Production of Cadmium

A project submitted to the department of chemical engineering of
Engineering College /AL-Qadisiya university

Supervised & overlooked by:

Dr. Ali Hussein Abbar

Prepared by:

Ali Abdul-Jaleel Lateef
Hussein Essam Hussein
الحمد لله الذي أثنا لنا درب العلم والمعرفة وأعاننا على
أداء هذا الواجب ووقتنا إلى انجاز هذا العمل...
نتوجه بجزيل الشكر والامتنان إلى كل من ساعدنا
من قريب أو من بعيد على انجاز هذا العمل ووضعيت
ما واجهناه من صعوبات، وشخص بالذكر الأستاذ
المشرف أ. م. علي حسين عبار الذي لم يبخل علينا
بتوجيهاتته ومصاوره القيمة التي كانت عونا لنا في
إتمام هذا البحث كما نشكره لجهوده البذلة لوضع
برنامج العمل واشرافه على سير التجارب ونصائحه
القيمة لاتخاذ نتائج البحث بالشكل الجيد، سائلين
المولى عز وجل أن يحفظه ويوفقه.
كما نتندب بجزيل الشكر والامتنان للسيد رئيس
قسم الهندسة الكيمياوية أ. م. صالح عبد الجبار صالح
لتشجيعه لنا وتوفر المستلزمات الضرورية لإنجاز
البحث.
ولا ننسى أن نشكر كل أساتذتنا الاعزاء الذين امدونا
المعلومات والنصائح في سبيل إنهاء البحث بأفضل
صورة.
الحمد لله رب العالمين والصلاة والسلام على خاتم الأنبياء والمرسلين وله الطيبين الطاهرين

أهدي هذا العمل : ........

إلى .... من لا يمكن للكلمات أن تؤدي حقهم إلى من ضحوا بانفسهم يه سبيل الوطن إلى شهداء العراق العظيم

إلى .... من ربيتي وأئارت دربي وأعانتني بالصلاة والدعاء، إلى أغلى إنسان يه هذا الوجود أمي الحبيبة

إلى .... من عمل بكد يه سبيلي وعلمني معنى الكفاح وأوصلني إلى ما أنا عليه أبكر الكريم أدمه الله لي

إلى .... من عمل معنا باخلاص بغية إتمام هذا العمل،

إلى استاذنا الفاضل حفظه الله
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Chapter One
INTRODUCTION

Cadmium, Cd, atomic number 48, Ar 112.40, belongs to the zinc subgroup of the periodic table along with mercury. In its compounds it has an oxidation state of +2. There are eight natural isotopes, with mass numbers ranging from 106–116. The most abundant are 114Cd (29 %) and 112Cd (24 %). There are also a number of metastable isotopes.

Strohmeyer discovered cadmium in 1817 in the course of investigating zinc carbonate. He recognized that the yellow color of a sample of zinc oxide produced by roasting was due to the presence of an unknown metal oxide. Because this new element also occurred in the zinc ore calamine, a name derived from the Latin word “cadmia,” he named the new element cadmium.

Unlike some other heavy metals, such as lead or mercury, which have been used since ancient times, cadmium has been refined and utilized only relatively recently. After its discovery more than a century elapsed before the metal or its compounds were employed to any significant extent. Only since the 1940’s production and consumption have risen distinctly. The primary uses are electroplated cadmium coatings, nickel–cadmium storage batteries, pigments, and stabilizers for plastics. Publicity about the toxicity of cadmium has affected the consumption.

1. Occurrence, Raw Materials
Cadmium is widely distributed. It occurs in the earth’s crust with a content estimated to be between 0.08 and 0.5 ppm. In top soil cadmium content usually lies between 0.1 and 1 ppm.

The best known cadmium mineral is greenockite, cadmium sulfide (77.6% Cd). One mineral form of cadmium carbonate (61.5% Cd) was named otavite after its discovery site, the Tsumeb Mine, in the Otavi deposit of South-West Africa. In Sardinia, pure cadmium oxide (87.5 % Cd) was found in a zinc deposit. However, none of these cadmium minerals is of industrial importance because the deposits are too small.

Only zinc minerals in which cadmium is found as an isomorphic component, with concentrations ranging from 0.05 to 0.8 %, averaging about 0.2 %, have economic significance for cadmium recovery [2]. In addition, lead and copper ores contain small amounts of cadmium, which can be separated during the roasting and smelting processes.
Chapter Two
Production Methods

1) Hydrometallurgical process

Cadmium occurs primarily as sulfide minerals in zinc, lead–zinc, and copper–lead–zinc ores. Beneficiation of these minerals, usually by flotation or heavy-media separation, yields concentrates which are then processed for the recovery of the contained metal values. Cadmium follows the zinc with which it is so closely associated. The zinc concentrate is first roasted in a fluid-bed roaster to convert the zinc sulfide to the oxide and a small amount of sulfate. Normally, roasting is carried out with an excess of oxygen below 1000°C so that comparatively little cadmium is eliminated from the calcined material in this operation (3). Since the advent of the Imperial Smelting Zinc Furnace, the preliminary roasting processes for zinc and zinc–lead concentrates result in cadmium recovery as precipitates from solution or as cadmium–lead fume, respectively, as shown in Figure 1.

Fig. 1. Preliminary cadmium roasting processes.

Air pollution problems and labor costs have led to the closing of older pyrometallurgical plants, and to increased electrolytic production. On a worldwide basis, 77% of total zinc production in 1985 was by the electrolytic process (4). In
electrolytic zinc plants, the calcined material is dissolved in aqueous sulfuric acid, usually spent electrolyte from the electrolytic cells. Residual solids are generally separated from the leach solution by decantation and the clarified solution is then treated with zinc dust to remove cadmium and other impurities.

**Cadmium Precipitates.** Processing (5) of the cadmium-bearing precipitate may follow the flow sheet shown in Figure 2. More recently, electrolytic zinc plant practice has developed such that precipitates containing up to 90% cadmium are produced and antimony is commonly used, instead of arsenic, as an additive in solution purification (7–11). The precipitates of Figure 2, containing 4 to 29 times more zinc than cadmium as well as other impurities, notably residual copper, is dissolved at 45–82°C in a mixture of spent electrolyte from the zinc plant, sulfuric acid, and spent cadmium electrolyte. The copper is removed by galvanic precipitation with a small amount of zinc dust. After filtering the copper cake, cadmium is reprecipitated in two stages, usually at pH 5.2 and using 0.6–2 kg zinc dust per kg cadmium, so that the product contains about 80% cadmium and less than 5% zinc. Steam oxidation of this sponge is optional. It is then dissolved at 45–82°C in spent cadmium electrolyte and make-up sulfuric acid to give a solution of about 200 grams Cd per liter, and is mixed with recirculating spent electrolyte to form the cell electrolyte. The electrowinning is carried out at 21–25°C in cells equipped with silver–lead anodes and aluminum cathodes at a current density varying between 26 and 240 A/m², and cell voltages of 2.5–2.8 V. Glue is added at rates of 0–2.5 kilogram per metric ton of cadmium deposited. Cathode deposits are stripped from the aluminum blanks every 6 to 24 hours, depending on the current density. They are washed, dried, and melted at 380–400°C under sodium hydroxide, which not only acts as a flux to prevent oxidation, but also effectively removes any zinc or arsenic that may still be present. Finally, the metal is cast into commercial shapes, ie, slabs, balls, ingots, rods, splatters, and powder.

Precautions have to be taken during the dissolution of cadmium precipitates or the galvanic precipitation of cadmium with zinc to remove possible mist and toxic gases such as arsine. Suitable exhaust hoods and scrubbers must be provided. The fume that may be formed during cathode melting must be removed similarly.
Fig. 2. Electrolytic production of cadmium from zinc electrolyte purification residue (5,6).

2-Pyrometallurgical Process

In the pyrometallurgical process, cadmium is volatilized during the roasting and sintering of zinc concentrates, and the resultant fume and dust are collected as flue dust in baghouses or electrostatic precipitators. The initial cadmium content of the flue dust can be as high as 10%. A great deal of cadmium collects with the zinc metal and may be removed by refining of zinc by fractional distillation (the boiling point of cadmium is 767°C and that of zinc and cadmium metal from zinc concentrates.

During the pyrometallurgical extraction of zinc, calcine from a roaster can be sintered with coke on a sintering machine to give a dense desulfurized product. The sintering operation results in considerable volatilization of cadmium and lead.
compounds, enhanced by the presence of chloride, leading to a 90–99% recovery of cadmium. The fume and dust from the sintering machine are collected in a baghouse (12,13). Cadmium not removed during sintering and subsequent operations follows the zinc metal and often is recovered during zinc metal purification by distillation.

The cadmium content in the feed to lead and copper smelters is lower than that generally encountered in zinc plants, and this necessitates upgrading the initial cadmium level in the fume by one or more refuming steps in a kiln or reverberatory furnace. The final fume may contain as much as 45% cadmium. In general, the composition of these fumes as well as those obtained from zinc sintering vary with respect to cadmium content and impurities. Fumes usually require more processing and purification steps for cadmium recovery than do purification residues from electrolytic zinc plants. Galvanic precipitation is the most frequently adopted method for the final recovery of cadmium in pyrometallurgical plants, but electrowinning may also be used.

The flow sheet in Figure 3 illustrates cadmium recovery from cadmium-bearing fumes. Depending on composition, the fume may have to be roasted with or without sulfuric acid or oxidized using sodium chlorate or chlorine in order to convert cadmium into a water- or acid-soluble form and to eliminate volatile constituents. However the leach solution is obtained, it must generally be purified to remove arsenic, iron, copper, thallium, and lead, using the various treatments shown in Figure 3 for the recovery of cadmium from baghouse fume. The cadmium may also be galvanically precipitated from the leach solution and then re-dissolved (see Alternative 1 in Fig. 3).

In the recovery of cadmium from fumes evolved in the Imperial Smelting process for the treatment of lead–zinc concentrates, cadmium is separated from arsenic using a cation-exchange resin such as Zeocarb 225 or Amberlite 120 (14,15). Cadmium is absorbed on the resin and eluted with a brine solution. The cadmium may then be recovered directly by galvanic precipitation.

Alternative 2 in Figure 3 indicates the most common method for the recovery of cadmium from purified leach solution by galvanic displacement with zinc in the form of dust, sheets, or even rods or rectangular anodes. The final processing depends on the grade of zinc. In most cases, the pH for galvanic precipitation is below 2, although one plant operates at pH 6.2. Temperatures range from ambient to 70°C and precipitation times vary from 30 min to 18 h, depending on temperature and aggregation of the zinc. The weight of zinc required to precipitate
one kilogram of cadmium varies between 0.65–0.95 kg. In most plants, the final cadmium sponge is washed to remove soluble impurities, and then compacted by briquetting. The briquettes may be melted under a flux of sodium hydroxide or ammonium chloride or be distilled for final purification.

In Alternative 3 (Fig. 3), the electrolysis may be operated on a semicontinuous basis with the cadmium eventually being stripped completely from the electrolyte, which is then discarded after suitable treatment. Instead of the usual silver–lead anodes, high silicon–iron anodes, such as Duriron, are commonly used.
3- Recycling

The purpose of the recycling procedures described here is the production of intermediate products – by special procedures – that can then be fed into the normal cadmium production processes. The recovery of cadmium – tin alloys, about 65%Sn and 35%Cd, usually begins by dissolving the scrap in nitric acid. The
tin stays in the residue as stannic acid. An impure carbonate is precipitated from the cadmium nitrate solution with soda.

The residue and the rejected batches from the production of cadmium pigments containing cadmium sulfide and selenide are dissolved in strongly oxidizing acid. The cadmium is recovered by cementation or as the carbonate. A variation of this process is described in [17].

In order to recycle the cadmium from nickel–cadmium batteries and similar production scrap, the parts are dissolved in agitated 2wt % sulfuric acid. A reasonably good separation from the dissolved nickel can be attained by taking advantage of the lower stability of the ammine complexes of cadmium: cadmium can be precipitated from ammonium hydroxide solution as carbonate, whereas nickel remains in solution [18]. Alternatively, nickel can be extracted from the ammonia solution with chelating reagents before cadmium carbonate is precipitated [19]. Sludge from the treatment of cadmium containing wastewater can often be injected directly into the refinery process. Preliminary, conventional wet chemical enrichment is possible but is usually omitted for economic reasons [20]. Several methods for conversion of electroplating wastewater to sludge not requiring special handling can be found in [21].

A plant was constructed by Sab-NiFe in Sweden for recycling used batteries and scrap at the same location where nickel–cadmium batteries are produced. Cadmium-containing waste and filter dust are leached with acid, and the cadmium is deposited by electrolysis. Used batteries are dismounted in a semiautomatic process. From the iron- and cadmium-containing plates, the cadmium is distilled under a reducing atmosphere at 850 °C and returned to the production department. Even sealed cells can be processed by distillation after the plastic is destroyed by pyrolytic pretreatment in a slightly oxidizing atmosphere [22].

The proposed method

(Production of cadmium from waste nickel-cadmium batteries)

The method of recovering cadmium (Cd) and nickel (Ni) from nickel-cadmium batteries, containing the following process steps:

1) Reducing the size of the batteries to small pieces with linear sizes smaller than approximately 15mm
2) Leaching out the fraction at approximately 90° C. with a strong acid, filtering and removing the residue
3) Extracting the Cd from the leaching liquid with the aid of solvent extraction;
4) Stripping the extracting loaded with Cd with a liquid and separating metallic Cd therefrom with the aid of electrolysis
5) Adjusting the acidity of the leaching liquid in such a manner, that iron F $^{+2}$ions are converted into solid Fe $^{+3}$ hydroxide, and filtering this off
6) Recovering Ni: metallic, or as Ni-hydroxide respectively, from the filtrate with the aid of electrolysis or by increasing the pH. Such a, so-called hydrometallurgical, method is known

The Ni-Cd batteries concerned consist of approximately 16-weight % of Cd, of approximately 34 weight% of Ni, of approximately 23 weight % of Fe, and for the rest of plastic and paper.

The size reduction takes place with the aid of a knife mill, equipped with a retaining screen. The contents of the batteries are thus laid bare. When leaching the conditions are chosen in such a manner, that the maximum amount of Cd and Ni goes into solution. The four independent parameters: the temperature, the ratio between the quantities of acid and solid material, the strength of the acid and the residence-time in the leaching barrel are the controlling factors.

The solvent extraction of Cd can be achieved with the aid of various extractants, for example with tributylphosphate or with a tertiary amine. After the Cd has been removed from the leaching liquid, the conversion of Fe $^{+2}$ into Fe $^{+3}$ therein can be achieved with the aid of hypochlorite and at a pH of approximately 4.

The aim of the method is twofold: on the one hand to prevent the dangerous Cd from ending up in the environment, on the other hand to be able to re-use the relatively expensive Cd and Ni. Re-chargeable batteries in particular contain much Ni and Cd. With the aid of this known process approximately 275 kg of metallic Ni and 150 kg of metallic Cd can be recovered from each1000 kg of batteries. An important factor for applying such a process is the economy of the process. Figure ( ) show the block diagram of the proposed method.
Chapter Three
Material Balance

**Plant production capacity:**

Capacity = 10000 ton/year

Operating Time = 300 day/year

Total amount of Nickl-Cadmium and waste battery = \( \frac{10000 \times 1000}{300 \times 24} = 1388.8 \, Kg/hr \)

**Notation:**

The following notation will be adapted in the material balance analysis:

- F=Feed to equipment.
- O.M.B: Overall Material Balance.
- C.M.B: Component Material Balance.
- Temp. : Temperature of the fluid.
- Press. : Pressure of the fluid.
- Vol. : Volume of the fluid.

Atomic weight of the most compound in the system (g/mol)

<table>
<thead>
<tr>
<th>CdO</th>
<th>NiO</th>
<th>Fe</th>
<th>HCl</th>
<th>H2O</th>
<th>CdCl2</th>
<th>NiCl2</th>
<th>FeCl2</th>
</tr>
</thead>
<tbody>
<tr>
<td>128.4</td>
<td>74.69</td>
<td>55.846</td>
<td>36.46</td>
<td>18</td>
<td>183.4</td>
<td>129.69</td>
<td>126.845</td>
</tr>
<tr>
<td>Cd</td>
<td>Ni</td>
<td>NaCl</td>
<td>NaOH</td>
<td>H2</td>
<td>Cl2</td>
<td>Fe(OH)2</td>
<td></td>
</tr>
<tr>
<td>112.4</td>
<td>57.9</td>
<td>58.433</td>
<td>39.997</td>
<td>2</td>
<td>30.0</td>
<td>89.846</td>
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</table>
Crusher (CR-101)

Material Balance on Crusher:

Amount of Feed >10 mm

\[
\frac{1388.8}{0.55} = 2525.09 \frac{kg}{r} = f2
\]

The amount of f4 = The amount of f1
<table>
<thead>
<tr>
<th>Material</th>
<th>F1</th>
<th>F2</th>
<th>F3</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Kg/hr</td>
<td>Wt%</td>
<td>Kg/hr</td>
</tr>
<tr>
<td>CdO</td>
<td>219.443</td>
<td>10.8</td>
<td>398.964</td>
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<tr>
<td>NiO</td>
<td>470.83</td>
<td>33.9</td>
<td>802.000</td>
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<tr>
<td>Fe</td>
<td>312.498</td>
<td>22.0</td>
<td>588.140</td>
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<tr>
<td>Impurities</td>
<td>386.249</td>
<td>27.8</td>
<td>701.976</td>
</tr>
<tr>
<td>Total</td>
<td>1388.8</td>
<td>100</td>
<td>2025.9</td>
</tr>
</tbody>
</table>

**Screen (SC-101)**

Material Balance on Screen:

F2 = F3 + F4
\[ 2525.09 \quad 1388.8 = 1136.29 \frac{kg}{r} = (f3) \]

**M.B. on F2:**

\[ Cd(Cdo) = 0.158 \quad 2525.09 = 398.694 \frac{kg}{r} \]

\[ Ni(Nio) = 0.339 \quad 2525.09 = 856.005 \frac{kg}{r} \]

\[ Fe(Fe) = 0.225 \quad 2525.09 = 568.145 \frac{kg}{r} \]

\[ Inert Material = 0.278 \quad 2525.09 = 701.976 \frac{kg}{r} \]

**M.B. on F3:**

\[ Cd(Cdo) = 0.158 \quad 1136.29 = 179.533 \frac{kg}{r} \]

\[ Ni(Nio) = 0.339 \quad 1136.29 = 385.202 \frac{kg}{r} \]

\[ Fe(Fe) = 0.225 \quad 1136.29 = 255.665 \frac{kg}{r} \]

\[ Inert Material = 0.278 \quad 1136.29 = 315.890 \frac{kg}{r} \]

**M.B. on F4:**

\[ Cd(Cdo) = 0.158 \quad 1388.8 = 219.443 \frac{kg}{r} \]

\[ Ni(Nio) = 0.339 \quad 1388.8 = 470.83 \frac{kg}{r} \]

\[ Fe(Fe) = 0.225 \quad 1388.8 = 312.498 \frac{kg}{r} \]

\[ Inert Material = 0.278 \quad 1388.8 = 386.029 \frac{kg}{r} \]
<table>
<thead>
<tr>
<th>Material</th>
<th>F2</th>
<th>F3</th>
<th>F4</th>
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<td></td>
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<td>380.23</td>
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<td>Fe</td>
<td>868.15</td>
<td>22.0</td>
<td>565.60</td>
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<td>Inert Material</td>
<td>701.97</td>
<td>27.8</td>
<td>340.89</td>
</tr>
<tr>
<td>Total</td>
<td>2025.0</td>
<td>100.0</td>
<td>1136.29</td>
</tr>
</tbody>
</table>
Leaching (LE-101)

\[ CdO + 2HCL \rightarrow CdCl2 + H2O \] \hspace{1cm} (1)
\[ NiO + 2H2O \rightarrow Nicl2 + H2O \] \hspace{1cm} (2)
\[ Fe + 2Hcl \rightarrow Fecl2 + H2 \] \hspace{1cm} (3)

\[ Moles of CdO = \frac{219.443}{128.4} = 1.709 \text{ kmol} \]
Conversion  =  90%

CdO reacted

\[ 1.709 \times 0.9 = 1.5381 \frac{kmol}{hr} = Cdcl2 = H2O \ldots \text{In reaction(1)} \]

NiO reacted

\[ NiO = 6.303 \times 0.9 = 5.6727 \frac{kmol}{hr} = Nicl2 = H2O \ldots \text{In reaction(2)} \]

Fe reacted

\[ Fe = 5.595 \times 0.9 = 5.0355 \frac{kmol}{hr} = Fecl2 = H2O \ldots \text{In reaction(3)} \]

\[ \text{Moles of CdO out} = 1.709 \times 1.5381 = 0.1709 \frac{kmol}{r} \]

\[ \text{Weight of CdO out} = 0.1709 \times 128.4 = 21.9143 \frac{kg}{r} \]

\[ \text{Moles of NiO out} = 6.303 \times 5.6727 = 0.6303 \frac{kmol}{r} \]

\[ \text{Weight of NiO out} = 0.6303 \times 74.69 = 47.08 \frac{kg}{r} \]

\[ \text{Moles of Fe out} = 5.595 \times 5.0355 = 0.5595 \frac{kmol}{r} \]

\[ \text{Weight of Fe out} = 0.5595 \times 55.845 = 31.245 \frac{kg}{r} \]

\[ \text{Weight of Cdcl2 out} = 1.5381 \times 183.4 = 282.087 \frac{kg}{r} \]
Weigt of Nicl2 out = 5.6727 kg
129.69 = 735.692 kg

Weigt of Fecl2 out = 5.0355 kg
126.845 = 638.72 kg

Hcl concentration=

\[ 6N = 6M = 6 \frac{mol \ HCl}{l \ sol} \]

Volume of the solution=13888.8 L

Moles of HCl=6*13888.8 =83332.8 mol

Weight of HCl=83332.8*36.5 =3041647.2 g
=3041.647 Kg

Basis= 1 L of solution

HCl=6 mol*36.5 =219 g HCl

HCl=\frac{219}{1098} = 0.1999 = 19.99%

\rho =1.089 Kg/L

\[ N = \frac{CONV}{e} = 6 \]

\[ \frac{Solid \ Hcl}{1 \ \ 10} \]

\[ 1388.8 \ \ X \]

\[ Hcl \ Sol. = 13888 \ \frac{l}{hr} \ \ Solution = f5 \]

\[ f5 \ weig = 13888 \ \frac{l}{hr} \ \ 1.098 \ \frac{Kg}{l} = 15249.024 \ \frac{kg}{r} \]
Weigt of HCl in \(3041.647 \frac{kg}{r}\)

\[\text{Weigt of H}_2\text{O in } = 15249.024 \times 3041.647\]

\[= 12207.377 \frac{kg}{r}\]

Moles of H\(_2\)O produced = \(1.5381 + 5.6727 = 7.2105 \frac{kmol}{r}\)

Mass of H\(_2\)O produced = \(7.2105 \times 18 = 129.789 \frac{kg}{r}\)

Total H\(_2\)O in F6 = \(12207.377 + 129.789\)

\[= 12337.166 \frac{kg}{r}\]

HCl Reacted

From reaction (1) = \(1.5381 \times 2 = 3.0762 \frac{kmol}{r}\)

From reaction (2) = \(5.6727 \times 2 = 11.3454 \frac{kmol}{r}\)

From reaction (3) = \(5.0355 \times 2 = 10.071 \frac{kmol}{r}\)

\[3.0762 \frac{kmol}{r} + 11.3454 \frac{kmol}{r} + 10.071 \frac{kmol}{r} = 24.4926 \frac{kmol}{r}\]

\[24.4926 \times 36.5 = 893.9799 \frac{kg}{r}\]

Total HCl out in f6

\[3041.647 \times 893.9799 = 2147.6671 \frac{kg}{r}\]
\[ Total \ f_6 = 16241.5971 + 386.029 = 16627.626 \ \text{kg} \]

Total weight of H2 out in F7

\[ 5.595 \times 2 = 11.19 \ \text{kg} \]

<table>
<thead>
<tr>
<th>Material</th>
<th>F4 Kg/hr</th>
<th>F4 Wt%</th>
<th>F5 Kg/hr</th>
<th>F5 Wt%</th>
<th>F6 Kg/hr</th>
<th>F6 Wt%</th>
<th>F7 Kg/hr</th>
<th>F7 Wt%</th>
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<tbody>
<tr>
<td>CdO</td>
<td>229.44</td>
<td>75.5</td>
<td>---</td>
<td>---</td>
<td>21.943</td>
<td>1.13</td>
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<tr>
<td>NiO</td>
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<td>33.9</td>
<td>---</td>
<td>---</td>
<td>22.77</td>
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<td>Fe</td>
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<tr>
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<td>---</td>
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<td>4.03</td>
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<tr>
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<td>---</td>
<td>638.77</td>
<td>3.93</td>
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<td>H2</td>
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<td>---</td>
<td>11.19</td>
<td>100</td>
<td>---</td>
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<tr>
<td>HCl</td>
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<td>---</td>
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<td>19.99</td>
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<tr>
<td>H2O</td>
<td>---</td>
<td>---</td>
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<td>50.1</td>
<td>75.96</td>
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<td>Total</td>
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<td>16627.626</td>
<td>100</td>
<td>11.19</td>
<td>100</td>
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Heat Exchanger(HE-101)

M.B. on Heat exchanger
Steady state
In=Out
F6=F6`

CdO 0.13
NiO 0.25
Fe 0.19
Inert Material 0.02
CdCl₂ 1.73
NiCl₂ 4.53
FeCl₂ 3.93
HCl 13.22
H₂O 75.96
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<td>٣٤٩.١٢</td>
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<td>٢٧.٠</td>
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<tr>
<td>Fe</td>
<td>٣٨.٢٠</td>
<td>٣٦٨.٠٢</td>
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<td>٠.٠٢</td>
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<td>٢٨٢.٠٨</td>
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<td>٣٧٣.٧٢</td>
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<td>٣٨٨.٧٢</td>
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<td>١٣٧.٢٢</td>
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<td>٧٠.٩٦</td>
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<tr>
<td>Total</td>
<td>١٦٦٢٧.٦٢</td>
<td>١٠٠</td>
</tr>
</tbody>
</table>
Filteration (F-101)

\[ f_9 = Cdo_{out} + NiO_{out} + Fe_{out} + inert \]

\[ = 491.569 \frac{kg}{r} \]

Assum 5% for \( H2O \)
\[ f_8 = \frac{100.265}{0.95} = 105.54 \frac{kg}{r} \]

Weight of H2O in \( f_8 = 5.277 \frac{kg}{r} \)

\[ f_8 = 16638.816 \quad 491.569 \]

\[ = 16136.05 \frac{kg}{r} \]

Weight of CdCl2 in \( f_8 = 16136.05 \quad 0.0174 = 282.087 \frac{kg}{r} \)

Weight of NiCl2 in \( f_8 = 16136.05 \quad 0.0456 = 735.692 \frac{kg}{r} \)

Weight of FeCl2 in \( f_8 = 16136.05 \quad 0.0395 = 635.72 \frac{kg}{r} \)

Weight of HCl in \( f_8 = 16136.05 \quad 0.133 = 2147.667 \frac{kg}{r} \)

Weight of H2O in \( f_8 = 12337.166 \quad 5.277 \)

\[ = 12331.889 \frac{kg}{r} \]
<table>
<thead>
<tr>
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<th>F8 Kg/hr</th>
<th>Wt%</th>
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<td>Total</td>
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<td>16636.805</td>
<td>100</td>
<td>491.569</td>
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</table>
Extraction (EX-101)

TBP = F8 = 16136.05 \frac{kg}{hr}

Amount of CdCl2 in TBP =

\[ Amount \ of \ CdCl2 \ in \ TBP = 282.087 \times 0.99 = 279.266 \frac{kg}{r} \]

\[ F10 = F8 - CdCl2 \ in \ TBP = 16136.05 - 279.266 \]
\[ F_{11} = f_8 \quad \text{CdCl}_2 \text{ in TBP} \]

\[ = 15856.784 \frac{kg}{hr} \]

\[ \text{Nicl}_2 = 735.692 \frac{k}{r} \]

\[ \text{FeCl}_2 = 638.72 \frac{k}{r} \]

\[ \text{cdcl}_2 = 282.087 \quad 279.266 \]

\[ = 2.821 \frac{kg}{r} \]

\[ \text{H}_2\text{O} = 12331.889 \frac{kg}{r} \]

\[ \text{HCl} = 2147.6671 \frac{kg}{r} \]
<table>
<thead>
<tr>
<th>Material</th>
<th>F7 Kg/hr</th>
<th>Wt%</th>
<th>F9 Kg/hr</th>
<th>Wt%</th>
<th>F10 Kg/hr</th>
<th>Wt%</th>
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<tr>
<td>NiO</td>
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<td>----</td>
<td>----</td>
<td>----</td>
<td>----</td>
<td>----</td>
</tr>
<tr>
<td>Fe</td>
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<td>----</td>
<td>----</td>
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<td>16433.316</td>
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<td>10862.784</td>
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</table>
Neutralization (N-101)

\[ \text{NaOH} + \text{HCl} \rightarrow \text{NaCl} + \text{H}_2\text{O} \]

Moles of HCl in F11 = \( \frac{2147.6671}{36.5} \)

= 58.8401 \( \frac{\text{kmol}}{r} \)

Total NaOH in F20 = 2353.604 + (402.3432)

= 2755.9472 \( \frac{\text{kg}}{r} = f20 \)

M. B. on F21
Weight of NiCl₂ = 735.692 \frac{kg}{r}

Weight of CdCl₂ = 2.821 \frac{kg}{r}

Weight of NaCl = 58.8401 \text{ kmol} \quad 58

= 3412.7258 \frac{kg}{r}

H₂O = 58.840 \quad 18 = 1059.1218 \frac{kg}{r}

Total H₂O in F21 = 1059.1218 + 12331.889

= 13391.0108 \frac{kg}{r}

FeCl₂ + 2NaOH → Fe(OH)₂ + 2NaCl

Moles of FeCl₂ = \frac{638.72}{127}

= 5.62929 \frac{kmol}{r}

NaCl reacted = 5.02929 \quad 2

= 10.058 \frac{kmol}{r} \quad 58

= 583.39 \frac{kg}{r}

NaCl Total = 583.39 + 3412.7258

= 3996.1235 \frac{kg}{r}
\[ NaOH = 2 \quad 5.02929 = 10.058 \quad 40 \]
\[ = 402.3432 \frac{kg}{r} \]
\[ Fe(OH)_2 = 5.02929 \quad 90 \]
\[ = 452.6361 \frac{kg}{r} \]

Total F21 = NiCl2 + CdCl2 + H2O + Fe(OH)2 + NaCl
\[ = 735.692 + 2.821 + 13391.0108 + 3996.1235 + 452.6361 \]
\[ = 18578.2834 \frac{kg}{r} \]
<table>
<thead>
<tr>
<th>Material</th>
<th>F11 Kg/hr</th>
<th>Wt%</th>
<th>F20 Kg/hr</th>
<th>Wt%</th>
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<td>٠.٠٤</td>
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<tr>
<td>HCl</td>
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<td>١٣.٣</td>
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<td>Fe(OH)₂</td>
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<td>----</td>
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<td>----</td>
<td>----</td>
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<tr>
<td>Total</td>
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<td>٢٧٥٥.٩٤٧٢</td>
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<td>٢٧٥٥.٩٤٧٢</td>
<td>١٠٠</td>
</tr>
</tbody>
</table>
Filteration (F-102)

\[\text{CdCl}_2 \ 0.04\]
\[\text{NiCl}_2 \ 4.96\]
\[\text{H}_2\text{O} \ 72.07\]
\[\text{Fe(OH)}_2 \ 3.43\]
\[\text{NaCl} \ 18.66\]

\[\text{CdCl}_2 \ 0.015\]
\[\text{NiCl}_2 \ 4.07\]
\[\text{H}_2\text{O} \ 73.8\]
\[\text{NaCl} \ 22.1\]

\[\text{Fe(OH)}_2 \ 3.43\]
\[\text{H}_2\text{O} \ 90\]

\[\text{M.B. on Filter}\]

\[\text{Weight of Fe(OH)}_2 \text{ in F23} = \frac{452.6361 \ kg}{0.9} \frac{kg}{r} = 502.929 \frac{kg}{r}\]

\[\text{Weight of H}_2\text{O in F23} = 502.929 \times 452.6361 \times \frac{kg}{r} = 50.2929 \frac{kg}{r}\]
\[ F22 = F21 \quad F23 \]
\[ = 18578.2834 \quad 502.929 \]
\[ F22 = 18075.3544 \frac{kg}{r} \]
\[ NiCl2 = 735.692 \frac{kg}{r} \]
\[ CdCl2 = 2.821 \frac{kg}{r} \]

H2O M.B. in Filter

Weight of H2O in F22 = 13391.0108 \quad 50.2929

\[ = 13340.7179 \frac{kg}{r} \]
\[ NaCl = 3996.1235 \frac{kg}{r} \]

<table>
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<tr>
<th>Material</th>
<th>F21 Kg/hr</th>
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<th>F22 Kg/hr</th>
<th>Wt%</th>
<th>F23 Kg/hr</th>
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<td>100</td>
<td>502.929</td>
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Electrolysis (EC-102)

\[ \text{CdCl}_2 \ 0.015 \]
\[ \text{NiCl}_2 \ 4.07 \]
\[ \text{H}_2\text{O} \ 73.8 \]
\[ \text{NaCl} \ 22.1 \]

Cl2 100

F22

F25

EC-102

F27

H2O 5

Ni 95

F26

H2 100

F22

CdCl2 0.01

H2O 76.43

NaCl 23.5

\[ \text{NiCl}_2 \rightarrow \text{Ni} + 2\text{Cl} \]

\[ \text{Ni} + 2e \rightarrow \text{Ni} \]
Moles of NiCl₂ = 5.6726

Weight of Ni = 5.6726 * 58.69

\[ Ni = 332.93 \frac{Kg}{r} \]

M.B. on Electrolysis 2

\[ F22 = F24 + F25 + F26 + F27 \]
\[ F23 = 350.4526 \frac{Kg}{hr} \]

H₂O in f22 = 350.4526 - 332.93
= 17.5226 Kg/hr

Composition of F24

\[ CdCl₂ = 2.821 \frac{kg}{r} \]
\[ H₂O = 13340.7179 \frac{kg}{r} \]
\[ NaCl = 3996.1235 \frac{kg}{r} \]

So

\[ F24 = 2.821 + 13340.7179 + 3996.1235 = 17229.6624 \frac{kg}{r} \]

NiCl₂ → Ni + 2Cl
\[ 2Cl → Cl₂ \]
\[ Cl₂ = 5.6726 \quad 71 \]

So
\[ F25 = 402.7546 \frac{kg}{r} \]

\[ H2O \rightarrow H + OH \]
\[ 2H \rightarrow H2 \]

\[ Moles \ of \ Ni = \frac{I}{2} \times \frac{0.5}{26.8} \]

So

\[ I = 608 \ A \]

\[ Moles \ of \ H2 = \frac{608}{2} \times 26.8 \]

=11.34 kmol/hr

So

\[ H2 = 11.34 \times 2 \]
\[ = 22.68 \frac{kg}{r} = F26 \]

M.B. on H2O

\[ H2O = 408.42 \frac{kg}{r} \]

H2O in F24 = H2O in F22 - H2O in F26 (Reacted)
\[ = 13340.7179 - 408.42 \]
\[ = 12932.2979 \frac{kg}{hr} \]

\[ F24 = CdCl2 + NaCl + H2O \]
\[ \text{Material} \quad \begin{array}{c|c|c|c|c|c|c|c|c|c} 
\text{Material} & \text{F22} & \text{F24} & \text{F25} & \text{F26} & \text{F27} \\
& \text{Kg/hr} & \text{Wt%} & \text{Kg/hr} & \text{Wt%} & \text{Kg/hr} & \text{Wt%} & \text{Kg/hr} & \text{Wt%} & \text{Kg/hr} & \text{Wt%} \\
\hline 
\text{CdCl2} & 2.821 & 0.016 & 2.821 & 0.16 & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} \\
\text{NiCl2} & 735.692 & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} \\
\text{H2O} & 13340.7179 & 73.8 & 12932.2979 & 76.43 & \text{----} & \text{----} & \text{----} & \text{----} & \text{17.5226} & \text{----} \\
\text{Fe(OH)2} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} \\
\text{NaCl} & 3996.1235 & 22.1 & 3996.1235 & 23.6 & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} \\
\text{Cl2} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} \\
\text{H2} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & 22.96 & 100 \\
\text{Ni} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{----} & \text{322.93} & \text{90} \\
\hline 
\text{Total} & 18075.3544 & 100 & 16931.2424 & 100 & \text{----} & \text{----} & \text{----} & \text{----} & 22.96 & 100 \\
\end{array} \]
Dryer (DR-102)

\[ F29 = Ni = 332.93 \frac{Kg}{r} \]

\[ F28 = H2o = 17.5226 \frac{Kg}{r} \]
Stripping (ST-101)

\[ F_{12} = 16415.316 \frac{\text{kg}}{\text{hr}} \]

M.B. on CdCl2
Weight of CdCl\(_2\) in F10 = \(279.266 \frac{\text{Kg}}{\text{hr}}\)

= 279.266 087

= 242.96142 \(\frac{\text{Kg}}{\text{hr}}\)

F12 = Amount of H\(_2\)O = \(16415.316 \frac{\text{Kg}}{\text{hr}}\)

:. F13 = 242.96142 + 16415.316

= 16658.27742 \(\frac{\text{Kg}}{\text{hr}}\) (CdCl\(_2\) + H\(_2\)O)

<table>
<thead>
<tr>
<th>Material</th>
<th>F30</th>
<th>F10</th>
<th>F12</th>
<th>F13</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Kg/hr</td>
<td>Wt%</td>
<td>Kg/hr</td>
<td>Wt%</td>
</tr>
<tr>
<td>CdCl(_2)</td>
<td>54.31</td>
<td>0.44</td>
<td>1.71</td>
<td>----</td>
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<tr>
<td>TBP</td>
<td>16136.05</td>
<td>49.34</td>
<td>16136.05</td>
<td>0.18</td>
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<tr>
<td>H(_2)O</td>
<td>----</td>
<td>----</td>
<td>----</td>
<td>----</td>
</tr>
<tr>
<td>Total</td>
<td>1318.336</td>
<td>100</td>
<td>16415.316</td>
<td>100</td>
</tr>
</tbody>
</table>
Electrolysis (EC-101)

EC-101

Cl₂ 100

CdCl₂ 1.45
H₂O 98.55

H₂O 99.995
CdCl₂ 0.005

H₂ 100

CdCl₂ 95
H₂O 5
CdCl₂ → Cd²⁺ + 2Cl⁻¹

Cd + 2e → Cd

2Cl⁻ → cl²⁻ + 2e

CdCl₂ = \frac{242.96142}{183.4} = 1.3247 \text{ Kmol/hr} = \text{Cd}

\text{Amount of F16} = \text{Weight of Cl2} = 1.3247 \text{ Kmol/hr} × 71 = 94.0537 \text{ kg/hr}

\text{H}_2\text{O} → \text{H}^+ + 0\text{H}^-

2\text{H}^+ + 2e → \text{H}2

\text{Cd} = 1.3247 \text{ Kmol/hr}

n = \frac{I}{2 \times 26.8} × 0.5

1.32 = \frac{I}{2 \times 26.8} × 0.5

I1 = 141.504 \text{ A}

I2 = 141.50 \text{ A}

n = \frac{141.50}{2 \times 26.8}

n = 2.639 \text{ kmol/hr} \text{H2}

\text{Amount of F17} = 5.279 \text{ kg/hr}

\text{Moles of Cd produced} = 1.3247 \times 0.997 = 1.32 \text{ Kmol/hr}
Amount of Cd in F14 = 112.4 \times 1.32 = 148.368 \frac{Kg}{hr}

Total amount of F14 = \frac{148.368}{0.95} = 156.17 \frac{kg}{hr} (Cd + H2O)

Amount of H2O in F14 = 156.17 \times 148.368 = 7.808 \frac{kg}{hr}

Moles of CdCl2 = 1.32 \times 10^{-3} \frac{kmol}{hr}

= 3.974 \times 10^{-3} \frac{kmol}{hr}

= 3.9741 \times 10^{-3} = 0.7288 \frac{kg}{hr}

M.B. on H2O

H2O in F15 = 16415.316 \times 7.808 = 16407.508 \frac{kg}{hr}

\therefore F15 = 0.7288 + 16407.508

= 16408.2368 \frac{kg}{hr}

H2O in F15 = 16407.508 \times 95.03

= 16312.478 \frac{kg}{hr}

F15 = 16313.2068 kg/hr
<table>
<thead>
<tr>
<th>Material</th>
<th>F13 Kg/hr</th>
<th>F13 Wt%</th>
<th>F15 Kg/hr</th>
<th>F15 Wt%</th>
<th>F16 Kg/hr</th>
<th>F16 Wt%</th>
<th>F17 Kg/hr</th>
<th>F17 Wt%</th>
<th>F14 Kg/hr</th>
<th>F14 Wt%</th>
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<tbody>
<tr>
<td>CdCl₂</td>
<td>242.96142</td>
<td>٦.٨٨٨</td>
<td>٦.٨٨٨</td>
<td>٦.٨٨٨</td>
<td>٦.٨٨٨</td>
<td>٦.٨٨٨</td>
<td>٦.٨٨٨</td>
<td>٦.٨٨٨</td>
<td>٦.٨٨٨</td>
<td>٦.٨٨٨</td>
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<tr>
<td>H₂O</td>
<td>16415.316</td>
<td>٩٨.٤٥</td>
<td>٩٨.٤٥</td>
<td>٩٨.٤٥</td>
<td>٩٨.٤٥</td>
<td>٩٨.٤٥</td>
<td>٩٨.٤٥</td>
<td>٩٨.٤٥</td>
<td>٩٨.٤٥</td>
<td>٩٨.٤٥</td>
</tr>
<tr>
<td>Cl₂</td>
<td>----</td>
<td>----</td>
<td>----</td>
<td>----</td>
<td>٩٤.٠٥٣</td>
<td>٩٤.٠٥٣</td>
<td>----</td>
<td>----</td>
<td>----</td>
<td>----</td>
</tr>
<tr>
<td>H₂</td>
<td>----</td>
<td>----</td>
<td>----</td>
<td>----</td>
<td>----</td>
<td>----</td>
<td>٥.٢٧٩</td>
<td>٥.٢٧٩</td>
<td>----</td>
<td>----</td>
</tr>
<tr>
<td>Total</td>
<td>١٦٣١٣.٢٠٦</td>
<td>١٠٠</td>
<td>١٦٣١٣.٢٠٦</td>
<td>١٠٠</td>
<td>١٦٣١٣.٢٠٦</td>
<td>١٠٠</td>
<td>١٦٣١٣.٢٠٦</td>
<td>١٠٠</td>
<td>١٦٣١٣.٢٠٦</td>
<td>١٠٠</td>
</tr>
</tbody>
</table>
Dryer (DR-101)

Amount of F19 = Amount of Cd = \(148.368 \, \text{Kg/hr}\)

Amount of F18 = Amount of H2O = \(7.808 \, \text{kg/hr}\)

F19 = 148.368 \(300 \times 24 = 1068249.6 \, \text{kg/hr}\)

= 1068.2496 \, \text{Ton/hr}\
<table>
<thead>
<tr>
<th>Material</th>
<th>F14</th>
<th>F18</th>
<th>F19</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Kg/hr</td>
<td>Wt%</td>
<td>Kg/hr</td>
</tr>
<tr>
<td>CdCl2</td>
<td>----</td>
<td>----</td>
<td>----</td>
</tr>
<tr>
<td>H2O</td>
<td>٧.٨٠٨</td>
<td>٥</td>
<td>٧.٨٠٨</td>
</tr>
<tr>
<td>Cd</td>
<td>٨٨٥٨</td>
<td>----</td>
<td>----</td>
</tr>
<tr>
<td>Total</td>
<td>١٠٠١</td>
<td>٧.٨٠٨</td>
<td>١٠٠</td>
</tr>
</tbody>
</table>
Chapter Four
Energy Balance

The calculations will be based on the first law of thermodynamic
(The total quantity of emerge is constant. When emerge disappear in one
form, it appears other forms).

\[ [H+(\frac{1}{2}U^2)+Zg]m=Q-W_s \] (for open system)

Where:
Q=heat gained by system (positive)
W_s =work done by system

Assumptions:
1-Negleet kinetic and potential energy.
   \[ H=Q-W_s \]
2-For open system shaft work\( (W_s) = 0 \)
3-For open system with physical operation
   \[ H=Q \]
4-For open system with chemical reaction
   \[ H+H_{reaction}=Q \]
   \[ H=H_{out} - H_{in} \]
   \[ H=m* C_{p,mean} * T \]
   \[ C_{p,mean} = \sum (X_i*Cp) \] or \[ C_{p,mean} = \sum (y_i*Cp_i) \]
   \[ T=T-T_{reference}, \]
   \[ T_{reference} = 298K \]

5-For physical mixing process we assume ideal solution therefore
\[ H_{\text{mixing}} = 0 \]

4-1 Energy Balance on Crusher:
Because of the temperature of input and output of the Crusher = 25 °C
So Q (for Crusher) = 0

4-2 Energy Balance on Screen:
Because of the temperature of input and output of the Screen = 25 °C
So Q (for Screen) = 0

4-3 Energy Balance on Leaching:
\[ Q = H - H_r \]
\[ H = H_{\text{out}} - H_{\text{in}} \]
\[ = (F_6 + F_7) - (F_4 + F_5) \]
\[ F_7 = m \cdot C_p \cdot T \]
\[ = 11.19 \cdot 14.44 \cdot (363 - 298) \]
\[ = 10502.934 \text{ Kj/hr} \]
\[ H_{\text{cdo}} = m \cdot C_p \cdot T \]
\[ = 21.943 \cdot 27.43 \cdot (363 - 298) \]
\[ = 39123.27 \text{ Kj/hr} \]
\[ H_{\text{nio}} = 47.077 \cdot 49.88 \cdot (363 - 298) \]
\[ = 152636.098 \text{ Kj/hr} \]
\[ H_{\text{fe}} = 31.245 \cdot 0.45 \cdot (363 - 298) \]
\[ = 913.91 \text{ Kj/hr} \]
\[ H_{\text{CdCl}_2} = 282.087 \times 74.468 \times (363 - 298) \]
\[ = 1365429.553 \text{ Kj/hr} \]
\[ H_{\text{NiCl}_2} = 735.692 \times -30.2 \times (363 - 298) \]
\[ = -1444163.396 \text{ Kj/hr} \]
\[ H_{\text{FeCl}_2} = 638.72 \times 75.791 \times (363 - 298) \]
\[ = 3146599.789 \text{ Kj/hr} \]
\[ F_4 = \]
\[ H_{\text{CdO}} = 219.443 \times 27.43 \]
\[ = 391255.8969 \text{ Kj/hr} \]
\[ H_{\text{NiO}} = 1526525 \text{ Kj/hr} \]
\[ H_{\text{Fe}} = 312.498 \times 0.85 \times (363 - 298) \]
\[ = 17265.5145 \text{ Kj/hr} \]
\[ H_{\text{HCl}} = 2147.6671 \times 143 \times (363 - 298) \]
\[ = 19969028.26 \text{ Kj/hr} \]

\[ \text{CdO} + 2\text{HCl} \rightarrow \text{CdCl}_2 + \text{H}_2\text{O} \]
\[ H_r = H_{\text{out}} - H_{\text{in}} \]
\[ = (n \times \text{HFH}_2\text{O} + n \times \text{HCl}_2) - (n \times \text{HCl} + n \times \text{CdO}) \]
\[ = (1 \times -241.8 + 1 \times -391.50) \times -2 \times -92.3 + 1 \times -258.4) \]
\[ = -190.3 \text{ Kj/hr} \]

\[ \text{NiO} + 2\text{HCl} \rightarrow \text{NiCl}_2 + \text{H}_2\text{O} \]
\[ H_r = (1 \times -241.8 + 1 \times -305.33) - (2 \times -92.3 + 1 \times -244.3) \]
\[ = -118.23 \text{ Kj/hr} \]
Fe + 2HCl → FeCl₂ + H₂

\[ H_r = (1\times-341.8+0)-(2\times-92.3+0) \]
\[ = -157.2 \text{ Kj/hr} \]

\[ H = 3260539.22 - 21904074.7 \]
\[ = -18633033 \text{ Kj/hr} \]

\[ H_r = -190.3 - 118.23 - 157.2 = 465.73 \text{ Kj/hr} \]

\[ Q = -18643069.975 \text{ Kj/hr} \]

4-4 Energy Balance on Heat Exchanger:

\[ Q = H \]
\[ H = m \times C_p \times T \]

\[ H_{\text{CdO}} = 21.943 \times 27.43 \times (363 - 298) \]
\[ = 39123.271 \text{ Kj/hr} \]

\[ H_{\text{NiO}} = 47.077 \times 49.88 \times (363 - 298) \]
\[ = 152636.098 \text{ Kj/hr} \]

\[ H_{\text{Fe}} = 31.245 \times 0.45 \times (363 - 298) \]
\[ = 913.91 \text{ Kj/hr} \]

\[ H_{\text{CdCl₂}} = 282.087 \times 74.468 \times (363 - 298) \]
\[ = 2927654.034 \text{ Kj/hr} \]

\[ H_{\text{NiCl₂}} = 735.692 \times -30.2 \times (363 - 298) \]
\[ = -1444163.396 \text{ Kj/hr} \]

\[ H_{\text{FeCl₂}} = 638.72 \times 75.791 \times (363 - 298) \]
\[ = 3146599.789 \text{ Kj/hr} \]
$H_{\text{HCl}} = 2147.6671 \times 143 \times (363-298)$
  
  \[= 19969028.26 \text{ Kj/hr}\]

$H_{\text{H}_{2}\text{O}} = 12337.166 \times 75.318 \times (363-298)$

\[= 60398693.47 \text{ Kj/hr}\]

**4-5 Energy Balance on Filteration:**

Because of the temperature of input and output of the Filter $= 25 \degree C$

So $Q$ (for Filter) $= 0$

**4-6 Energy Balance on Extraction:**

Because of the temperature of input and output of the Extractor $= 25 \degree C$

So $Q$ (for Extractor) $= 0$

**4-7 Energy Balance on Electrolysis:**

$Q = H$

\[= H_{\text{out}} - H_{\text{in}} \]

\[=(F17+F18+F15+F14)-F13\]

$HF_{13} = H\text{CdCl}_2 + \text{HH}_2\text{O}$

$H\text{CdCl}_2 = 242.961 \times 74.468 \times (313-298)$

\[= 271392.296 \text{ Kj/hr}\]

$\text{HH}_2\text{O} = 16415.316 \times (313-298) \times 89.64$

\[= 22071609 \text{ Kj/hr}\]

$F16 = \text{CL}_2$

$H\text{Cl}_2 = 94.0537 \times 78.8(313-298)$

\[= 111171.4734 \text{ Kj/hr}\]
HH₂ = 28.78*5.279(313-298)
= 2279006.812 Kj/hr
F15 = H₂O+CdCl₂
F15 = HDCl₂
= 0.7288*(313-298)*74.468
= 814.084 Kj/hr
F15 = HH₂O
= 16312.478*(313-298)*75.267
= 18417010.07 Kj/hr
F14 = HD
HD = 148.368*(313-298)*26.2
= 58320.647 Kj/hr
HH₂O = 7.808*(313-298)*75.267
= 8815.271 Kj/hr
Q = H
= -1467860.9 Kj/hr

4-8 Energy Balance on Neutralizer:
Because of the temperature of input and output of the Neutralizer = 25 °C
So Q (for Neutralizer ) = 0

4-9 Energy Balance on Filter:
Because of the temperature of input and output of the Filter = 25 °C
So Q (for Filter ) = 0
4-10 Energy Balance on Electrolysis2:

\[ Q = H = H_{\text{out}} - H_{\text{in}} \]

\[(Hf_{23} + F_{26} + F_{25}) - (F_{22} + F_{24})\]

HF25 = HCl2
\[= 402.7546 \times (313 - 298) \times 78.8\]
\[= 475809.372 \text{ Kj/hr}\]

H26 = HH2
\[= 22.69 \times (313 - 298) \times 5.279\]
\[= 1796.70765 \text{ Kj/hr}\]

HF22 = HNiCL2 + HCdCL2 + HH2O + HNaCL

HNiCL2 = 735.692 \times (313 - 298) \times 26.145
\[= 288520.01 \text{ Kj/hr}\]

HCdCL2 = 2.821 \times (313 - 298) \times 29.679
\[= 1255.866885 \text{ Kj/hr}\]

HH2O = 13340.7179 \times (313 - 298) \times 75.267
\[= 15061737.21 \text{ Kj/hr}\]

HNaCl = 3996.1235 \times (313 - 298) \times 50.446 = 3023826.691 \text{ Kj/hr}\]

F24 = CdCL2 + NaCL + H2O

HCdCL2 = 2.821 \times (313 - 298) \times 29.679
\[= 1255.866885 \text{ Kj/hr}\]

HNaCL = 50.446 \times (313 - 298) \times 3996.1235
=3023826.691Kj/hr
HH₂O=133407179*(313-298)*75.267
=15061737.21Kj/hr
F23 =HNi+HH₂O
HNi=332.93*(313-298)*26.145
=130566.8228Kj/hr
HH₂O=17.5226*(313-298)*75.267
=19783.10301Kj/hr
Q=-17974203.54Kj/hr
4-11 Energy Balance on Drying:
H=(F19+F21)-F14
HF19=148.368*(383-313)*26.817=278514.92 Kj/hr
HH₂O=7.808*(383-313)* 75.318=41165 Kj/hr
F14=HCd+HH₂O
HCd=148.368*(383-313)*26.817
=278514.92Kj/hr
HH₂O=7.808*(383-313)*75.821
=41440.752Kj/hr
# Chapter Five

## Equipment Design

<table>
<thead>
<tr>
<th>Material</th>
<th>Property of material</th>
<th>Stream at 90 °C</th>
</tr>
</thead>
<tbody>
<tr>
<td>cdo</td>
<td>Specific heat KJ/kg.K</td>
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<tr>
<td></td>
<td>Density kg/m3</td>
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<td></td>
<td>Thermal conductivity W/m.K</td>
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<tr>
<td></td>
<td>Viscosity N.S/m²</td>
<td>...........</td>
</tr>
<tr>
<td>Nio</td>
<td>Specific heat KJ/kg.K</td>
<td>49.88</td>
</tr>
<tr>
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<td>Density kg/m3</td>
<td>6670</td>
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<td>Thermal conductivity W/m.K</td>
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<td>Viscosity N.S/m²</td>
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<tr>
<td>Fe</td>
<td>Specific heat KJ/kg.K</td>
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<td>Thermal conductivity W/m.K</td>
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<td>Density kg/m3</td>
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<td>Thermal conductivity W/m.K</td>
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<td>Viscosity N.S/m²</td>
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</tr>
<tr>
<td>NiCl₂</td>
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<td>Density kg/m3</td>
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<tr>
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<td>Thermal conductivity W/m.K</td>
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<td>Viscosity N.S/m²</td>
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<td>H₂</td>
<td>Specific heat KJ/kg.K</td>
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<td>Density kg/m3</td>
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<td>Thermal conductivity W/m.K</td>
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<tr>
<td></td>
<td>Viscosity N.S/m²</td>
<td>...........</td>
</tr>
</tbody>
</table>
1- Leaching design

\[(\mu_{\text{mean}}) = \mu_L \times (1 + 2.5 \times \Theta)\]
\[= \frac{C_v}{100 \Theta}\]
\[\mu_{\text{mean}} = 1.055 \times 10^{-2}\]

\[m' = 32097.1652/\text{hr}\]
\[\rho_{\text{mix.}} = (\rho \times x)_{\text{CdO}} + (\rho \times x)_{\text{Nio}} + (\rho \times x)_{\text{Fe}} + (\rho \times x)_{\text{CdCl2}} + (\rho \times x)_{\text{NiCl2}} + (\rho \times x)_{\text{H2}} + (\rho \times x)_{\text{HCl}}\]
\[\rho_{\text{mix.}} = (8150 \times 0.0013) + (6670 \times 0.0025) + (7700 \times 0.0019) + (4050 \times 0.0173) + (3000 \times 0.0453) + (70.85 \times 1) + (1180 \times 0.132) + (3160 \times 0.0393) + (1000 \times 0.7596) = 1376.926\]

\[Q = \frac{m'}{\rho_{\text{mix.}}}\]
\[= \frac{32097.1652}{1367.623}\]
\[= 23.469 \text{ m}^3/\text{hr}\]

Basis = 1 hr.
V = 23.469 m³.

- Agitator selection guide:
From figure down to determine types of agitator:
V = 23.469 m³/hr.
Type of agitator (turbine)
Agitator power consumption:
An estimate of the power requirements for various applications can be obtained from Table 10.14 Vol. 6

<table>
<thead>
<tr>
<th>Agitation</th>
<th>Applications</th>
<th>Power, kW/m³</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mild</td>
<td>Blending, mixing</td>
<td>0.04–0.10</td>
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<tr>
<td></td>
<td>Homogeneous reactions</td>
<td>0.01–0.03</td>
</tr>
<tr>
<td>Medium</td>
<td>Heat transfer</td>
<td>0.03–1.0</td>
</tr>
<tr>
<td></td>
<td>Liquid-liquid mixing</td>
<td>1.0–1.5</td>
</tr>
<tr>
<td>Severe</td>
<td>Slurry suspension</td>
<td>1.5–2.0</td>
</tr>
<tr>
<td></td>
<td>Gas absorption, Emulsions</td>
<td>1.5–2.0</td>
</tr>
<tr>
<td>Violent</td>
<td>Fine slurry suspension</td>
<td>&gt; 2.0</td>
</tr>
</tbody>
</table>

Severe slurry suspension 1.5-2.0 kW/m³
P = 1.5-2.0 kW/m³
= 1.5 + 2.0 / 2
= 1.75 kW/m³

P = 1.75 kW/m³ * 51.668632 m³
P = 90.4205 kw
P = 90420.5 w

---------

HL = DT ---------------------- (1)

V = \pi / 4 * DT ^ 2 * HL ------------------------ (2)

Sub equ. 1 in 2

V = \pi / 4 * DT ^ 3

23.469 = \pi / 4 * DT ^ 3

DT ^ 3 = 29.896

DT 3.097 m

L / DT = 1.5

L = 1.5 * 3.09
= 4.635 m

A = \pi * DT * L
= \pi * 3.09 * 4.635
= 44.97 m²

Six-blade flat blade turbine agitator:

DA = DT / 3
= 3.09 / 3
= 1.03 m

Agitator tip speed u_T given by:

u_T = \pi * DA * N

It is commonly used as a measure of degree of agitation in a liquid mixing system.

Tip speed ranges for turbine agitators are recommended as follows:-

2.5 to 3.3 m/s for low agitation
3.3 to 4.1 m/s for medium agitation
4.1 to 5.6 m/s for high agitation
From figure down to determine Po
Po=7
Assume the velocity range from 2.5 to 5.6 for determine power
From try and error to determine N, Re, uT, P

\[ u_T = 4.3014 \text{ m/s} \]
\[ Re = 130866.2732 \]
\[ N = 1.33 \text{ rps.} \]
\[ N = 1.33 \text{ rps} \times 60 \]
\[ = 79.8 \text{rpm} \]
\[ P = 44438.2028 \text{w power requirement} \]

**Heat transfer of mixer:**

\[ L = 4.635 \text{m} \]
\[ DA = 1.03 \text{m} \]
\[ u_T = 4.3014 \text{ m/s} \]
\[ Re = 130866.2732 \]
\[ N = 0.9975 \text{ rps} \]
\[ Cp \text{ mean}=kJ/Kg.K \]
\[ q = -18643069.975 \text{KJ/hr} \]
\[ q = -5178.630486 \text{kJ/sec for one unit} \]
Agitated vessels:

Unless only small rates of heat transfer are required, as when maintaining the temperature of liquids in storage vessels, some form of agitation will be needed. The various types of agitator used for mixing and blending described, are also used to promote heat transfer in vessels. The correlations used to estimate the heat transfer coefficient to the vessel wall, or to the surface of coils, have the same form as those used for forced convection in conduits. The fluid velocity is replaced by a function of the agitator diameter and rotational speed, D & N, and the characteristic dimension is the agitator diameter.

\[ Nu = C Re^a Pr^b \left( \frac{\mu}{\mu_w} \right)^c \]

For agitated vessels:

\[ \frac{h_i D}{k_f} = C \left( \frac{ND^2 \rho}{\mu} \right)^a \left( \frac{C_p \mu}{k_f} \right)^b \left( \frac{\mu}{\mu_w} \right)^c \]

where \( h_i \) = heat transfer coefficient to vessel wall or coil, Wm\(^{-2}\)C\(^{-1}\)
\( D \) = agitator diameter, m
\( N \) = agitator speed, rpm (revolutions per second)
\( \rho \) = liquid density, kg/m\(^3\)
\( k_f \) = liquid thermal conductivity, Wm\(^{-1}\)C\(^{-1}\)
\( C_p \) = liquid specific heat capacity, J kg\(^{-1}\)C\(^{-1}\)
\( \mu \) = liquid viscosity, Nm\(^{-2}\)s.

The values of constant C and the indices a, b and c depend on the type of agitator, the use of baffles, and whether the transfer is to the vessel wall or to coils. Some typical correlations are given below. Baffles will normally be used in most applications.
Flat blade disc turbine, baffled or unbaffled vessel, transfer to vessel wall, $Re < 400$:

$$Nu = 0.54Re^{0.67}Pr^{0.33} \left( \frac{\mu}{\mu_w} \right)^{0.14}$$

Flat blade disc turbine, baffled vessel, transfer to vessel wall, $Re > 400$:

$$Nu = 0.74Re^{0.67}Pr^{0.33} \left( \frac{\mu}{\mu_w} \right)^{0.14}$$

$Re = 105326.082$

$Re >> 400$

Flat blade disc turbine, baffled vessel, transfer to vessel wall, $Re > 400$:

$$Nu = 0.74Re^{0.67}Pr^{0.33} \left( \frac{\mu}{\mu_w} \right)^{0.14}$$

$K_{mix} = (0.0013*8) + (0.0025*3.57) + (0.0019*79.5) + (0.0173*9) + (0.0453*0.056) + (1*0.172) + (0.132*0.0139) + (0.0393*7) + (0.75*1.4)$

$= 1.86 \text{w/m*k}$

$C_p_{mean} = 3.1324 \text{KJ/Kg.K}$

$Pr = \frac{\mu}{cp/k}$

$Pr = 1.055*10^{-2} * 3.1324 * 10^3 / 1.86$

$= 24.299$

$Nu = 0.74*(130866.2732)^{0.67} * (24.299)^{0.33} * (1.055*10^{-2}/1*10^{-3})^{0.14}$

$= 7906.89$

$Nu = \frac{i}{DA/k}$

$7906.89 = hi * 1.03 / 1.86$

$hi = 14278.46 \text{w/m².k}$

Physical properties of steam at mean temperature of 90°C from steam tables:

$C_p = 2.1226 \text{kJ/Kg.K}$

$K = 0.02674 \text{w/m.k}$
\[ \mu = 0.01306 \text{ mN.s/m}^2 \]
\[ \text{Pr} = 0.847 \]
\[ \rho = 1.129 \text{ kg/m}^3 \]
\[ \lambda = 2201.59 \text{ kJ/kg} \]

The baffle forms a continuous spiral channel, section 75 mm * 200 mm.
Number of spirals = height of jacket/pitch = \[ \frac{4.635}{200 \times 10^{-3}} \]
= 23.175

Length of channel = \[ 23.175 \times \pi \times 3.09 \]= 224.8577 m

Cross-sectional area of channel = \[ 75 \times 200 \times 10^{-6} \]= 15 \times 10^{-3} \text{ m}

hydraulic mean diameter \( d_e \) = \[ \frac{4 \times \text{cross-sectional area}}{\text{wetted perimeter}} \]

= \[ 4 \times (75 \times 200) / 2 \times (75 + 200) \]= 109

**Condensation of steam:**

Steam is frequently used as a heating medium. The film coefficient for condensing steam can be calculated using the methods given in the previous sections; but, as the coefficient will be high and will rarely be the limiting coefficient, it is customary to assume a typical, conservative, value for design purposes. For air-free steam a coefficient of 8000 W/m²°C (1500 Btu/h ft² °F) can be used
ho=8000 \text{ w/m}^2{\text{°C}}

From table 12.2 Vol.6 fouling factor
\( h_{od} = 4000 \text{ w/m}^2{\text{°C}} \)
\( h_{id} = 5000 \text{ w/m}^2{\text{°C}} \)

From Table 12.6 Vol.6
Thermal conductivity for stainless steel (kW) = 16 w/m°C

\[ \frac{1}{U_o} = \frac{1}{h_o} + \frac{1}{h_{od}} + \frac{d_o \ln \left( \frac{d_o}{d_i} \right)}{2k_w} + \frac{d_o}{d_i} \times \frac{1}{h_{id}} + \frac{d_o}{d_i} \times \frac{1}{h_i} \]

The overall coefficient based on the outside area of the tube W/m
ho = outside fluid film coefficient W/m·C
hi = inside fluid film coefficient W/m·C
hod = outside dirt coefficient (fouling factor) W/m·C
hid = inside dirt coefficient W/m·C
Kw = thermal conductivity of the tube wall material W/m²·C

di = tube inside diameter m
do = tube outside diameter m

\[ \frac{1}{U} = \frac{1}{8000} + \frac{1}{4000} + 3.11 \ln \left( \frac{3.11}{3.09} \right) / 2 + 16 \]
\[ + \frac{3.11}{3.09} \times \frac{1}{5000} + \frac{3.11}{3.09} \times \frac{1}{16464} \]

U = 790.87 w/m²·C

*ΔT = 120 - (25 + 90)/2
= 62.5°C
q = U * A * ΔT
5178.630486 * 1000 = 790.87 * A * 60.2
A = 108.77 > 44.97 m² so we will use composite wall

Pressure drop

\[ \Delta P = 8j_f \left( \frac{L'}{d_i} \right) \frac{\rho u'^2}{2} \]

Where:- if is the dimensionless friction factor and L is the effective pipe length
From figure (12.24) Vol.6 up $j_f = 0.0027$

$\Delta P = 8 \times 0.0027 \times (232.03/109) \times 1.129 \times 3.23^2 / 2$

$= 0.27 \text{kN/m}^2$

**Mechanical design:**

take as 5 per cent above operating pressure

Design pressure = $1 \times 1.05$

Design pressure = 1.05 bar

Design pressure = $0.105 \text{N/mm}^2$

Design temperature = 65°C

From Table 13.2, typical design stress = $115 \text{N/mm}^2$

Calculate the thickness by the equation:

$e = \pi d_i / 2 - \pi = 0.1 \times 3.09 / 2 \times 115 - 0.1 = 0.001 \text{m} = 1 \text{mm}$

in our calculation we take the thickness from the standard $t = 10 \text{mm}$

$D_i = 3.09 \text{m} = 3090 \text{mm}$

$H_v = 3.09 \text{m} = 3090 \text{mm}$

$t = 0.010 \text{m} = 10 \text{mm}$

**Dead weight of vessel:**
This equation only applies strictly to vessels with uniform thickness, it can be used to get a rough estimate of the weight of this vessel by using the average thickness 16mm

\[
W_v = C_v \pi \rho_m D_m g (H_v + 0.8D_m) t \times 10^{-3}
\]

For steel vessel

\[
W_v = 240 C_v D_m (H_v + 0.8D_m) t
\]

\(W_v\) = total weight of the shell, excluding internal fittings, such as plates, \(N\),
\(C_v\) = a factor to account for the weight of nozzles, manways, internal supports, etc; which can be taken as
  = 1.08 for vessels with only a few internal fittings,
  = 1.15 for distillation columns, or similar vessels, with several manways, and with plate support rings, or equivalent fittings.

\(H_v\) = height, or length, between tangent lines (the length of the cylindrical section), m,
\(g\) = gravitational acceleration, 9.81 m/s²,
\(t\) = wall thickness, mm
\(\rho_m\) = density of vessel material, kg/m³,
\(D_m\) = mean diameter of vessel = \((D_i + t \times 10^{-3})\), m.

Take \(C_v= 1.15\), vessel with several fitting,
\(D_m= 3.09 + 10*10^{-3} = 3.1\) m, \(H_v= 3.09\) m, \(t = 10\) mm
\(W_v = 240 * 1.15 * 3.09*(3.09+ 0.8* 3.09)*10 = 47434.96\) N

**Weight of insulation :**

Mineral wool density = 130 kg/m³
Approximate volume of insulation = \(2\times3.09\times75*10^{-3} = 1.45539\) m³
Weight = \(1.45539 \times 130 \times 9.81 = 1856.058\) N
Double this to allow for fittings = \(2\times1856.058 = 3712.1177\) N
Total weight = \(47434.96 + 3712.1177 = 51147.07\) N

**Wind loading :**

Take dynamic wind pressure as 1280 N/m²
Mean diameter, including insulation = \(3.09 + 2\times(10+75)*10^{-3} = 3.26\) m

\(F_w = P_w \text{Deff}\)
Loading (per linear meter)\( F_w = 1280 N/m^2 \times 3.26 m = 4172.8 N/m \)

Bending moment at bottom tangent line

\[ M_x = F_w \times H_v^2 / 2 = 4172.8 \times 3.09^2 / 2 = 19921.15 N.m \]

**Analysis of stresses:**

At bottom tangent line Pressure stresses

\[ \sigma_h = P D_i / 2t \]
\[ \sigma_L = 0.101 \times 3090 / (2 \times 10) = 15.619 N/mm^2 \]
\[ \sigma_L = P D_i / 4t \]
\[ \sigma_L = 0.105 \times 3090 / (4 \times 12) = 6.759 N/mm^2 \]

Dead weight stress

\[ \sigma_w = W / (D_i + t) \times t \]
\[ \sigma_w = 51147.07 / (3090 + 10) \times 10 = 0.52544 N/mm^2 \text{ (compressive)} \]

**Bending stresses:**

\[ IV = (\pi / 64) \times (D_o^4 - D_i^4) \]
\[ = (\pi / 64) \times (3110^4 - 3090^4) = 1.17 \times 10^{11} mm^4 \]
\[ \sigma_b = \pm (M / IV) \times (D_i / 2 + t) \]
\[ = (19921.15 \times 10^3 / 1.17 \times 10^{11}) \times (3090 / 2 + 10) \]
\[ = \pm 0.0438 N/mm \]

The resultant longitudinal stress is

\[ \sigma_z = \sigma_L + \sigma_w + \sigma_b \]
\( \sigma_w \) is compressive and therefore negative.

\[ \sigma_z (\text{upwind}) = 6.759 + 0.52544 + 0.0438 = 7.32824 N/mm \]
\[ \sigma_z (\text{downwind}) = 6.759 + 0.52544 - 0.0438 = 7.24164 N/mm \]

As there is no torsional shear stress, the principal stresses will be \( \sigma_z \) and \( \sigma_h \)

The radial stress is negligible, \( \approx \pi i / 2 = 0.1 / 2 = 0.0505 N/mm \)

The greatest difference between the principal stresses will be on the down-wind side

\[ \sigma_h - \sigma_z (\text{downwind}) = 15.619 - 7.24164 = 8.37736 N/mm^2 \]

well below the maximum allowable design stress
Check elastic stability (buckling):

Critical buckling stress
\( \sigma_c = 2 \times 10^4 \left( \frac{t}{D_o} \right) = 20000 \left( \frac{10}{3110} \right) \)
\( \sigma_c = 64.308 \text{ N/mm}^2 \)

The maximum compressive stress will occur when the vessel is not under pressure
\( \sigma_b + \sigma_w = 0.0438 + 0.52544 = 0.56924 \text{ N/mm}^2 \)
well below the critical buckling stress. So design is satisfactory.

Vessel Supports:

Try a straight mixer skirt (\( \theta_s = 65 \)) of plain stainless steel, design stress 115 N/mm²
and Young’s modulus 200000 N/mm² at ambient temperature.
The maximum dead weight load on the skirt will occur when the vessel is full of water
at ambient temperature. Young’s modulus 200000 N/mm² and Young’s modulus 200000 N/mm²
The maximum dead weight load on the skirt will occur when the vessel is full of water.
Approximate weight = \( \frac{1}{4} D^2 h \rho_{\text{average}} g \)
= \( \frac{1}{4} \times 3.09^2 \times 3.09 \times 2016 \times 9.81 \)
Approximate weight = 148.23 N
Weight of vessel = 51147.07 N
Total weight = Approximate weight + Weight of vessel
Total weight = 148.23 + 51147.07 = 51295.30 N
Wind loading = 19921.15 N/m
Bending moment at base of skirt = W * x^2 / 2
Let skirt height = 1 m
X = 3.09 + skirt height = 3.09 + 1 = 4.09 m
Bending moment at base of skirt = 19921.15 * 4090^2 / 2
= 1.666 * 10^{11} Nm
\( \sigma_{bs} \) = bending stress in the skirt
\( \sigma_{bs} = \frac{4 * M_s}{(D_s + t_s) * t_s * D_s} \)
Fig(4.2) straight skirt support
\( M_s \) = maximum bending moment, evaluated at the base of the skirt (due to Wind, seismic and eccentric loads)
\( D_s \) = inside diameter of the skirt at the base
\( t_s \) = skirt thickness.
\( \theta_s \) = base angle of a conical skirt, normally 80° to 90°.

\[ \sigma_{bs} = 4 \times 1.666 \times 10^{11} / [(3090+10) * 10 * 3090] = 2.2155 \text{ N/mm}^2 \]
\( \sigma_{ws} \) = the dead weight stress in the skirt
\[ \sigma_{ws(test)} = \frac{W}{(D_s + t_s) * t_s} \]
\( W \) = total weight of the vessel and content (Approximate weight)
\[ \sigma_{ws(test)} = \frac{148.23}{(3090+10) * 10} = 0.0015 \text{ N/mm}^2 \]
\[ \sigma_{ws(operating)} = \frac{51147.07}{(3090+10) * 10} = 0.525 \]
\( \sigma_s \) (compressive) = \( \sigma_{bs} + \sigma_{ws(test)} \)
= 2.2155 + 0.0015 = 2.217 N/mm² (compressive)
\( \sigma_s \) (tensile) = 2.2155 - 0.525 = 1.69 N/mm
the joint factor J as 0.85
The skirt thickness should be such that under the worst combination of wind and dead-weight loading the following design criteria are not exceeded:
\( \sigma_s \) (tensile) < J * f_s * sin \( \theta_s \)
1.69 < 0.85 * 89 sin(90)
1.69 < 67.63
\( \sigma_s \) (compressive) < 0.125E(t_s/D_s)sin\( \theta_s \)
2.217 < 0.125 * 2 * 10^4 (18/3090) sin(90)
2.217 < 14.56
Both criteria are satisfied, add mm² for corrosion, gives a design thickness of 130 mm

**Base ring and anchor bolt design:**

- **Approximate pitch circle diameter:** 1.2 mm
- **Circumference of bolt circle:** $1.2 \times 3090 \approx 11643.12$ mm²
- **Number of bolts required at minimum:** 8 bolt
- **Bolt pitch:** $11643.12 / 8 = 1455.39$ mm
- **The bolt design stress:** 125 N/mm²
- **Ms** = $1.666 \times 10^{11}$, **W** = 51147.07 N

$$A_b = \frac{1}{N_b f_b} \left[ \frac{4M_s}{D_b} - W \right]$$

$A_b$ = area of one bolt at the root of the thread, mm²,
$N_b$ = number of bolts,
$f_b$ = maximum allowable bolt stress, N/mm²;
    typical design value 125 N/mm² (18,000 psi),
$M_s$ = bending (overturning) moment at the base, Nm,
$W$ = weight of the vessel, N,
$D_b$ = bolt circle diameter, m.

$$A_b = \frac{1}{8 \times 125} \left[ (4 \times 1.666 \times 10^{11} / 1.2) - 51147.07 \right] = 25.88 \text{ mm}$$

**Bolt root diameter:** $(19.63 \times 4 / 0.5) = 0.174 \text{ mm}$

25 mm so we will take bolt diameter (diameter = 25 mm)

$$F_b = \left[ \frac{4M_s}{\pi D_s^2} + \frac{W}{\pi D_s} \right]$$

$F_b$ = the compressive load on the base ring, Newton’s per linear meter
$D_s$ = skirt diameter, m

$$F_b = \left\{ 4 \times 1.666 \times 10^{11} / (3.09^2) + 51147.07 / (3.09) \right\} = 2.22 \times 10^6 \text{ N/m}$$

The minimum width of the base ring is given by:

$$L_b = F_b / (f_c \times 10^3)$$

also actual width $L_b = L_r + ts + 50 = 150 + 10 + 50 = 210 \text{ mm}$

**Actual person concrete foundation**

$$f_c = F_b / (\text{actual width} \times 10^3) = 2.22 \times 10^6 / (210 \times 10^3) = 10.57 \text{ N/mm}$$

The minimum thickness is given by
\[ t_b = L_r \sqrt{\frac{3f'_c}{f_r}} \]

\( L_r \) = the distance from the edge of the skirt to the outer edge of the ring, mm:
\( t_b \) = base ring thickness, mm,
\( f'_c \) = actual bearing pressure on base, N/mm²,
\( f_r \) = allowable design stress in the ring material, typically 140 N/mm².

\[ t_b = 150 \times (3 \times 10.57 / 140)^{0.5} = 71.388 \text{mm} \]

Taking corrosion allowances = 2 + 71.388 = 73.388 mm

Figure (4.3) Flange ring dimension

Gasket materials

<table>
<thead>
<tr>
<th>Gasket material</th>
<th>Gasket factor ( m )</th>
<th>Min. design seating stress ( \sigma (\text{N/mm}^2) )</th>
<th>Sketches</th>
<th>Minimum gasket width (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Asbestos with a suitable binder for the operating conditions</td>
<td>3.2 mm thick</td>
<td>2.00</td>
<td>11.0</td>
<td>10</td>
</tr>
<tr>
<td>1.6 mm thick</td>
<td>2.75</td>
<td>25.5</td>
<td></td>
<td></td>
</tr>
<tr>
<td>0.8 mm thick</td>
<td>3.50</td>
<td>44.8</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Rubber with cotton fabric insertion</td>
<td>1.25</td>
<td>2.8</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Table Gasket material (based on a similar table in BS5500:1991; see BS PD 5500 2003)

For stream (1) we find the optimum diameter as:
\[ d_{\text{optimum}} = 260G^{0.52} \rho^{-0.37} \]
Stream (1) \( G = 1.2766 \text{ kg/s} \), \( \rho = 1000 \text{ kg/m}^3 \)

\[ d_{\text{optimum}} = 260(1.2766)^{0.52}(1000)^{-0.37} = 22.9151 \text{ mm} \]

Optimum diameter with allowance

\[ d_1 = d_{\text{outside}(1)} = 22.9151 + 10 = 32.9151 \text{ mm} = 0.032 \text{ m} \]

For stream (3) we find the optimum diameter as

\[ d_{\text{optimum}} = 260G^{0.52} \rho^{-0.37} \]

Stream (3) \( G = 0.0840 \text{ kg/s} \), \( \rho = 1804.4 \text{ kg/m}^3 \)

\[ d_{\text{optimum}} = 260(0.0840)^{0.52}(1804.4)^{-0.37} = 4.47 \text{ mm} = 0.00447 \text{ m} \]

Optimum diameter with allowance

\[ d_1 = d_{\text{outside}(3)} = 4.47 + 10 = 14.47 \text{ mm} \]
Figure 4.5. Economic pipe diameter; chart constructed to connect values of flow rate and density to obtain economic
4.1- Design of Heat exchanger:

Figure (4.1) Clarification the Heat Exchanger

\[ Q = \frac{8255596.44}{3600} = 22932.11012 \text{ Kw} \]

\[ Tm = \frac{(T1 - t2) - (T2 - t1)}{\ln\left(\frac{T1 - t2}{T2 - t1}\right)} \]

\[ = \frac{(90 - 60) - (25 - 10)}{\ln\left(\frac{90}{25}\right)} = 21.64 \text{ c} \]

The correction factor

\[ R = \frac{T1 - T2}{t2 - t1} = \frac{90 - 25}{60 - 10} = 1.3 \]

\[ S = \frac{t2 - t1}{T1 - t1} = \frac{60 - 10}{90 - 10} = 0.625 \]
From figure (12.19) page (567)  

$$F = 0.8$$

$$T_m = F \times T_m = 21.64 \times 0.8 = 17.31 \, ^\circ C$$

First we assume Overall all coefficient from table 12-1 vol.6

$$U = 1150 \, \text{w/m}^2.\text{c}$$

$$Q = U \times A \quad T_m \Rightarrow A = \frac{Q}{U} \quad T_m$$

$$A = \frac{22932.11012}{1150} = 1.151 \, \text{m}^2$$

The standard dimension for steel tube we obtain it from table in vol.6 12-3

Outside diameter = 20 mm

Inside diameter = 16.8 mm

Length of tube = 4.88 mm

Area of one tube = 3.14 * do * L

$$= 3.14 \times 20 \times 10^{-3} \times 4.88 = 0.303 \, \text{m}^2$$
\[ Nt = \text{trial Area} / \text{area at one tube} \]
\[ = \frac{1.151}{0.303} = 3.801 \]

**Tube bundle diameter:**
\[ Db = \frac{d_o \cdot (N_t/k)^{\frac{1}{n_1}}}{K_1 \cdot n_1} \]
Where:
- \( N_t \) = number of tube, \( D_b \) = bundle diameter mm, \( d \) = tube outside diameter mm
- \( K_1 \) & \( n_1 \) can be founded from table (12.4) page (649), (vol.6)
- \( K = 0.249 \)
- \( n = 2.207 \)

\[ D_b = 20 \cdot \left( \frac{4}{0.249} \right)^{\frac{1}{2.207}} = 145.575 \text{ mm} \]

Use a split-ring floating head type.

From figure (12.10), (vol.6), bundle diameter clearance = 58
Shell diameter, \( D_s = D_b + D_b \) clearance
\[ = 145.575 + 58 = 203.575 \text{ mm} \]

**Tube-side coefficient**

Mean water temperature = (60+10)/2 = 35°C
Tube cross-sectional area = \( (\pi/4) \cdot 16^2 = 201 \text{ mm}^2 \)
Tube per pass = 4/2 = 2
Total flow area = 2 \cdot 201 \cdot 10^{-6} = 0.00402 \text{ m}^2
Water mass velocity = 247.1722/0.00402 = 61485.621 Kg/s m²
Density of water = 995 Kg/m³
Water linear velocity = \( \frac{\text{mass velocity per area}}{\text{density of water}} \)
\[ = \frac{61485.621}{995} = 64.114 \text{ m/s} \]

\[ hi = \frac{4200 \cdot (1.35 + 0.02 \cdot t) \cdot Ut^{0.8}}{d^{0.2}} \]
where

\( h_i = \text{inside coefficient for water, } \text{w/m}^2\text{.c} \)
\( t = \text{water mean temperature, } \text{c} \)
\( U_t = \text{water velocity, } \text{m/s} \)
\( d_i = \text{inside diameter} \)

\[
hi = \frac{4200(1.35+0.02) (35) (6.4114)^{0.8}}{(16.8 \times 10^{-3})^{0.2}} = 543880.631 \text{ w/m}^2\text{.c}
\]

Viscosity of water = 0.8 mNs/m², Thermal conductivity = 0.59 W/m °C

\[
\text{Re} = \frac{pud_i}{\mu} = \frac{995 \times 64.114 \times 16 \times 10^{-3}}{(0.8 \times 10^{-3})}
\]

\( \text{Re} = 1275868.6 \)

\[
\text{Pr} = \frac{(C_p \mu)}{K_f}
\]

\[
= \frac{(91.63 \times 0.8 \times 10^{-3})}{0.59}
\]

\( \text{Pr} = 0.1242 \)

\( L/d_i = 4.83 \times 10^3 / 16 = 302 \)

From figure (12.23), (vol.6), \( j_h = 3.2 \times 10^{-3} \)

\[
hi = \frac{0.59/16 \times 10^{-3}}{3.2 \times 10^{-3} \times 1275868.6 \times 0.1242^{0.33}}
\]

\( = 75639.4 \text{ W/m}^2\text{.°C} \)

**Shell-side coefficient**

Choose baffle spacing = \( D_s/5 \)

\( = 203.575 / 5 = 40.715 \text{ mm} \)

Tube pitch = 1.25 \( \times 20 = 25 \text{ mm} \)

Cross-flow area, \( A_s = ((25 - 20)/25) \times 203.575 \times 40.715 \times 10^{-6} \)

\( = 0.00165 \text{ m}^2 \)
Mass velocity, \( G_s = \frac{16627.626}{3600} \times \frac{1}{0.00165} \)
\[= 279.92 \text{ Kg/m}^2\cdot\text{s} \]

Equivalent diameter, \( d_e = \frac{(1.1/d_o)(pt^2 - 0.917 \times d_o^2)}{2} \)
\[= \frac{(1.1/20)(25^2 - 0.917 \times 20^2)}{2} = 14.4 \]

Mean shell side temperature = \( \frac{(90+25)}{2} \)
\[= 57.5 ^\circ C \]

Physical properties for mixture.

\( X_{CdO} = 0.13 \)
\( X_{NiO} = 0.25 \)
\( X_{Fe} = 0.19 \)
\( X_{CdCl_2} = 1.73 \)
\( X_{NiCl_2} = 4.53 \)
\( X_{FeCl_2} = 3.93 \)
\( X_{HCl} = 13.22 \)
\( X_{H_2O} = 75.96 \)

\[ \rho_{av} = X_{CdO} \times \rho_{CdO} + X_{NiO} \times \rho_{NiO} + X_{Fe} \times \rho_{Fe} + X_{CdCl_2} \times \rho_{CdCl_2} + X_{NiCl_2} \times \rho_{NiCl_2} + X_{FeCl_2} \times \rho_{FeCl_2} + X_{HCl} \times \rho_{HCl} + X_{H_2O} \times \rho_{H_2O} \]
\[= (0.0013 \times 8150) + (0.0025 \times 6670) + (0.0019 \times 7700) + (0.0173 \times 4050) + (3000 \times 0.0453) + (1180 \times 0.132) + (3160 \times 0.0393) + (1000 \times 0.7596) = 1306.076 \]

Thermal conductivity:

\[ K_{f(\text{av})} = X_{CdO} \times K_{CdO} + X_{NiO} \times K_{NiO} + X_{Fe} \times K_{Fe} + X_{CdCl_2} \times K_{CdCl_2} + X_{NiCl_2} \times K_{NiCl_2} + X_{FeCl_2} \times K_{FeCl_2} + X_{HCl} \times K_{HCl} + X_{H_2O} \times K_{H_2O} \]
\[= (0.0013 \times 8) + (0.0025 \times 3.57) + (0.0019 \times 79.5) + (0.0173 \times 9) \]
\[(0.0453*0.056) + (0.132*0.0139) + (0.0393*7) + (0.75*1.4)\]
\[= 1.688 \text{w/m}^2\text{k}\]

Heat capacity:

\[C_p_{av} = X_{CdO} \cdot C_{pCdO} + X_{NiO} \cdot C_{pNiO} + X_{Fe} \cdot C_{pFe} + X_{CdCl_2} \cdot C_{pCdCl_2} + X_{NiCl_2} \cdot C_{pNiCl_2} + X_{FeCl_2} \cdot C_{pFeCl_2} + X_{HCl} \cdot C_{pHCl} + X_{H_2O} \cdot C_{pH_2O}\]

\[C_{p_{mean}} = 3.1324 \text{KJ/Kg.K}\]

\[Re = \frac{(Gs \ de)}{\mu}\]
\[= \frac{(279.92 \times 14.4)}{0.001} = 4030848\]

\[Pr = \frac{(0.001 \times 42.0)}{0.6} = 6.5\]

From figure below calculate,

\[J_h = 8 \times 10^1\]

For 25% baffle cut

Figure (4.3) Shell-side heat-transfer factors, segmental baffles
The overall all transfer coefficient evidence:

\[
\frac{1}{U} = \frac{1}{h_o} + \frac{1}{h_{od}} + \frac{\text{doln} (\frac{d_o}{d_i})}{2 \text{ kw}} + \frac{d_o}{d_i} \frac{1}{h_d} + \frac{d_o}{d_i} \frac{1}{h_i}
\]

where

- \(U\) = the overall coefficient, \(\text{w/m}^2\text{.c}\)
- \(o\) = outside fluid film coefficient, \(\text{w/m}^2\text{.c}\)
- \(i\) = inside fluid film diameter, \(\text{w/m}^2\text{.c}\)
- \(od\) = outside dirt coefficient, \(\text{w/m}^2\text{.c}\)
- \(id\) = inside dirt coefficient, \(\text{w/m}^2\text{.c}\)
- \(kw\) = thermal conductivity of the tube wall material, \(\text{w/m.c}\)
- \(do\) = tube outside diameter, \(\text{m}\)
- \(di\) = tube inside diameter, \(\text{m}\)

\(U = 1235\ \text{w/m.c} \Rightarrow \text{which is near from the assumption value}\)

Tube side pressure drop:

\[Pt = Np \left\{ 8jf \left( \frac{L}{d_i} \right) \left( \frac{U}{\mu w} \right) + 2.5 \right\} \frac{\rho u^2}{2}\]

We neglect the \(\left( \frac{U}{\mu w} \right) \) because \(\mu\)

Where

- \(Np\) = number of passes
- \(Mw\) = viscosity of wall
- \(P\) = density of water
- \(Ut\) = velocity of water in tube

\[pt = 2 \left\{ 8 \times 0.8 \times \frac{4.83 \, 10^{-3}}{16.8 \, 10^{-3}} + 2.5 \right\} \frac{995}{2} \frac{6 \, 4114}{2}\]

\[pt = 3882.4 \ \text{N/m}^2 = 3.882 \ \text{kpa}\]
**Mechanical design:**

1-thickness of shell:

\( P = 12 \text{ bar} \)

Design pressure take 10% above operating pressure

\( P = (12 - 1) \cdot 1.1 = 12.1 \text{ bar} \)

\( P = 1.21 \text{ N/mm}^2 \)

Design temperature = 413°C

From table 13.2 vol.6 typical design stress(f)=89 \( \text{N/mm}^2 \)

\[
e = p \cdot \frac{D_i}{2 \cdot f - p}
\]

\[
= 1.21 \cdot 0.58474 \cdot \frac{1000}{2 \cdot 98 - 1.21}
\]

\[
= 3.63 \text{ mm}
\]

Add corrosion allowance = 2 mm

\( e = 3.63 + 2 \)

\( e = 5.63 \text{ mm} \)

2-demand head

Flat heat

Used a full face gasket \( cp = 0.4 \)

\( De = \text{diameter let} = 0.584 \text{ m} \)

\[
e = cp \cdot De \cdot \frac{\sqrt{p}}{\sqrt{f}}
\]

\[
e = 0.4 \cdot 0.584 \cdot 1000 \cdot \frac{\sqrt{584}}{\sqrt{83}}
\]
\( e = 19.59 \, \text{mm} \)

Add corrosion allowance = 2 mm

\( e = 2 + 19.59 \)

\( e = 21.59 \, \text{mm} \)

3-weight load:

For the steel

\[
W_c = 240 \times C_v \times D_m (H + 0.8 \times D_m) \times t
\]

\( W_c \) = total weight

\( C_v = 1.08 \)

\( H \) = height

let \( H = 4.83 \, \text{m} \)

\( t \) = thickness, \( \text{mm} \)

\( D_m \) = mean diameter

\[
D_m = (D_i + t \times 10^{-3})
\]

\[
D_m = (0.584 + 5.103 \times 10^{-3})
\]

= 0.5891 m

\[
W = 240 \times 1.08 \times 0.5891 (4.83 + 0.8 \times 0.5891) \times 8.53 \times 10^{-3}
\]

\[ W = 6.904842 \, \text{KN} \]

\[ W = 6904.842 \, \text{N} \]
4-skirt supports:

For straight cylindrical $\theta = 90$ made from steel

![Diagram of skirt supports](image)

Figure (4.6) skirt-support welds

Wind loading

Take dynamic wind pressure $= 1280 \, N/mm^2$

$$D_{eff} = D_s + D_s \times [(t + e)]$$

Assume $t = 5.103$ mm

e = 50 mm (insulation)

$$D_{eff} = 0.397 + 0.397[(5.105 + 50) \times 10^{-3}]$$

$$= 0.419 \, m$$
Figure (4.7) Typical skirt-support designs (a) Straight skirt

\[ F_w = P_w + D_{eff} \]

\[ = 1280 \times 0.419 = 536.32 \text{ N/m}^2 \]

Bending moment at bottom

\[ M_x = F_w \times \frac{h^2}{2} \]

\[ M_x = 536.32 \times \frac{4.83^2}{2} \]

\[ = 6255.877 \text{ N} \]

Mx=6.255877 KN

Bending moment at base of skirt

Take height of skirt= 1 m

Height of heat exchanger=4.83 m

Total height =4.83+1=5.83 m
\[
Bending = Mx \cdot \frac{h^2}{2} \\
= 6.255877 \cdot \frac{5.83^2}{2} = 106.31512 \text{ KNm}
\]

\[
\sigma_{bs} = \frac{4M_s}{\pi(Ds + ts) \cdot ts \cdot Ds}
\]

\(\sigma_{bs}\) = bending stress in the skirt

Ms = maximum bending moment

Ds = inside diameter let = 0.5 m

\[
\sigma_{bs} = 4 \cdot 106.31512 \cdot 1000 \cdot \frac{1000}{[\pi(500 + 5.103) \cdot 5.103 \cdot 500]}
\]

= 105.0872 N/mm\(^2\)

\[
\sigma_{ws} = \frac{w}{\pi(Ds + ts) \cdot ts}
\]

\(\sigma_{ws}\) = the dead weight stress in the skirt

ts = thickness = 5.23 mm

W = total weight

Ds = 0.5 m

Approximate weight = \[
\frac{\pi}{4} [0.5^2 \cdot 4.83] \cdot 7800 \cdot 9.81
\]

Approximate weight = \[
\frac{\pi}{4} [0.5^2 \cdot 4.83] \cdot 7800 \cdot 9.81
\]

Approximate weight = 72567.24 N

Total weight = 72567.24 + 6904.84 = 79472.24 N

\[
\sigma(ws) = \frac{72567.24}{\pi(500 + 5.103) \cdot 5.103}
\]

= 8.96 N/mm\(^2\)
\[
\sigma_{ws(operete)} = \frac{79472.08}{\pi(500 + 5.103) \times 5.103}
\]

\[= 9.814 \, N/mm^2\]

Criteria desing:

Taking joint factor \((j) = 0.85\)

Desing stress \((f) = 100 \, N/mm^2\)

\[\theta = 90^\circ\]

\(\sigma_s\) less than \((f \times j \times \sin \theta)\)

\((1.94\) less than \((100 \times 0.85 \times \sin 90)\)

\(9.814\) less than \(85\)

So, the criteria is satisfied
Chapter Six

Cost Estimation

Fixed capital investment for cost index (2010) = 550.8
Cost index for (2017) = 567.5
The cost in 2010 = $9,000,000.5

Therefore present fixed capital investment
Present cost = original cost in 2010 (index value at 2017/index value at 2010)

Present cost = $9,000,000.5 * (567.5/550.8) = $9,272,876.332

Estimation of total investment cost:
1- Direct cost:
   a- Purchased equipment cost:
      (15 - 40% of FCI) assume 30% of FCI

      = 0.3 * $9,272,876.332
      = $2,781,862.9

   b- Installation cost:
      (35 - 45% of PEC) assume 40% of PEC
      = 0.4 * $2,781,862.9
      = $1,112,745.16

   c- Instrument and control installed:
      (6 - 30% of PEC) Assume 20% of PEC
      = 0.2 * $2,781,862.9
      = $556,373.0


d- Piping installation cost: (10 - 80% of PEC) Assume 60 %
= 0.6 * 2,781,862.9
= 1,669,117.74$

e- Electrical installation cost: (10 - 40% of PEC) Assume 30% of PEC
= 0.3 * 2,781,862.9
= 834,558.87$

f- Building process and auxiliary (10-70% of PEC) Assume 50 %
= 0.5 * 2,781,862.9
= 1,390,931.45$

g- Service facilities: (30-80% of PEC) Assume 65 %
= 0.65 * 2,781,862.9
= 1,808,210.885$

h- Yard improvement: (10-15% of PEC) Assume 12 %
= 0.12 * 2,781,862.9
= 333,823.548$

i- Land: (4-8% of PEC) Assume 5 %
= 0.05 * 2,781,862.9
= 139,093.145$

Therefore direct cost = a + b + c + d + e + f + g + h + i

= 2,781,862.9 + 1,112,745.16 + 556,372.58 + 1,669,117.74 + 834,558.87 + 1,390,931.45 + 1,808,210.885 + 333,823.548 + 139,093.145
= 10,626,716.28$

Indirect cost:

Expenses which are not directly involved with material and labour of actual installation or complete facility
a- Engineering and supervision (5-30% of DC) Assume 25%

\[ = 0.25 \times 10626716.28 \]

\[ = 2656679.07 \]$ 

b- Expenses: (10% of DC)

\[ = 0.1 \times 10626716.28 \]

\[ = 1062671.628 \]$ 

c- Contractors fee (2-7% of DC) Assume 5%

\[ = 0.05 \times 10626716.28 \]

\[ = 531335.814 \]$ 

d- Contingency: (8-20% of DC) Assume 18%

\[ = 0.18 \times 10626716.28 \]

\[ = 1912808.93 \]$

Therefore total indirect cost = a + b + c + d

\[ = 2656679.07 + 1062671.628 + 531335.814 + 1912808.93 \]

\[ = 6163495.442 \]$

**Fixed capital investment**

Fixed capital investment (FCI) = DC + IC
=10626716.28+6163495.442
=16790211.72$

Working capital investment: 10% of FCI Assume 15%
=0.15*16790211.72
=2518531.758$

2- Total capital investment

Estimation of total product cost (TPC):

Fixed charges:

a- Depreciation: (10% of FCI for machinery)
=0.1*16790211.72
=1679021.172$

b- Local taxes: (3-4% of TPC = FCI) Assume 3.5%
=0.035*16790211.72
=587657.4102$

c- Insurances (0.4-1% of FCI) Assume 0.7%
=0.007*16790211.72
=117531.482$

d- Rent: (8-12% of FCI) Assume 10%
=0.1*16790211.72
=1679021.172$

Therefore total fixed charges = a + b + c + d

=1679021.172+587657.4102+117531.482+1679021.172
=4063231.236$
But, Fixed charges= (10-20% of TPC) Assume 20%
Therefore Total product cost = total fixed charges / 0.2 or * 100/20
\[ = \frac{4063231.236}{0.2} \]
\[ = 20316156.18\] $ \\

**Direct production:**

a- Raw material: (10-50% of TPC) Assume 40%
\[ = 0.4 \times 20316156.18 \]
\[ = 8126462.472\] $

b- Operating labor (OL): (10-20% of TPC) Assume 15%
\[ = 0.15 \times 20316156.18 \]
\[ = 3047423.427\] $

c- Direct supervisory and electric labor (10-25% of OL) Assume 20%
\[ = 0.2 \times 3047423.427 \]
\[ = 609484.6854\] $

d- Utilities (10-20% of TPC) Assume 15%
\[ = 0.15 \times 20316156.18 \]
\[ = 3047423.427\] $

e- Maintenance (2-10% of FCI) Assume 7%
\[ = 0.07 \times 16790211.72 \]
\[ = 1175314.82\] $

f- Operating supplies (OS): (10-20% of maintenance) Assume 15%
=0.15*1175314.82
=176297.2231$

g- Laboratory charges(10-20% of OL) Assume 15%
=0.15*3047423.427
=457113.5141$

h- Patent and royalties(2-6% of TPC) Assume 5%
=0.05*20316156.18
=1015807.809$

**Plant overhead cost:** 50-70% of (OL+OS+M) Assume 60%

=0.6(3047423.427+176297.2231+1175314.82)
=2639421.282$

**General expenses:**

a- Administration cost:(40-60% of OL) Assume 50 %

=0.5*3047423.427
=1523711.714$

b- Distribution and selling price(2-30% of TPC) Assume 25 %

=0.25*20316156.18
=5079039.045$

c- Research and development cost:(3% of TPC)

=0.03*20316156.18
Therefore general expenses (GE) = A + B + C

\[= 1523711.714 + 5079039.045 + 609484.6854 = 7212235.444\]

Therefore manufacturing cost (MC) = Product cost + fixed charges + Plant overhead expenses

\[= 20316156.18 + 4063231.236 + 2639421.282 = 27018808.7\]

**Total production cost:**

Total production cost = MC + GE

\[= 7212235.444 + 27018808.7 = 34231044.14\]

**Gross earnings and rate of return:**

The plant is working for say 320 days a year

Selling price =

Total income = produce day rate * number production days (in year) * price per unit

\[= 23400 \times 320 \times 8 = 59904000\]

Gross profit = Total income - total product

\[= 59904000 - 34231044.14 = 25672955.86\]

Tax = 50%

Net profit = Gross profit – (Gross profit * 50%)
=25672955.86-(25672955.86\times0.5)
=12836477.93$

Rate of return = \frac{net \ profit}{total \ capitalinvestmen}
=\frac{12836477.93}{4063231.236}$
=3.159179
References:


