Delta H(\text{Ca(OH)}_2) = 0$$

$$\Delta H(\text{Reactants}) = 1240.05 \text{ MJ/day}$$

$$\Delta H(\text{Products}) = 7711.76 \text{ MJ/day}$$

$$\Delta H = \Delta H_{\text{out}} - \Delta H_{\text{in}} + \Delta H_{\text{rxn}} = 0$$

$$\Delta H_{\text{rxn}} = \Delta H_{\text{in}} - \Delta H_{\text{out}}$$

$$= 1240.05 - 7711.76$$

$$= -6471.71 \text{ MJ/day}$$

$$Q = m C \Delta T$$

$$m = \frac{Q}{c \Delta T}$$

$$m = \frac{6471.71 \times 1000}{4.18(100)}$$

$$m = 15482 \text{ kg/h}$$

$$\Delta H = \Delta H_{out} - \Delta H_{in} + \Delta H_{reaction} = 0$$

### **CLARIFIER 1**

$$\Delta H = \Delta H(\text{Products}) - \Delta H(\text{Reactants})$$

$$= 0 - 7711.76$$

$$= 7711.76$$

### **CSTR-2**

$$\Delta H(\text{Reactants}) = 0$$

$$\Delta H(\text{CaOH}_2) = 0$$

$$\Delta H(\text{Products}) = 2894.44$$

$$\Delta H_{reaction} = \Delta H_{in} - \Delta H_{out}$$

$$= 0 - 2899.44$$

$$= -2899.44$$

$$\text{Duty } Q = m c p \Delta T$$

$$m = 2899.44 \times 103 / 4.18 \times 100$$

$$m = 6936.45933 \text{ kg/h}$$

### **BELT FILTER 1**

$$\Delta H(\text{Reactants}) = 2899.44$$

$$\Delta H(\text{Products}) = 966.31$$

$$\Delta H = \Delta H_{out} - \Delta H_{in}$$

$$\Delta H = 966.31 - 2899.44$$

$$\Delta H = -1933.13$$

### **CSTR-3**

$$\Delta H(\text{Reactants}) = 966.31$$

$$\Delta H(\text{Ca(OH)}_2) = 0$$

$$\Delta H(\text{Products}) = 2408.88$$

$$\Delta H = \Delta H_{out} - \Delta H_{in} + \Delta H_{rxn} = 0$$

$$\Delta H_{rxn} = \Delta H_{in} - \Delta H_{out}$$

$$\Delta H_{rxn} = 966.31 - 2408.88$$

$$\Delta H_{rxn} = -1442.57$$

### **BELT FILTER 2**

$$\Delta H(\text{Reactants}) = 2408.88$$

$$\Delta H(\text{Products}) = 802.646$$

$$\Delta H = \Delta H_{\text{out}} - \Delta H_{\text{in}}$$

$$\Delta H = 802.646 - 2408.88$$

$$\Delta H = -1606.234 \text{ MJ/day}$$

$$\Delta H = \Delta H_{\text{out}} - \Delta H_{\text{in}}$$

$$\Delta H = 59.61 - 19.9$$

$$= 39.71 \text{ MJ/day}$$

#### CSTR-4

$$\Delta H(\text{Reactants}) = 802.646$$

$$\Delta H(\text{Ca(OH)}_2) = 0$$

$$\Delta H(\text{Products}) = 2158.2$$

$$\Delta H = \Delta H_{\text{out}} - \Delta H_{\text{in}} + \Delta H_{\text{rxn}} = 0$$

$$\Delta H_{\text{rxn}} = \Delta H_{\text{in}} - \Delta H_{\text{out}}$$

$$\Delta H_{\text{rxn}} = 802.646 - 2158.2$$

$$\Delta H_{\text{rxn}} = -1355.554$$

$$Q = m C \Delta T$$

$$m = \frac{Q}{C \Delta T}$$

$$m = \frac{39.71 \times 1000}{29.4(100)}$$

$$m = 13.5 \text{ kg/h}$$

**EVAPORATOR**                                   and  
**CRYSTALIZER**    were    simulated  
using aspen

#### ROTARY DRYER

$$\Delta H(\text{Reactants}) = 19.9 \text{ MJ/day}$$

$$\Delta H(\text{Products}) = 59.61 \text{ MJ/day}$$

Since the computation of energy flow or enthalpy in an evaporator and crystallizer for multiple streams require complex calculations with iterations, simulations using aspen solids was used for a more accurate result for the evaporator and crystallizer units.

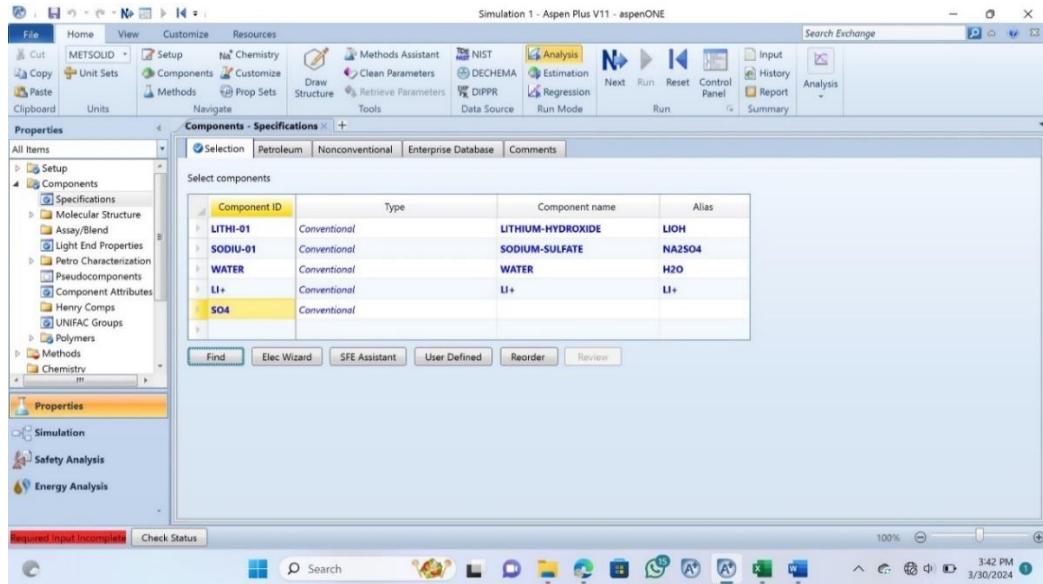
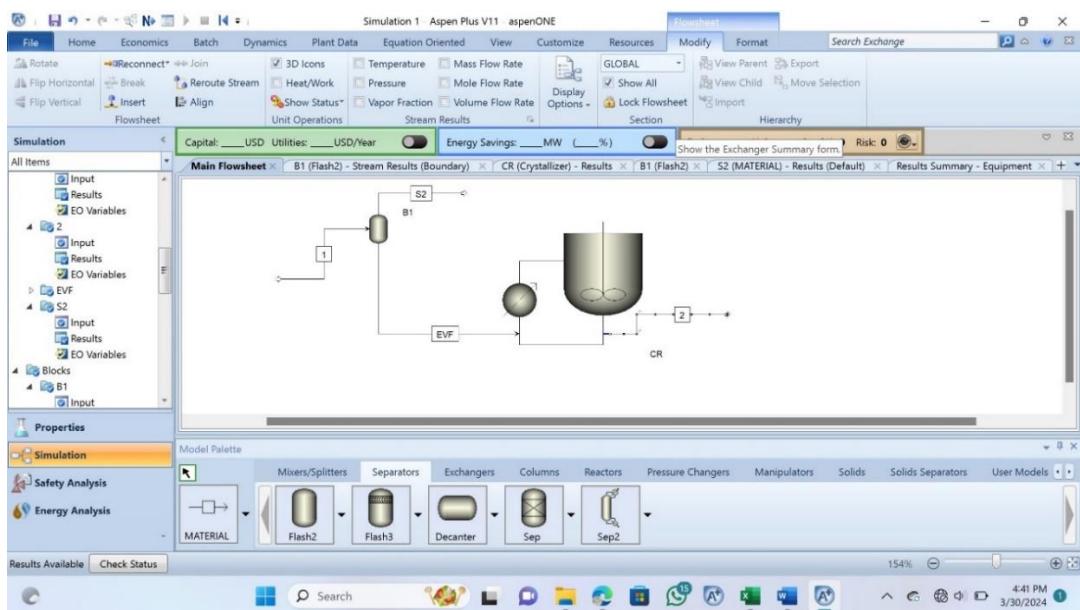


Figure 68: Specification of components for simulation



Simulation of process equipment (evaporator and crystallizer)

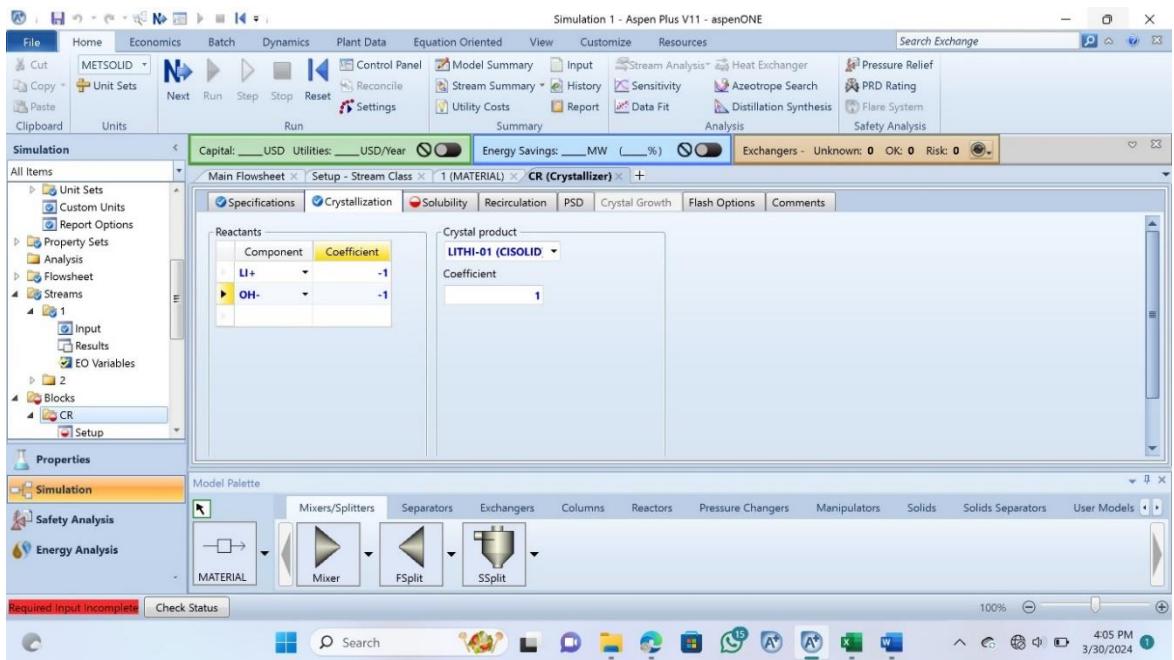
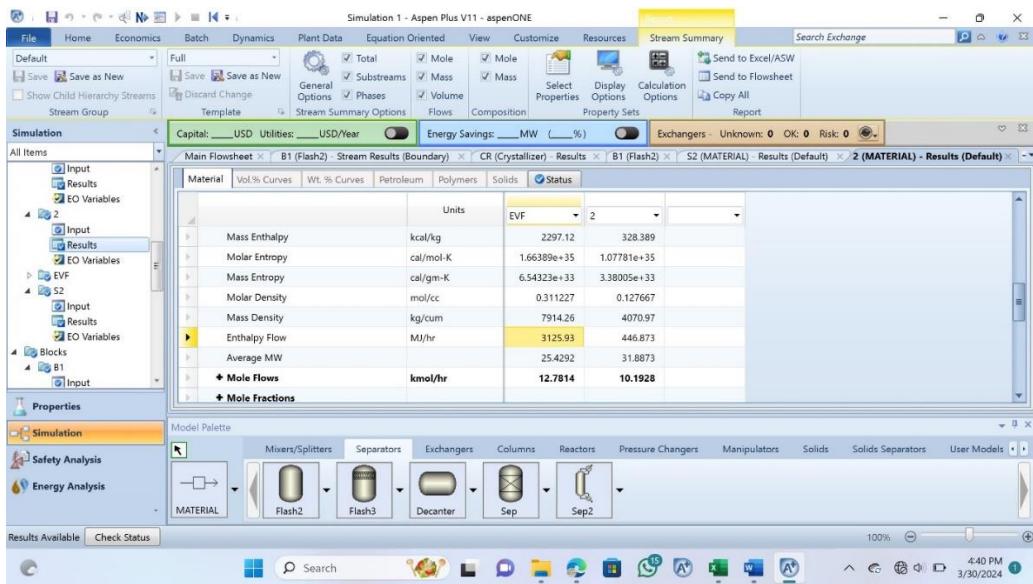


Figure 69: Specification of the ions of component to be crystallized.



Value of Enthalpy flow out of the crystallizer

