

Energy Management for Pollution Control

Paul Mac Berthouex & Linfield Brown



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ENERGY MANAGEMENT FOR POLLUTION CONTROL

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1st edition

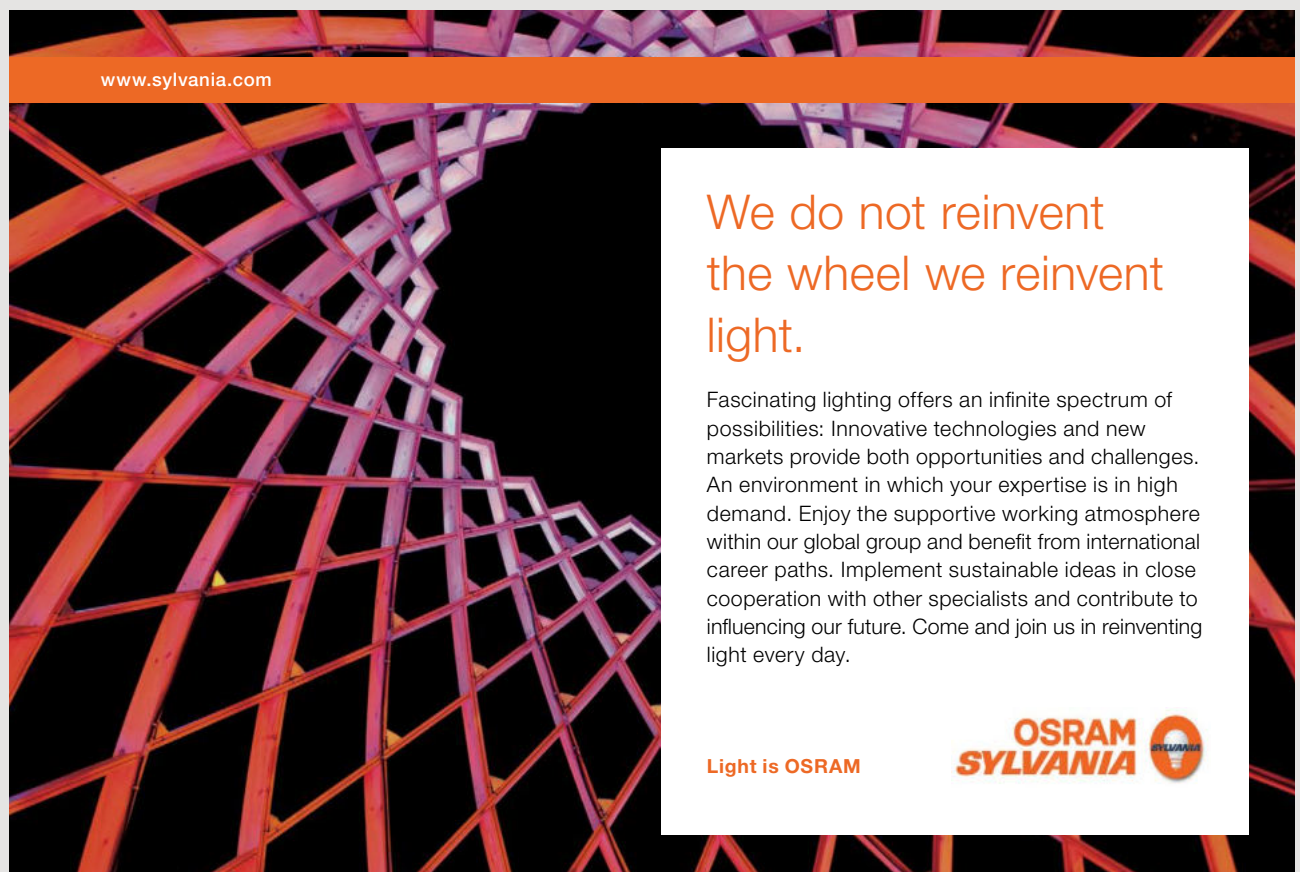
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CONTENTS

	Preface	8
1	Energy Management	10
1.1	Conservation of Mass and Energy	12
1.2	Process Integration	14
1.3	Conclusion	15
2	The Laws of Thermodynamics	16
2.1	Three Laws of Thermodynamics	16
2.2	Conservation of Energy – The First Law of Thermodynamics	17
2.3	The Heat Trap – The Second Law of Thermodynamics	20
2.4	Conclusion	23




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3	Energy Units and Energy Conversion	24
3.1	Work, Power and Energy	24
3.2	Arithmetic Equivalence of Energy Units	25
3.3	Energy Conversion Efficiency	26
3.4	Combined Heat and Power (CHP)	30
3.5	Wind Energy	33
3.6	Solar Energy	36
3.7	The Future of Renewable Energy	40
3.8	Conclusion	42
4	The Energy Balance and Enthalpy	43
4.1	Enthalpy	43
4.2	Specific Heat	45
4.3	The Energy Balance	50
4.4	Cooling Towers	53
4.5	Refrigeration	58
4.6	Boiler Efficiency and Water Use	60
4.7	Conclusion	63
5	Combustion of Municipal Refuse	64
5.1	Stoichiometry	64
5.2	Combustion Stoichiometry	68
5.3	Composition of Solid Waste	72
5.4	Heating Value of Waste Materials	75
5.5	Incineration of Solid Waste and Sludge	80
5.6	Case Study – Palm Beach Renewable Energy Facility No. 2	82
5.7	Conclusion	85
6	Energy Recovery from Biogas	86
6.1	Composition of Biogas	87
6.2	Energy Recovery from Landfill Gas	89
6.3	Model for Landfill Gas Production	94
6.4	Energy Recovery from Anaerobic Sludge Digestion	97
6.5	Gas Treatment Technologies	106
6.6	Conclusion	108
7	Incineration of Waste Gas	110
7.1	Safety – The Explosive Limits	110
7.2	Gas Composition	111
7.3	Thermal Incineration	116
7.4	Catalytic Incineration of Waste Gases	117
7.5	Case Study - Recovery of Heat from Combustion of Waste Gases	118
7.6	Case Study - Energy Balance on a Regenerative Thermal Oxidizer	125
7.7	Conclusion	129

8	Energy Conservative Design	130
8.1	Heat Exchangers	132
8.2	Heat Exchanger Networks	137
8.3	Pinch Analysis for Heat Exchanger Network Design	140
8.4	Conclusion	141
9	Energy for Pumping	142
9.1	Pump Efficiency	143
9.2	Pressure and Head	144
9.3	The Pump Curve, Efficiency Curve, and System Curve	145
9.4	Motor Efficiency	148
9.5	Power Requirements	149
9.6	Calculating Head Losses from K Values	150
9.7	Pipe Networks	154
9.8	Conclusion	156
10	Energy for Blowers and Compressors	157
10.1	Blowers	158
10.2	Aeration in Wastewater Treatment	160
10.3	The Demand for Oxygen and Air	161
10.4	Air Delivery Pressure	165
10.5	Blower Power	168
10.6	Variations in Air Flow Rates	171
10.7	Conclusion	174
11	The Economics of Energy Management	175
11.1	Energy Use in Water and Wastewater Treatment	176
11.2	Costs Estimates	181
11.3	Capital Cost Estimates	181
11.4	Economy of scale	182
11.5	Planning Costs for Pumps	187
11.6	Cost of Electricity	188
11.7	Peak Energy Demand and Peak Charges	189
11.8	Saving by Shedding Peak Loads	192
11.9	The Cost of Steam	193
11.10	The Time Value of Money	195
11.11	Payback Time	196
11.12	Adjusting Cost Between Future and Present Time	199
11.13	Uniform Annual Payments	201
11.14	Comparing Two Projects	205
11.15	Life Cycle Cost	206
11.16	Inflation	211
11.17	Conclusion	213

12	References	214
13	Appendix 1 – Abridged Table of Atomic Numbers and Atomic Masses	220
14	Appendix 2 – Conversion Factors	221
15	Appendix 3 -Densities and Specific Weights	224
16	Appendix 4 – Heating Values	227
17	Appendix 5 – Enthalpy of Water and Steam	232
18	Appendix 6 – Enthalpy of Air	233
19	Appendix 7 – Lower and Upper Explosive Limits for selected volatile compounds	234
20	Appendix 8 – Financial Tables	235
	Index	244

PREFACE

Material and energy balances are an essential part of process synthesis and analysis. The material balance is essential to the design and operation of every system. Since most processes use energy for pumping or heating, or recover heat from combustion processes, the energy balance is also essential. The energy balance can only be made if the kinds and mass of materials moving through the system are known, so material and energy balances go hand in hand.

Pollution Prevention and Control: Material and Energy Balances was published by Bookboon.com in 2014. This new edition separates material balances and energy balances into two books, *Material Balances for Pollution Prevention and Control* and *Energy Management for Pollution Prevention and Control*.

This book has eleven chapters

- Chapter 1 introduces the material and energy balance concepts with some examples that preview some ideas that are explained in the following chapters.
- Chapter 2 is an introduction to the laws of thermodynamics, the forms of energy, and energy conversion efficiency. Solar and wind energy are discussed briefly.
- Chapter 3 explains how energy is converted from one form to another in order to produce heat and electricity. The units of measuring energy are also explained.
- Chapter 4 is how enthalpy and specific heat are used with the material balance to calculate the energy balance.
- Chapter 5 covers the combustion of municipal refuse, including the composition and heating values. Using the chemical reactions of combustion to make the material balance on reactants and products is explained.
- Chapter 6 is about energy recovery from biogas derived from landfills and anaerobic processes that are used to manage wastewater treatment plant sludge, agricultural wastes, and strong industrial wastewater.
- Chapter 7 is the incineration of waste gas, which is an air pollution control method that offers the possibility of heat recovery.
- Chapter 8 is about organizing heat exchangers and the flow of heat through systems to minimize the use of steam for heating and water for cooling.
- Chapter 9 is about energy use for pumping.
- Chapter 10 is about energy use for air blowers and aeration for activated sludge treatment.
- Chapter 11 is about costs and the economic calculations that are used to compare alternate projects.

Chapter 11 is new. New material has been added to chapters 3, 5, 6, 9 and 10. An important addition is the publication of a Study Guide for Energy Management with problems and solutions. We hope that the Study Guide will stimulate the practice that is necessary to develop skill in solving problems.

The book is designed as an engineering text, but the concepts and calculations are accessible to students in non-engineering disciplines. The goal is to build problem-solving strategies and skills that are widely useful in water pollution control, air pollution control, and solid waste control. Instruction is mostly by example calculations and case studies.

The course is quantitative, but not mathematically difficult. The main challenge is formulating the problem; if that is done properly calculating the solution will not be overly difficult.

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1 ENERGY MANAGEMENT

Engineering design is a blend of *synthesis* and *analysis*. “*Synthesis* deals with the creation of artificial things that have desired properties by combining often diverse elements into a coherent whole. *Analysis* examines the elements and their relations. Each synthesis creates an analysis problem, the solution of which often provides insights that create a new synthesis”

– Rudd et al 1973

Analysis can be understood and practiced without knowing about the machinery that is used to accomplish the transformations and separations. We only need to know what change is accomplished or required. The essential tools are the *material balance* and the *energy balance*. A material balance will be needed for virtually every pollution prevention and control problem. The material balance *and* the energy balance are needed when accounting for the use and flow of energy.

Energy and material flow through processes and system are linked. Sometimes the changes in energy are inconsequential, like the change in heat energy (temperature) of water as it flows through a settling tank, so an energy balance is not needed. The reverse is not true. It is impossible to make an energy balance without knowing the material balance. Important quantities are the mass of water converted to steam, the mass of fuel entering a boiler or incinerator, and the volume and mass rate of flow of flue gas and its components.

Most energy and water conservation problems involve the flow of heat energy in water or steam that are used for heating and cooling. The simultaneous flow of heat and mass explains why reducing water use usually reduces energy use, and using less energy often reduces water.

The basic elements that are organized into pollution prevention and control systems are reactors and separations, shown in Figure 1.1. The material balance is an accounting of the mass of every material that enters and leaves each stage of the processing system. The mass in must equal the mass out for each process and for the system overall. The same is true for the energy balance – energy inputs, in all the various forms, must equal energy outputs.

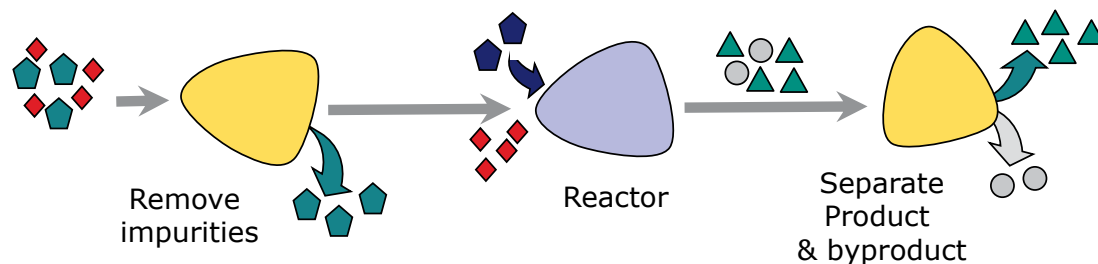


Figure 1.1 Integration of separators and reactors

Reactors are used to promote and control chemical and biochemical reactions for the purpose of transforming toxic, offensive, unstable, or low value materials into non-toxic, inoffensive, stable, and useful materials. Some reactors, such as the activated sludge process for treating wastewater, operate at ambient temperatures. It is important to know how the temperatures may change with season and manufacturing practices, but it is not necessary to make a balance on heat energy. Even though the process is not heated it does consume great amounts of energy to run the blowers that deliver air to the process. This is usually the single largest use of energy in a conventional wastewater treatment plant.

Anaerobic sludge digesters are heated and the amount of heat to be added is proportional to the mass of sludge, which is mostly water, and to the required increase in temperature, and heat losses from the reactor.

Heat energy is needed in combustion processes to bring the air, fuel, and combustible materials (organic chemicals, refuse derived fuel, dried sludge, etc.) to a temperature that assures complete destruction. The chemical energy of combustible materials is converted to heat energy which can be converted to electricity or used to heat buildings and processes.

Separations selectively remove one species of material from others for the purpose of concentrating or upgrading the material. Some separations, such as reverse osmosis and ultrafiltration, require energy for pumping. Pumping liquids and gases consumes electricity in large quantities so this offers opportunities for energy conservation.

Condensation and evaporation require heat energy to be added or removed. This is done by using steam, cooling water, or air to add or remove energy. Heat removed from the hot gas or liquid is transferred to a cold stream in a heat exchanger. The goal in heat exchanger systems is to minimize the uses of steam and cooling water because this maximizes the recovery of energy.

1.1 CONSERVATION OF MASS AND ENERGY

The process designer and operator must know the amounts of material and energy that will be managed. Flow rates, physical properties, and composition can be monitored in existing systems and this information can be converted into a material balance. This will help to reduce operating costs, but it does not solve the design problem.

We are concerned not only with the way things are but with *the way things ought to be*. Many systems exist only as alternatives in our mind, or on paper. We cannot measure that which does not yet exist but, we must have accurate estimates of the flow rates and compositions in order to assess alternate designs. This is done by using the two fundamental tools for process design.

Conservation of Mass - Mass is neither created nor destroyed (except in nuclear reaction which are not discussed in this text). The masses of all material flowing into and out of a system must be equal (unless some material is stored within the system). If there is no accumulation within the system it is said to operate at *steady-state*.

Materials that move through the system may be changed physically or chemically. Water becomes steam, steam becomes water, fuel becomes gas, particles dissolve, solutes precipitate, gases are absorbed by or stripped from liquids, and so on. Molecules are decomposed and the atoms are rearranged to make molecules of new materials. Whatever happens within the system, the total mass that enters the system either leaves it or is stored within the system. The same is true for each element. If a solution of chloride and sodium is put into a system, the mass of chloride and the mass of sodium that leaves, in whatever form, must equal the mass that entered. Conservation of mass holds absolutely for all substances, and for the total system and each processing step in the system.

Conservation of Energy - The same is true for energy. Energy is neither created nor destroyed. The energy flowing into a system must equal the energy that leaves the system.

Energy can be dissipated from a useful form into waste heat by friction or heat loss from steam pipes. Heat energy becomes mechanical energy as when steam drives a generator shaft. Mechanical energy becomes kinetic energy when a pump imparts motion to a fluid. After nature has done all the manipulations the total amount of energy must be the same. Some careful analysis may be needed to account for everything, but the First Law of Thermodynamics says that the account must balance.

Figure 1.2 shows the material balance and the energy balance for a simple system of two heat exchangers. Cooling water is used to condense steam. The hot water from condenser (the condensate) is used to heat air. The mass entering and leaving each process is equal. The same is true for the heat energy. The method for calculating the energy of each stream uses the energy balance.

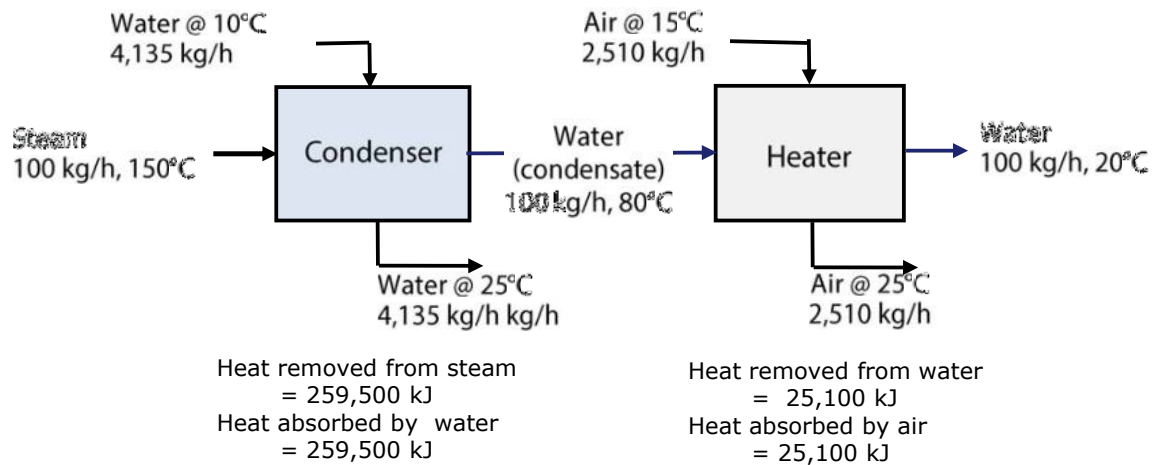


Figure 1.2 The inputs of heat energy and mass of water equal the outputs for the condenser, the heater, and the overall system

The concepts are easy to understand and the calculations are readily learned. In practice the more difficult work is estimating or collecting the necessary information about the flow and composition of the input and output streams.



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1.2 PROCESS INTEGRATION

Figure 1.3 shows the energy balance for a primitive and inefficient design and a more efficient integrated system. Conservation of mass and conservation of energy require for the heat exchangers, the reactor, and the overall system that:

$$\text{Total mass in} = \text{Total mass out}$$

$$\text{Energy in} = \text{Energy out}$$

The balance for heat energy is

$$\begin{aligned} \text{Feed} + \text{Added from steam} + \text{Produced in reactor} \\ = \text{Removed by cooling water} + \text{Product} \end{aligned}$$

The energy balance is satisfied for each unit and the overall balance is satisfied for each system. The balance on mass of material is also satisfied; 1000 kg of feed leaves as 1000 kg of products and by-products (some of which may be wastes). The efficient system eliminates the need for steam by recovering heat energy from the product stream to heat the reactor feed. This also reduces the mass of cooling water needed.

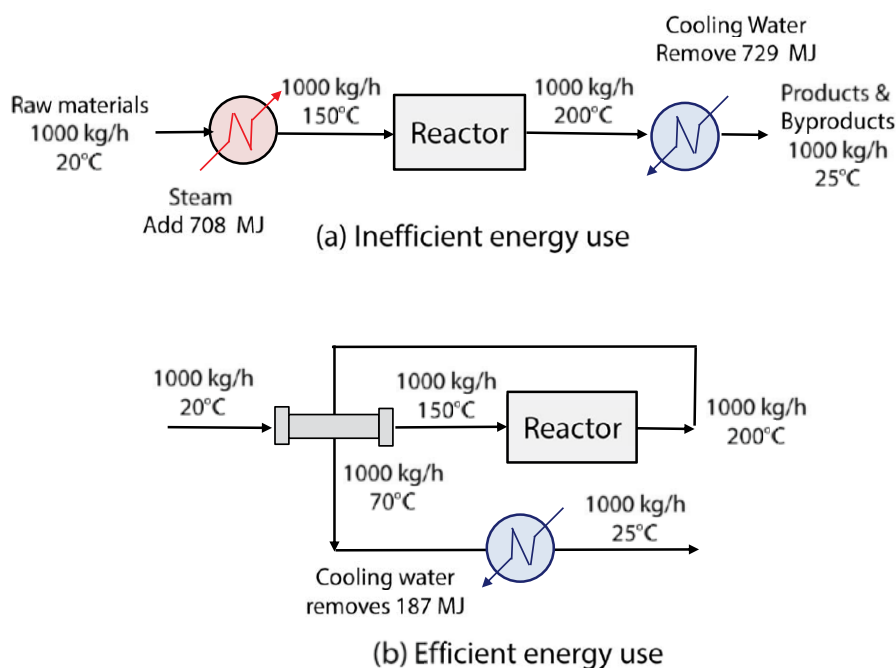


Figure 1.3 Two designs for a reactor system.

This is an example of why integrating the study of material and energy balances is important for environmental engineers. Reducing the heating and cooling of materials, including water is a satisfying way of reducing energy use and emissions to the atmosphere. Reducing energy use reduces water use and the reverse is true as well.

1.3 CONCLUSION

Energy management is part of process synthesis and analysis, and an important part of process operation. Energy costs are a large part of operating costs and this creates opportunities to use energy efficiently and save money.

An accurate energy balance depends on having an accurate accounting of the mass and kinds of materials moving through the system. In pollution prevention and control work two important materials are air and water. Heating and cooling water, and pumping water, are intense energy demands. Therefore, water conservation and energy conservation go hand in hand. Reduce one and you will reduce the other.

2 THE LAWS OF THERMODYNAMICS

Designing systems to manage the flow of energy is much like designing systems to manage the flow of materials. In existing systems we can measure temperatures, electrical use, steam flow, and fuel burned and make a balance on the energy entering and leaving the system. The energy flow of newly-imagined systems on the drawing board (in the CAD system) cannot be measured, so it is calculated. Because energy is not lost, the amount entering a system or process equals the amount leaving. The energy balance is the designer's counterpart to the material balance.

The energy balance is used to answer such questions as:

- How much heat energy can be obtained from one ton of fuel oil?
- How much steam can be produced from one ton of fuel oil?
- How much electricity can be produced from one ton of fuel oil?
- How much energy is required to pump water into an elevated tank?
- How much heat energy can be recovered from the hot exhaust gas from a gas engine?
- Does a sludge digester produce enough biogas to heat the sludge that is fed to the digester?

This and the following chapters deal with the law of conservation of energy and the basic principles of thermodynamics, but only in simple situations. We will be able to account for the flow of energy as materials flow through a processing system and to approximate the heat available from burning different fuels, including waste materials. More complicated problems are in the realm of the mechanical and chemical engineer.

2.1 THREE LAWS OF THERMODYNAMICS

The laws of thermodynamics are:

- *First law:* Energy can be changed into various forms, but the total amount of energy is unchanged and a balance can always be made to account for it. This is the principle of *conservation of energy*.
- *Second law:* No machine is one hundred percent efficient in converting energy input into work output. It is impossible to devise a machine which working in a cycle shall produce no other effects than the extraction of heat from a reservoir and the performance of an equal amount of work.

- *Third law.* It is impossible to reduce the temperature of a system to absolute zero in a finite number of steps.

2.2 CONSERVATION OF ENERGY – THE FIRST LAW OF THERMODYNAMICS

There are many forms of energy: gravitational, kinetic, heat, elastic, electrical, chemical, radiant, nuclear, and mass energy. Energy can be changed into various forms, but a balance can always be made to account for it. This is the *first law of thermodynamics* - the *principle of conservation of energy*.

"There is no known exception to the law of conservation of energy ... It is not a description of a mechanism, or of anything concrete; it is just a strange fact that we can calculate some number and when we finish watching nature go through her tricks and calculate the number again, it is the same"

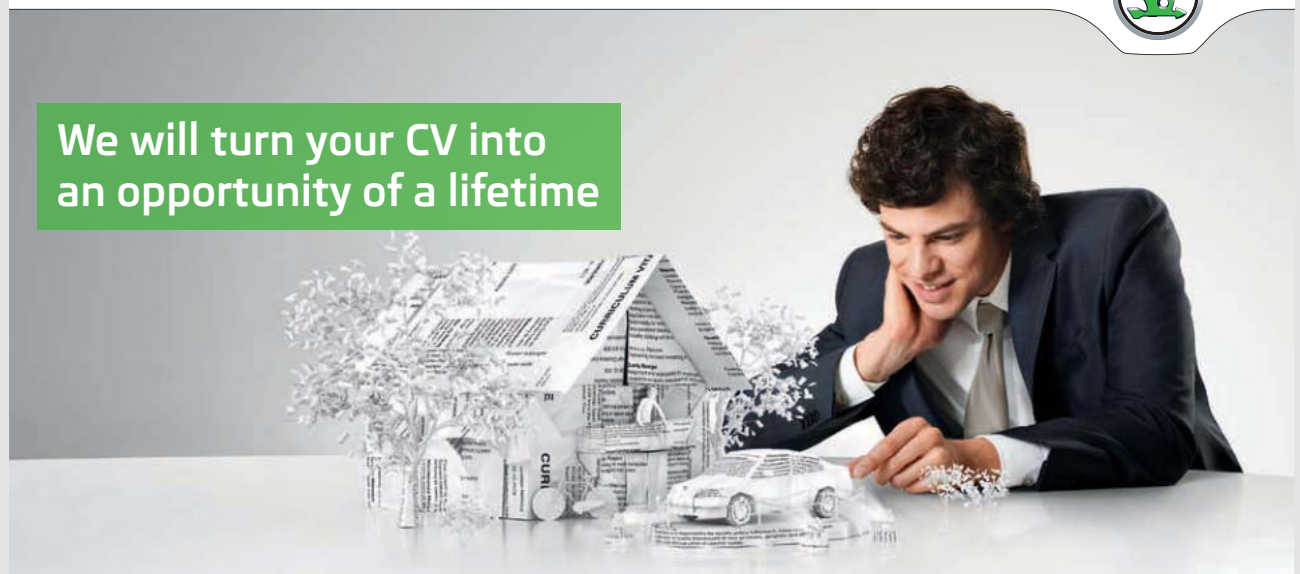
– Feynman 1996

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Much of our concern with energy has to do with converting it from one form to another. Chemical energy in natural gas is converted to thermal energy in steam, which is converted to mechanical energy in a turbine, which is converted into electrical energy. Each conversion is inefficient, due in part to restrictions imposed by Nature and in part to technology. This needs to be understood to make predictions about how much heat can be obtained from burning different kinds of fuels, and how much useful energy can be obtained from machinery.

Figure 2.1 shows an enclosed volume or system over which the energy balance is to be made. We are not concerned with the internal details, only with the passage of energy across the system boundaries. Energy crossing the boundary may be associated with a mass of flowing material, or it can cross the boundary as the flow of heat, for example heat loss by radiation from hot surfaces. Or, it could enter in the form of fuel that releases its energy upon combustion, or as chemicals that react to consume or release heat. All forms of energy, both useful and otherwise, are included.

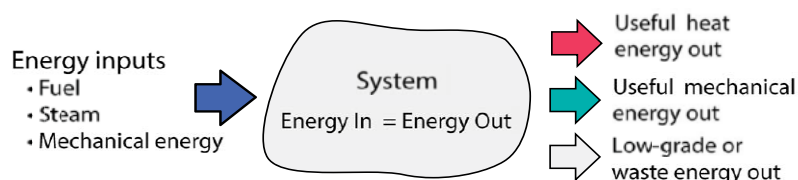


Figure 2.1 All energy that enters an enclosed volume of the system must accumulate in or leave the system.

We will study energy balances that involve heating, cooling, and combustion processes that operate at constant pressure. In constant pressure processes the energy balance is analogous to the material balance, with enthalpy (kJ/kg or kJ/L) taking on the role of concentration (mg/kg or mg/L). If the process is not at constant pressure, vapors and gases expand and contract and a thermodynamic function called entropy must be used in the analysis. These processes are important but they will not be covered in this book.

For the case where no energy accumulates within the process boundary, the energy balance is

$$\text{Total energy in} = \text{Total energy out}$$

Each term in the equation must have the same units (Joule, kWh, Btu, etc.).

EXAMPLE 2.1 POWER PLANT WASTE HEAT

A thermal electric power plant, Figure 2.2, burns 1,000,000 kg/day of coal that has a heating value of 9.35 kWh/kg (33.66 MJ/kg). The thermal efficiencies of the boiler, turbine, and generator are 80%, 45%, and 98%, respectively. The losses are waste heat.

The basis is 1 day of operation. All quantities are for a period of one day.

Coal input = $(1,000,000 \text{ kg coal})(9.35 \text{ kWh/kg}) = 9,350,000 \text{ kWh} = 33,660,000 \text{ MJ}$

Boiler efficiency = 80%

Energy transmitted to the turbine = $(0.80)(9,350,000 \text{ kWh})$
 $= 7,480,000 \text{ kWh} = 26,930,000 \text{ MJ}$

Energy lost with the exhaust gas = $9,350,000 \text{ kWh} - 7,480,000 \text{ kWh}$
 $= 1,870,000 \text{ kWh} = 6,730,000 \text{ MJ}$

Turbine efficiency = 45%

Energy transmitted to the generator = $(0.45)(7,480,000 \text{ kWh})$
 $= 3,366,000 \text{ kWh} = 12,120,000 \text{ MJ}$

Energy lost with cooling water = $7,480,000 \text{ kWh} - 3,366,000 \text{ kWh}$
 $= 4,114,000 \text{ kWh} = 14,810,000 \text{ MJ}$

Generator efficiency = 98%

Energy produced = $(0.98)(3,366,000) = 3,300,000 \text{ kWh} = 11,880,000 \text{ MJ}$

Energy lost as waste heat = $66,000 \text{ kWh} = 240,000 \text{ MJ}$



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About $100(3,300,000 \text{ kWh}/9,350,000 \text{ kWh}) = 35.3\%$ of the energy input is converted to electricity. Almost two-thirds of the energy input is lost from the power plant to the environment, mostly by rejection to cooling water in the condensers. The waste heat absorbed by the cooling water is subsequently dissipated in the river or to the atmosphere via evaporation in a cooling tower.

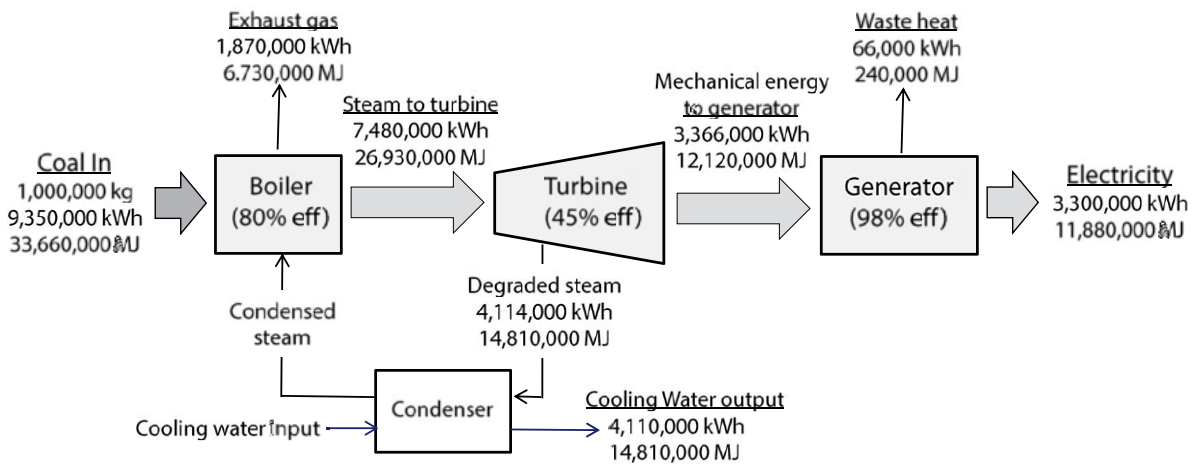


Figure 2.2 Thermal electric power plant.

2.3 THE HEAT TRAP – THE SECOND LAW OF THERMODYNAMICS

The second law explains the efficiency, or lack of efficiency, of energy conversions in heat engines. A heat engine converts thermal energy to mechanical energy, as in a fossil fuel power plant or internal combustion engine. The application to heat engines is shown in Figure 2.3, in which heat energy (Q_h) is withdrawn from a hot reservoir and converted into work (W) and waste heat (Q_c). Work is energy that can in principle be quantitatively converted to the lifting of a weight. Heat is that part of total energy flow that is caused by a temperature difference.

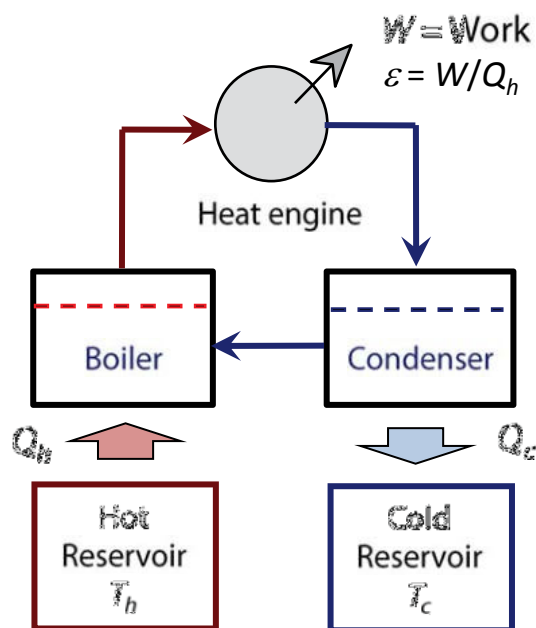


Figure 2.3 General heat engine.

The efficiency of the heat engine, ε , is the useful work produced per unit of energy withdrawn from the hot reservoir.

$$\varepsilon = W/Q_{h0}$$

The inefficient conversion of thermal energy is a consequence of the second law of thermodynamics and not of inept engineering. The thermodynamic limit of the heat engine is a heat trap that cannot be avoided in any energy conversion scheme that involves operations with thermal energy. No heat engine can be more efficient than the theoretical Carnot engine and the efficiency of the Carnot engine is determined by the temperatures of the hot and cold reservoirs, T_h and T_c , measured in kelvins ($K = ^\circ C + 273$).

$$\varepsilon_{\text{Carnot}} = \frac{T_h - T_c}{T_h}$$

In theory the efficiency of a Carnot heat engine can be 100 percent only if the temperature reservoir to which heat is ejected is at absolute zero. (Scientists have been able to reach a temperature of 1 nanokelvin, which is one-billionth of a degree Kelvin. At this temperature it takes an atom half a minute to move one inch. The average temperature of the universe is 2.73 K.)

Efficiency is improved when the temperature at which an engine receives its energy is as high as possible and the temperature at which it rejects its waste heat must be as low as possible. The upper limit is determined by the endurance of materials of construction and is on the order of $600^\circ C$. The lower limit of the cold reservoir for most applications (power plants, automobiles, etc.) is the ambient temperature of the surrounding air, land, or water.

EXAMPLE 2.2 THEORETICAL CARNOT EFFICIENCY OF AN ENGINE

The Carnot efficiency of theoretical engine that operates between a high temperature of 600°C and a low temperature of 15°C is

$$\varepsilon_{\text{Carnot}} = \frac{T_h - T_c}{T_h} = \frac{(600^\circ\text{C} + 273^\circ\text{C}) - (15^\circ\text{C} + 273^\circ\text{C})}{(600^\circ\text{C} + 273^\circ\text{C})} = 0.67$$

This describes a theoretical heat engine that must reject one-third of its energy as waste heat. Real engines reject as much as two-thirds of their input as waste heat.

Any real device that converts heat into work must be less efficient than a Carnot engine that operates over the same temperature range.

EXAMPLE 2.3 ENERGY RECOVERY IN INTERNAL COMBUSTION ENGINES

The chemical energy released from fuel that is burned in an internal combustion engine leaves the engine in these ways.

Useful mechanical work	30 - 38%
Hot jacket water	20 - 26%
Hot lubricating oil	2 - 8%
Hot exhaust gas	30 - 37%
Radiation from metal parts	1 - 10%

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If the hot jacket water is passed through a radiator that cools the water by dumping heat to the air, the energy in the jacket water is wasted. In stationary installations such as engines that drive pumps, generators, compressors, it is possible to recover about 20 percent of the heat from the jacket water, about 13 percent from the exhaust, and perhaps some more from the hot lubricating oil, to bring the overall thermal efficiency from 30 - 38% up to the range of 50 to 70%.

2.4 CONCLUSION

The First Law of Thermodynamics says that energy, in all of its forms, must balance over a system. The total energy coming in must equal the total energy going out. Energy is not created, but it can change form and appear to be created. For example, burning natural gas to make electricity may seem to create energy because the natural gas contains chemical potential energy which is not sensed as heat.

Heat may seem to be lost when the electrical energy is compared with the heat energy of the natural gas. It still exists in a form that may not be useful, hence the sense of being lost.

A heat engine operates between two temperatures that limit the theoretical efficiency, and the actual efficiency cannot match the theoretical efficiency

3 ENERGY UNITS AND ENERGY CONVERSION

Water trapped behind a dam has potential energy and that is converted to kinetic energy as the water flows through the raceway of a hydroelectric plant. The hydroturbine converts kinetic energy to mechanical energy and the generator converts mechanical energy to electric energy. The electric energy is converted to mechanical energy in electric motors, into chemical energy in charging batteries, and into heat for cooking and heating. The measurement units for these energy flows and transformations are varied because they involve units of force, mass, distance, time, and temperature.

3.1 WORK, POWER AND ENERGY

Energy is defined as the capacity to do work. In physical terms

$$\text{Energy (work)} = \text{Force} \times \text{Displacement}$$

The *Newton* is that force which will give an acceleration of one meter per second per second to a mass of one kilogram. The *Joule* is the work done by a force of one Newton when its point of application is moved through a distance of one meter in the direction of the force. Mechanical work energy has units of N-m = Joule (J); the Newton-meter (N-m) and Joule (J) are identical in value.

Power is the rate of expending energy, or the rate of energy flow

$$\text{Power} = \text{change in (or flow of) energy per unit time}$$

The *Watt* (W) is the power experienced by an energy flow of one Joule per second (J/s). The Watt is used for the measurement of every kind of power. Power has units of J/s, = Watt (W), kilowatt (kW), ft-lb/sec, or horsepower (hp). Electric power is typically measured in kilowatts (kW) or megawatts (MW).

Mechanical power in the U.S. is normally reported as *horsepower* (hp), where 1 hp = 550 ft-lb/s = 0.7457 kW = 2.685 MJ. Heat energy in the U.S is usually measured as British thermal unit (Btu).

Finally, energy is often expressed as

$$\text{Energy} = (\text{Power})(\text{Time})$$

which has units of Watt-second (W-s), kilowatt-hour (kWh), or Joules (J), kiloJoules (kJ), or megaJoules (MJ).

3.2 ARITHMETIC EQUIVALENCE OF ENERGY UNITS

The arithmetic equivalence of energy units indicates how numerical values are converted from one set of units to another. It provides no useful information about the conversion of energy from one form to another. Table 3.1 lists some frequently needed energy and power equivalence factors.

Energy unit		Btu	kWh	Joules
1 kilowatt-hour (kWh)	=	3412	1	3.6×10^6
1 Joule (J)	=	9.478×10^{-4}	2.778×10^{-7}	1
1 kilocalorie (kcal)	=	3.968	1.163×10^{-3}	4184
1 British thermal unit (Btu)	=	1.0	2.931×10^{-4}	1055
1 horsepower-hour (hp-h)	=	2544	0.7457	2.685×10^6
Fuel Unit		Btu	kWh	Joules
1 barrel petroleum (bbl) [42 US gal]	=	5.8×10^6	1700	6.117×10^9
1 cubic foot of natural gas	=	1050	0.308	1.108×10^6
1 gallon of gasoline	=	126,000	36.92	133×10^6
Power Unit		kW	hp	Btu/h
1 kW	=	1.0	1.341	3.412
1 hp	=	0.7457	1.0	2,544
1 Btu/h	=	2.931×10^{-4}	3.930×10^{-4}	1.0
Notes: 1 Btu raises the temperature of 1 pound of water 1°F 4184 J raises the temperature of 1 kg of water by 1°C				

Table 3.1 Energy and Power Arithmetic Equivalence Factors

Most of the world measures energy in Joules and kilowatt-hours. The Joule is a small amount of energy. One Joule raises the temperature of 1 g of water by 0.239°C and it takes 4184 J to raise the temperature of 1 kg of water by 1°C. Also, 1 kWh = 3,600,000 J = 3,412 Btu. kiloJoules (kJ) or megaJoules (MJ) are more convenient units in most engineering problems.

The most commonly used units in the U.S. are kWh and Btu. Btu is a convenient measure of energy when mass is measured in pounds (lb), as one Btu will raise the temperature of one pound of water by one degree Fahrenheit (°F). Horsepower (hp) is widely used to measure the mechanical power of a motor or engine.

The examples in this book will use both metric units (Joules, Watt, °C) and customary U.S. units (hp, Btu, °F).

EXAMPLE 3.1 HEAT ENERGY AND ELECTRIC ENERGY

The energy released by burning one kilogram of West Virginia bituminous coal (4.5% ash, 1% S) is 33.66×10^6 J. The arithmetic equivalent of electrical energy, using $1 \text{ kWh} = 3.6 \times 10^6$ J, is

$$(33.66 \times 10^6 \text{ J}) (1 \text{ kWh} / 3.6 \times 10^6 \text{ J}) = 9.35 \text{ kWh}$$

The heat energy released by the combustion of one kilogram of coal is arithmetically equivalent to the operation of a 100 W light bulb for 93.5 hours. Unfortunately, the efficiency of converting coal to electrical energy is only about 35%. The amount of electricity delivered from the generator is closer to 3.3 kWh. Some of that is used in the power plant and some (about 6%) is lost in transmission from the plant to the consumer. Therefore, the more realistic yield from 1 kg of coal is enough electrical energy to operate a 100 W light bulb for less than 30 hours.

3.3 ENERGY CONVERSION EFFICIENCY

Energy can be converted from one form to another, for example from the chemical energy in natural gas to thermal energy to make steam to mechanical energy in a turbine-generator to make electrical energy. Each conversion degrades some energy to a form that is not useful. We say that this energy is 'lost' or 'wasted'. (It is not lost. It exists in a different form.)



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Figure 3.1 shows the energy conversion efficiencies of a variety of devices. Newer technologies, such as wind turbines, photovoltaic solar cells, etc., are well established and will continue to become more efficient in the future.

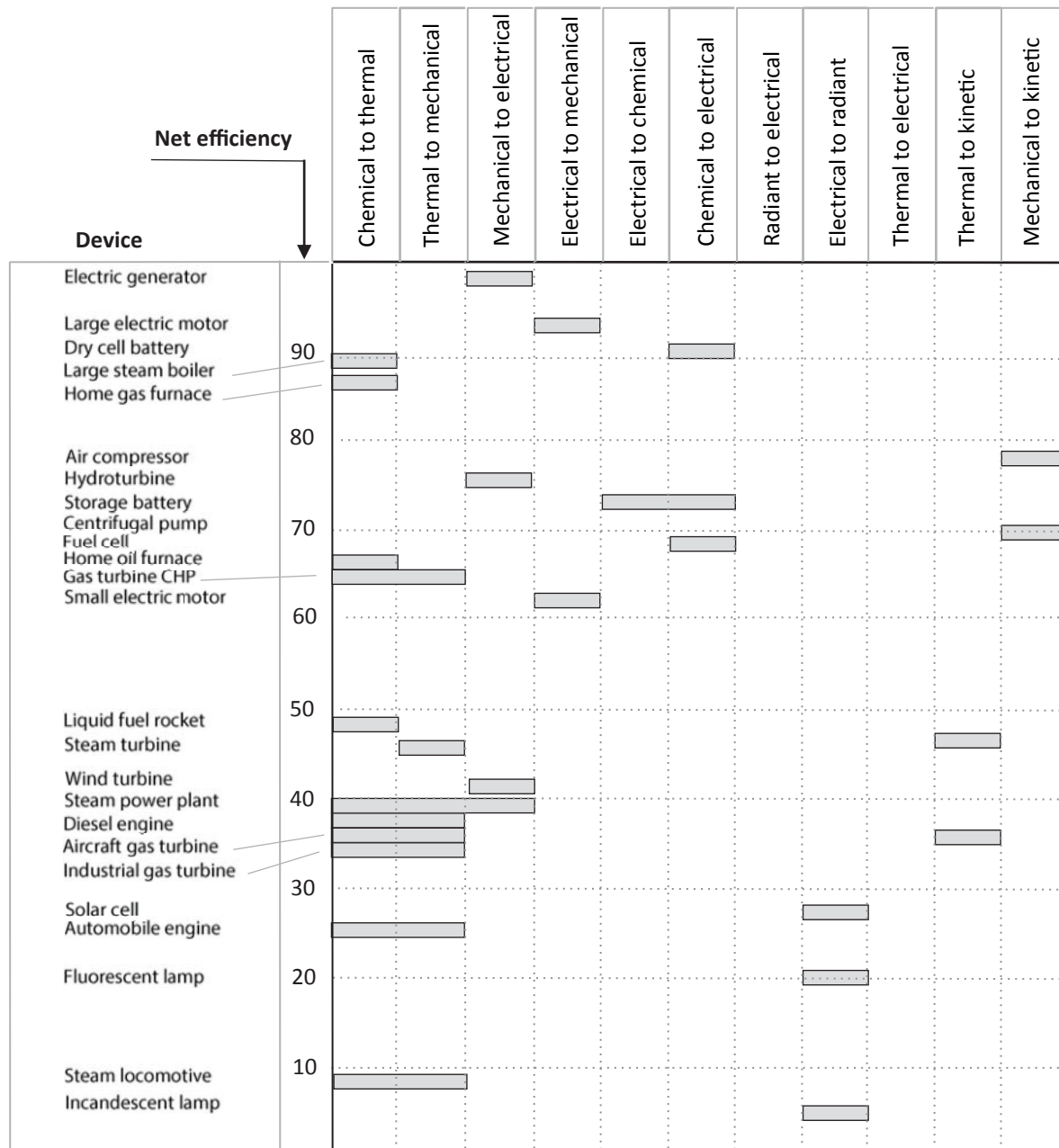


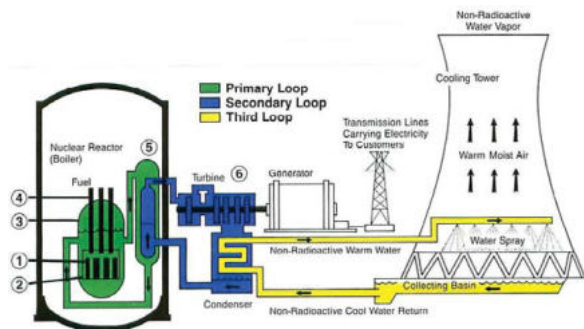
Figure 3.1 Efficiency of energy conversion runs from less than 10 percent to 99 percent for large generators. The efficiency of each device has a wider range than shown in the graph, which is useful to rank the devices and not to precisely evaluate their performance.



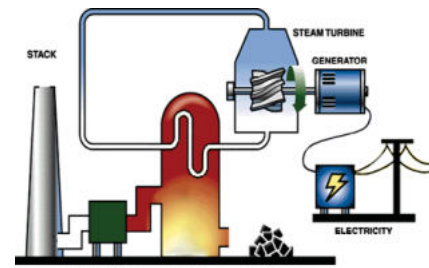
Hydroturbine



Gas turbine (courtesy of Siemens)



Nuclear power plant



Coal-fired steam power plant

Figure 3.2 Four traditional methods of converting energy into electricity: Hydro-turbine, Gas turbine, Nuclear energy, Coal-fired steam power plant.

Figure 3.2 shows the four traditional methods of producing electricity. A *hydro-turbine* converts about 75 percent of the potential energy of the passing water into electricity.

A *fossil fuel plant* converts only 35 - 40% percent of the heat generated by burning fossil fuel is into electrical energy. The conversion of chemical energy in coal to thermal energy in a large boiler is 88% efficient; the conversion of thermal energy into mechanical energy in a steam plant is about 45% efficient, and the conversion of mechanical energy into electricity by a generator is 98% efficient. The complete conversion of chemical energy to electric energy has an efficiency of $(0.88)(0.45)(0.98) = 0.388$, or 39%. About two-thirds of the chemical energy stored in the coal is released to the environment, primarily as hot gases and cooling water. The laws of thermodynamics indicate that it is impossible to do much better than this.

Nuclear power is a low-carbon generation method. A *nuclear power plant* generates electricity from steam that is produced by the controlled fission of Uranium 235. Nuclear reactors have an extra layer of protection against leaks that is provided by the heat exchanger between the reactor and the steam-driven turbine. The cooling water for steam condensation does not contact the cooling fluid that has passed through the reactor.

As of 2018 there are 449 nuclear plants in the world with a combined capacity of 394 GW (Wikipedia). There are 58 under construction and 154 planned. Over 400 more have been proposed. The new plants are mostly in Asia. They produce 11% of the world's electricity,

somewhat less than the 16% from hydro-electric stations. In 2016, 61 nuclear plants in the U.S. produced 20% of the electricity. Sixteen countries get at least 25% of their electricity from nuclear power. France is at about 75%.

An attractive feature of nuclear power is that no carbon dioxide is released by nuclear fission. The arguments against new nuclear plants include nuclear waste disposal (from ore processing and disposal of spent fuel rods) and possibility that a failure could release radiation. The new plants are generation III designs, which are entirely different from earlier designs.

Most experts are skeptical that other carbon-free sources (e.g. wind and solar) can replace the nuclear capacity and this creates issues about climate change and meeting the international goals for reduced carbon dioxide emissions.

EXAMPLE 3.2 WATER HEATING EFFICIENCY

Water can be heated by burning natural gas for direct heating or by using natural gas to make electricity to heat a resistance element, as shown in Figure 3.3. Each method will heat 50 gallons of water from 45°F to 160°F. Electrical heating consumes substantially more gas even though the efficiency of the



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electric heater is nearly 100 percent. The amount of energy needed to raise the temperature of the water from 45°F to 160°F is 48,000 Btu. The 48,000 Btu input to the electric heater requires 150,000 Btu in the form of natural gas into the electric power plant. Delivering natural gas directly to the water heater consumes only 77,000 Btu of energy.

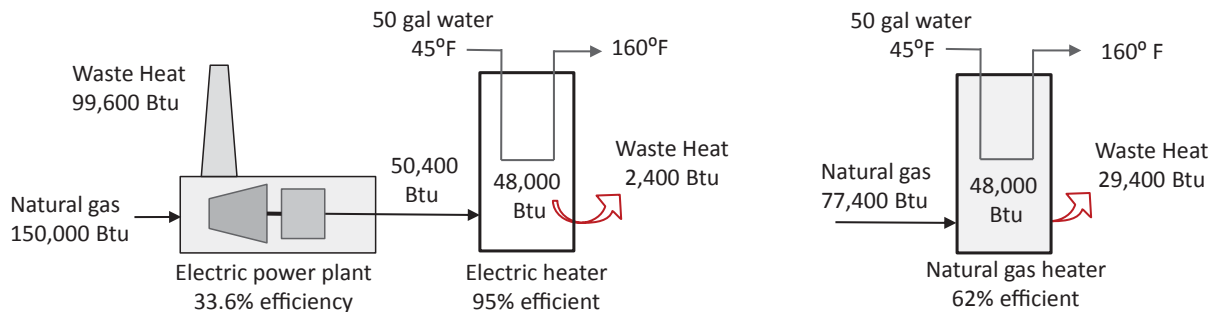


Figure 3.3 Two methods of heating water

3.4 COMBINED HEAT AND POWER (CHP)

Combined heat and power (CHP), also known as co-generation, is an integrated system that converts *waste heat* from the thermal process for generating electricity, usually a gas turbine, into usable heat. Figure 3.4 is the CHP scheme. Figure 3.5 shows a gas turbine that is designed for CHP. The technology is versatile. Micro-turbines can be used in residential homes or commercial buildings, and large scale engines for electric utilities. More than 50% of the electricity in Denmark is produced by CHP and the use is expected to increase in other European countries and in the U.S.

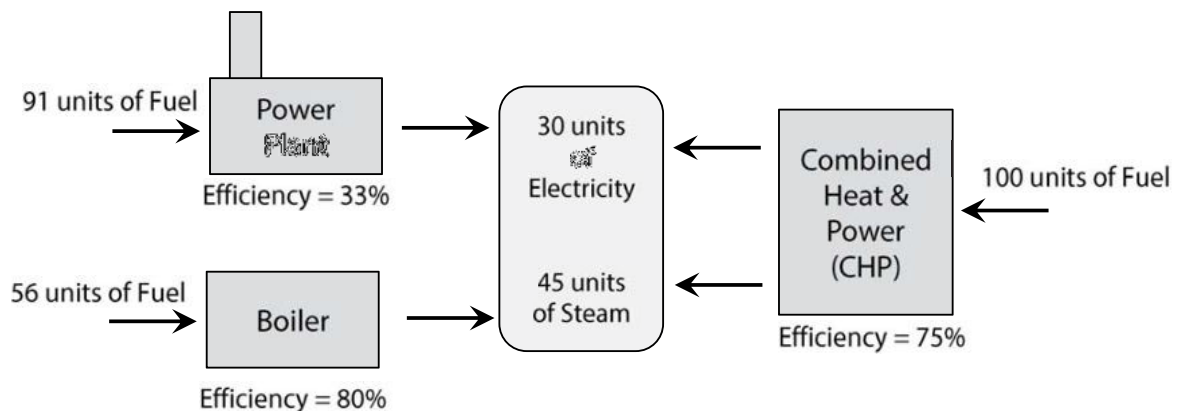


Figure 3.4 Comparison of separate heat and power production and combined heat and power (CHP)

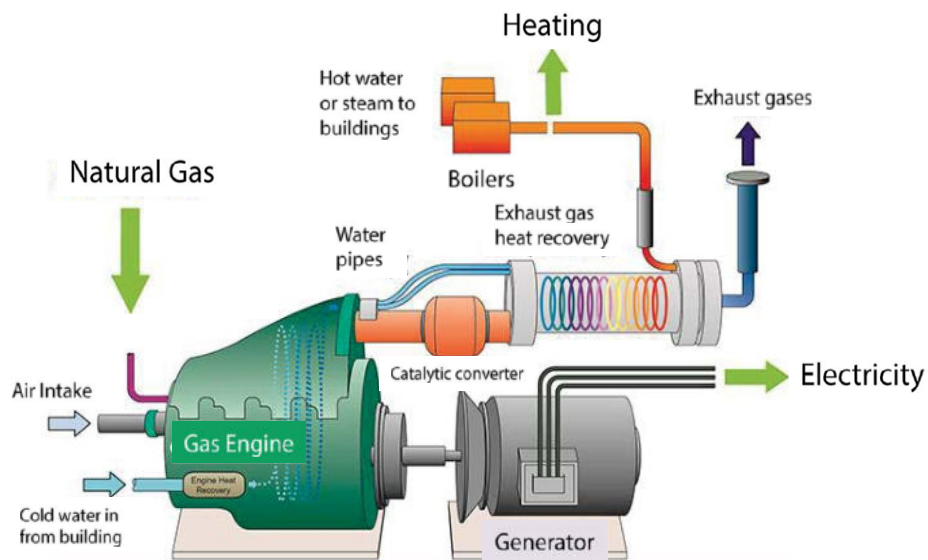


Figure 3.5 A gas turbine designed for combined heat and power production.

Figure 3.6 is the process flow diagram for the combined heat and power (CHP) plant at the University of Wisconsin-Madison. The electrical generating capacity is 150 MW. Waste heat is converted into 500,000 lb/h of steam and 20,000 tons of chilled water. The overall net efficiency rate of 70 percent is double that of conventional plants. Figure 3.7 shows the CHP building with Lake Mendota in the background.

Natural gas and air are fed to the gas turbines. The hot turbine exhaust gas is used to make steam to drive the steam turbine that is coupled with an electric generator. The steam turbine drives electric generators and it also provides hot exhaust to the heat recovery steam generators to make low pressure steam to drive a secondary steam turbine and electrical generator. Steam from the steam turbine can be used to heat university buildings, or it can be exhausted to a condenser. Water circulates between the condenser and a cooling tower. The generated electricity can go to the electrical grid or to the university chilled water system. A second cooling tower services the chiller.

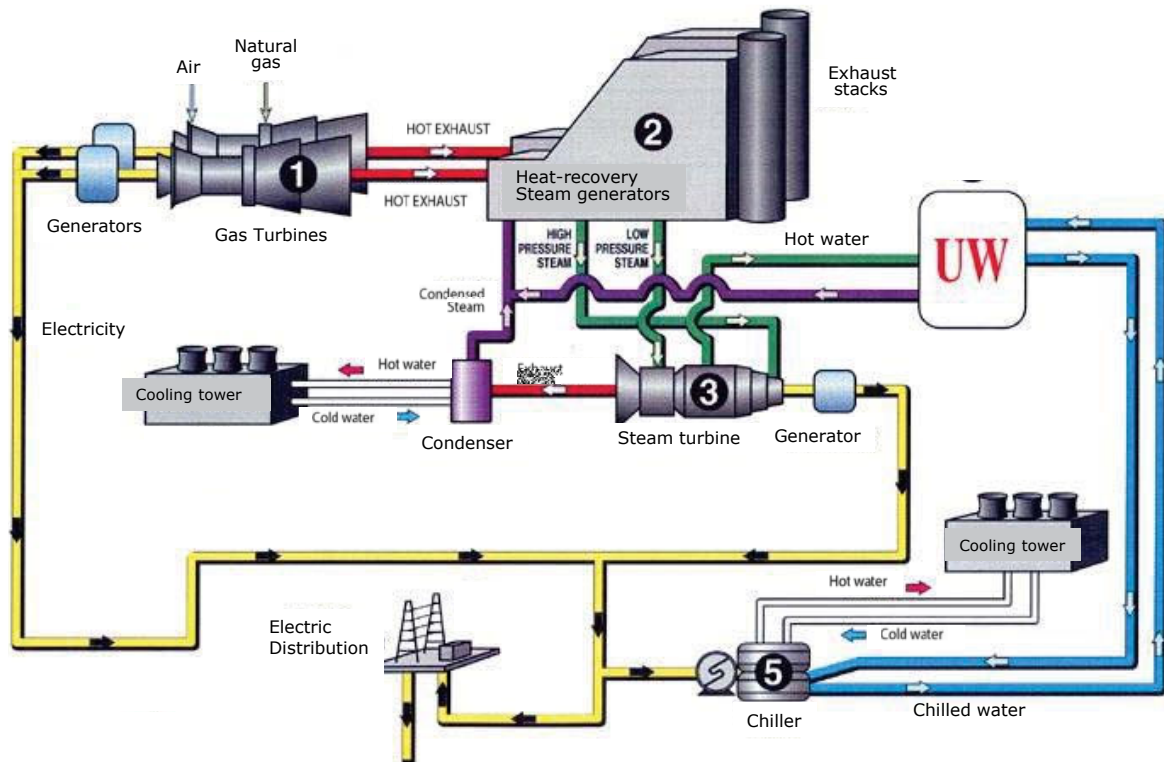


Figure 3.6 Process flow diagram for a modern combined heat and power plant at the University of Wisconsin-Madison (UW-Madison).

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Figure 3.7 Combined heat and power plant at the University of Wisconsin-Madison. The cooling towers are on the left side of the building.

3.5 WIND ENERGY

A *wind farm* is a group of wind turbines in the same location that are used for the production of electrical energy. A *wind turbine*, Figure 3.8, is wind-driven rotor that drives an electric generator.

A turbine can convert about 30% of the kinetic energy of wind to electric energy. In ideal conditions it may produce electricity 75–80% of the time. Output is typically highest in the late afternoon in the hottest part of the year and in mid-mornings and mid-evenings in the coldest part of the year. This makes the power output vary by the season and the time of day.

The largest wind farm (in 2018) is 6,900 MW in China; the largest in the U.S. is 1,320 MW. The available wind energy is the kinetic energy of air in motion that passes through the effective disk area, A , of the turbine blades. For wind velocity v , the kinetic energy is

$$E_{\text{kinetic}} = \frac{1}{2} Mv^2$$

The power is kinetic energy per unit time

$$P_{\text{available}} = \frac{E_{\text{kinetic}}}{t} = \frac{1}{2} mv^2 = \frac{1}{2} (\rho Av)v^2 = \frac{1}{2} \rho Av^3$$

where M = mass of air passing through the effective disk area of the turbine

m = mass *rate* of air flow passing through the effective disk area = (ρAv)

ρ = air density

v = wind velocity

The available wind power is much greater than the rated efficiency of the wind turbine for several reasons. The velocity of the wind leaving the rotor cannot be zero so it is impossible to extract all of the wind energy. According to Betz's Law no turbine can capture more than 59.3% of the kinetic wind energy. This could be achieved only with an ideal propeller operating at a very high rotor tip speed. Output power is a fraction of the available power and given by

$$P_{\text{output}} = C_p P_{\text{available}} = C_p \left(\frac{1}{2} \rho A v^3 \right)$$

where C_p = power factor. The Betz limit is $C_p = 16/27 = 0.59$. In actual installations C_p ranges from about 0.3 to 0.4.

If the incoming air has 100 units of energy (kinetic energy), about 40 units will be transferred into mechanical energy by the rotor blades. Friction in the brake and gearbox reduces the mechanical energy delivered to the generator to 35 units. The electrical output from the generator will be about 33 units and the final output is about 30 units, due to energy loss in voltage conversion and distribution.



Figure 3.8 Installation of wind turbines

EXAMPLE 3.3 WIND ENERGY TO ELECTRICAL ENERGY

A wind turbine has a rotor diameter of 164 m, an overall height of 200 m, and a rated capacity of 10 MW. Calculate the available and output power for a wind velocity of 14 m/s and an air density of $\rho = 1.2 \text{ kg/m}^3$.

The rotor swept area is

$$A = \pi D^2/4 = \pi (164 \text{ m})^2/4 = 21,124 \text{ m}^2$$

The wind power is calculated using

$$P_{\text{available}} = 0.5 \rho A v^3$$

$$P_{\text{available}} = 0.5 (1.2 \text{ kg/m}^3) (21,124 \text{ m}^2) (14 \text{ m/s})^3 = 34.8 \text{ MW}$$

The turbine can deliver about 30-35% of this total wind power as electricity. Thus for a power factor, $C_p = 0.30$, the output power is

$$P_{\text{output}} = C_p P_{\text{available}} = 0.30 (34.8 \text{ MW}) = 10.4 \text{ MW}$$

hence the rated capacity of 10 MW.

Figure 3.9 shows the wind power output expected for turbines of 25% and 40% efficiency. Wind speeds of 15 to 26 m/s produce the maximum efficiency. Below 10 m/s power output is negligible.

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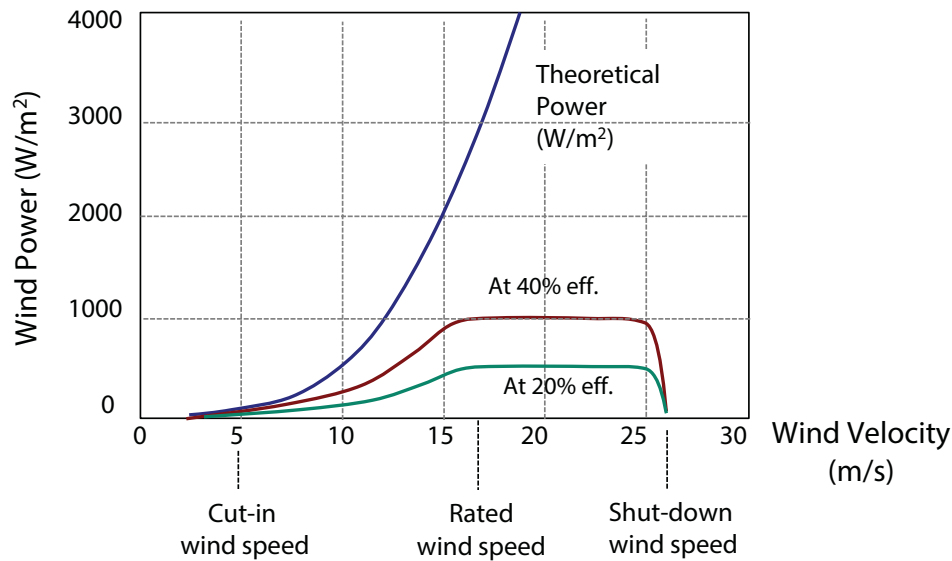


Figure 3.9 Power rating curve for a wind turbine.

3.6 SOLAR ENERGY

Insolation is the rate of *solar energy* arriving on a specific flat surface perpendicular to the line of the sun. The average rate of solar energy arriving at the outer edge of the earth's atmosphere, before any losses is 1.366 kW/m^2 (429 Btu/h-ft^2). At sea level, the maximum possible insolation is about 70% of the incident value, or 0.956 kW/m^2 ($320 \text{ Btu/ft}^2\text{-h}$). An actual solar collector will deliver a fraction of this, say 10-30%, depending on the technology used, the geographic location of the collector, and local conditions such as shade and snow cover. The best states for solar energy are California, Hawaii, Arizona, Maryland, and Delaware. The last two may be surprising but electric energy is expensive and the states have policies that encourage renewable energy (IRENA 2015, 2016).

Solar heating of water is relatively simple. A solar absorber, usually a flat black surface with high absorbance, receives the radiant energy from the sun and converts it to heat. Water is warmed as it passes through the absorber. At one time the largest market for solar heaters in the U.S. was to heat swimming pools in California; see Figure 3.10. Their use for residential heating, Figure 3.11, is becoming more common.

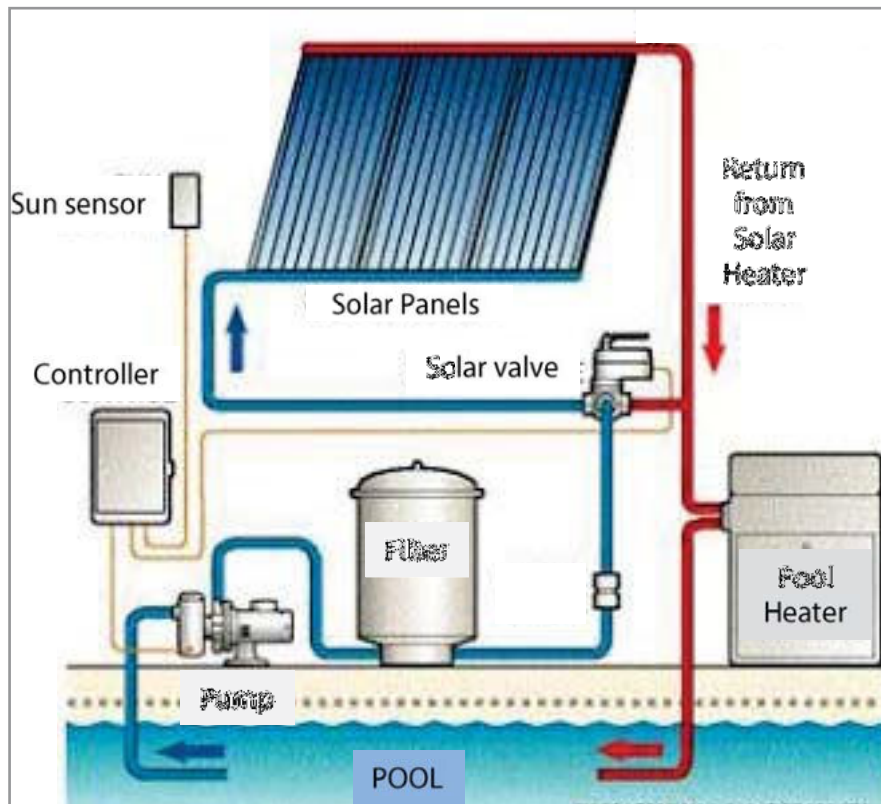


Figure 3.10 Solar heating for a swimming pool



Figure 3.11 Solar panels on a residence can heat air or water, or produce electricity.

Solar electric or *photovoltaic cells* convert sunlight directly into electricity. Electricity can be imported or exported to the grid as required by the demand and the available solar energy. A one-kilowatt (1 kW) solar electric system will generate one kilowatt hour (1 kWh) of electricity after exposure to one hour of sunlight. In Wisconsin, an unshaded 1-kW panel, facing south and inclined between 25° and 50°, will generate about 1,200 kWh/y. A 1-kW pole-mounted tracking panel that constantly faces the sun will generate about 1,600 kWh/y. Figure 3.12 shows the Desert Sunlight Solar Farm (Mojave Desert, CA).



Figure 3.12 Desert Sunlight Solar Farm (Mojave Desert, CA) – 8,000,000 solar panels power 160,000 California homes.







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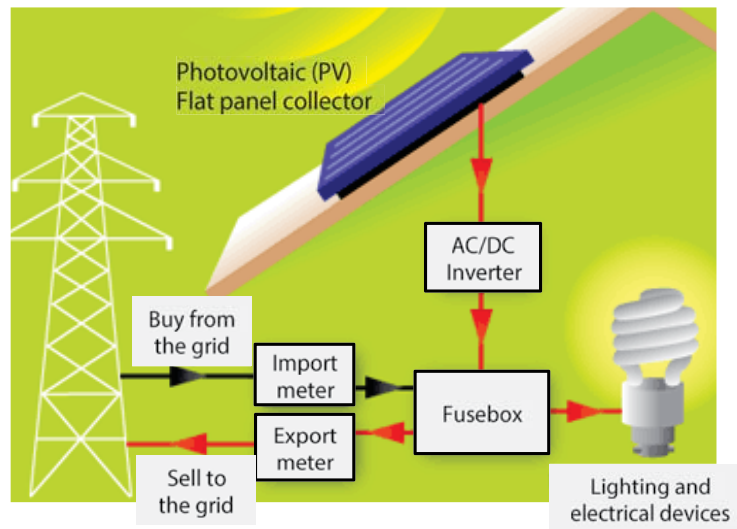


Figure 3.13 Household system for photovoltaic (PV) conversion of solar energy to electricity

The *Watt-peak* (Wp) or *kilowatt peak* (kWp) is a standardized measure that excludes solar conditions. This must be translated into a kWh value that takes account of the solar conditions at the particular location of the solar collector. As a rough idea, a 1 kWp system will produce 1,800 kWh/year in Southern California and 850 kWh/year in Northern Germany. The average Wisconsin home consumes 8,500 kWh of electricity per year, which would require a 10 kWp system.

EXAMPLE 3.4 PHOTOVOLTAIC CONVERSION OF SOLAR ENERGY TO ELECTRICAL ENERGY

Solar electric power, as shown in Figure 3.13, will be installed in house that receives an average of 4 hours of sun per day to provide an output of 1,500 Watt-h/day (Wh/d). The wattage output is 80% of the collected solar energy, so the collection requirement is $(1,500 \text{ Wh/d})/(0.8) = 1,825 \text{ Wh/d}$. Divide the daily power requirement by the average of 4 h/d of sun to get the required system output of $(1,825 \text{ Wh})/(4 \text{ h}) = 470 \text{ W}$ (rounded up). The proposed installation is four 120 W panels or eight 60 W panels.

The maps in Figure 3.14 and Figure 3.15 show the ground solar energy on a clear day with a horizontal flat-plate solar collector. The units are watt hours per square meter per day (Wh/m²d) and kilowatt hours per square meter per year (kWh/m²y), respectively, but this is solar energy received and not electricity produced. Also, solar panel are installed at an angle to increase energy collection, and, of course, not all days are clear and sunny.

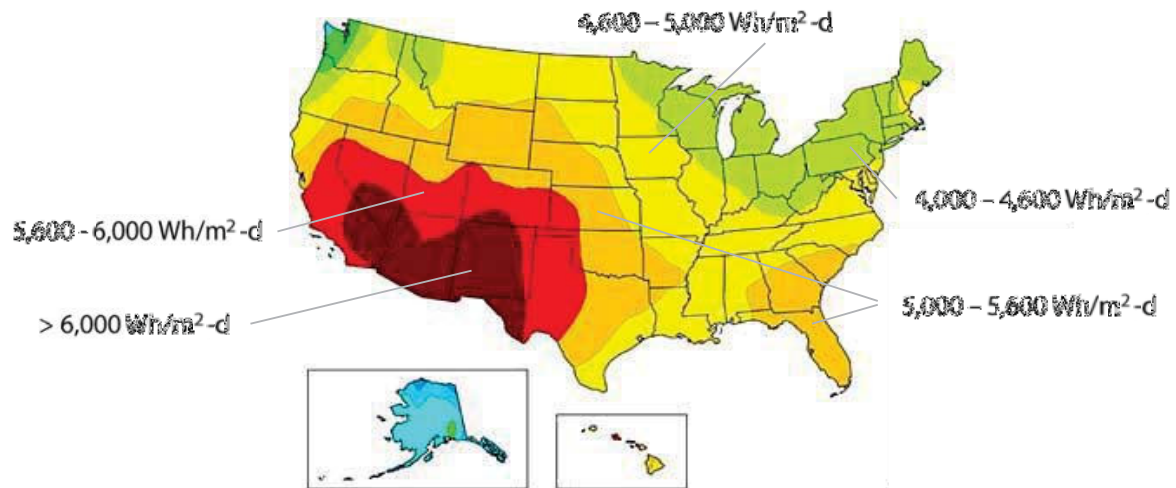


Figure 3.14 Solar energy available on a clear day with a horizontal flat-plate collector, measured in $\text{Wh/m}^2\text{d}$. (Source NASA)

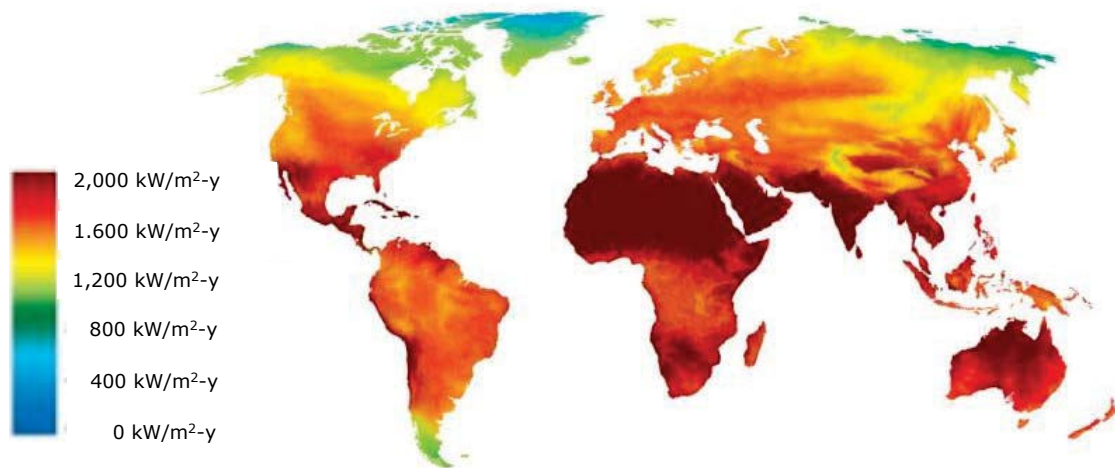


Figure 3.15 Map of world solar energy potential (incident solar radiation or insolation), measured in $\text{kWh/m}^2\text{-y}$. (Source: NASA 2008)

3.7 THE FUTURE OF RENEWABLE ENERGY

Generating economical power is a proven reality. Hawaii will soon eliminate fossil fuels and use only geothermal, solar, and wind energy. California now requires new houses to have solar energy.

The last 10 years have seen the falling costs for wind and solar energy and corresponding increases in installed capacity. Installed solar energy capacity is more than 500,000 MW in 2018 compared with 10,000 MW in 2008. Installed wind energy was 85,000 MW at the end of 2017, up from 25,000 MW in 2008. Wind energy is routinely purchased in bulk for two cents per kWh (Scientific American, Aug. 2017). Solar energy prices have been reported for less than five cents per kWh.

The next challenge is energy storage. Wind and solar are intermittent sources – here one minute and gone the next. They are not always available when they are needed and, at times, they are not needed when they are here.

Deep cycle batteries provide energy storage for solar, wind and other renewable energy systems. Different from a car battery, a deep cycle battery is capable of surviving prolonged, repeated and deep discharges which are typical in renewable energy systems that are off grid (disconnected from the electric utility company). Lead and lithium ion storage batteries are available for household use; lithium ion batteries are lighter, more compact and have a longer life (about 10 years). The Tesla home battery can provide seven days on continuous power.

Large storage batteries are being developed. Tesla installed a 100 MW battery set at an Australian wind farm. The New York Times (August 3, 2018) reported a proposal by Los Angeles Water and Power to integrate solar and wind energy with a pumped storage project at Lake Mead and Boulder Dam. Solar and wind energy would be used to pump water from below the dam, having been once used to generate hydroelectric power, 20 miles back upstream to be stored in Lake Mead until more hydroelectric energy is needed. Lake Mead in essence becomes a storage battery for wind and solar energy.

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3.8 CONCLUSION

This chapter has introduced the concept of the energy balance, the units commonly used to measure energy and power, the efficiency of converting energy from one form to another, and a few ways of producing electricity.

The law of conservation of energy is analogous to the law of conservation of mass. The energy balance is analogous to the material balance. Furthermore, making an energy balance requires knowledge of the material balance in order to account for energy carried with materials as they move through a process or system. Understanding the conservation of energy guides design decisions that are effective in energy conservation.

Solar and wind energy production is growing as markets expand, costs come down, and as the unsustainability of burning carbon-based fossil becomes more and more clear. The remaining problem is that the sun may not shine and the wind may not blow when the power is needed. Lithium batteries to store electricity until it is needed to meet peak demands are being developed, but at this time there are questions about their useful life. A recent interesting news article (New York Times, 29 July 2018) describes a proposal put forth by the Los Angeles Department of Water and Power to use solar energy at night to pump water upstream to Lake Mead and the Hoover Dam. The value of electricity at peak demand is about three times more than at off-peak hours. The water would be put through the hydroturbines at Hoover Dam to meet peak demands.

4 THE ENERGY BALANCE AND ENTHALPY

A method is needed to quantify the energy that is associated with the flow of a mass of material. Enthalpy is a measure of the amount of energy stored in a material. This energy is stored in chemical bonds that hold a molecule together, by the internal forces that hold solids and liquids as a cohesive mass, by the random motion of molecules in a gas, and by other internal phenomena. The enthalpy changes during processing, as the temperature and form of material changes, and by computing the enthalpy change we can estimate the amount of energy consumed or generated.

Because Btu and °F are still widely used in the U.S., in contrast to the rest of the world, these units are used in some examples, while others will use metric units, and some examples will give both units. This should not confuse understanding of the underlying concepts and general methods for making the energy balance.

4.1 ENTHALPY

It is not correct to speak of the heat content of a substance because from the thermodynamic point of view it is heat only when it is being transferred. Something is needed which is a property of the material and this is enthalpy.

Enthalpy (H) is a thermodynamic quantity that is used to describe the change in heat energy when a material changes from one temperature to another. It cannot be detected directly and it has no absolute value. Only changes in enthalpy can be measured, and the change is calculated relative to some initial condition or standard reference condition.

Enthalpy increases or decreases depending on the direction of the temperature change. If the enthalpy of a substance is known at different temperatures it is simple to calculate the heat required to bring the substance from one condition to another.

The change in enthalpy is referenced to an arbitrarily chosen standard state or reference state. At the standard state the enthalpy is assumed to be zero. For air, 20°C (68°F) or 60°F and 1 atm pressure is commonly used as the standard state. For steam the standard state is liquid water at 0°C (32°F). Obviously the choice of a standard state affects the number assigned to the enthalpy, but it does not affect the change in enthalpy for a given process.

In a simple batch operation in which a mass of material is being changed from state 1 to state 2, the difference between the heat energy added to the material, and the mechanical work done on the material, is

$$Q - W = m(H_2 - H_1)$$

where Q = heat energy (kJ, Btu)

W = mechanical work (kJ, Btu)

m = mass of material (kg, lb)

H_1 and H_2 = enthalpy of the material at states 1 and 2 (kJ/kg, Btu/lb, or kWh)

The energy balance is

$$(\text{Total enthalpy in}) + (\text{Heat in}) = (\text{Total enthalpy out}) + (\text{Heat out})$$

Each term must have the same units (e.g. kJ, Btu, or kW).

The accounting for energy flow rates (kJ/m, kJ/h, Btu/h, or kW) is

$$\left(\begin{array}{c} \text{Rate of} \\ \text{enthalpy in} \end{array} \right) + \left(\begin{array}{c} \text{Rate of} \\ \text{heat in} \end{array} \right) = \left(\begin{array}{c} \text{Rate of} \\ \text{enthalpy out} \end{array} \right) + \left(\begin{array}{c} \text{Rate of} \\ \text{heat out} \end{array} \right)$$

where the units are kJ/min, kJ/h, BTU/h, or kW.



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Sources: Keuzegids Master ranking 2013; Elsevier 'Beste Studies' ranking 2012; Financial Times Global Masters in Management ranking 2012

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The enthalpy for liquids and solids depends on the temperature and mass of the material. The value of H for gases or vapor depends on pressure, temperature, and mass. This dependency (for gases) is a subject for a more advanced course.

Specific enthalpy is the enthalpy per unit mass, with units of kJ/kg, kJ/g-mol, Btu/lb, or Btu/lb-mol. The customary symbol for specific enthalpy is h , but we shall use H to avoid confusion since h has been used for hours. The context and the units, which are always given, will make it clear.

4.2 SPECIFIC HEAT

Sensible heat effects are changes in temperature that can be sensed, say, by the touch of the hand, and for this reason the energy involved with temperature change is called sensible heat.

The amount of sensible heat, Q , that must be added (or removed) to change mass m from temperature T_1 to temperature T_2 is

$$Q = c_p m (T_2 - T_1)$$

where c_p is the specific heat (or heat capacity) of the substance.

The *specific heat* of a substance is the amount of heat per unit mass required to raise its temperature by one degree. It is the basis of the definition of the Btu (British thermal unit) and the calorie. In the SI system of units, the calorie is rarely used, and has been largely replaced by the Joule.

1 Btu will raise the temperature of 1 lb of water by 1°F

1 calorie will raise the temperature 1 g of water by 1°C

4.184 kJ will raise the temperature of 1 kg of water by 1°C.

Thus, the specific heat of water is 1 Btu/lb°F, 1 cal/g°C, or 4.184 kJ/kg°C.

Derived quantities that specify heat capacity as an *intensive* property (independent of the size of the sample) are the *molar heat capacity* (heat capacity of a mole of pure substance) and the *specific heat*, which is the heat capacity per unit mass of material. The *specific heat* is sometimes called the *heat capacity*, c_p , and is measured in units of J/g°C, kJ/kg°C, or Btu/lb°F.

When substances melt, evaporate, or change from one crystalline phase to another, there is a heat effect known as the *heat of fusion* or *heat of transition*. Sometimes this heat effect is called *latent heat* to distinguish it from *sensible heat*. Latent heat is that required to cause a change in state with no change in temperature.

The heat effect when a substance changes from a solid to a liquid or from a liquid to a vapor is the *latent heat of fusion* or *latent heat of evaporation*. The latent heats of fusion and evaporation are always positive. For pure substances this heat effect takes place at constant temperature; with mixtures the temperature changes. In calculating the enthalpy of a substance, the latent heat must be added if there is a change of phase.

For example, water at 100°C (212°F) can be changed to steam by adding 2,257 kJ/kg (970 Btu/lb) of water. If this is done at constant pressure, the temperature of the steam is 100°C. Adding an amount of heat equal to the heat of vaporization (2,257 kJ/kg or 970 Btu/lb) changes the liquid to vapor but does not change the temperature. Likewise, changing 100°C steam to 100°C water requires the removal of 2,257 kJ/kg (970 Btu/lb) of heat.

Because of the importance of steam for heating, its enthalpy has been calculated for various temperatures and pressures, based on liquid water at 0°C (32°F). Tables of the thermodynamic properties of steam list vapor pressure (usually absolute pressure), specific volume (ft³/lb or m³/kg) of liquid and vapor, enthalpy (Btu/lb or kJ/kg) of liquid and vapor, and entropy (Btu/lb°F or kJ/kg°C) of liquid and vapor. The boiling point of a liquid depends on the pressure. The heat of vaporization decreases as the pressure increases and therefore as the temperature increases. Appendix 5 gives some data for saturated water and steam.

Table 4.1 gives the specific heat of air and some common gases at different temperatures. The heat capacity of gases varies only slightly with temperature, but differs with pressure because the density of the gas changes. The specific heat of air is about 1.006 kJ/kg°C (0.24 Btu/lb°F) from 0°C – 300°C (0°F - 500°F). Several other gases, with the notable exception of hydrogen, have a similar value. Water vapor has a higher specific heat, about 1.88 kJ/kg°C (0.45 Btu/lb°F) over a wide range. All these values will increase if the gases are pressurized.

Temp. (°C)	Dry Air	Nitrogen (N ₂)	Oxygen (O ₂)	Methane (CH ₄)	Carbon dioxide (CO ₂)	Hydrogen (H ₂)
0	1.006	1.039	0.915	2.165	0.815	14.195
100	1.006	1.042	0.934	2.448	0.914	14.448
200	1.006	1.052	0.963	2.807	0.993	14.533
300	1.006	1.069	0.995	3.175	1.057	14.581
400	1.014	1.091	1.024	3.529	1.110	14.662
500	1.030	1.115	1.048	3.86	1.155	14.779
Conversion Factor: 1 Btu/lb°F = 4.187 kJ/kg						

Table 4.1 Specific heat, c_p , (kJ/kg°C) of gases at constant pressure (1 atm) for various temperatures.

Table 4.2 gives the specific heat and heat of vaporization of selected liquids. The specific heat for liquid organic chemicals is about 2.1 kJ/kg°C (0.5 Btu/lb°F). Water is the exceptional liquid, with a much higher specific heat.

Substance	Specific heat, c_p		Boiling point		Heat of vaporization	
	(Btu/lb°F)	(kJ/kg°C)	(°C)	(°F)	(Btu/lb)	(kJ/kg)
Acetic acid	0.522	2.186				
Acetone	0.514	2.152	56.2	133.2	224.0	521
Benzene	0.419	1.754	80.1	176.2	169.5	394.3
n-Butanol	0.582	2.437	116.8	242.2	254.4	591.7
Carbon tetrachloride	0.201	0.842	76.8	170.2	83.6	194.5
Ethyl alcohol	0.68	2.85	78.3	172.9	367.8	855.5
Toluene	0.44	1.84	110.6	231.1	156.3	363.6
Water	1.00	4.187	100	212	970.3	2,257
Conversions: 1 Btu/lb°F = 4.187 kJ/kg°C ; 1 Btu/lb = 2.326 kJ/kg						

Table 4.2 Specific heat and heat of vaporization for some liquids



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EXAMPLE 4.1 MAKING STEAM

Five thousand pounds per hour of water at 100°F is heated to make steam at 212°F and 1 atmosphere pressure. The amount of heat required will be calculated in two ways.

Method 1. Use the specific heat (1 Btu/lb°F) and the latent heat of vaporization (970 Btu/lb)

Basis: $m = 5,000 \text{ lb}$

Heating the water from 100°F to 212°F requires

$$Q = mc_p(T_2 - T_1) = (5000 \text{ lb/h}) (1 \text{ Btu/lb°F})(212^\circ\text{F} - 100^\circ\text{F}) = 560,000 \text{ Btu/h}$$

Changing the 212°F water to 212°F steam requires

$$Q = mH_v = (5000 \text{ lb/h}) (970 \text{ Btu/lb}) = 4,850,000 \text{ Btu/h}$$

The total heat required is

$$560,000 \text{ Btu/h} + 4,850,000 \text{ Btu/h} = 5,410,000 \text{ Btu/h}$$

Method 2. Use enthalpy.

The amount of heat required is equal to the change in enthalpy of the incoming water and the exiting steam.

From Appendix 5:

$$\text{Enthalpy of liquid water at } 100^\circ\text{F and } 1 \text{ atm} \quad H_1 = 67.97 \text{ Btu/lb}$$

$$\text{Enthalpy of water vapor (steam) at } 212^\circ\text{F and } 1 \text{ atm} \quad H_2 = 1,150.4 \text{ Btu/lb}$$

Change in enthalpy and the total amount of heat required to make the steam

$$Q = m(H_2 - H_1) = (5000 \text{ lb/h}) (1,150.4 \text{ Btu/lb} - 67.97 \text{ Btu/lb}) = 5,412,000 \text{ Btu/h}$$

This answer is slightly different than the 5,410,000 Btu/h calculated by Method 1 because the enthalpy values from the steam tables are more precise than we calculate using a specific heat of 1.00 Btu/lb°F and a latent heat of 970 Btu/lb. The difference is not important in our understanding of the basic principles of making an energy balance.

EXAMPLE 4.2 ENTHALPY OF HUMID AIR

Calculate the enthalpy of moist air at saturation and at 50% saturation. (Saturation is 100% relative humidity and 50% saturation is 50% relative humidity.)

Air saturated with water vapor air at 25°C, has a mass fraction water vapor of $x = 0.0203 \text{ kg H}_2\text{O/kg}$ air. The enthalpy of dry air at $T = 0^\circ\text{C}$ is 0.0 kJ/kg.

H for saturated air at 25°C

$$H_{Sat, 25^\circ\text{C}} = c_{p, \text{air}} \Delta T + x \left(c_{p, \text{air}} \Delta T + H_{\text{water}, 25^\circ\text{C}} \right)$$

$$\begin{aligned} H_{Sat, 25^\circ\text{C}} &= \left(1.006 \frac{\text{kJ}}{\text{kg}^\circ\text{C}} \right) (25^\circ\text{C}) + \left(0.0203 \frac{\text{kg H}_2\text{O}}{\text{kg air}} \right) \left[\left(1.84 \frac{\text{kJ}}{\text{kg}^\circ\text{C}} \right) (25^\circ\text{C}) + 2,442 \frac{\text{kJ}}{\text{kg}} \right] \\ &= 25.15 + 0.93 + 49.57 = 75.65 \frac{\text{kJ}}{\text{kg}} \end{aligned}$$

The first term in the brackets is the sensible heat due to heating water vapor, and is often neglected.

At 50% saturation, $x = 0.01 \text{ kg H}_2\text{O/kg air}$, and the enthalpy is

$$\begin{aligned} H_{RH=50\%, 25^\circ\text{C}} &= \left(1.006 \frac{\text{kJ}}{\text{kg}^\circ\text{C}} \right) (25^\circ\text{C}) + \left(0.01 \frac{\text{kg H}_2\text{O}}{\text{kg air}} \right) \left[\left(1.84 \frac{\text{kJ}}{\text{kg}^\circ\text{C}} \right) (25^\circ\text{C}) + 2,442 \frac{\text{kJ}}{\text{kg}} \right] \\ &= 25.15 + 0.46 + 24.42 = 50.03 \frac{\text{kJ}}{\text{kg}} \end{aligned}$$

EXAMPLE 4.3 SPECIFIC HEAT OF A GAS MIXTURE

Gas from an anaerobic sludge digester is 70% methane (CH_4) and 30% carbon dioxide (CO_2), ignoring traces of other gases such as hydrogen and sulfur dioxide. These are volume percentages. The volume fraction of a gas equals its mole fraction, so a volume fraction of 0.70 for methane means 0.7 moles N_2 per mol of gas mixture.

$$\text{Specific heats (kJ/kg}^\circ\text{C)} \quad \text{CH}_4 = 2.165 \quad \text{CO}_2 = 0.815$$

The specific heat of the mixture is the weighted mass average of specific heats of the two gases.

Mass of CH_4 in 1 kg-mol of gas mixture

$$= (0.7 \text{ kg-mol CH}_4)(16 \text{ kg CH}_4/\text{kg-mol}) = 11.2 \text{ kg CH}_4$$

Mass of CO_2 in 1 kg-mol of gas mixture

$$= (0.3 \text{ mol CO}_2)(44 \text{ kg CO}_2/\text{kg-mol}) = 13.2 \text{ kg CO}_2$$

Total mass of 1 kg-mol of gas mixture = 24.4 kg

$$\text{Mass fraction of CH}_4 = (11.2 \text{ kg CH}_4)/(24.4 \text{ kg mixture}) = 0.459$$

$$\text{Mass fraction of CO}_2 = (13.2 \text{ kg CH}_4)/(24.4 \text{ kg mixture}) = 0.541$$

Specific heat of mixture

$$= (2.165 \text{ kJ/kg}^\circ\text{C})(0.459) + (0.815 \text{ kJ/kg}^\circ\text{C})(0.541)$$

$$= 0.994 \text{ kJ/kg}^\circ\text{C} + 0.441 \text{ kJ/kg}^\circ\text{C} = 1.435 \text{ kJ/kg}^\circ\text{C}$$

4.3 THE ENERGY BALANCE

The energy balance is

$$(\text{Total enthalpy in}) + (\text{Heat in}) = (\text{Total enthalpy out}) + (\text{Heat out})$$

Each term must have the same units (e.g. kJ, Btu, or kW).

The accounting for energy flow rates (kJ/min, kJ/h, Btu/h, or kW) is

$$\left(\text{Rate of total enthalpy in} \right) + \left(\text{Rate of heat in} \right) = \left(\text{Rate of total enthalpy out} \right) + \left(\text{Rate of heat out} \right)$$

EXAMPLE 4.4 HEAT RECOVERY

Steam that has been used in a process or a boiler is condensed and the condensate is used to heat air. The arrangement is shown in Figure 4.1. The cooling water entering the condenser is 10°C. The steam entering the condenser is at 150°C. The system is designed to have 25°C cooling water leave the condenser. The condensate enters the heater at 80°C and leaves at 20°C. Calculate the amount of cooling water, X , that is needed and the amount of air, Y , that can be heated. Ignore any loss of heat from the walls of the condenser, air heater, or piping.



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The enthalpies of water and steam are found in Appendix 4.

Water at 10°C	$H = 42.02 \text{ kJ/kg}$
Water at 20°C	$H = 83.92 \text{ kJ/kg}$
Water at 25°C	$H = 104.84 \text{ kJ/kg}$
Water at 80°C	$H = 334.95 \text{ kJ/kg}$
Steam	$H = 2745.9 \text{ kJ/kg}$

The heat capacity of air, from Table 4.1, is $c_p = 1.006 \text{ kJ/kg}^\circ\text{C}$

Three energy balance equations can be written: an energy balance on the condenser, an energy balance on the heater, and an energy balance for the overall system.

There are two unknowns, the mass of cooling water and the mass of air, so two energy balance equations are needed. The possibilities are:

- Energy balance on the condenser and an energy balance on the heater
- Overall energy balance plus an energy balance on the condenser
- Overall energy balance plus an energy balance on the heater.

The problem will be solved using option (a).

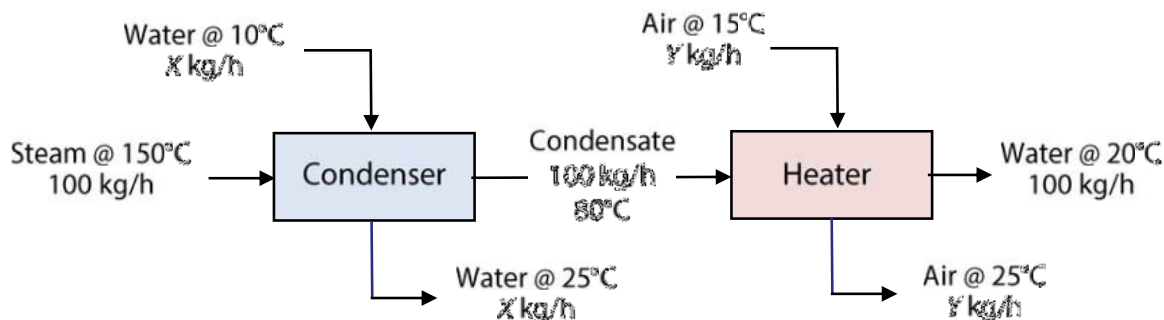


Figure 4.1 Heating air with heat recovered from a steam condenser.

Energy balance on the condenser will give the cooling water flow rate, $X \text{ kg/h}$

Enthalpy absorbed by water = enthalpy given up from steam

$$X(104.84 \text{ kJ/kg} - 42.02 \text{ kJ/kg}) = (100 \text{ kg/h})(2745.9 \text{ kJ/kg} - 334.95 \text{ kJ/kg})$$

$$X = (100 \text{ kg/h})(2411 \text{ kJ/h}) / (62.82 \text{ kJ/h}) = 3,838 \text{ kg/h water}$$

Energy balance on the heater will give the airflow rate, $Y \text{ kg/h}$

Enthalpy absorbed by air = enthalpy given up from condensate

$$Y(1.006 \text{ kJ/kg}^\circ\text{C})(25^\circ\text{C} - 15^\circ\text{C}) = (100 \text{ kg/h})(334.95 \text{ kJ/kg} - 83.92 \text{ kJ/kg})$$

$$Y = (25,103 \text{ kJ/h}) / 10.06 \text{ kJ/kg} = 2,495 \text{ kg/h air}$$

EXAMPLE 4.5 CONDENSING ETHANOL

A flow of 1000 kg/h of ethyl alcohol (ethanol) vapor at 78.3°C is to be condensed and cooled to 20°C. The boiling point of ethanol is 78.3°C so the vapor does not need additional cooling before condensation will occur. The latent heat of vaporization is 855.5 kJ/kg and this is the amount of heat that must be removed to accomplish the phase change from vapor to liquid. The specific heat of liquid ethanol is $c_p = 2.85$ kJ/kg°C. Cooling water is available at 10°C.

Energy balance on ethanol:

$$(1000 \text{ kg/h})(2.85 \text{ kJ/kg } ^\circ\text{C})(78.3^\circ\text{C} - 20^\circ\text{C}) + (1000 \text{ kg/h})(855.5 \text{ kJ/kg}) \\ = 166,155 \text{ kJ/h}$$

Energy balance on water, assuming the water is heated to 65°C. ΔT is $65 - 10 = 55^\circ\text{C}$

$$(X \text{ kg/h})(65 - 10^\circ\text{C})(4.187 \text{ kJ/kg } ^\circ\text{C}) = 166.155 \text{ kJ/h}$$

$$X = (166.155 \text{ kJ/h}) / (55^\circ\text{C} - 10^\circ\text{C})(4.187 \text{ kJ/kg}) = 722 \text{ kg/h}$$

EXAMPLE 4.6 HEATING FOR WASTEWATER SLUDGE

Sludge removed from a thermophilic anaerobic digester that operates at 60°C must be cooled to 45°C so it can be given further treatment in a mesophilic digester that operates at 45°C; see Figure 4.2. The heat exchanger inlet and outlet water temperatures are 20°C and 33°C. The digested sludge flow is 20 m³/h. The sludge enters the heat exchanger at 60°C and leaves at 45°C. Assume equal densities for the sludge and the water. The heat capacities are: $c_p = 4.1$ kJ/kg°C for sludge and $c_p = 4.187$ kJ/kg°C for water. The sludge contains solids that have a heat capacity less than water so the heat capacity of the mixture is less than water.

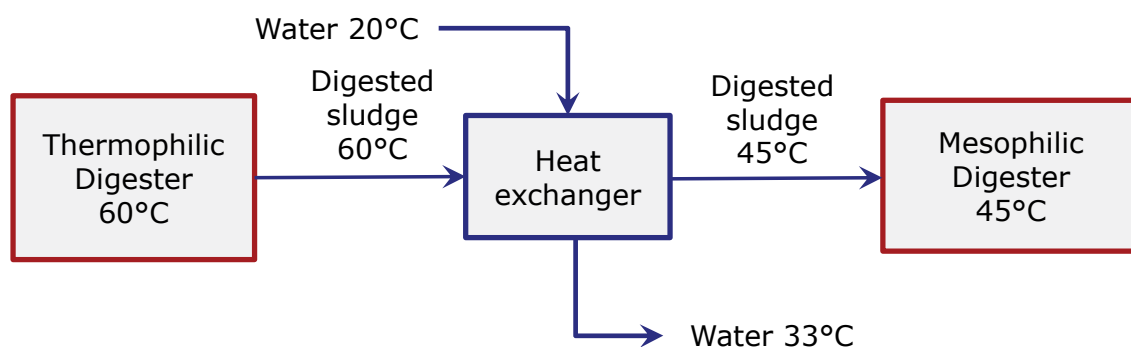


Figure 4.2 Heating water with sludge from a thermophilic digester

Basis: 1 hour of operation = 20 m³/h = 20,000 kg/h sludge

Define m = mass of water passing through the heat exchanger.

Energy balance on heat exchanger

Heat transferred to water = Heat transferred from sludge

$$m(4.187 \text{ kJ/kg}^\circ\text{C})(33^\circ\text{C} - 20^\circ\text{C})$$

$$= (20,000 \text{ kg sludge/h})(4.1 \text{ kJ/kg}^\circ\text{C})(60^\circ\text{C} - 45^\circ\text{C})$$

$$m(54.431) = 1,230,000 \text{ kg/h}$$

$$m = 22,597 \text{ kg/h}$$

4.4 COOLING TOWERS

Industry uses water in the manufacturing process, and for cooling and heating, as shown in Figure 4.3.

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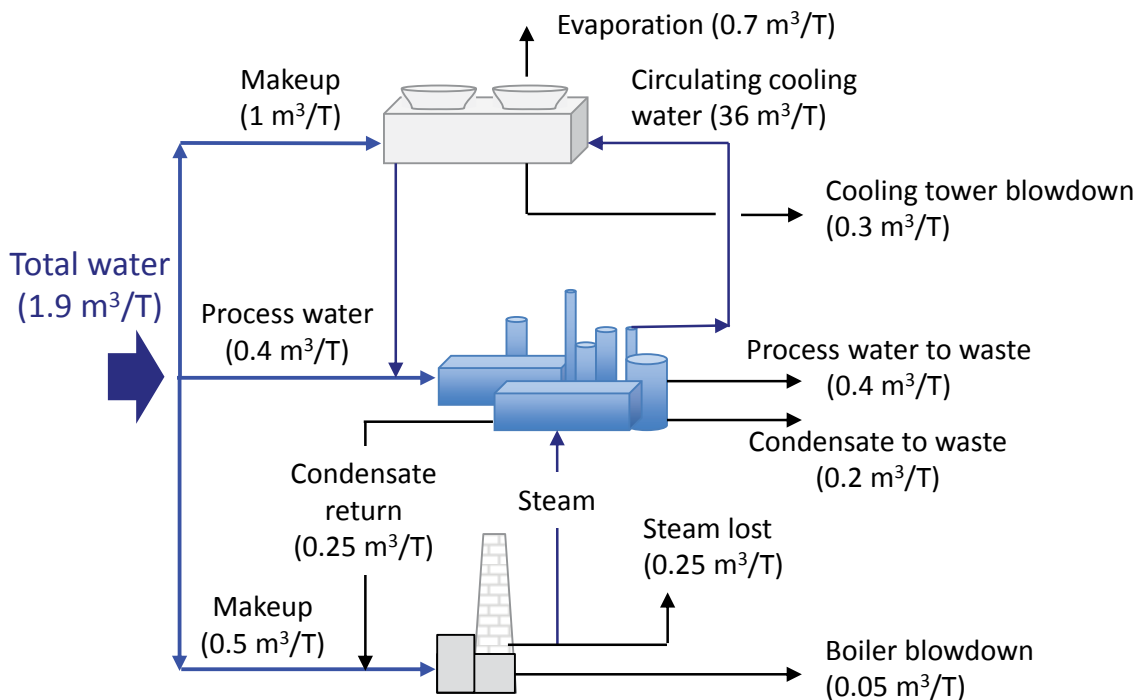


Figure 4.3 The industrial water cycle includes process water, cooling water, and steam for heating.

Cooling towers are recirculating systems. The warm water input is cooled by evaporation to be reused. The evaporated water carries no minerals so the concentration of total dissolved solids (TDS), often called the salt concentration, in the recirculating water increases with each cycle of reuse. The salt concentration is controlled by removing salt-laden circulating water and replacing it with fresh water.

Figure 4.4 defines the variables for cooling tower operation. Cool water, at temperature T_{Cool} , is delivered from the tower from the industry's hot processes and condensers. Warm water, at temperature T_{Warm} , is returned to the cooling tower. Ambient air rises through the tower, either by natural draft or by forced draft using large fans in the tower. This contact causes water to be lost as *drift* (D) and *evaporation* (E). The heat required to evaporate the water is derived from the water itself, so the mass of water evaporated is directly proportional to the amount of heat removed from the recirculating water.

Evaporation removes water and heat, but it does not remove dissolved minerals. *Drift*, the uncontrolled blowing of water droplets from the tower, removes both water and minerals; it is functionally the same as blowdown. A new tower in good condition, with drift eliminators, loses about 0.005% of the recirculating water via drift. The evaporation rate, and the cooling capability of the tower, depend on the air and water temperatures.

Blowdown (BD) is the intentional, controlled removal of mineral-laden water from the cooling system. Blowdown is quantified by the term *cycles of concentration* or simply *cycles*. Cycles are equal to the ratio of make-up water volume to blowdown water volume, assuming that drift and spills are negligible.

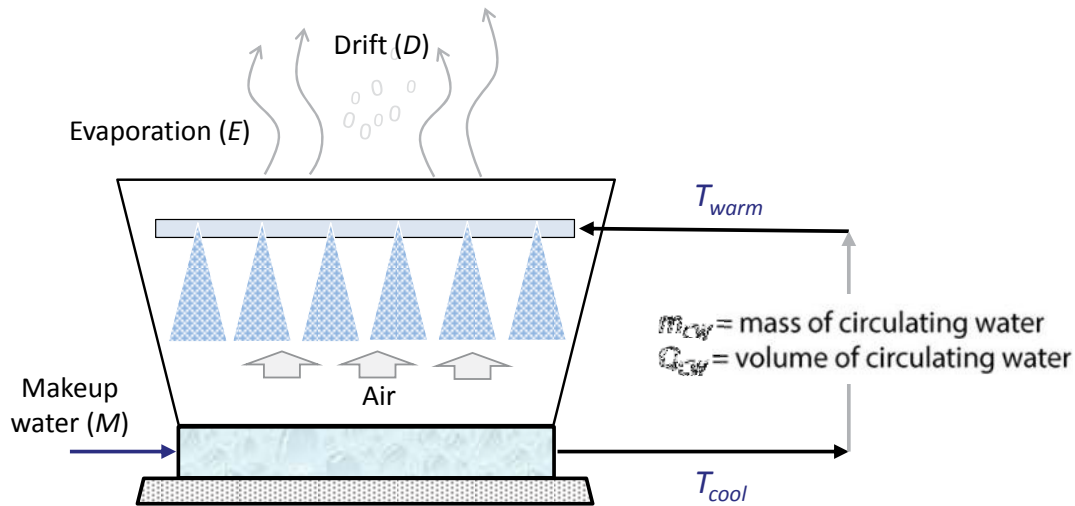


Figure 4.4 The water balance for a fan-induced draft counter-flow cooling tower circuit. Makeup (M) = Blowdown (BD) + Evaporation (E) + Drift (D).

Fresh water makeup (M) is supplied to replace evaporation loss (E), drift loss (D), and blowdown (BD). For convenience, drift is considered as part of the blowdown. The water balance is:

$$M = E + BD$$

Since the evaporated water (E) has no salts, the material balance on dissolved solids is simple. The dissolved solids entering with the makeup water must leave with the blowdown.

$$(M)(\text{TDS}_M) = (BD)(\text{TDS}_{BD})$$

where TDS_{BD} = Blowdown concentration of TDS (ppm or mg/L)

TDS_M = Makeup water concentration of TDS (ppm or mg/L)

The average number of times the water is circulated through the cooling tower is the *cycles of concentration*, or simply *cycles*.

$$\text{Cycles} = \frac{\text{TDS}_{BD}}{\text{TDS}_M} = \frac{M}{BD} = \frac{M}{M - E} = 1 + \frac{E}{BD}$$

If the minerals in the recirculating water were four times as concentrated as in the make-up water, the tower would be running at 4 cycles. Most cooling towers operate in the range of 3 to 7 cycles. This depends on the quality of the makeup water supply. Well water usually has high levels of dissolved solids. On the other hand New York City has a surface rainwater source that is low in minerals and cooling towers in that city are often able to operate at 7 or more cycles.

Dry-bulb temperature (T_{DB}) is the most familiar property of air. When people refer to air temperature, they are normally referring to its dry-bulb temperature. This is the sensible temperature.

The *wet-bulb temperature* (T_{WB}) is the lowest temperature that can be reached by evaporation of water only. Cooling towers depend on achieving water temperatures below the *dry bulb temperature* of the cooling air.

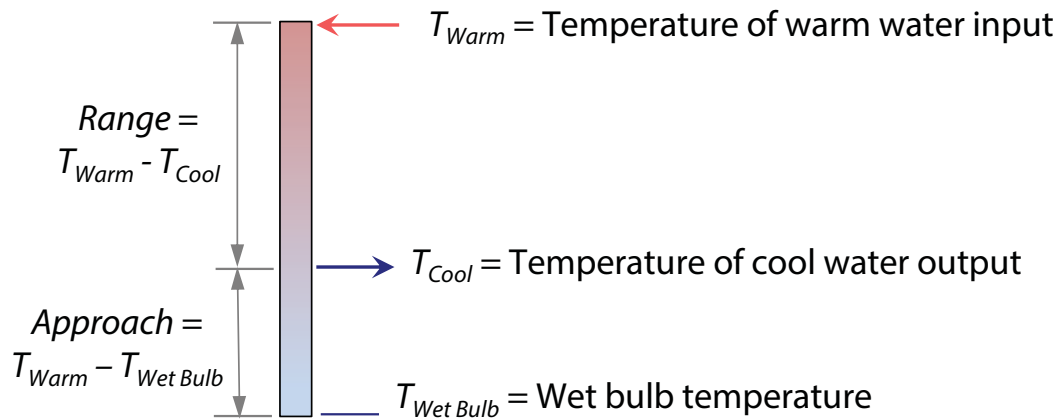


Figure 4.5 Definition of Range and Approach of a cooling tower.

Cooling towers are rated in terms of *range* and *approach*, as shown in Figure 4.5. The *range* is the temperature difference of the warm water entering the cooling tower and the cool water that is leaving. The *approach* is the difference in temperature between the cooled-water temperature

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and the entering-air *wet bulb temperature*. It is impossible for the cooled water temperature to be less than the wet bulb temperature and uneconomical to approach it too closely.

$$\text{Range} = T_{\text{Warm}} - T_{\text{Cool}}$$

$$\text{Approach} = T_{\text{Cool}} - T_{\text{WB}}$$

where T_{Warm} = inlet temperature of water to the tower (°C, °F)

T_{Cool} = outlet temperature of water from the tower (°C, °F)

T_{WB} = wet bulb temperature of entering air (°C, °F)

The range is normally 6°C to 16°C (approx. 10°F - 30°F).

The approach is a good indicator of cooling tower performance. The lower the approach, the better the cooling tower performance. A larger cooling tower (that is, more air or more fill) will produce a colder outlet water for a given heat load, flow rate and entering air condition. The lower the wet-bulb temperature, which indicates either cool air, low humidity or a combination of the two, the lower the cooling tower can cool the water.

The cooling tower efficiency, η_{CT} , is defined by the temperatures shown in Figure 4.5 and is usually 70% to 75%.

$$\eta_{CT} = \frac{100(T_{\text{Warm}} - T_{\text{Cool}})}{T_{\text{Warm}} - T_{\text{WB}}} = \frac{100(\text{Range})}{T_{\text{Warm}} - T_{\text{WB}}}$$

EXAMPLE 4.7 COOLING TOWER RANGE AND APPROACH

A cooling tower operates with a warm water input at 35°C, a cool water output of 20°C, and a wet bulb temperature of 14°C. Calculate the Range and the Approach for the tower.

$$\text{Range} = T_{\text{Warm}} - T_{\text{Cool}} = 35^\circ\text{C} - 20^\circ\text{C} = 15^\circ\text{C}$$

$$\text{Approach} = T_{\text{Cool}} - T_{\text{WB}} = 20^\circ\text{C} - 14^\circ\text{C} = 6^\circ\text{C}$$

EXAMPLE 4.8 COOLING TOWER EFFICIENCY

The cooling tower efficiency, η_{CT} , is usually 70 - 75%. Will a cooling tower operating at these temperatures have an efficiency within that range?

T_{Warm} = inlet temperature of water to the tower = 38°C

T_{Cool} = outlet temperature of water from the tower = 22°C

T_{WB} = wet bulb temperature of entering air = 16°C

$$\eta_{CT} = 100 \left(\frac{T_{\text{Warm}} - T_{\text{Cool}}}{T_{\text{Warm}} - T_{\text{WB}}} \right) = 100 \left(\frac{38^\circ\text{C} - 22^\circ\text{C}}{38^\circ\text{C} - 16^\circ\text{C}} \right) = 73\%$$

Cooling tower *effectiveness* is quantified by the ratio of the actual range to the ideal range, that is, the difference between cooling water inlet temperature and ambient wet-bulb temperature. It is expressed as a percentage.

$$\text{Effectiveness (\%)} = 100 \left(\frac{\text{Range}}{\text{Range} + \text{Approach}} \right) = 100 \left(\frac{T_{\text{Warm}} - T_{\text{Cool}}}{T_{\text{Cool}} - T_{\text{WB}}} \right)$$

The *cooling capacity* of a tower is the heat rejected in kJ/h or Btu/h.

$$\text{Cooling capacity} = m_{\text{CW}} c_p (T_{\text{Warm}} - T_{\text{Cool}})$$

where m_{CW} = mass flow rate of water (kg/h, lb/h)

c_p = specific heat of water (kJ/kg °C, Btu/lb °F)

T_{Warm} = inlet temperature of water to the tower (°C, °F)

T_{Cool} = outlet temperature of water from the tower (°C, °F)

4.5 REFRIGERATION

Air conditioning and refrigeration equipment capacity in the U.S. and much of North America is often specified in “tons” of refrigeration. Many manufacturers also specify Btu/h, especially for smaller equipment.

A ton of refrigeration (commonly abbreviated as *TR*) is a unit of power that describes the heat extraction capacity of cooling equipment. It is equivalent to the consumption of one ton (2000 lb) of ice per day and originated during the transition from stored natural ice to mechanical refrigeration. More precisely, it is the heat of fusion (144 Btu/lb or 333 kJ/kg) absorbed by melting 1 ton (2,000 lb or 907 kg) of pure ice per day at 0°C (32°F). A ton of cooling is the removal of 12,000 Btu/h (3,517 W = 12,660 kJ/h = 12.66 MJ/h) from water.

$$TR = mH_f = (2000 \text{ lb ice/d})(144 \text{ Btu/lb ice})(\text{d}/24\text{h}) = 12,000 \text{ Btu/h}$$

where H_f = heat of fusion

A mechanical chiller adds approximately 3,000 Btu/h (879 W or 3.165 MJ/h) of parasitic heat load. Therefore, a ton of cooling in a cooling tower is the removal of 15,000 Btu/h (4,396 W or 15.83 MJ/h). An absorption chiller generates 18,000 Btu of heat for each ton of chilled water produced. Thus a cooling tower for an absorption chiller must remove 30,000 Btu/h (8,792 W or 31.65 MJ/h).

From a simplified heat balance around the cooling tower, the energy required for evaporation must equal the energy supplied from cooling the circulating stream (as shown in Figure 4.4):

$$EH_v = m_{\text{CW}} c_p \Delta T \text{ or } E = m_{\text{CW}} c_p \Delta T / H_v$$

where E = evaporation rate (lb/h or kg/h)

m_{CW} = mass flow rate of circulating water (lb/h or kg/h)

H_v = latent heat of vaporization of water (970 Btu/lb or 2260 kJ/kg)

ΔT = water temperature difference from tower top to tower bottom ($^{\circ}\text{F}$ or $^{\circ}\text{C}$)

c_p = specific heat of water (1 Btu/lb $^{\circ}\text{F}$ or 4.184 kJ/kg $^{\circ}\text{C}$)

Rule-of-thumb 1. One ton of cooling means that 15,000 Btu/h of heat must be removed from the water, and for a $\Delta T = 10^{\circ}\text{F}$, that requires a 3 gal/min flow of water (1 gal water = 8.34 lb).

$$15,000 \text{ Btu/h} = m_{CW} c_p \Delta T = m_{CW} (1 \text{ Btu/lb} - ^{\circ}\text{F})(10^{\circ}\text{F})$$

$$m_{CW} = (15,000 \text{ Btu/h}) / (10 \text{ Btu/lb}) = 1,500 \text{ lb/h} = 25 \text{ lb/min} = 3 \text{ gal/min}$$

Rule of thumb 2. Removing 15,000 Btu/h requires evaporating 1.85 gal/h. Evaporation of 1 lb of water takes about 970 Btu of heat.

$$15,000 \text{ Btu/h} = (E)H_v = (E)(970 \text{ Btu/lb})$$

$$E = (15,000 \text{ Btu/h}) / (970 \text{ Btu/lb}) = 15.5 \text{ lb/h} = 1.85 \text{ gal/h}$$

Operating an absorption chiller requires the removal of twice the amount of heat and twice the amount of water must be evaporated.



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4.6 BOILER EFFICIENCY AND WATER USE

Water absorbs more heat for a given temperature rise than any other common inorganic substance. It expands 1600 times as it evaporates to form steam at atmospheric pressure. The steam carries large quantities of heat. These unique properties of water, and its ready availability, make it an ideal raw material for heating and power generating processes.

The power output of a boiler, ignoring adjustment for steam pressure, is

$$\text{BHP} = \frac{(S)(F_E)}{34.5}$$

where BHP = boiler horsepower (hp)

S = steam production (lb/h)

$F_E = (H_{SH} + H_v)/970.3$ = factor of evaporation

H_{SH} = sensible heat (Btu/lb)

H_v = latent heat of evaporation of water (970.3 Btu/lb at 212°F and 1 atm).

34.5 = conversion factor (1 BHP = 34.5 lb steam/h, at 212°F and 1 atm)

The boiler fuel consumption depends on the feedwater temperature, steam temperature and pressure, boiler efficiency, and the fuel heating value.

$$F = \frac{S(H_S - H_{FW})}{\eta_{\text{boiler}} F_{HV}}$$

where F = fuel consumption (gal/h)

S = steam production (lb/h)

H_S = enthalpy of steam at stated temperature and pressure (Btu/lb)

H_{FW} = enthalpy of feedwater at saturation temperature (Btu/lb)

η_{boiler} = boiler efficiency (typically 80%)

F_{HV} = fuel heating value (Btu/gal)

EXAMPLE 4.9 BOILER HORSEPOWER

A boiler generating 21,500 lb/h of steam at 155 lb/in² has a factor of evaporation of $F_E = 1.02$. The boiler horsepower is

$$\text{BHP} = \frac{(21,500 \text{ lb/h})(1.02)}{34.5} = 636 \text{ hp}$$

EXAMPLE 4.10 FACTOR OF EVAPORATION

Water enters a boiler at 225°F. The boiler pressure is 100 lb/in² and the boiler water temperature is 338°F. The latent heat is 881 Btu/lb. The factor of evaporation is

$$F_E = \frac{H_{SH} + H_{LH}}{970.3 \text{ Btu/lb}} = \frac{(338^\circ\text{F} - 225^\circ\text{F})(1 \text{ Btu/lb}^\circ\text{F}) + 881 \text{ Btu/lb}}{970.3 \text{ Btu/lb}} = 1.02$$

EXAMPLE 4.11 BOILER FUEL CONSUMPTION

A boiler produces 10,000 lb/h of steam using diesel fuel ($F_{HV} = 130,000 \text{ Btu/gal}$) with a feedwater temperature of 154°F. The enthalpy of the steam is 1190 Btu/lb at 100 lb/in².

$$F = \frac{S(H_S - H_{FW})}{\eta_{\text{boiler}} F_{HV}} = \frac{(10,000 \text{ lb/h})(1,190 \text{ Btu/lb} - 122 \text{ Btu/lb})}{0.8(130,000 \text{ Btu/gal})} = 103 \text{ gal/h}$$

All natural waters contain varying amounts of dissolved and suspended matter and dissolved gases. The amount of minerals dissolved in water varies from 30 g/L in sea water to anything from 0.005 to 1500 mg/L in fresh water supplies. Since water impurities cause boiler problems, careful consideration must be given to the quality of the water used for generating steam. Table 4.3 gives the American Society of Mechanical Engineers (ASME) guidelines for water quality in modern industrial water tube boilers for reliable continuous operation.

The boiler feed water must be of such quality that it can be concentrated a reasonable number of times inside the boiler, without exceeding the tolerance limits of the particular boiler design. If the feed water does not meet these requirements it must be treated to remove impurities. The impurities need not be completely removed if chemical treatment can effectively and economically control scaling and corrosion.

Drum pressure (psi)	Iron (ppm Fe)	Copper (ppm Cu)	Total hardness (ppm CaCO ₃)	Silica (ppm SiO ₂)	Total alkalinity (ppm CaCO ₃)	TDS (mg/L)
0-300	0.100	0.050	0.300	150	149-700	700-3500
301-450	0.050	0.025	0.300	90	120-600	600-3000
451-600	0.030	0.020	0.200	40	100-500	500-2500
601-750	0.025	0.020	0.200	30	40-200	200-1000
751-900	0.020	0.015	0.100	20	30-150	150-750
901-1000	0.020	0.015	0.050	8	25-125	125-625
1001-1800	0.010	0.010	0.0	2	100	100
1800-2350	0.010	0.010	0.0	1	50	100

Source: American Society of Mechanical Engineers (ASME)

Table 4.3 Guidelines for water quality in modern industrial water tube boilers for reliable continuous operation.

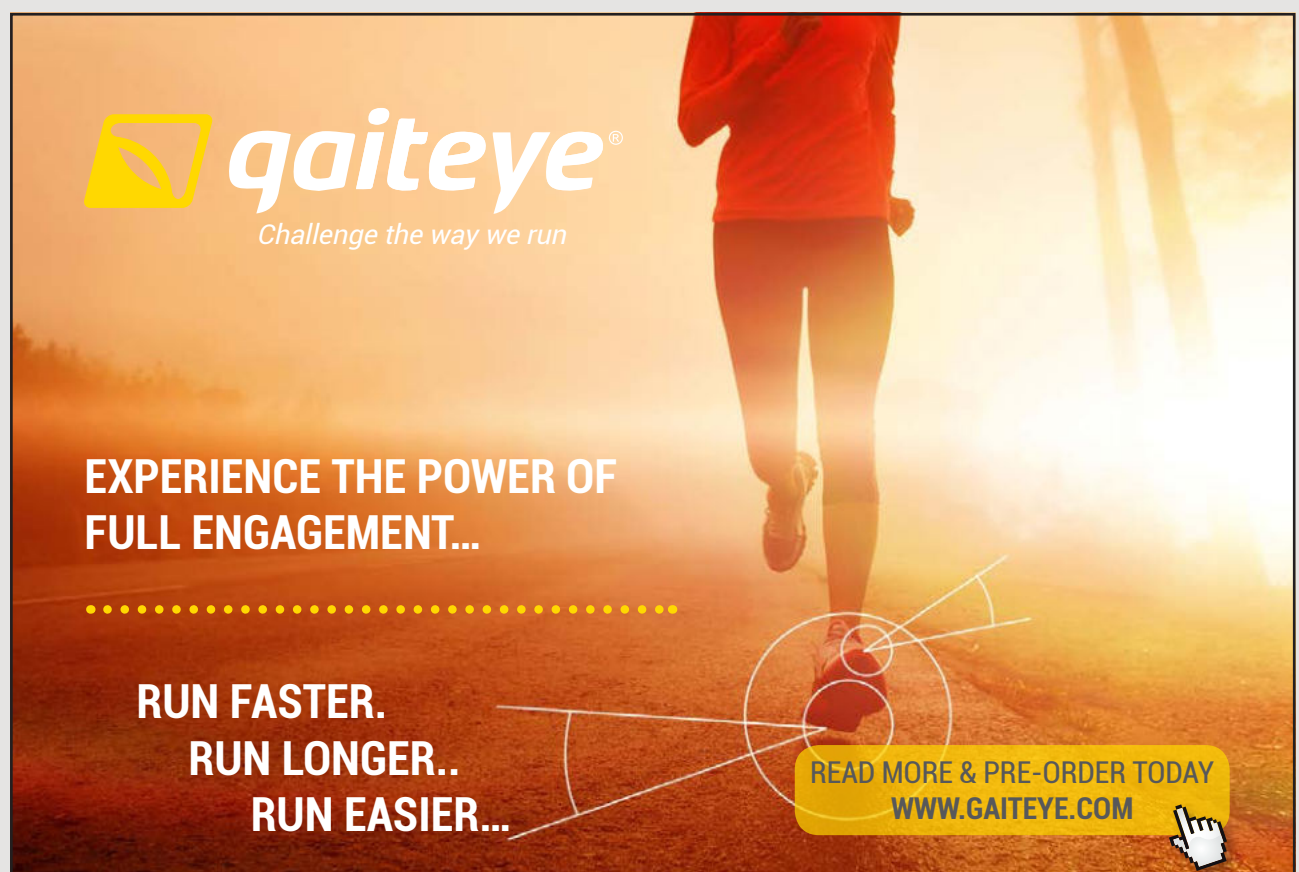
Boiler *blowdown* is the intentional removal of boiler water to control the level of total dissolved solids (TDS) in the boiler during the steam production. Makeup water is added to balance the steam losses and blowdown. The material balance is

$$\text{Feed water} + \text{Steam} = \text{Blowdown}$$

The blowdown rate of a boiler depends on

- 1) steam consumption (steam used in the process that is not returned as condensate to the boiler)
- 2) concentration of impurities in the feed water, and
- 3) maximum allowable TDS in the boiler.

Boiler blowdown is expressed as a fraction of feedwater mass flow and can range from less than 1% to more than 10% depending on water chemistry, boiler operating pressure, and other factors. However, a 10% blowdown rate does not mean 10% energy loss because it is moderate-energy water and not high-energy steam. The energy added to the blowdown water is the blowdown flow rate times the difference in the enthalpy of the blowdown and the feedwater. Therefore, 10% blowdown rate can translate into 5% fuel energy input.



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The fraction of energy loss associated with blowdown, L_{BD} , is the heat energy lost divided by the fuel energy put into the boiler.

$$L_{BD} = \frac{m_{BD}(H_{BD} - H_{FW})}{m_{\text{Fuel}}(\text{HHV})}$$

where L_{BD} = loss due to blowdown (%)

m_{BD} = mass flow rate of blowdown (lb/h)

m_{Fuel} = mass flow rate of fuel (lb/h)

H_{BD} = enthalpy of blowdown (Btu/lb)

H_{FW} = enthalpy of feedwater at saturation temperature (Btu/lb)

HHV = higher heating value of fuel (Btu/lb)

4.7 CONCLUSION

The law of conservation of energy is analogous to the law of conservation of mass. The energy balance is analogous to the material balance. Furthermore, making an energy balance requires knowledge of the material balance in order to account for energy carried with materials as they move through the system.

Enthalpy, H , is a thermodynamic quantity that is used to describe the change in heat energy when the temperature of a material changes from one temperature to another. Enthalpy cannot be detected directly and it has no absolute value. What can be measured is the change in enthalpy relative to some initial condition or to a standard reference condition. It is measured in units of kJ or Btu. Specific enthalpy is measured in units of kJ/kg, kJ/kg-mol, Btu/lb, or Btu/lb-mol.

5 COMBUSTION OF MUNICIPAL REFUSE

When a waste material, such as municipal refuse or landfill gas, is considered as a fuel the design problems are to determine its composition so the pollution potential of the end products are known and to determine how much useful energy will be released so the economics can be calculated.

Combustion products may be completely harmless or harmful, and this is often determined by the combustion conditions. Carbon will burn to carbon monoxide (CO) or carbon dioxide (CO₂). For many years we thought CO₂ was harmless, but the increasing amount of CO₂ in the atmosphere is causing climate change. Combustion can convert nitrogen to nitrous oxides (NO_x), which are air pollutants and contribute to smog formation. Sulfur burns to form sulfur dioxide (SO₂) and sulfate (SO₄²⁻).

The next three chapters involve fuel-to-energy transformations. The first sections of this chapter review how balanced chemical equations are used to calculate the mass of reactants and products. The reactants may be solids, liquids, or gases. The products are gases.

The rest of the chapter shows how these basics are used to analyze the combustion of municipal refuse. Chapters 6 and 7 are about energy recovery from biogas and the incineration of gases.

5.1 STOICHIOMETRY

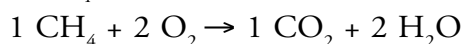
Molecules and compounds are merged and split and rearranged in chemical reactions. A molecule is formed when two or more atoms chemically join together. If the atoms are different elements, a compound is formed. Not all molecules are compounds, since some, such as hydrogen gas or ozone, consist only of one element or type of atom.

Stoichiometry is the science of deciphering the qualitative and quantitative information that is given by a *balanced stoichiometric equation*. A balanced stoichiometric equation shows the chemical composition of the reactants and products and the molar proportions in which they react. The key to deciphering the reaction stoichiometry is the conversion of molecular units to mass units so we know how much of each reactant is consumed in the reaction and the yield of each product.

By convention, the reactants are on the left-hand side and the products are on the right. All products are shown, even those that are inert. Inert substances that may be carried along with the reactants do not react, they produce nothing, and they are ignored *for the purpose*

of the stoichiometry. They do, however, have bulk and mass, and they are not ignored in the material balance and energy balance of the reactor system.

The meaning of *balanced* is explained by example. The stoichiometry for combustion of methane (CH_4) is



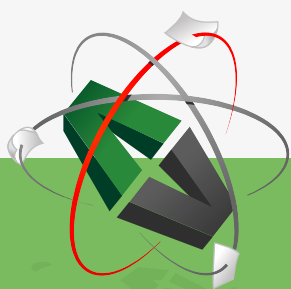
The reaction equation gives the exact molecular proportions of the reaction. The stoichiometric coefficients refer to molecular units (not mass units). Read the equation like a sentence:

*1 mole of methane reacts with 2 moles of oxygen gas
to form 1 mole of carbon dioxide and 2 moles of water.*

More precisely, the balanced equation shows that the carbon atom from the methane molecule appears in the molecule of carbon dioxide. Also, four atoms of oxygen react; two of them appear in a molecule of carbon dioxide (CO_2) and two appear in 2 molecules of water (H_2O).

The equation is *balanced* because the number of reacting carbon atoms is the same number of carbon atoms in the products; one carbon atom on the left and one on the right. The same balance must exist for hydrogen and oxygen atoms. There are four hydrogen atoms

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(H) on each side and four oxygen atoms (O) on each side. The atoms have been rearranged, but they have not disappeared.

Because of conservation of mass, the masses of each atomic species are balanced, and the mass of reactants must therefore equal the mass of products. The carbon atom that reacts has the same mass as the carbon atom that appears in the molecule of CO_2 . This is true for each species of atom, and it is, therefore, true for the total mass.

The mole is a measure of mass. One gram-mole (g-mol) contains about 6×10^{23} atoms or molecules (Avagadro's number). The mass of a compound is the mass of 6×10^{23} molecules of each element in the molecule.

The mass of one gram-mole of carbon, C, (6×10^{23} atoms C) is 12.011 g. The molecular mass of one mole of carbon is 12.011 g/g-mol, usually rounded off to 12.0 g/g-mol for engineering calculations. The masses of other elements are referenced to the atomic mass of carbon. Appendix 1 is an abbreviated list of atomic masses.

In many engineering calculations it is more convenient to use kg-moles, lb-moles, or T-moles. One kilogram mole (kg-mol) contains 6×10^{26} molecules. One pound mole (lb-mol) about 2.7×10^{28} molecules; it has more molecules because one pound mass is 453.6 times greater than one gram mass. Table 5.1 lists the atomic and molecular masses for some common gases.

Element		Atomic Mass (g/g-mol)	Compound		Molar Mass (g/g-mol)
Carbon	C	12.01	Oxygen O_2	O_2	$2(16) = 32$
Hydrogen	H	1.01	Carbon dioxide	CO_2	$12 + 2(16) = 44$
Oxygen	O	16.0	Water	H_2O	$1(2) + 16 = 18$
Nitrogen	N	14.01	Methane	CH_4	$12 + 4(1) = 16$
Sulfur	S	32.06	Hydrogen sulfide	H_2S	$2(1) + 32.06 = 34$
			Sulfur dioxide	SO_2	$32.06 + 2(16) = 64.1$

Table 5.1 Atomic and molecular masses

EXAMPLE 5.1 BURNING METHANE

The balanced stoichiometric reaction for the burning of methane is given in the first line of Table 5.2. The other sections of the table interpret this chemical sentence in terms of molecules, moles, and mass.

Balanced Stoichiometric Equation for Burning Methane						
CH_4	+	2 O_2	\rightarrow	CO_2	+	$2 \text{ H}_2\text{O}$
Molecular Basis						
1 molecule of methane	reacts with	2 molecules of oxygen	to give	1 molecule of carbon dioxide	and	2 molecules of water
$1 (6.023 \times 10^{23})$ molecules C_7H_{16}	+	$2 (6.023 \times 10^{23})$ molecules O_2	\rightarrow	$1 (6.023 \times 10^{23})$ molecules CO_2	+	$2(6.023 \times 10^{23})$ molecules H_2O
Molar Basis						
1 g-mol CH_4	+	2 g-mol O_2	\rightarrow	1 g-mol CO_2	+	2 g-mol O_2
1 kg-mol CH_4	+	2 kg-mol O_2	\rightarrow	1 kg-mol CO_2	+	2 kg-mol O_2
1 lb-mol CH_4	+	2 lb-mol O_2	\rightarrow	1 lb-mol CO_2	+	2 lb-mol O_2
1 T-mol CH_4	+	2 T-mol O_2	\rightarrow	1 T-mol CO_2	+	2 T-mol O_2
Molecular mass of each compound						
16 g/g-mol		32 g/g-mol		44 g/g-mol		18 g/g-mol
Mass Basis						
$1(16 \text{ g}) \text{ CH}_4$ $= 16 \text{ g CH}_4$	+	$2(32 \text{ g}) \text{ O}_2$ $= 64 \text{ g O}_2$	\rightarrow	44 g CO_2	+	$2(18 \text{ g}) \text{ H}_2\text{O}$ $= 36 \text{ g H}_2\text{O}$
16 kg CH_4	+	64 kg O_2	\rightarrow	44 kg CO_2	+	36 kg H_2O
16 lb CH_4	+	64 lb O_2	\rightarrow	44 lb CO_2	+	36 lb H_2O
16 ton CH_4	+	64 ton O_2	\rightarrow	44 ton CO_2	+	36 ton H_2O

Table 5.2 Equivalent interpretations of the stoichiometric equation.

Converting from a molar to a mass basis uses the molar masses of the compounds, which are:

Atomic masses (g): C = 12, H = 1, and O = 16

$\text{CH}_4 = 1(12) + 4(1) = 16 \text{ g/g-mol} = 16 \text{ lb/lb-mol} = 16 \text{ ton/ton-mol}$

$\text{CO}_2 = 1(12) + 2(16) = 44 \text{ g/g-mol} = 44 \text{ lb/lb-mol} = 44 \text{ ton/ton-mol}$

$\text{O}_2 = 2(16) = 32 \text{ g/g-mol} = 32 \text{ lb/lb-mol} = 32 \text{ ton/ton-mol}$

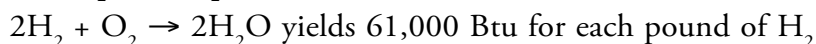
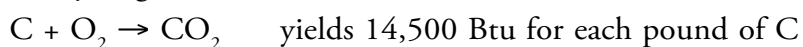
$\text{H}_2\text{O} = 2(1) + 16 = 18 \text{ g/g-mol} = 18 \text{ lb/lb-mol} = 18 \text{ ton/ton-mol}$

The mass, in grams of substance reacting, is calculated by multiplying the molar mass (g/g-mol) of each substance by the number of moles. The proportions of reactants and products hold for any mass units.

Notice that all the mass is accounted for. The mass of reactants equals the mass of products formed: 452 g of reactants yield 452 g of products.

5.2 COMBUSTION STOICHIOMETRY

The most important product of combustion is heat. The basic combustion reactions for carbon and hydrogen are:



The amount of heat released can be calculated from the stoichiometry but the physical chemistry is not needed here. Instead, there data are on the heating value of liquids and solids that can be used as fuels in Section 5.4, Table 5.6, and Appendix 4.

The chemical formulas are usually known for gases (for example, methane is CH_4) and so are the compositions of gas mixtures (for example, 70% CH_4 and 30% CO_2 as volume percent).

Solid materials are more variable and complex mixtures that are often given a categorical description (the material is 50% paper and 50% plastic) or an *empirical chemical formula* that shows the proportions of the dominant elements. As an example, an empirical compound might be $\text{C}_{59}\text{H}_{93}\text{O}_{37}\text{N}$, which states that one ‘molecule’ of this “compound” contains carbon, hydrogen, oxygen, and nitrogen in the proportions shown by the subscripts. There is no such compound; empirical means the atomic proportions are correct so the formula can be used to do stoichiometric calculations.



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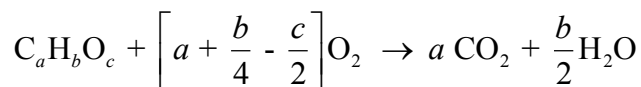
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A combustible waste can be analyzed for carbon, hydrogen, oxygen, and other elements. A simplified empirical formula of the main constituents of a dry material is represented as $C_aH_bO_c$ and a generic combustion reaction as



Nitrogen, sulfur, or other elements could be included, but these are usually negligible fraction of the total mass. They may be important for calculating trace gases in the flue gas (sulfur dioxide for example), but they are not important in the energy balance and are often omitted from the empirical formulas and the balanced chemical equations.

The empirical stoichiometric equation must satisfy the law of conservation of mass. That is, the number of atoms must balance on both sides, and the mass of products must equal the mass of reactants. The stoichiometric coefficients (a , b , c) are assigned so that the atoms of C, H, and O balance.

All of the carbon goes to CO_2 so there are a atoms of carbon in the fuel to create a molecules of CO_2 . This requires $2a$ atoms of oxygen. The hydrogen from the fuel forms water. Specifically, b atoms of H are used to create $b/2$ molecules of H_2O . This requires $b/2$ atoms of oxygen. The stoichiometric coefficient for oxygen must make the number of atoms of oxygen on the reaction side equal the number of atoms on the products side. The atoms of oxygen coming from O_2 is $(2a + b/2 - c)$. The oxygen atoms coming from $C_aH_bO_c$ (that is c oxygen atoms) are subtracted. Each molecule of oxygen gas provides 2 atoms of O and the coefficient is $2(2a + b/2 - c) = a + b/4 - c/2$.

More directly, say the coefficient of O_2 is x . The equation is balanced when

$$2x + c = 2a + b/2 \text{ and } x = a + b/4 - c/2$$

EXAMPLE 5.2 COMBUSTION OF MUNICIPAL REFUSE

A dried municipal refuse has the following empirical chemical composition, $C_{59}H_{93}O_{37}N$. Find the mass of oxygen required to completely combust 1000 kg/h of this material.

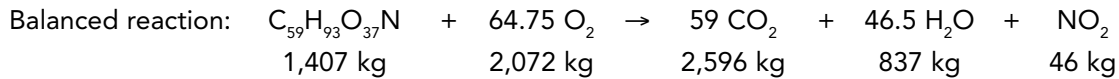
The balanced combustion reaction is



where

$$a = 59, b = 93, c = 37, \text{ and } d = 1$$

$$x = a + b/4 - c/2 + d = 59 + 93/4 - 37/2 + 1 = 64.75$$



The stoichiometric oxygen requirement is

$$\text{mass O}_2 = \left(\frac{1,000 \text{ kg/h refuse}}{1,407 \text{ kg refuse/kg-mol refuse}} \right) \left(2,072 \frac{\text{kg O}_2}{\text{kg-mol refuse}} \right) = 1,773 \text{ kg O}_2/\text{h}$$

Air is 23.2% O₂ by weight, and the mass fraction of oxygen in air is 0.232

Mass of air that contains 1,773 kg of O₂ = (1,773 kg O₂/h)/0.232 = 7,642 kg air/h

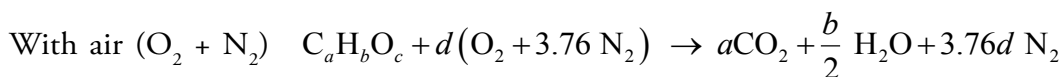
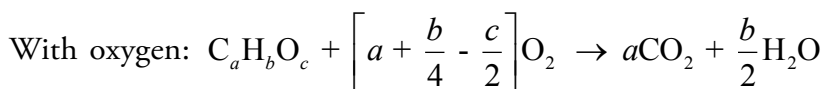
Some excess air beyond the stoichiometric requirement will be needed to have complete combustion.

The oxygen in the excess air will not burn and the nitrogen in the excess air passes through unchanged.

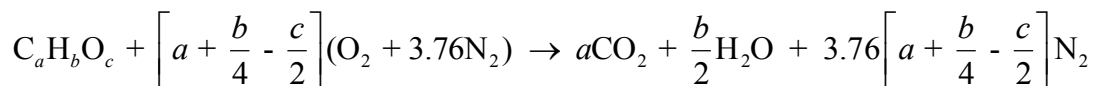
By convention, the stoichiometric reaction does not show impurities and inert substances in the fuel, such as trace elements (i.e., Hg and Cl), moisture or ash. Trace elements (<1% of total by dry weight) may become gases, acids, fumes, or ash.

This would also include nitrogen since it is inert, but it can be convenient to overlook this convention for omitting inert material when calculating the air required for the combustion, which contains roughly three volumes of nitrogen for each volume of oxygen. The mass and volume of nitrogen need to be known and the direct way is to include it in the stoichiometric calculation.

The mole fractions of gasses in a mixture are equal to the volume fractions. The volume fractions are 79% N₂ and 21% O₂, and the mole fractions are the same, 79% N₂ and 21% O₂. This means that 79/21 = 3.76 moles of nitrogen accompany each mole of oxygen in the air. The balance combustion equation can be modified to account for this. The reactions for combustion are:



Once again it is required that $c + 2d = 2a + b/2$ and $d = a + b/4 - c/2$ and

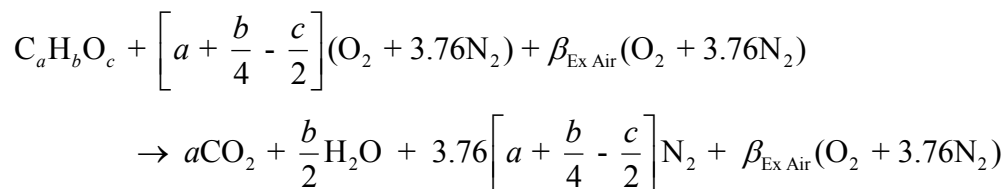


The *stoichiometric ratio* is the ideal where all fuel and air are consumed without any excess left over. In practice, burning conditions are never ideal and more air than ideal must be supplied for complete combustion. The amount of air above the theoretical requirement is referred to as *excess air*.

Insufficient air supply to the burner causes unburned fuel, soot, smoke, and carbon monoxide exhausts, and other problems that result in heat transfer surface fouling, pollution, lower combustion efficiency, flame instability and a potential for explosion.

These problems are reduced by operating with *excess air* so there is a percentage of oxygen in the exhaust gas to make sure combustion is complete. Boilers and high temperature process furnaces usually incorporate a modest amount of excess air, about 10 to 20% more than what is needed to burn the fuel completely. Power plant boilers normally run about 10 to 20 percent excess air. Natural gas-fired boilers may run as low as 5 percent excess air. Pulverized coal-fired boilers may run with 20 percent excess air. Gas turbines run very lean with up to 300 percent excess air.

The general combustion reaction can be modified with a factor, $\beta_{\text{Ex Air}}$, that accounts for excess air



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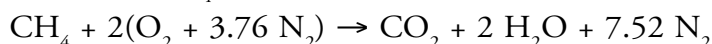
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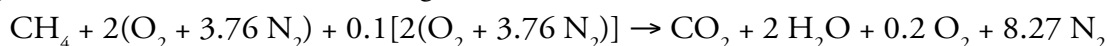


The term $\beta_{\text{Ex Air}} (\text{O}_2 + 3.76 \text{ N}_2)$ is the excess air. The oxygen in the excess air passes through because it is not consumed. For 10% excess air $\beta_{\text{Ex Air}} = 0.1$; for 25% excess air $\beta_{\text{Ex Air}} = 0.25$.

Burning methane (CH_4) in air is



Burning methane with 10% excess air gives



Knowing the combustion products is important for selecting materials of construction to avoid corrosion, and in selecting additional air pollution control equipment to neutralize the acidic compounds.

If the fuel ($\text{C}_a\text{H}_b\text{O}_c$) contains a few elements that form gases, such as N, or S, these will be liberated from the parent molecule and the combustion products will include SO_2 , SO_3 , NO_2 , NO_x . Sulfuric acid (H_2SO_4) and nitric acid (HNO_3) are formed when these products react with water vapor.

Elements such as mercury (Hg), chlorine (Cl), fluorine (F), copper (Cu) can form toxic metal fumes (Hg and Cu) and acids (HCl and HF). Elements such as iron (Fe), phosphorus (P), calcium (Ca) do not form gases or fumes and they become part of the ash.

5.3 COMPOSITION OF SOLID WASTE

The composition of residential, municipal, commercial, or industrial solid waste is highly variable with season, city, state and country. Even so, it is helpful to have an idea of the composition of the most common components in residential solid waste. This information is in Table 5.3. The percentages for carbon, hydrogen, etc. apply to the dry weight fraction of the solid waste component. Everything that is not C, H, O, N or S is assigned to the mass fraction for ash.

Example 5.3 shows how these data are used to calculate the empirical stoichiometric formula for a mixed waste.

Component	Dry solids content (%)	Water Content (%)	Elemental Composition of Dry Waste (mass %)					
			Carbon (C)	Hydrogen (H)	Oxygen (O)	Nitrogen (N)	Sulfur (S)	Ash
Organic								
Food wastes	30	70	48.0	6.4	37.6	2.6	0.4	5.0
Paper	94	6	43.5	6.0	44.0	0.3	0.2	6.0
Cardboard	95	5	44.0	5.9	44.6	0.3	0.2	5.0
Plastics	99	1	60.0	7.2	22.8	--	--	10.0
Textiles	90	10	55.0	6.6	31.2	4.6	0.1	2.5
Rubber	100	0	78.0	10.0	--	2.0	--	10.0
Leather	80	20	60.0	8.0	11.6	10	0.4	10.0
Yard wastes	35	65	47.8	6.0	38.0	3.4	0.3	4.5
Wood	80	20	49.5	6.0	42.7	0.2	0.1	1.5
Inorganic								
Glass	97	3	0.5	0.1	0.4	<0.1	--	98.9
Metals	97	3	4.5	0.6	4.3	<0.1	--	90.5
Dirt, ash, etc.	97	3	26.3	3.0	2.0	0.5	0.2	68.0

Table 5.3 Empirical composition of the combustible components of residential solid waste in the U.S. (Source: Vesiland et al. 2001, Reinhart 2004).

EXAMPLE 5.3 EMPIRICAL COMPOSITION OF MIXED SOLID WASTE

Large quantities of mixed solid waste are available as a fuel. A representative sample of the waste has this as-delivered (wet) composition: 40 kg paper, 10 kg cardboard, 15 kg plastics, and 10 kg wood. The moisture content and elemental composition of each component from Table 5.3 were used to calculate the mass of dry solids, water, and the elemental composition of each component given in Table 5.4. The total masses for each element were used in Table 5.5 to calculate the empirical composition.

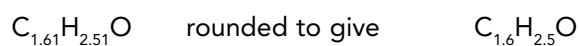
Material	Total mass (kg)	Water (%)	Dry solids (kg)	Water (kg)	Elemental Composition of Dry Solids (kg)					
					C	H	O	N	S	Ash
Paper	40	6	37.6	2.4	16.356	2.256	16.544	0.1128	0.075	2.256
Cardboard	10	5	9.5	0.5	4.180	0.5605	4.237	0.028	0.019	0.475
Plastics	15	1	14.85	0.15	8.910	1.0692	3.386	0	0	1.485
Wood	10	20	8	2	3.960	0.480	3.416	0.016	0.008	0.12
Total	75	5.05	69.95	5.05	33.406	4.366	27.583	0.1573	0.102	4.336

Table 5.4 Calculation of the elemental composition of a waste mixture

Element	Mass (kg)	Atomic Mass (kg/mol)	Moles	Basis for Mole Ratios		
				O = 1	N = 1	S = 1
C	33.406	12.01	2.782	1.61	247.74	872.8
H	4.366	1.01	4.322	2.51	384.98	1356.4
O	27.583	16.00	1.724	1.00	153.54	540.0
N	0.157	14.01	0.011		1.00	3.5
S	0.102	32.07	0.003			1.00


Table 5.5 Calculation of mole ratios for the stoichiometric formulation of the dry waste mixture.

The most useful empirical formula is the simplest




Including nitrogen and sulfur gives $\text{C}_{248}\text{H}_{385}\text{O}_{154}\text{N}$ and $\text{C}_{873}\text{H}_{1356}\text{O}_{540}\text{N}_{3.5}\text{S}$. The sulfur and nitrogen are irrelevant to the fuel value.

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5.4 HEATING VALUE OF WASTE MATERIALS

The production of *municipal solid waste* (MSW) in the U.S. in 2010 was about 225 million metric tons (2.25×10^{11} kg). The composition of this waste is about 13% inert material (metal, glass, etc.), 25% moisture, and 50% combustible paper, wood, rubber, and plastic. By definition, municipal solid waste does not include industrial, agricultural, medical wastes or sewage sludge.

The *lower heating value* (LHV), also known as the *net heating value*, is the enthalpy (kJ/kg, Btu/lb) available after adjusting for water vapor produced during combustion. Water is a product of combustion. This water must be heated to the boiling point, vaporized, and the vapor must be heated to the combustion temperature. This consumes some of the heat released by the fuel. An increase in the moisture content of a given combustible material will reduce the LHV. The LHVs are the useful calorific values in boiler combustion plants.

The *higher heating value* (HHV) (also known as the *gross energy*) of a fuel is defined as the amount of heat released by a specified quantity (initially at 25°C) once it is combusted and the products have returned to a temperature of 25°C, which takes into account recovery of the latent heat of vaporization of water in the combustion products. The HHVs are derived only under laboratory conditions.

When considering combustion for a waste material, the first question is will the waste material burn without the addition of auxiliary fuel. The second question is, if the combustion is self-sustainable, how much heat energy will be released.

The shaded region on the Tanner diagram, Figure 5.1, predicts a self-sustainable region of combustion when the ash content is less than 60%, the moisture content is less than 50%, and the organic content is more than 25%. The percentages are based on the wet mass of the solid waste sample. Organic content is measured by weight loss upon combustion. The available heat energy is greatest in the lower right corner of the diagram where the high heating value (HHV) can reach 19 MJ/kg wet mass.

In practice the lower heating value (LHV) is the useful energy the waste can yield. Komilis (2013) estimates the lower heating value (LHV), with units of kJ/kg wet waste, as

$$\text{LHV}_w \text{ (MJ/wet kg)} = 17.1 OM_d - 14.9 M$$

$$\text{LHV}_w \text{ (MJ/wet kg)} = 16.5 OM_w$$

where OM_d = fraction of organic matter on a dry weight basis

OM_w = fraction of organic matter on a wet weight basis

M = moisture content as a fraction of wet weight

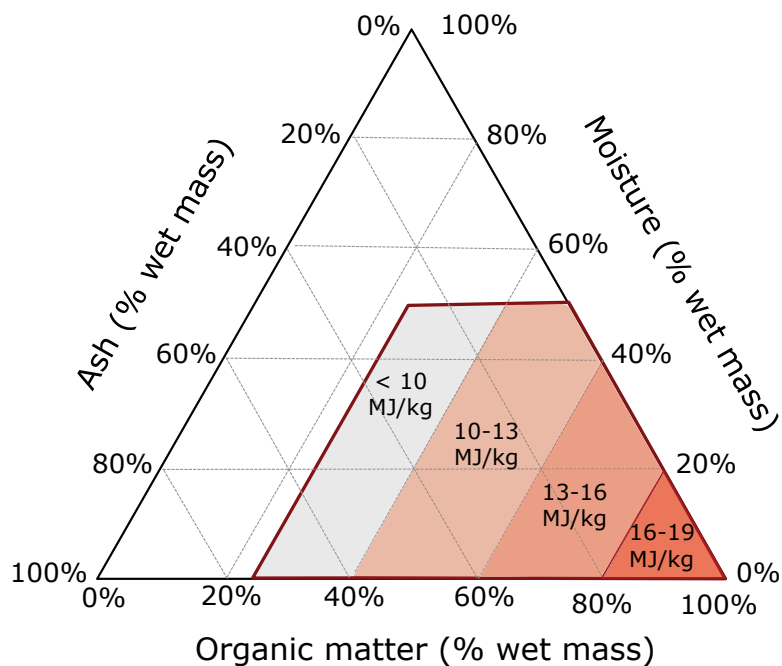


Figure 5.1 The Tanner diagram shows a region of self-sustaining combustion. Values inside the shaded region are high heating values (HHV) in MJ per kg of wet waste. (Redrawn from Komilis et al 2013)

Table 5.6 lists the *lower heating value* (LHV), also known as the *net heating value*, of several fuels, including some waste materials. The U.S. EPA considers any material with a LHV of 12,000 – 18,000 kJ/kg (5,000 to 8,000 Btu/lb) to be a bona fide fuel.

Fuel	Btu/lb	kWh/lb	kJ/kg	kWh/kg
Carbon	14,100	4.13	32,800	9.61
Coal				
Bituminous (8 % ash, 4% S)	11,500	3.37	26,700	7.84
Bituminous (4.5 % ash, 1% S)	14,500	4.25	33,700	9.89
Anthracite (8 % ash, 0.5% S)	13,500	3.96	31,400	9.20
Liquids				
Fuel oil (87.3% carbon)	18,000	5.28	41,900	12.27
Crude oil	19,000	5.57	44,200	12.95
Gasoline	20,000	5.86	46,500	13.64
Gases				
Landfill gas (50% CH ₄)	11,500	3.37	26,700	7.84
Sludge digester gas (70% CH ₄)	16,000	4.69	37,200	10.91
Natural gas*	20,300	5.95	47,200	13.84
Methane	21,500	6.30	50,000	14.66
Ethane	20,300	5.95	47,200	13.84
Propane	19,800	5.80	46,100	13.50
Butane	20,000	5.86	46,500	13.64
Municipal solid waste				
Unsorted moist refuse	3,000 - 5,000	0.88 - 1.46	7,000 - 11,600	2.05-3.41
Dry combustibles	8,000	2.34	18,600	5.45
Waste materials (dry weight basis)				
Newsprint	7,800	2.29	18,100	5.32
Digested sewage sludge (40% ash)	5,300	1.55	12,300	3.61
Wastewater grease & scum (12% ash)	16,700	4.89	38,800	11.39
Rags (10% moisture, 2.5 % ash)	7,650	2.24	17,800	5.22
Waste sulfite liquor solids	7,900	2.32	18,400	5.39
Sugar cane bagasse (Dry)	8,000	2.34	18,600	5.45
Vinyl scrap	11,400	3.34	26,500	7.77
Foam scrap	12,290	3.60	28,600	8.38
Tires	15,000	4.40	34,900	10.23
Used oil	18,000	5.28	41,900	12.27
Polyethylene film	19,160	5.62	44,600	13.06
* Lower heating value of natural gas = 983 Btu/ft ³				
Conversion Factors: 1 Btu/lb = 2.931 x 10 ⁻⁴ kWh/lb = 6.463 x 10 ⁻⁴ kWh/kg = 2.326 kJ/kg				

Table 5.6 Lower heating value (LHV) of selected fuels and waste materials

EXAMPLE 5.4 REFUSE AS FUEL

A refuse processing plant shreds 980 T/d of raw refuse in a hammermill. Metals are removed at the rate of 74 T/d. The remaining 906 T/d of solid material is mixed with coal according to heating value in proportions of 10% refuse and 90% coal and burned in an electric generating plant boiler. The refuse heating value is 5000 Btu/lb and the coal heating value is 12,000 Btu/lb. In order to burn all the processed refuse, the total daily energy input to the boiler is:

Refuse:	$(906 \text{ T/d})(2000 \text{ lb/T})(5000 \text{ Btu/lb}) = 9,060 \times 10^6 \text{ Btu/d}$
Coal:	$(9 \text{ Btu coal/Btu refuse})(9,060 \times 10^6 \text{ Btu/d}) = 81,540 \times 10^6 \text{ Btu/d}$
Total:	$90,600 \times 10^6 \text{ Btu/d}$

At 40% energy conversion, this should generate about $10.6 \times 10^6 \text{ kWh}$ of electricity over a 24-h day.

$$0.4(90,600 \times 10^6 \text{ Btu/d}) / (3412 \text{ Btu/kWh}) = 10.6 \times 10^6 \text{ kWh/d}$$

The energy from the refuse, accounts for ten percent of this total. The true energy yield of the refuse is lower because of the energy consumed in the shredding, sorting, and hauling.



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EXAMPLE 5.5 UPGRADING THE FUEL VALUE OF SOLID WASTE

Figure 5.2 shows the material flow in a 30,000 kg/h (30 T/h) municipal solid waste (MSW) test processing facility. Ferrous metals are removed by magnetic separation before the waste is shredded. Air classification separates lighter and heavier particles. The lighter material, which has a high organic content, is separated into three size fractions with a trommel screen. The trommel overflow (material larger than 100 mm) is the lightest and has the highest organic content. This is the most desirable material for sale to power plants as refuse derived fuel (RDF). The mid-size fraction (25-100 mm) also can be used as RDF.

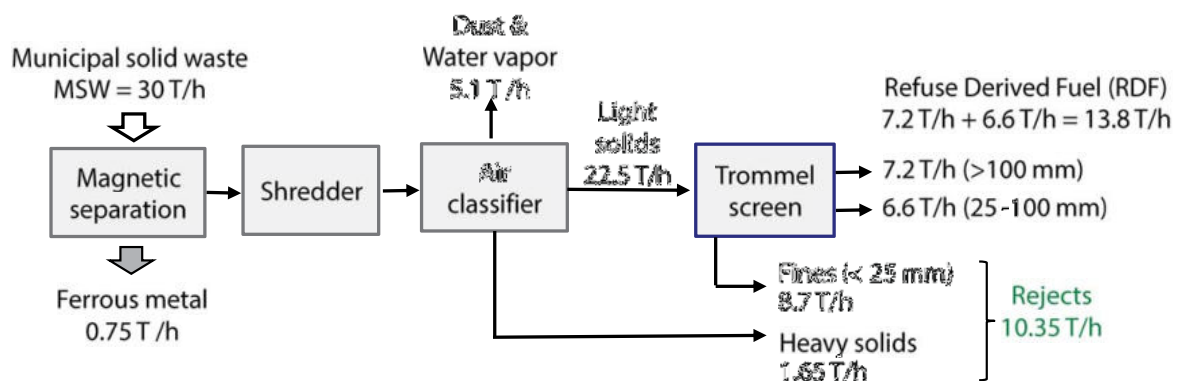


Figure 5.2 Municipal solid waste sorting process.

The RDF yield from a 30 T/h input is the two larger size fractions discharged from the trommel screen. That is

$$\text{RDF} = 7.2 \text{ T/h} + 6.6 \text{ T/h} = 13.8 \text{ T/h}$$

The sorting has produced these changes from MSW to RDF:

- Plastics content increased from 26% in the MSW to 68% in the RDF, on a dry weight basis.
- Food waste, metal, glass, and ceramics are almost absent in the trommel overflow.
- Water content of the trommel overflow decreased to 40% from 51% in the air classifier feed.
- Combustible content increased from 20% in the MSW to 29% in the RDF.
- Sulfur content decreased from 0.8% to 0.05%.
- Chlorine content increased from 0.18% to 0.23% because plastics have been concentrated in the RDF.
- Higher heat value increased from 2.65 kWh/kg in the MSW to 4.32 kWh/kg in the RDF.

The total mechanical power requirement for the 30 T/h test facility was 955 kW, distributed as follows

Shredder = 825 kW

Magnetic separator = 5 kW

Trommel screen = 12 kW

Air classifier = 33 kW

Conveyors = 15 kW

Cyclone = 65 kW (air pollution control)

Energy used to produce 13.8 T/h (13,800 kg/h) of RDF, assuming an 8-hour operating day, was
 $(955 \text{ kW})(8 \text{ h}) = 7,640 \text{ kWh}$

Energy used per kg of RDF produced was
 $(955 \text{ kW})(8 \text{ h})/(13,800 \text{ kg}) = 0.554 \text{ kWh/kg}$ of RDF produced.

Energy content of the 13.8 T/d of RDF, assuming an 8-hour operating day and a fuel value of 4.32 kWh/kg
 $(13,800 \text{ kg/h})(8 \text{ h})(4.32 \text{ kWh/kg}) = 477,000 \text{ kWh}$

Net energy output of the process = $477,000 \text{ kWh} - 7,640 \text{ kWh} = 469,360 \text{ kWh}$

Net energy output per kg RDF produced = $4.32 \text{ kWh/kg} - 0.554 \text{ kWh/kg} = 3.765 \text{ kWh/kg}$

The energy used was less than 13% of the 4.32 kWh/kg fuel value of the RDF.

5.5 INCINERATION OF SOLID WASTE AND SLUDGE

Incineration is the controlled combustion to reduce the volume and weight of waste. In Europe, most municipal refuse is incinerated. In the U.S. certain toxic chemicals and hospital wastes are incinerated, but municipal refuse generally is not, partly because of the commonly held image of inefficient old-style systems, but also because of cost. New technology makes it possible to operate at higher temperatures with better control of combustion conditions. Also, new incinerators have highly efficient air pollution control systems. Burning wastes for heat recovery should be technically feasible in many situations. Economic feasibility is a separate question.

The heart of an incinerator system is the combustion chamber. The most used *mass burn* technologies for municipal solid wastes are moving grate and rotary kiln furnaces. Mass burn means the refuse is burned as it arrives, or with minimal pretreatment. Fluidized beds are an option but pretreatment (size reduction and sorting) or selective collection of the waste will be required.

Sludge incineration is done with rotary kilns, multiple hearth, and fluidized bed systems.

Table 5.7 summarizes the successful applications in the European Community.

Technique	Untreated Municipal Waste	Pretreated MSW & RDF	Hazardous waste	Sewage Sludge	Clinical Waste
Moving grate	+++	+++	+	+	++
Rotary kiln	+	++	+++	++	+++
Static hearth	+	+	++	+	+++
Static furnace	+	+	+++	+	++
Fluid bed, bubbling		++	+	++	+
Fluid bed, circulating		++	+	+++	+
Key: + means widely used, ++ means frequently used, +++ means sometimes used, and blank means rarely used.					

Table 5.7 Summary successful application of thermal treatment techniques to the main waste types at dedicated installations (European Commission 2006)

Figure 5.3 is a schematic diagram of a moving grate furnace with a boiler and a wet scrubber and baghouse filter for air pollution control. A variety of air pollution control technologies can be used depending on the characteristics of the emissions.

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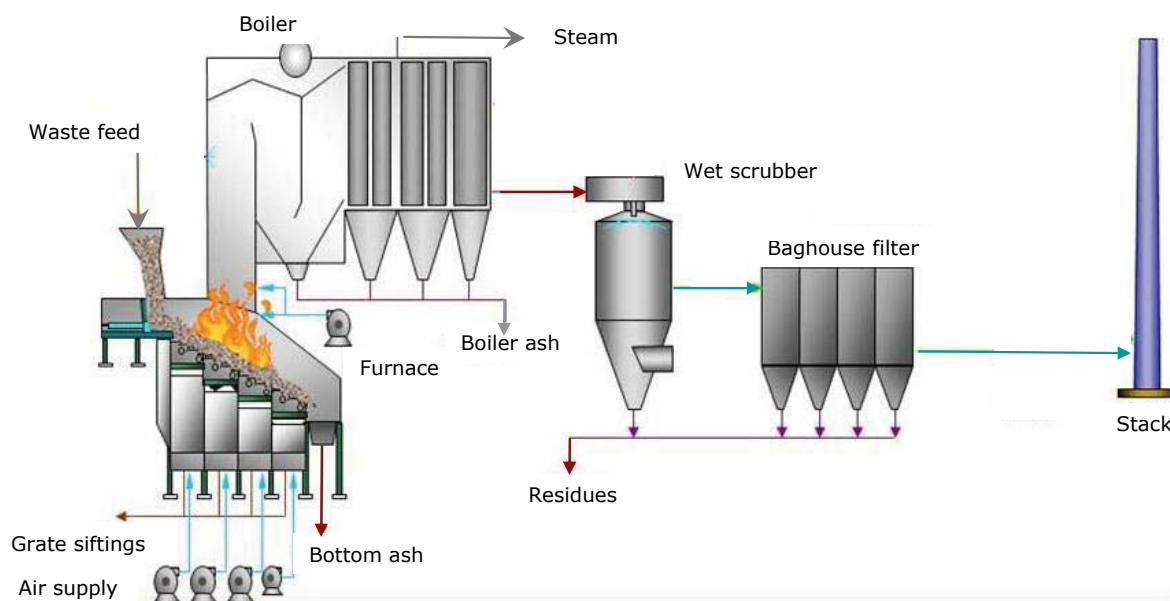


Figure 5.3 Moving grate furnace with boiler and air pollution control equipment

The energy released by the combustion of the organics in the waste may not be sufficient to reach the desired reactor temperature, especially if the material is moist, and auxiliary fuel such as natural gas may be needed. Auxiliary air is needed to get complete combustion.

Incineration produces bottom ash, slag, and fly ash. Depending on the methods used for air pollution control, there may also be wastewater from ash handling and flue gas scrubbing.

Mass burning takes a non-homogeneous waste with little or no pretreatment (sorting, grinding, etc.). The usual furnace technology is rotary kiln or moving grate.

Burning pretreated and homogenized waste, as discussed earlier in this chapter, requires size reduction, shredding and sorting. Moving grate furnaces are the proven technology.

5.6 CASE STUDY – PALM BEACH RENEWABLE ENERGY FACILITY NO. 2

This case study will illustrate the complexity of a real project and show how some of the elements previously discussed are implemented.

The Palm Beach Renewable Energy Facility No. 2 is a waste-to-energy (WTE) power plant built for Solid Waste Authority (SWA) of Palm Beach County in West Palm Beach, Florida. It is the first WTE project of its kind in the U.S. in more than 20 years. Construction started in 2012 and commercial operation started in 2015 (Kitto et al 2016, Schauer 2016).

This was a design/build contract and an O&M agreement. The design/build lump sum contract cost was \$668,000,000. The first-year operation and maintenance fee was \$20,490,000.

The plant processes 3000 tons per day (1,000,000 ton/y) of mostly unprocessed municipal solid waste, enough so the amount of waste landfilled is reduced by more than 90%. Metals recovery is 27,000 tons/year.

The heat generated by the combustion is used to produce steam (284,400 lb/h) in three Babcock & Wilcox boilers that drive a turbine generator to produce 95 MW of electric power that is sold to Florida Power & Light. Natural gas is used as an auxiliary fuel.

Air pollution controls are scrubbers to remove acid gases, pulse-jet fabric filters for particulate removal, and selective catalytic reduction to remove nitrogen oxides. The uncontrolled NO_x emission level is 250 ppmv in an actual flue gas volumetric flow of 5,219 m^3/min . The catalytic reduction system will remove 541,182 kg NO_x/year .

Figure 5.4 shows moving grate furnace with a steam boiler and economizer to recover heat (economizers are discussed in Chapter 7) and the first two stages of the air pollution control system. The spray dryer absorber injects water to capture acid gases (primarily sulfur and chlorine) and other air pollutants. The pulse-jet fabric filter removes particulates. 'Pulse-jet' refers to the method of cleaning the fabric filters. Figure 5.5 shows the selective catalytic reduction (SCR) for removing NO_x . This is one of the cleanest renewable energy facilities in the U.S.

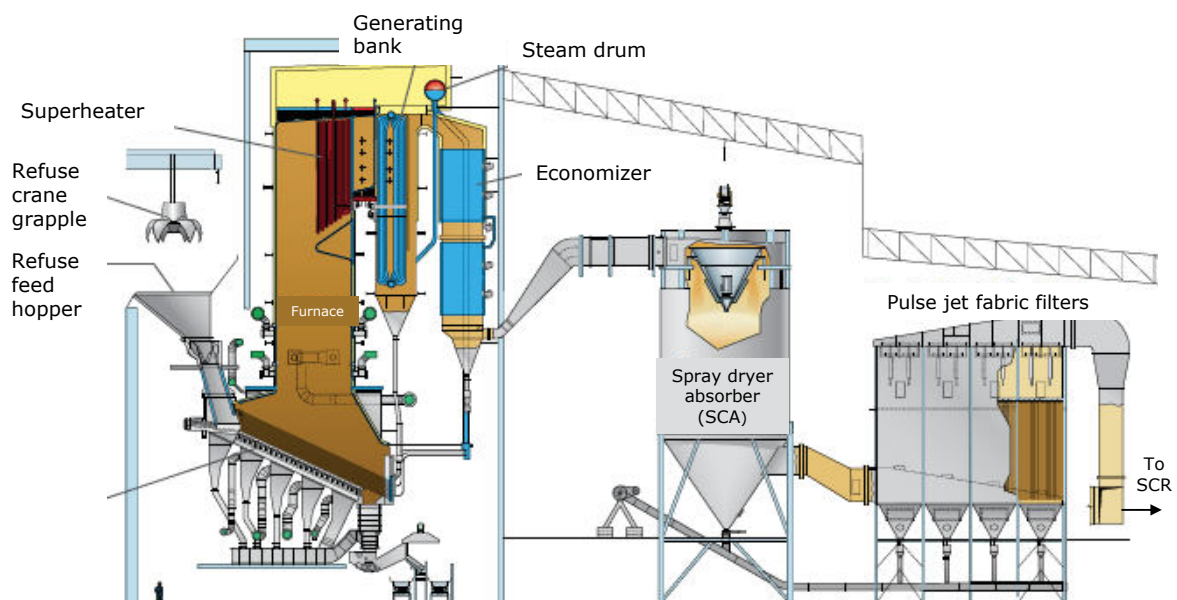


Figure 5.4 Palm Beach Renewable Energy Facility No. 2 furnace, boiler, economizer for heat recovery and fabric filters for particulates removal. Turbine generator is not shown.

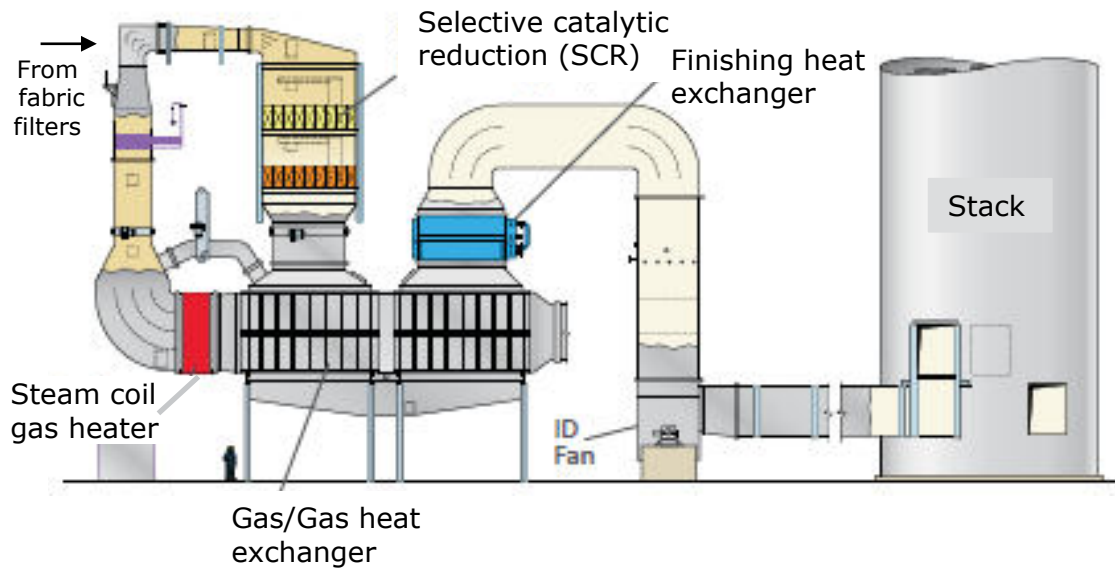


Figure 5.5 Selective catalytic reduction for NO_x control.

Bottom and fly ash are delivered, along with any metallic items, to an ash management system. A rotary magnet removes ferrous metals and an eddy current separator removes nonferrous metals from the ash to be sold on the scrap metal market.



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5.7 CONCLUSION

Solid waste and sludge incineration is one way of recovering heat energy, steam or electricity from the waste disposal process. The kind of incinerator determines how much pretreatment is required before waste is fed to the furnace. Some installations are so efficient they are called waste-to-energy plants.

6 ENERGY RECOVERY FROM BIOGAS

Biogas, a mixture of methane and carbon dioxide with traces of other gases, is fuel quality gas that is produced by a biological process called *anaerobic fermentation*. The process is used for municipal sludge digestion, agricultural manure management, and treatment of strong organic industrial wastes, and it occurs naturally in landfills. In many application the gas is cleaned of impurities before the methane is burned.

A variety of materials can be used as feedstock, such as food waste, sewage sludge, animal manure, and agricultural biomass. Figure 6.1 is a generic process that shows gas generation in a sludge digester or landfill with supernatant and residual solids disposal, and four alternatives for beneficial use of the biogas. Storage is needed because the rates of production and use are not constant.

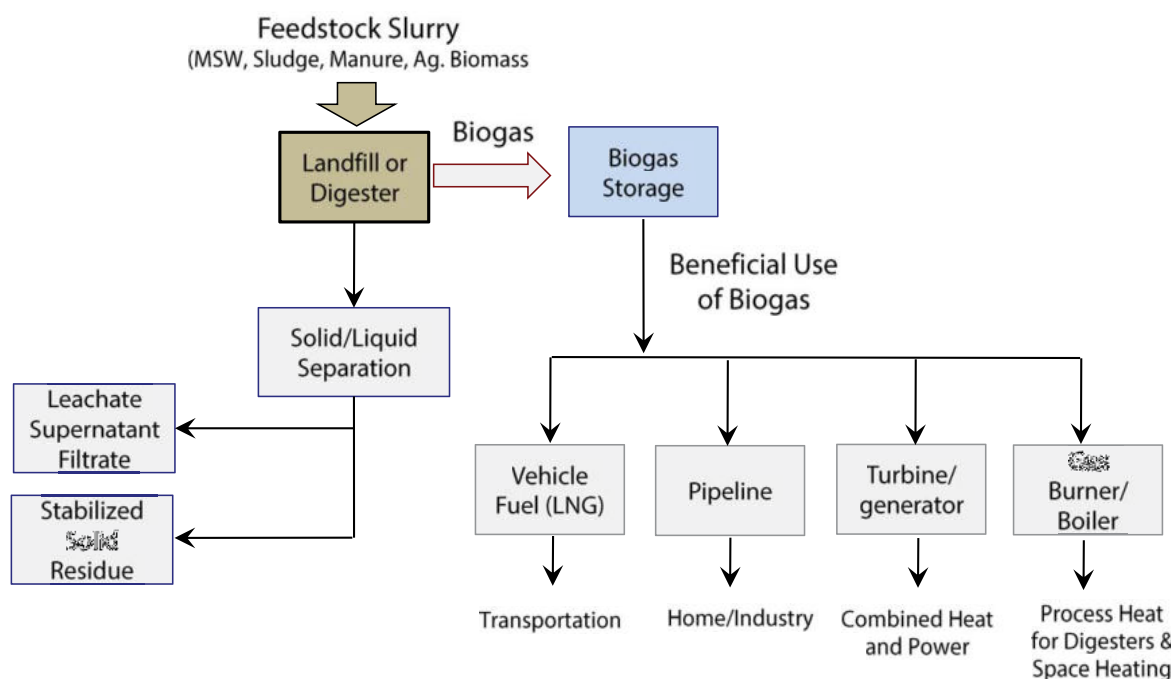


Figure 6.1 Biogas production and options for biogas use.

Whatever the feedstock, biogas production includes a sequence of chemical and biochemical reactions that solubilize some materials, breakdown complex molecules into simple organic acids (e.g., acetic), and convert the organic acids to methane and carbon dioxide. The output from the process, in addition to biogas, is stabilized organic solids, and liquid supernatant filtrate from sludge concentration processes.

There are four general options for using the biogas. Biogas can be used on-site for heating or generating electricity, or it can be converted to pipeline quality methane gas by removing most of the carbon dioxide and hydrogen sulfide. The traditional use is fuel for heaters and boilers, or fuel for engines to drive air compressors or electric generators. These kinds of energy recovery systems are well established at landfills and wastewater treatment plants. They are becoming popular for managing animal wastes. Upgrading biogas for use in vehicles is a new and promising technology.

6.1 COMPOSITION OF BIOGAS

The methane (CH_4) content of the gas can range from 50% (in landfills) to 80% (in sludge digesters). These are volume percentages. Methane and carbon dioxide typically account for 98-99% of the gas; the remaining fraction will contain hydrogen sulfide, hydrogen, nitrogen, and other trace gases. More detail is given in Table 6.1.



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Gas Characteristic	Units	Landfill gas	Anaerobic Digester Gas	Natural gas
Lower heating value (LHV)	MJ/Nm ³	16	23	39
	kWh/Nm ³	4.4	6.5	11.0
	MJ/kg	12.3	20	48
Density	kg/Nm ³	1.3	1.1	0.82
Relative density	-	1.1	0.9	0.63
Methane number (a)		>130	>135	73
Methane	vol %	45	65	90
Methane, range	vol %	35-65	60-70	85-92
Heavy hydrocarbons	vol %	0	0	9
Hydrogen	vol %	0-3	0	-
Carbon dioxide	vol %	40	35	0.7
Carbon dioxide, range	vol %	15-40	30-40	0.2-1.5
Nitrogen	vol %	15	0.2	0.3
Nitrogen, range	vol %	5-40	-	0.3-1.0
Oxygen	vol %	1	0	-
Oxygen, range	vol %	0-5	-	-
Hydrogen sulfide	ppm (vol.)	<100	<500	3.1
Hydrogen sulfide, range	ppm (vol.)	0-100	0-4000	1.1-5.9
Ammonia	ppm (vol.)	5	100	-
Total chlorine as Cl ⁻	mg/Nm ³	20-200	0-5	
<p>a) The methane number is a measure of fuel quality for a gas mixture that contains methane. Methane has a value of 100; other gases have a higher value so a gas mixture can be valued at more than 100. Engine manufacturers will set a minimum acceptable value for methane number, lower heating value, and maximum H₂S content. The methane number for natural gas is typically between 70 and 92.</p> <p>Standard conditions: scf = standard cubic foot = 1 ft³ at 60 °F, 1 atmosphere and 0% relative humidity Nm³ = normal cubic meter = 1 m³ at 0 °C, 1 atmosphere, and 0% relative humidity Volumetric conversion factor for gases: 1 Nm³ = 37.326 scf</p>				

Table 6.1 Composition of natural gas and biogas from landfills and anaerobic digesters (Source: Swedish Gas Center 2012.)

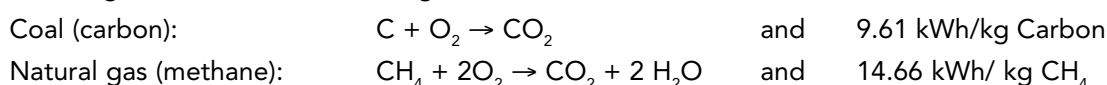
Natural gas has the highest energy/carbon ratio of any fossil fuel, and thus produces less carbon dioxide per unit of energy. The primary component of natural gas is methane (CH₄),

the lightest hydrocarbon molecule, but it may also contain ethane (C_2H_6), propane (C_3H_8), butane (C_4H_{10}) and other gases in lesser amounts. Combustion of one cubic meter of natural gas yields 38 MJ (10.6 kWh or 36,000 Btu).

EXAMPLE 6.1 CLEAN ENERGY

Natural gas is often touted as the 'clean energy' alternative to coal. Both fuels produce the same amount of CO_2 per mass of carbon burned, but natural gas yields more energy per mass of CO_2 produced.

Natural gas is about 95% methane and coal is about 95% carbon. For simplicity, use 100% carbon and methane as surrogates for coal and natural gas. The balanced chemical reactions and fuel values are:



Coal (carbon) combustion releases 44 kg CO_2 for every 12 kg carbon burned and
 $(9.61 \text{ kWh/kg C})(12 \text{ kg C}/44 \text{ kg } CO_2) = 2.62 \text{ kWh released/ kg } CO_2 \text{ produced}$

Natural gas (methane) combustion produces 44 kg CO_2 for every 16 kg methane burned and
 $(14.66 \text{ kWh/kg } CH_4)(16 \text{ kg } CH_4/44 \text{ kg } CO_2) = 5.33 \text{ kWh released/kg } CO_2 \text{ produced}$

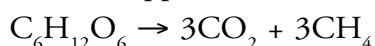
Natural gas combustion produces about twice the energy per mass CO_2 produced and is 'the cleaner fuel'. Burning coal also has associated problems with sulfur and particulate emissions as well as disposal of ash (from the 5% or so of coal that is not carbon), not including serious environmental problems with coal mining.

6.2 ENERGY RECOVERY FROM LANDFILL GAS

The U.S. EPA estimated that the approximately 6,000 landfills in the U.S. produce an estimated 13 - 18 billion m^3 (450 – 650 billion ft^3) of methane. Modern practice is to extract methane from the landfill and use it. Methane gas has twenty times the global warming potential (GWP) of carbon dioxide, so collecting it for use as fuel is beneficial in terms of energy produced and reduction of greenhouse gases. Just over half of landfill-produced CH_4 is recovered and burned through flaring or electricity generation. The number of landfill gas projects using the methane gas from decomposing refuse went from 399 in 2005 to 594 in 2011 (U.S. EPA).

Landfill gas also contains varying amounts of nitrogen, oxygen, water vapor, hydrogen sulfide, and other contaminants. The U.S. EPA has identified 94 non-methane organic compounds (NMOCs), including toxic chemicals like benzene, toluene, chloroform, vinyl chloride, and carbon tetrachloride. At least 41 of the non-methane organic compounds contain chlorine, fluorine, or bromine. These compounds usually make up less than one percent of landfill gas but they cannot be neglected.

Cellulose and hemicellulose, the most common biodegradable components in refuse, decompose under anaerobic conditions to yield equal parts of methane and carbon dioxide. A simplified and approximate reaction, using $C_6H_{12}O_6$ as an empirical 'refuse molecule' is



The yield is 3 moles of CO_2 and 3 moles of CH_4 , which is 50% methane and 50% carbon dioxide as volume percent. Landfill gas from the most productive phase is approximately 50% to 60% methane, and 40% to 50% carbon dioxide. When the landfill is young (< 5 years) or very old (> 25 years), the methane content may be only 10-20%.

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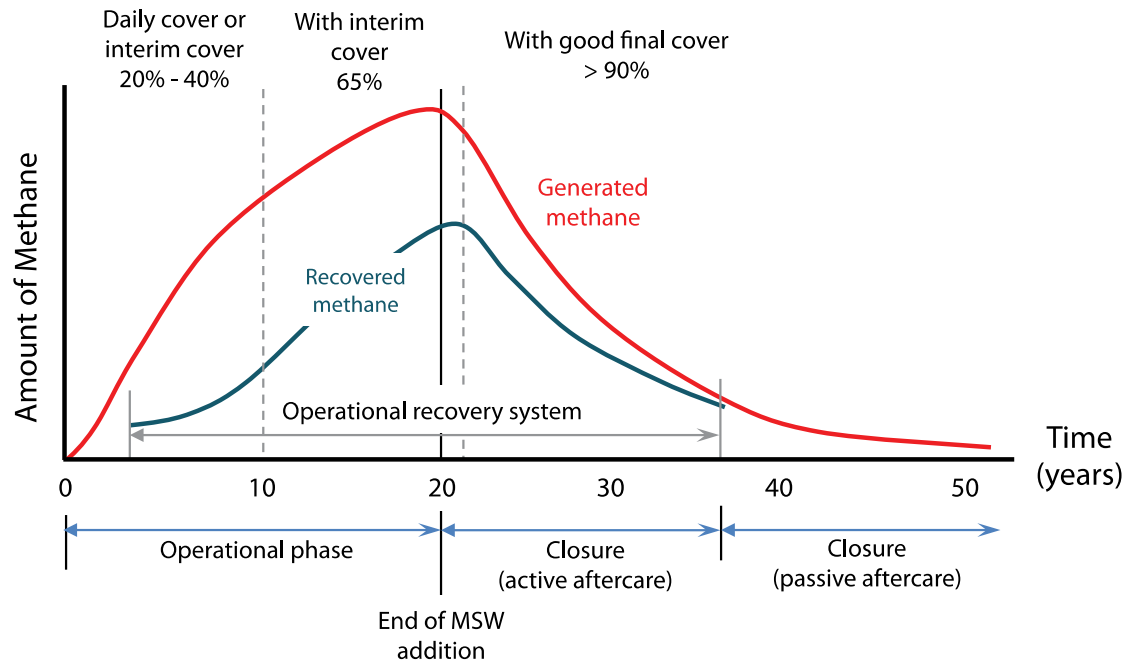


Figure 6.2 Conceptual model of the development of methane production in a landfill.

Figure 6.2 shows that the volume and quality of gas from a landfill changes over time. They also vary from location to location within a landfill as new cells are developed and older ones are sealed. When the refuse is first placed and covered, the gas within the buried material is essentially air (oxygen and nitrogen). Within a fairly short time the oxygen is consumed and the initial aerobic conditions turn anaerobic, the pH and moisture content become tolerable for the methane forming bacteria and fermentation starts to produce organic acids (acetic, etc.). The economically interesting phase has useful volumes of methane being produced continuously over a number of years. Finally, the organic substrate necessary for methane production is exhausted, and the volume of gas produced and the methane content of the gas decline below useful levels.

Constant gas production increases the pressure within the landfill and causes lateral and vertical gas migration. Migration to the surface adds to the inventory of atmospheric greenhouse gases, which should be prevented. Landfills are capped with an impervious layer to reduce the entry of water. This also reduces the emission of gas.

Lateral gas migration can cause environmental, hygiene, and security problems. Accidents have occurred when gas moved out of the landfill and into nearby buildings and houses. There have been explosions that destroyed property and killed people.

The best way to control gas migration is extraction. The extracted gas needs to be burned in a flare or used as a fuel. Figure 6.3 shows gas extraction wells installed in a mature landfill with the gas being conveyed by a vacuum system to a point of beneficial use.

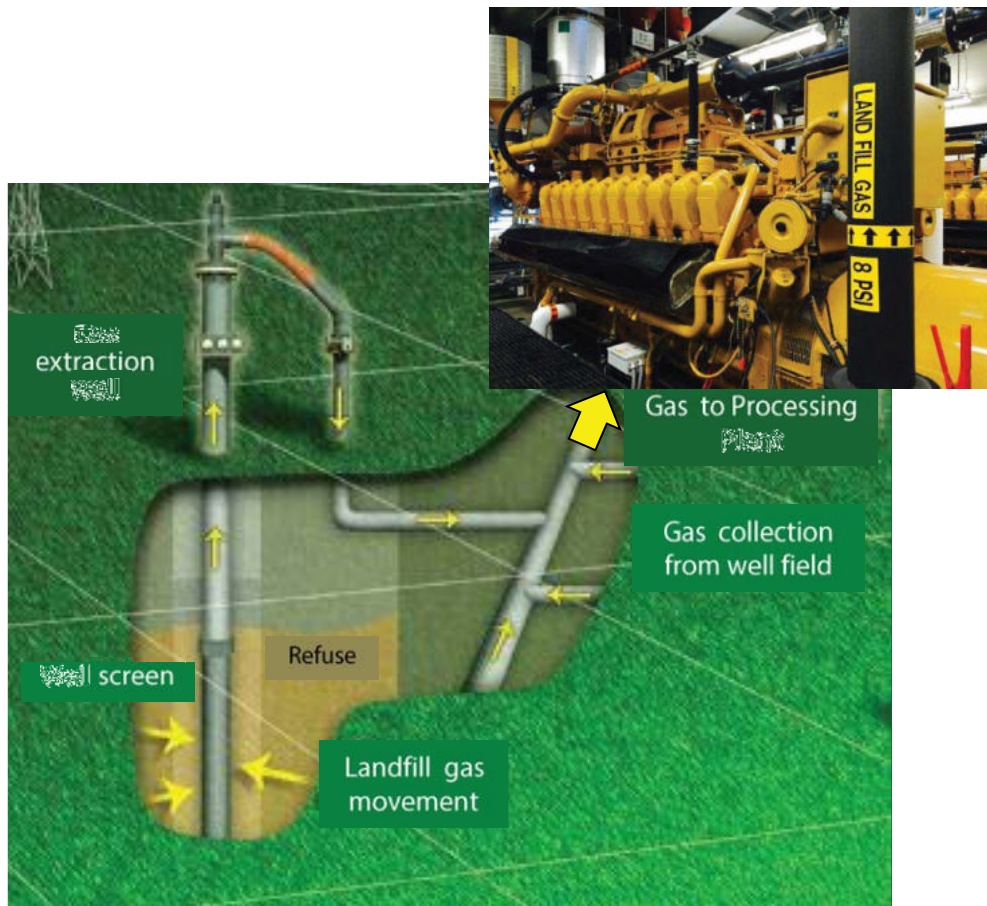


Figure 6.3 Landfill gas collection system

A Dow Chemical plant, Figure 6.4, uses methane from a local landfill in a manufacturing plant in Dalton, GA. The gas is used to generate steam for the manufacture of latex carpet backing. This site is expected to use approximately 169 million MJ (160 billion Btu) per year of landfill gas (the energy equivalent of 5.3 million liters of gasoline (1.4 million gallons)).

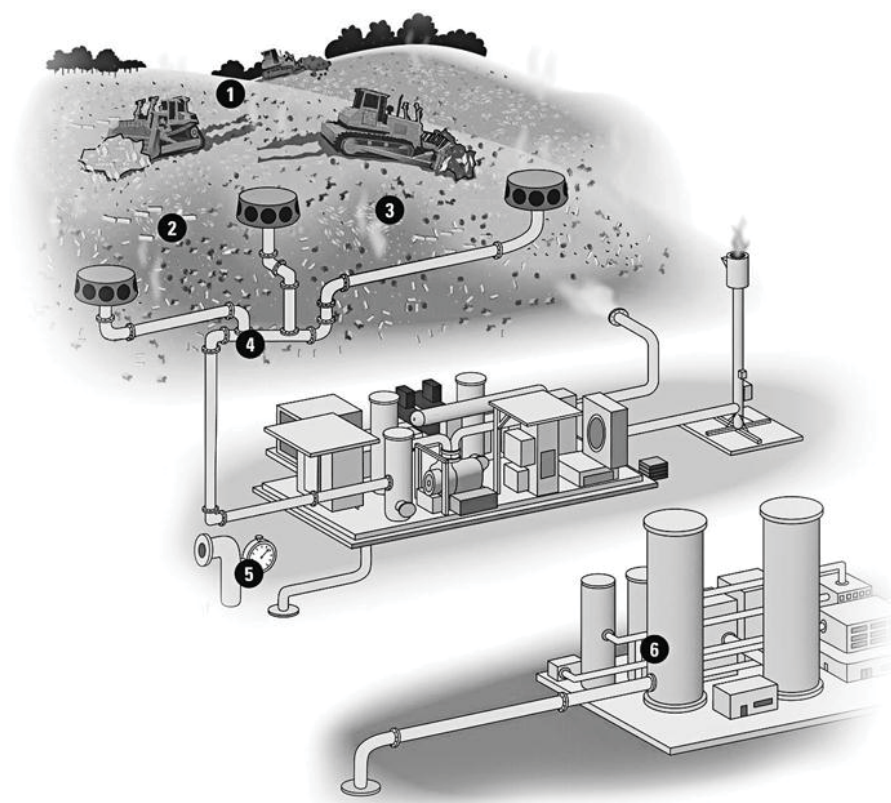


Figure 6.4 Landfill gas is extracted from wells, processed to remove hydrogen sulfide and other corrosive or harmful compounds and burned to produce heat or electricity. The landfill gas system for energy in Dalton, Georgia is shown in this diagram. (1) Waste is deposited in the landfill. (2) Anaerobic bacteria decompose the municipal solid waste. (3) Methane off-gas is generated. (4) A system of pipes and blowers collects gas and delivers it to a central location. (5) Gas is used as fuel to make steam. (6) Steam is used to manufacture latex carpet backing. (Patt & Banholzer, 2009)

Landfill gas can be managed by flaring (wasting by burning) or it can be used as fuel for boilers, internal combustion engines, gas turbines, or fuel cells. Methane can be converted to methyl alcohol, or it can be cleaned and piped to other industries or into natural gas lines. Hydrogen sulfide (H_2S) must be removed prior to most uses so sulfuric acid is not formed during combustion.

EXAMPLE 6.2 LANDFILL GAS

Gas will be collected from a landfill that receives the solid waste from a city of 200,000 people. Assuming 900 kg of refuse per year per person, the landfill has been receiving 180,000,000 kg/y, or about 500,000 kg/d. Each kg of municipal refuse put into a landfill yields 0.41 m^3 of landfill gas that is about 50 percent methane and 50 percent carbon dioxide. The lower heating value of methane is about 50,000 kJ/kg, or 33.6 MJ/m^3 (9.33 kWh/m^3).

Expected gas production = $(500,000 \text{ kg/d})(0.41 \text{ m}^3/\text{kg}) = 205,000 \text{ m}^3/\text{d}$

Lower heating value of methane = 9.33 kWh/m^3

Landfill gas is approximately 50% methane

Approximate lower heating value of the landfill gas = 4.7 kWh/m^3

Energy value of the gas, based on the lower heating value, is

$$(205,000 \text{ m}^3/\text{d})(4.7 \text{ kWh/m}^3) = 964,000 \text{ kWh/d}$$

In a landfill the liquid is a strong concentrate called *leachate*. Landfills are constructed with drainage systems to collect and remove leachate. The stabilized solids, of course, remain in the landfill.

6.3 MODEL FOR LANDFILL GAS PRODUCTION

Economic decisions related to beneficial use of landfill gas depend on predictions of the volume and quality of the gas that can be brought to a point of use. A simple model is given to predict methane production. If the gas is fifty percent methane the total gas production is double the model's predicted value for methane. This model does not predict how much of the biogas can be extracted from the landfill.

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The rate of decomposition of a waste mass in a landfill is initially fast but it slows over time. A useful approximation is the exponential decay model, also known as a first-order model.

$$M_t = M_0 e^{-kt}$$

where M_0 is the initial mass and M_t is the mass remaining at time t . The mass of refuse decomposed in the first year is

$$M_0 - M_1 = M_0 - M_0 e^{-k(1y)} = M_0(1 - e^{-k(1y)})$$

Define L as the potential landfill generation of CH_4 at time t , in $\text{m}^3 \text{CH}_4/\text{T}$ of wet refuse placed in the landfill. The methane generation potential at time $t = 0$ is $M_0 L$. The methane potential at the end of year 1 is $M_1 L$. The amount of methane generated in year 1 is

$$Q_{\text{CH}_4,1} = LM_0 - LM_1 = L(M_0 - M_1)$$

If the landfill gas is 50% methane, the total gas production is

$$Q_{\text{Gas}} = 2Q_{\text{CH}_4}$$

This calculation can be repeated for each year. The cumulative gas production is the sum of the annual amounts.

The U.S. EPA Landfill Gas Emissions Model (LandGEM) model is also based on exponential decomposition and gas production rates (U.S. EPA 2009).

Example 6.3 shows how methane production is predicted for a landfill that contains one million tons of municipal refuse. A more elaborate calculation is required for a working landfill where refuse is deposited over a period of time and then the landfill is closed and capped. Methane production will increase during the working years and then decrease when the mature landfill is closed. This is the pattern shown in Figure 6.2. The basic calculations are the same, but an accounting system is needed to keep track of when the deposits are made and how much methane is generated from each deposit over time.,

EXAMPLE 6.3 LANDFILL GAS PRODUCTION

Use the landfill gas generation model to predict methane production from a landfill that has $M_0 = 1,000,000 \text{ T}$, $k = 0.045/\text{y}$, and initial methane generation potential of $L_0 = 170 \text{ m}^3/\text{T}$.

Prediction accuracy is improved by using steps of 0.1 years; one-year steps over predict. For simplicity of instruction one year steps will be used.

Sample calculation

Mass of refuse remaining in the landfill

$$\text{End of Year 1 } M_1 = M_0 e^{-kt} = (1,000,000 \text{ T})e^{-(0.045/\text{y})(1 \text{ y})} = 955,997 \text{ T}$$

$$\text{End of Year 2 } M_2 = M_0 e^{-kt} = (1,000,000 \text{ T})e^{-(0.045/\text{y})(2 \text{ y})} = 913,931 \text{ T}$$

$$\text{End of Year 3 } M_3 = M_0 e^{-kt} = (1,000,000 \text{ T})e^{-(0.045/\text{y})(3 \text{ y})} = 873,716 \text{ T}$$

$$\text{End of Year 4 } M_4 = M_0 e^{-kt} = (1,000,000 \text{ T})e^{-(0.045/\text{y})(4 \text{ y})} = 834,270 \text{ T}$$

Mass of refuse decomposed

$$\text{Year 1 } M_0 - M_1 = 1,000,000 - 955,997 = 44,033 \text{ T/y}$$

$$\text{Year 2 } M_1 - M_2 = 955,997 - 913,931 = 42,066 \text{ T/y}$$

$$\text{Year 3 } M_2 - M_3 = 913,931 - 873,716 = 40,125 \text{ T/y}$$

$$\text{Year 4 } M_3 - M_4 = 873,716 - 834,270 = 38,446 \text{ T/y}$$

Gas Yield

$$\text{Year 1 } L_0(M_0 - M_1) = (170 \text{ m}^3/\text{T})(44,033 \text{ T/y}) = 7,480,428 \text{ m}^3/\text{y}$$

$$\text{Year 2 } L_0(M_1 - M_2) = (170 \text{ m}^3/\text{T})(42,066 \text{ T/y}) = 7,151,270 \text{ m}^3/\text{y}$$

$$\text{Year 3 } L_0(M_2 - M_3) = (170 \text{ m}^3/\text{T})(40,125 \text{ T/y}) = 6,836,597 \text{ m}^3/\text{y}$$

$$\text{Year 4 } L_0(M_3 - M_4) = (170 \text{ m}^3/\text{T})(38,446 \text{ T/y}) = 6,248,179 \text{ m}^3/\text{y}$$

Year	Solids remaining (T)	Methane produced (m ³ /y)	Year	Mass remaining (T)	Methane produced (m ³ /y)
0	1,000,000	0			
1	955,997	7,480,428	11	609,571	4,769,732
2	913,931	7,151,270	12	582,748	4,559,851
3	873,716	6,836,597	13	557,106	4,359,206
4	834,270	6,535,769	14	532,592	4,167,390
5	798,516	6,248,179	15	509,156	3,984,015
6	763,379	5,973,243	16	486,752	3,808,708
7	729,789	5,710,405	17	465,334	3,641,115
8	697,676	5,459,133	18	444,858	3,480,897
9	666,977	5,218,918	19	425,283	3,327,729
10	637,628	4,989,272	20	406,570	3,181,300

Table 6.2 Methane produced in a landfill

6.4 ENERGY RECOVERY FROM ANAEROBIC SLUDGE DIGESTION

Sludge digesters, Figure 6.5, in wastewater treatment plants convert biodegradable organic matter into methane and carbon dioxide. Figure 6.6 shows an agricultural system. The anaerobic digesters are lagoons with a flexible cover to capture and store the gas. The stabilized manure or digested sludge solids are stored until it can be spread on farmland or otherwise disposed.

Wastewater treatment plants use *anaerobic sludge digestion* to stabilize sludge that is highly offensive and unhealthful. Digestion makes the sludge easier to dewater so the volume can be reduced before composting, drying, or spreading on farmland.



Figure 6.5 Anaerobic sludge digesters in wastewater treatment plants (Photos courtesy of Shutterstock)

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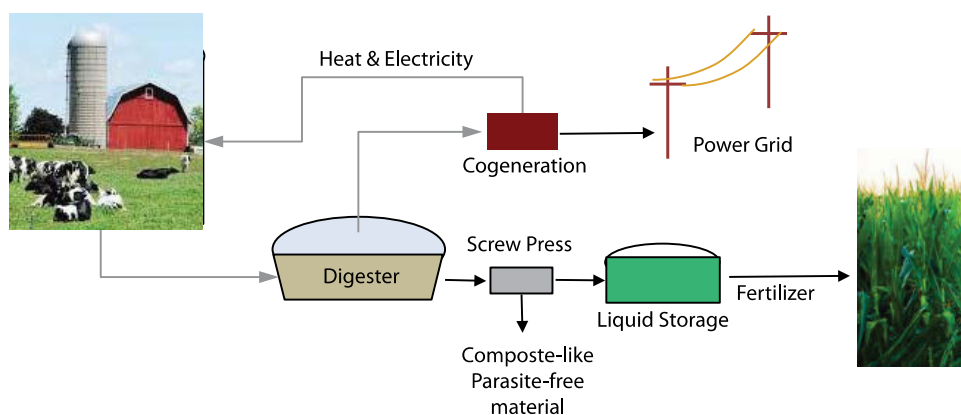


Figure 6.6 A biological process for managing animal manure to produce usable gas.

Biogas is 50-70% methane, and 30-50% carbon dioxide, with traces of hydrogen sulfide (1%-3%), small amounts of nitrogen and hydrogen, and traces other gases (mercaptans, etc.). The higher percentage of methane is for municipal sludge digestion; the lower is for manure digestion systems.

The gas is used to heat raw sludge as it enters the digester and to maintain the desired internal temperature of the digester, which is 35°C or 55°C, depending on the type of process that is used. Excess gas can be used to heat buildings, to fuel pump or blower engines, or to generate electricity.

The digested sludge is a slurry of stabilized solids and strong liquid. Solids are removed from the liquid by filtering or centrifuging. The solids are often spread onto farmland. The filtrate and centrate are recycled to the wastewater treatment plant.

The conventional *mesophilic* anaerobic sludge digester (MAD) operates at 35°C with a detention time of 20-30 days and achieves 45-50% volatile solids reduction. *Thermophilic* anaerobic digestion (TAD) operates at a temperature of 55°C and a detention time of 20 days has been gaining favor over the traditional mesophilic process because it accomplishes more solids destruction, yields more biogas, greatly reduces pathogenic organisms in the sludge, and the digested sludge is more easily dewatered. TAD and MAD can be combined as a two-stage digestion process called *temperature-phased anaerobic digestion* (TPAD).

The design of anaerobic processes is based on sludge age or loading rates. Loading rates for sludge digesters are given as kilograms of volatile solids per day per unit of digester volume (kg VS/m³-d). The empirical relations for gas production in anaerobic sludge digestion are based on VS destruction.

1 kg VS destroyed → 0.8 - 1.1 m³ gas produced (70% CH₄ + 30% CO₂)

1 kg VSS destroyed → 0.55 - 0.75 m³ CH₄ produced

Organic loadings are used for anaerobic treatment of soluble industrial wastes. Waste strength is measured as COD (or BOD) even though there is no reaction with oxygen in an anaerobic fermentation. Methane production is approximately 0.35 m^3 of methane produced per kg COD removed ($5.6 \text{ ft}^3/\text{lb}$ COD removed). The fermentation gas is typically 65-70% methane, so the total gas volume produced is about $0.5 \text{ m}^3/\text{kg}$ COD removed. An estimate for total gas production, G , for anaerobic industrial waste treatment can be made using

$$1 \text{ kg COD destroyed} \rightarrow 0.30 - 0.45 \text{ m}^3 \text{ CH}_4 \text{ produced}$$

EXAMPLE 6.4 SOLIDS BALANCE FOR AN ANAEROBIC SLUDGE DIGESTER

Figure 6.7 shows sludge entering an anaerobic digester at a total solids concentration of 4% solids by dry weight; 70% of the solids are organic and 30% are inert. This is typical of municipal sewage sludge. The digested sludge solids are 50% organic and 50% inert.

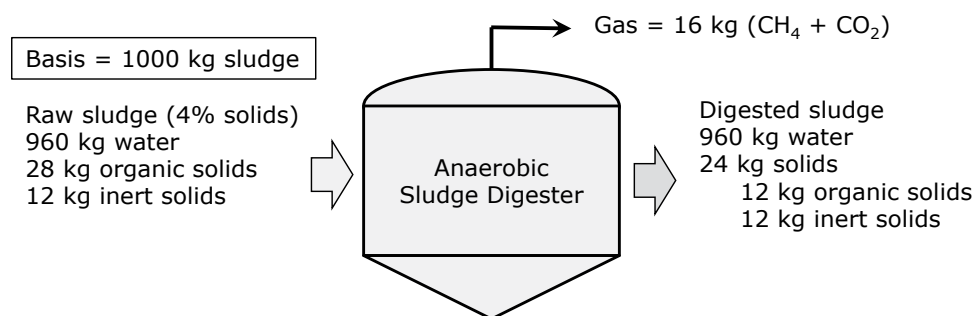


Figure 6.7 Anaerobic sludge digester.

Assumptions

- Flow of sludge into and out of the digester is continuous.
- Volume of sludge in the digester does not change with time.
- All inert solids leave in the digested sludge.
- Organic solids (volatile solids) that are not converted leave in the digested sludge
- Digester mixing keeps all solids in suspension.
- Water entering with the sludge leaves with the sludge.
- Evaporation can be neglected.

Basis: Feed of 1000 kg of sludge that is 4% total solids by weight.

Tie components: inert solids (fixed solids) and water.

Inputs:

1000 kg wet sludge = 960 kg water + 40 kg total solids

The solids (TS) are 25% fixed and 75% volatile (organic).

The digestion converts 50% of the volatile solids to gas.

Fixed solids (FS) = $(0.25 \text{ kg FS/kg TS})(40 \text{ kg TS}) = 10 \text{ kg FS}$

Volatile solids (VS) = $(0.75 \text{ kg VS/kg TS})(40 \text{ kg TS}) = 30 \text{ kg VS}$

Gas Outputs (VS destruction = 50%):

Conversion of VS to gas = $0.5(30 \text{ kg VS}) = 15 \text{ kg VS destroyed}$
= 15 kg VS converted to gas

Sludge Outputs:

Water = 960 kg

Fixed solids = 10 kg

Volatile solids = $30 \text{ kg input} - 15 \text{ kg converted to gas} = 15 \text{ kg}$

Totals = $10 \text{ kg FS} + 15 \text{ kg VS} = 25 \text{ kg TS}$

Sludge mass = $960 \text{ kg water} + 25 \text{ kg TS} = 985 \text{ kg}$

Solids concentration = $(25 \text{ kg TS}) / (985 \text{ kg sludge}) = 2.54\%$

Gas production = $0.8 - 1.1 \text{ m}^3/\text{kg VS destroyed}$.

Assume $1 \text{ m}^3/\text{kg VS destroyed}$

Gas production = $(1 \text{ m}^3/\text{kg VS destroyed})(15 \text{ kg VS destroyed}) = 15 \text{ m}^3 \text{ gas}$

Assumes gas is 75% methane

Methane produced = $0.75(15 \text{ m}^3) = 11.25 \text{ m}^3 \text{ CH}_4 \text{ produced}$.

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The solids concentration of the sludge leaving the digester is 2.54% (25 kg solids in 985 kg of digested sludge). The solids concentration leaving the process is less than entering. There has been no concentration of solids in the digester, but the digested sludge will be thickened or dewatered before disposal.

The net loss of solids is 15 kg. This is not much. The important gains have been the production of useable gas and the production of a less odorous material than can be rather easily dewatered for a trouble-free and perhaps even beneficial ultimate disposition of the material.

EXAMPLE 6.5 DIGESTER GAS FOR MICROTURBINE ELECTRICITY GENERATION

The Village of Grafton, WI, burns digester gas in a microturbine that will generate 30 kW at a digester gas feed rate of 13.13 ft³/min (0.374 m³/min). The microturbine will operate 90 percent of the time and has an O&M cost of \$0.025/kWh. Electric energy is worth \$0.08 kWh and recovered heat energy is worth \$6 per million Btu. (Source: Vik 2003)

Considering only operating costs, calculate the payback time for installing a microturbine.

Total solids to digester = 1,714 kg TS/d = 3,773 lb TS/d

Volatile solids fraction = 75%

Volatile Solids to digestion = (0.75 kg VS/kg TS)(1,714 kg TS/d)
= 1,286 kg VS/d = 2,834 lb VS/d

VSS reduction by digestion = 50%

VSS Destroyed = (0.5)(1,286 kg VS/d)
= 643 kg VS destroyed/d = 1,417 lb VS destroyed/d

Digester gas yield factor = 0.93 m³/kg VS destroyed = 15 ft³/lb VS destroyed

Gas Production = (0.93 m³/kg VS destroyed)(643 kg VS destroyed/d)
= 598 m³/d = 21,100 ft³/d

Methane production (gas = 70% methane) = (0.7)(598 m³/d) = 419 m³/d

Energy value of methane = (419 m³/d)(9.33 kWh/m³) = 3,909 kWh/d
= 14,070,000 kJ/d = 13,340,000 Btu/d

Microturbine (Capstone C30 Biogas) power generating potential = 30 kW

Microturbine electricity generating potential = 720 kWh/d

Annual utilization at 90% = (0.9)(24 h/d)(365 d/y) = 7884 h/y

Value of electricity generated at 7,884 h/y and \$0.08/kWh
= (30 kW)(7,884 h/y)(\$0.08/kWh) = \$18,922/y (use \$ 19,000/y)

Heat recovered from the microturbine at 25% efficiency = (0.25)(14,070,000 kJ/d)
= 3,518,000 kJ/d = 3,334,000 Btu/d

Value of heat recovered from the microturbine at 7,884 h/y utilization and \$6/1,000,000 Btu
= (3,334,000 Btu/d)(d/24h)(7,884 h/y)(\$6/1,000,000 Btu)
= \$6,571/y (use \$6,600/y)

Annual O&M costs of operating a microturbine at \$0.025/kWh
 $= (\$0.025/\text{kWh})(30 \text{ kW})(7,884 \text{ h/y}) = \$5,913/\text{y}$ (use \$5,900)

Microturbine purchase and installation = \$140,000

Payback period = $\$140,000/(\$19,000 + \$6,600 - \$5,900)$
 $= \$140,000/\$19,700 = 7.1 \text{ y.}$

EXAMPLE 6.6 ENERGY BALANCE FOR ANAEROBIC DIGESTION

The three digestion systems are compared in Figure 6.8. In each case the digester feed is $1,000 \text{ m}^3/\text{d} = 1,000 \text{ T/d}$. The volume of the sludge leaving the digester is $1,000 \text{ m}^3/\text{d}$, but the mass is less than $1,000 \text{ T/d}$ because some volatile solids are converted to gas.

The sludge is 3% solids by weight and the volatile solids fraction is 0.7, giving $30,000 \text{ kg/d}$ of total solids, $21,000 \text{ kg/d}$ of volatile solids and $9,000 \text{ kg/d}$ of fixed solids. The fixed solids pass through the digesters. The volatile solids are reduced by 40% or 50% as shown in Figure 6.8.

The assumed values for methane production and heating value are

Methane production = $0.7 \text{ m}^3 \text{ CH}_4/\text{kg VS destroyed}$
 Lower heating value of methane (CH_4) = 0.037 GJ/m^3

The energy balances for the entering and exiting sludge streams are

Sludge heating requirement (GJ/d) = $c_p m_{\text{sludge}} \Delta T_{\text{heating}}$

Heat recovery (GJ/d) = $c_p m_{\text{sludge}} \Delta T_{\text{recovery}}$

where specific heat of sludge = $c_p = 0.0042 \text{ GJ/T}^\circ\text{C}$

$\Delta T_{\text{heating}}$ applies to the sludge that is heated as it enters the digester

$\Delta T_{\text{recovery}}$ applies to the digested sludge that flows through the heat exchanger

The overall energy balance summarized in Table 6.3 give the energy value of the biogas, the energy requirement to heat the sludge, the heat energy recovered from the exiting warm sludge, and the heat losses through the walls, floor and cover of the digesters. The recovered energy is used to partially heat the feed sludge. Energy recovered from beneficial use of the methane gas is not included in the heat recovery values.

Energy Balance Component	Mesophilic (MAD)	Thermophilic (TAD)	Two-stage (MAD + TAD)
Energy from biogas (MJ)	+218	+272	+272
Heat energy recovered (MJ)	+84	+168	+168
Sludge heating (MJ)	-105	-189	-189
Heat losses (MJ)	-16	-28	-18
Overall energy balance	+181	+223	+272

Table 6.3 Energy balance comparison for three anaerobic digestion systems



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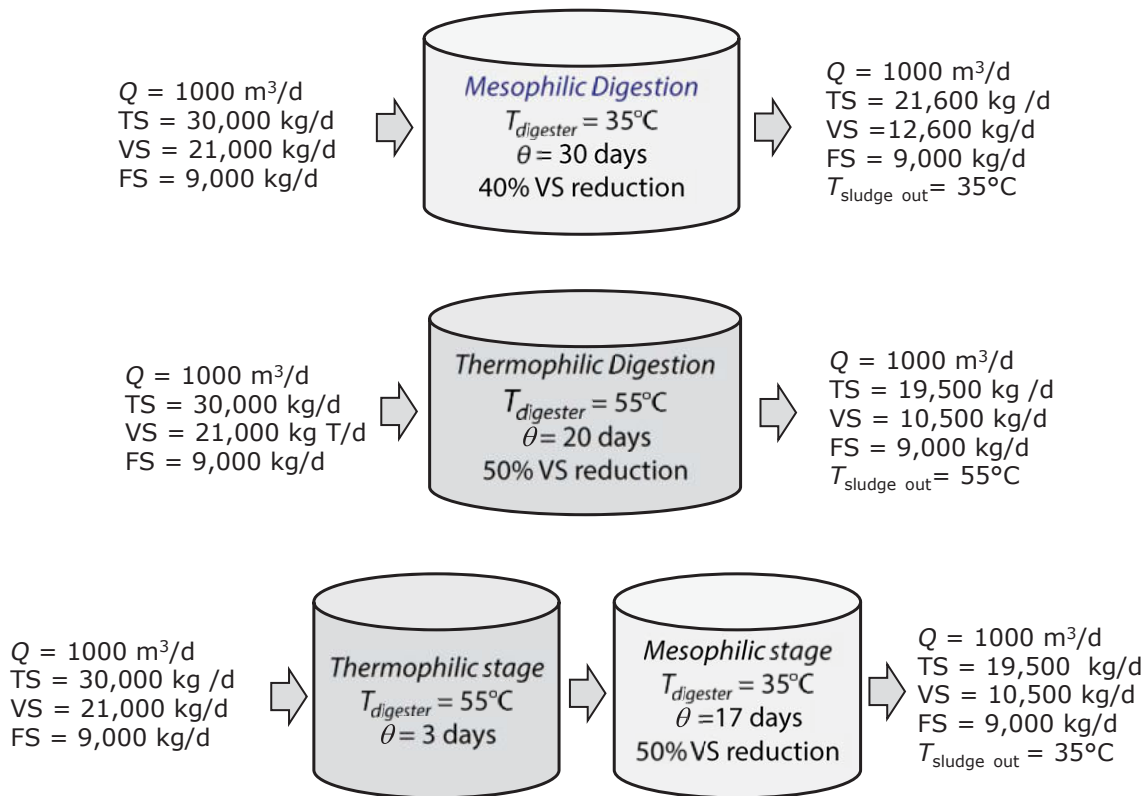


Figure 6.8 Three arrangements of anaerobic sludge digestion.

Heat is lost through the floor, cover, and walls of the digester in proportion to these areas and the difference between the ambient temperature of the air and earth and the temperature of the digester contents. The losses are small in comparison with the quantities of the energy balance so the details can be omitted.

The energy loss from the thermophilic digester is larger than the mesophilic digester because the internal temperature is higher. This is based on an assumption that the wall, floor, and cover construction is the same for both kinds of digesters; that is, no special insulation was added for the thermophilic process.

Local conditions (air temperatures, sludge quality, etc.) do affect the energy balances and may cause the most favorable digestion arrangement to differ from place to place. The air and earth temperatures used were 10°C , which is appropriate for northern regions. Obviously higher temperatures would be more suitable in many parts of the world (Puchajda & Oleszkiewicz 2008).

EXAMPLE 6.7 HEATING AN ANAEROBIC DIGESTER

This example explains the heat recovery energy balance for the mesophilic digester in Figure 6.8. Figure 6.9 shows raw sludge at 10°C heated to 30°C in a countercurrent heat exchanger, using digested sludge at 35°C as the heating fluid. A boiler raises the sludge temperature from 30°C to 35°C. A portion of the 216 GJ/d biogas energy is used in the boiler and 197 GJ/d is available for other uses (e.g. electricity generation)

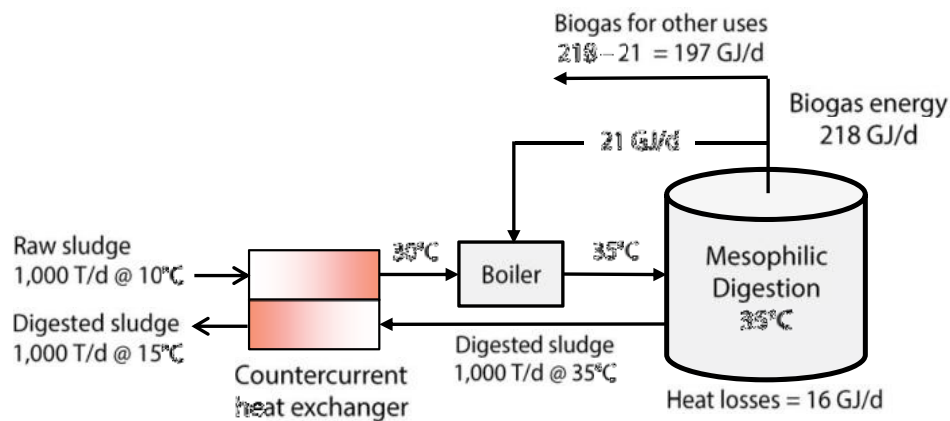


Figure 6.9 Energy balance with heat recovery for a mesophilic anaerobic digester.

$$\begin{aligned} \text{Total heating requirement } (T = 10^{\circ}\text{C to } T = 35^{\circ}\text{C}) \\ = (1000 \text{ T/d})(0.0042 \text{ GJ/T}^{\circ}\text{C})(35^{\circ}\text{C} - 10^{\circ}\text{C}) = 105 \text{ GJ/d} \end{aligned}$$

$$\begin{aligned} \text{Energy supplied from boiler } (T = 30^{\circ}\text{C to } T = 35^{\circ}\text{C}) \\ = ((1000 \text{ T/d})(0.0042 \text{ GJ/T}^{\circ}\text{C})(35^{\circ}\text{C} - 30^{\circ}\text{C}) = 21 \text{ GJ/d} \end{aligned}$$

$$\begin{aligned} \text{Energy supplied from heat recovery} \\ = (1000 \text{ T/d})(0.0042 \text{ GJ/T}^{\circ}\text{C})(35^{\circ}\text{C} - 15^{\circ}\text{C}) = 84 \text{ GJ/d} \end{aligned}$$

$$\begin{aligned} \text{Biogas production} \\ = (8,400 \text{ kg/d VS destroyed})(0.7 \text{ m}^3 \text{ CH}_4/\text{kg VS destroyed})(0.037 \text{ GJ/m}^3 \text{ CH}_4) \\ = 218 \text{ GJ/d} \end{aligned}$$

$$\text{Heat losses} = 16 \text{ GJ/d}$$

(Note: Heat is lost through the floor, cover, and walls of the digester in proportion to these areas and the difference between the ambient temperature of the air and earth and the temperature of the digester contents. These losses have been ignored to simplify the overall energy balance, but they are not ignored in digester design.)

$$\begin{aligned} \text{Overall energy balance} \\ = \text{Energy from biogas} + \text{Heat recovered} - \text{Sludge heating} - \text{Heat Losses} \\ = 218 \text{ GJ/d} + 84 \text{ GJ/d} - 105 \text{ GJ/d} - 16 \text{ GJ/d} = +181 \text{ GJ/d} \end{aligned}$$

6.5 GAS TREATMENT TECHNOLOGIES

Biogas can be used as it comes from the landfill or the anaerobic digester, but it is becoming more common to remove non-combustibles and impurities to produce a high Btu fuel that is equivalent to natural gas, or that can be compressed for use as a vehicle fuel (liquefied natural gas, LNG).

- Electric Power – Fuel for engines or turbines in landfill power plants or anaerobic digestion.
- Medium Btu – Delivered in a dedicated pipe to a single user to offset fossil fuel in steam boilers, kilns, burners, utility power plants, etc.
- High Btu – Cleaned to natural gas specifications and delivered to a natural gas pipeline.
- Vehicle Fuel (LNG and CNG) – Extension of High BTU: Cleaned to natural gas fuel specifications, compressed or liquefied, and stored in an on-site fueling station

The calorific or heat value of the fuel is determined predominantly by the percentage of methane present. Typically, this is 35–55% by volume for landfill gas and 50–70% by volume for digester gas. The non-methane content is carbon dioxide with traces of other gases. The high CO₂ content reduces the heating value, but otherwise is not a problem. The trace gases, which include sulfur and halogen (Cl and F) compounds and acids, form corrosive acids when they are burned.



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Pure methane, which has a heat value 9.97 kWh/m^3 , is the only significant hydrocarbon constituent in landfill gas converted to mechanical/electrical energy by the engine combustion process. The lower the methane content, the greater the volume of gas that must pass through the engine to achieve the same power output. This in turn means that potentially more aggressive gas constituents could enter the engine. This is why limits for aggressive gas constituents are defined 'per 100% methane'.

Biogas can be purified to almost 100% methane by removing carbon dioxide and nitrogen. Sulfur compounds and VOCs are also removed. Figure 6.10 is a block diagram for a purification system.

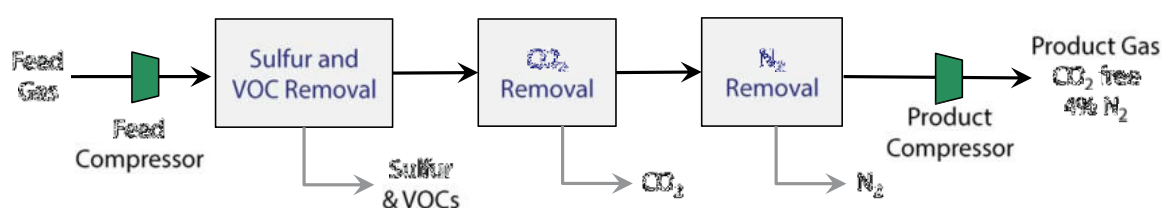


Figure 6.10 Biogas can be purified to remove carbon dioxide, nitrogen, sulfur compounds and VOCs.

The caloric value (CV) gives no indication of the aggressiveness of the supply gas or likely emissions. Supplying the input gas at higher pressure typically occurs with low calorific value gas. The higher inlet pressure of the gas will generally result in increased emissions of methane, non-methane VOCs, and other products of incomplete combustion.

Biogas from landfills and sludge digesters contains a variety of sulfur compounds, in particular hydrogen sulfide (H_2S). Most sulfur compounds are highly odorous (e.g. hydrogen sulfide, dimethyl sulfide, dimethyl disulfide, diethyl disulfide and carbon disulfide) and contain thiols, e.g. methanethiol (methyl mercaptan), ethanethiol and propanethiol. The typical H_2S concentration is less than 100 ppm, but in certain landfill situations it can be several thousand ppm. H_2S is easily removed by passing the gas through a bed of iron-impregnated wood chips.

Sulfur compounds are corrosive in the presence of free water or the moisture found within the engine oil and/or the biogas. These compounds can lead to wear on engine piston rings and cylinder linings. Gas recirculation systems may increase the availability of moisture within the engine system. This also affects oil quality, leading to the need for more frequent oil changes.

Halogenated compounds containing chlorine, bromine and fluorine (e.g. carbon tetrachloride, chlorobenzene, chloroform and trifluoromethane) are broken down during the combustion process and can form the acid gases, HCl and HF , in the presence of moisture. These are responsible for corrosion of metal piping and engine components.

Most halogenated compounds in landfill gas are the result of direct volatilization from solid waste components. The rate of absorption of chlorine compounds into the engine oil usually determines the frequency of oil changes in landfill gas engines.

Semi-volatile silicon compounds can be another problem in biogas combustion. Discarded consumer products (including cosmetics) in the landfill tend to be the main source of silicon in the supply gas. Many consumer products (hair care, skin care, underarm deodorants) and commercial lubricants contain silicones. *Organosiloxanes* are semi-volatile compounds that are not aggressive in the gaseous form. The problem comes from their conversion during combustion to solid inorganic silicon compounds. They form a coating on all surfaces contacted by the lubricating oil and can alter the oil retaining surface finish of engine cylinder liners. The deposits severely reduce engine life and increase maintenance costs.

Figure 6.11 shows a module for removing siloxane from biogas that feeds a combined heat and power (CHP) microturbine. Microturbines require a lot of gas pretreatment, but as a result they require far less ongoing maintenance than reciprocating internal combustion engines.

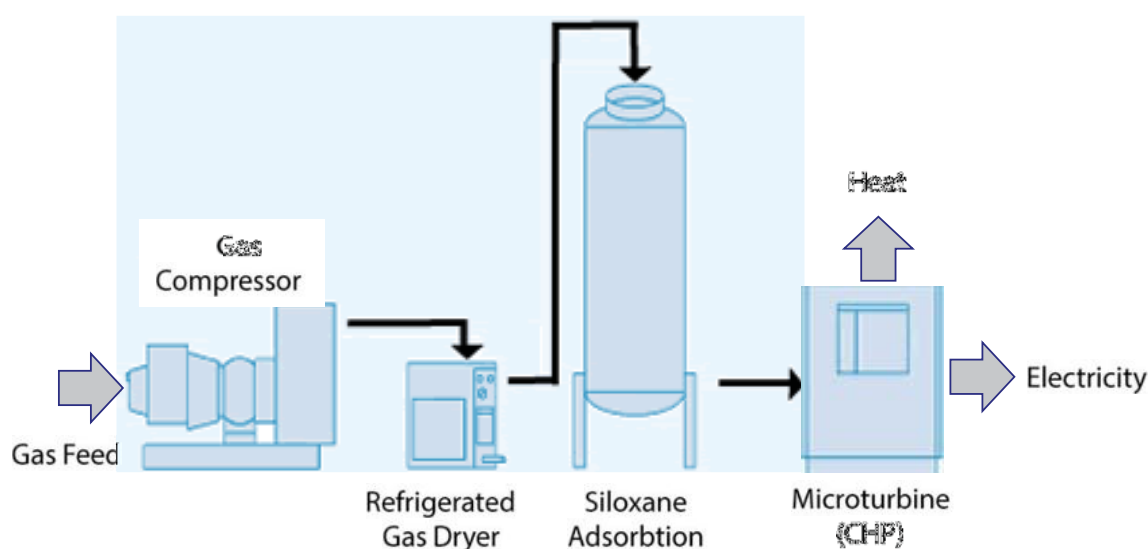


Figure 6.11 Siloxane can be removed from biogas by adsorption.

The energy used for biogas upgrading is from 0.1 to 0.27 kWh/m³ of raw biogas. Typical methane losses are less than one percent (IRENA 2018).

6.6 CONCLUSION

Materials once considered wastes are now seen as sources of useful energy. Municipal refuse has been used for fuel in boilers and power plants since the 1970s. Landfill gas is extracted and used for heating and power generation instead of being vented to the atmosphere and waste gas flares. Biogas from anaerobic wastewater treatment processes, in particular municipal and agricultural sludge digesters, allows many treatment facilities to be self-sufficient for heating with excess biogas energy and for powering gas engines to run air blowers and pumps.

Biogas can be burned as it comes from the landfill or the digester, but usually it is treated to avoid the formation of corrosive compounds (sulfuric and hydrochloric acid) and abrasive compounds. The fuel value can be upgraded by removing carbon dioxide. Nitrogen can also be removed, but this is not always justified.



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7 INCINERATION OF WASTE GAS

Waste gases that contain *volatile organic chemicals* (VOCs), such as solvents, and other hazardous organic compounds are regulated under air pollution rules. There are several ways to remove organic chemicals from a gas, for example adsorption onto activated carbon and condensation. These do not destroy the chemicals, leaving the possibility that they might be recovered and reused. Incineration destroys the chemicals. All that can be recovered is heat from the hot exhaust gases of thermal and catalytic incinerators.

7.1 SAFETY – THE EXPLOSIVE LIMITS

The workplace, ventilation systems, and the area around a thermal or catalytic incinerator are at risk of explosion if fuel and oxygen (air) exist in certain proportions and there is a spark or flame to cause ignition. The ratio of fuel and oxygen that is explosive varies with each combustible gas or vapor.

The minimum concentration of a particular combustible gas or vapor necessary to support its combustion in air is defined as the *Lower Explosive Limit* (LEL). Below this LEL, the mixture is too *lean* to burn because there is not enough fuel value. The maximum concentration of a gas or vapor that will burn in air is the *Upper Explosive Limit* (UEL). Above this level, the mixture is too *rich* to burn because there is not enough oxygen.

Waste gases are usually dilute and that makes the LEL the important factor. Safety regulations limit VOC concentrations in an exhaust or vent stream to a fraction, typically 25%, of the Lower Explosive Limit (LEL). If the gas is a mixture of VOCs, the gas with the lowest LEL controls. Table 7.1 gives the limiting concentration (25% of LEL) for some common VOCs and dusts. A larger table of LEL and UEL values for selected volatile compounds that will burn in air can be found in Appendix 7.

VOC	25% LEL (ppmv)	Dust	25% LEL (g/m ³)
Butane	4,650	Polystyrene	400
Ethane	7,500	Urea	125
Ethylene	6,875	Grain dust	125
Acetone	6,500	Sugar	100
Benzene	3,375	Coal	60
p-Xylene	2,500	Sewage sludge	250
Toluene	3,175	Sulfur	280

Table 7.1 Volatile Organic Compound (VOC) concentrations corresponding to 25% of Lower Explosive Limits (LELs)

The most violent explosions are produced when the proportion of oxygen present is not far from that which will result in complete combustion. The range of explosive concentrations of a dust cloud is not simply a function of the chemical composition of the dust. The limits vary with the size and shape of the particles in the dust cloud, with smaller sizes generally being more dangerous.

The LEL of a gas mixture is calculated using Le Chatelier's formula

$$\frac{\sum x_i}{LEL_{\text{mixture}}} = \frac{x_1}{LEL_1} + \frac{x_2}{LEL_2} + \dots + \frac{x_n}{LEL_n}$$

where x_i = volume fraction of flammable gas species i

EXAMPLE 7.1 LOWER EXPLOSIVE LIMIT OF VOCS

Solvent vapors are drawn from workplace ventilation hoods at concentrations of 4000 ppmv benzene (BZ) and 500 ppmv methyl chloride (MC). The LELs are 13,500 ppmv for BZ and 107,000 ppmv for MC.

The volume fraction of all combustible gases is

$$\sum x_i = x_{\text{BZ}} + x_{\text{MC}} = 4,000 \text{ ppmv} + 500 \text{ ppmv} = 4,500 \text{ ppmv}$$

From Le Chatelier's formula, the LEL of the mixture is

$$\frac{4,500 \text{ ppmv}}{LEL_{\text{mixture}}} = \frac{4,000 \text{ ppmv BZ}}{13,500 \text{ ppmv}} + \frac{500 \text{ ppmv MC}}{107,000 \text{ ppmv}} = 0.2963 + 0.0047 = 0.3010$$

$$LEL_{\text{mixture}} = \frac{4,500 \text{ ppmv}}{0.3010} = 14,950 \text{ ppmv}$$

$$25\% \text{ of } LEL_{\text{mixture}} = (0.25)(14,950 \text{ ppmv}) = 3,738 \text{ ppmv}$$

This is less than the total combustible fraction 4,500 ppmv. There is negligible risk of an explosion.

7.2 GAS COMPOSITION

Gases expand as temperature is increased and compress as pressure is increased, so gas volume or volumetric flow rate have no useful meaning until the corresponding gas temperature and pressure are known. These must be referenced to *standard temperature and pressure* (STP), which are defined differently by various organizations, as given in Table 7.2. The most used definitions are those of the International Union of Pure and Applied Chemistry (IUPAC) and the National Institute of Standards and Technology (NIST).

Normal cubic meters per hour (Nm^3/h) is the volumetric flow rate for gases at 0°C and 1 atm (101.325 kPa). In the U.S. a common measure is scfm to indicate 'standard cubic feet per minute' or acfm to indicate 'actual cubic feet per minute'.

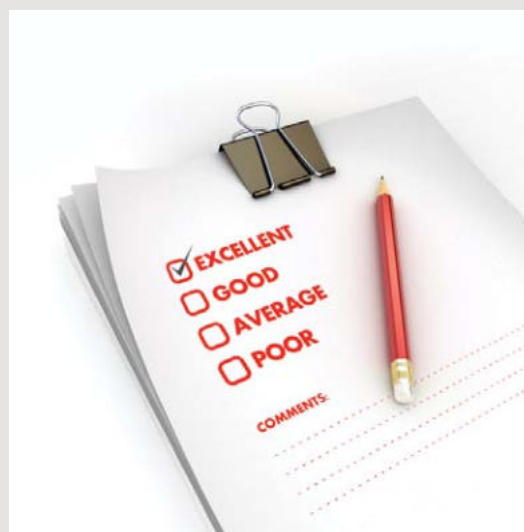
Discipline	Standard Conditions			Organization
	Temp.	Absolute Temp.	Pressure	
Chemistry/physics (STP)	0°C	273.15 K	100.000 kPa (1 atm)	IUPAC
Chemistry/physics (STP)	20°C	293.15 K	101.325 kPa (1 atm)	NIST
Air pollution (NTP)	20°C	293.15 K	101.325 kPa (1 atm)	U.S. EPA
Industrial hygiene	60°F	519.67°R	14.696 psi (1 atm)	U.S. OSHA
IUPAC = International Union of Pure and Applied Chemistry NIST = National Institute of Standards and Technology				

Table 7.2 Four definitions for standard conditions. (Source: Wikipedia)

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A mass concentration, say $\mu\text{g}/\text{m}^3$ or mg/m^3 , is needed to make a material balance. The mass concentration is calculated using the molecular mass of the gaseous pollutant. Corrections for pressure and temperature may be required. These corrections are made using the *ideal gas law*:

$$PV = nRT$$

where P = pressure, atm

T = absolute temperature, K

V = gas volume, L

n = number of moles of gas

R = universal gas constant = 0.08205 L atm/mole K

The value of R depends on the *units* used for pressure, temperature, and volume. The volume of one mole of gas at standard conditions volumes are given in Table 7.3.

Volume of 1 mole of gas	Temp.	Pressure
$8.3145(273.15/101.325) = 22.414 \text{ m}^3/\text{kg mol}$	0°C	101.325 kPa (1 atm)
$10.7316(519.67/14.696) = 379.48 \text{ ft}^3/\text{lb mol}$ $= 0.8366 \text{ ft}^3/\text{g mol}$	60°F	14.696 psi (1 atm)

Table 7.3 Volume of one mole of gas calculated at two standard reference conditions

For an equal number of moles of gas

$$\frac{P_1 V_1}{T_1} = \frac{P_2 V_2}{T_2}$$

Gas concentrations are usually measured as a volumetric ratio, typically parts per million by volume (ppmv). The ppmv concentration is independent of changes in pressure and temperature because all gases in a mixture expand or contract to the same extent. For example, if the concentration of SO_2 in air is 15 ppmv, then every million volumes of air contains 15 volumes of SO_2 regardless of how the gas mixture is compressed or expanded. This is one advantage of using ppmv units for air pollution work.

Another convenient measure is the mole fraction, which is the same as the volume fraction. Volume measures of composition do not change when temperature or pressure are changed because all gases expand and contract by the same amount. This is not true for mass concentrations; the mass of a component remains the same when the gas volume changes.

Volume and mass concentrations (ppmv and mg/m^3) can be converted using the molar mass (MM) of the pollutant and the ideal gas law. One g-mole of an ideal gas occupies a volume 0.02241 m^3 (22.41 L) at standard temperature and pressure ($0^\circ\text{C} = 273 \text{ K}$ and 1 atm). Also, 1 m^3 of an ideal gas contains $1/0.02241 \text{ m}^3 = 44.623 \text{ g-moles}$ of the gas. (One lb-mole occupies a volume of 359 ft^3 at STP).

The mass (mg) of a gas occupying 1 m³ is

$$\frac{\text{mass of gas (mg)}}{\text{volume of gas (m}^3\text{)}} = \left(\frac{\text{MM (g/mol)}}{0.02241 \text{ (m}^3\text{/mol)}} \right) \left(\frac{1,000 \text{ mg}}{\text{g}} \right) = \frac{\text{MM}(1,000)}{0.02241}$$

where MM is the molar mass of the gas (g/mol).

If a gas mixture contains a pollutant at a concentration of 1 ppmv, or 1 m³ of pollutant in 1,000,000 m³ of mixture, the mass concentration will be given by

$$\begin{aligned} \frac{\text{mg pollutant}}{\text{m}^3 \text{ mixture}} &= \left(\frac{\text{m}^3 \text{ pollutant}}{10^6 \text{ m}^3 \text{ mixture}} \right) \left(\frac{\text{MM g/g-mol}}{0.02241 \text{ m}^3\text{/g-mol}} \right) \left(\frac{1,000 \text{ mg}}{\text{g}} \right) \\ \frac{\text{mg pollutant}}{\text{m}^3 \text{ mixture}} &= \text{ppmv} \left(\frac{\text{MM} \times 10^3}{0.02241} \right) \left(\frac{1}{10^6} \right) = \text{ppmv} \left(\frac{\text{MM}}{22.41} \right) \end{aligned}$$

EXAMPLE 7.2 MASS OF METHANE GAS

Calculate the mass of the methane that occupies a volume of 4000 L at 20°C and 1.2 atm. The molar mass of methane is 16 g/g-mol. For these units the universal gas constant is 0.08205 L atm/mol K. Absolute temperature: = 20°C + 273 = 293 K.

From the ideal gas law:

$$PV = nRT$$

$$(1.2 \text{ atm})(4,000 \text{ L}) = n (0.08205 \text{ L atm/mol K})(293 \text{ K})$$

$$n = 199.7 \text{ mol}$$

$$\begin{aligned} \text{Total mass of the gas} &= (\text{moles of gas})(\text{molar mass of gas}) \\ &= (199.7 \text{ g-mol}) (16 \text{ g/g-mol}) = 3,195 \text{ g} \sim 3,200 \text{ g} \end{aligned}$$

EXAMPLE 7.3 VENTING TOLUENE.

A ventilation airflow of 20,000 m³/h (at STP) contains 1,800 ppmv toluene.

By definition, the volume fraction of toluene = 1,800 ppmv = 1,800 m³/1,000,000 m³

Volumetric flow of toluene = (20,000 m³/h)(1,800 m³/1,000,000 m³) = 36 m³/h

The density of toluene is 4.12 kg/m³

Mass flow of toluene = (36 m³/h)(4.12 kg/m³) = 148.3 kg/h

EXAMPLE 7.4 SULFUR DIOXIDE

What is the concentration, expressed as ppmv, of nitrogen, carbon dioxide, and sulfur dioxide (SO_2) in a combustion gas that has this composition as mole fractions: $\text{N}_2 = 0.75$, $\text{O}_2 = 0.05$, $\text{CO}_2 = 0.12$, $\text{SO}_2 = 0.0015$, and $\text{H}_2\text{O} = 0.0785$.

For a gas, mole fraction = volume fraction, and

$$\text{ppmv} = \text{volume fraction} \times 10^6$$

$$\text{Nitrogen} = 750,000 \text{ ppmv} \quad \text{CO}_2 = 12,000 \text{ ppmv} \quad \text{SO}_2 = 1,500 \text{ ppmv}$$

**EXAMPLE 7.5 CONVERT VOLUME CONCENTRATION
TO MASS CONCENTRATION**

The concentration of a gaseous pollutant in air is 15 ppmv. The molar mass of the pollutant is 16 g/mol. Find the concentration as mg/m^3 at standard conditions.

$$\frac{\text{mg pollutant}}{\text{m}^3 \text{ mixture}} = \left(\frac{15 \text{ m}^3 \text{ pollutant}}{10^6 \text{ m}^3 \text{ mixture}} \right) \left(\frac{16 \text{ g/g-mol}}{0.02241 \text{ m}^3/\text{g-mol}} \right) \left(\frac{1,000 \text{ mg}}{\text{g}} \right) = 10.7 \frac{\text{mg}}{\text{m}^3}$$



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7.3 THERMAL INCINERATION

Thermal incineration, sketched in Figure 7.1, is the combustion of gaseous mixtures in a refractory lined vessel that raises the temperature of the waste gas from its inlet temperature, about 20-35°C (approx. 68-100°F), to the combustion temperature which varies from 650-1100°C (1200- 2000°F) for different compounds.

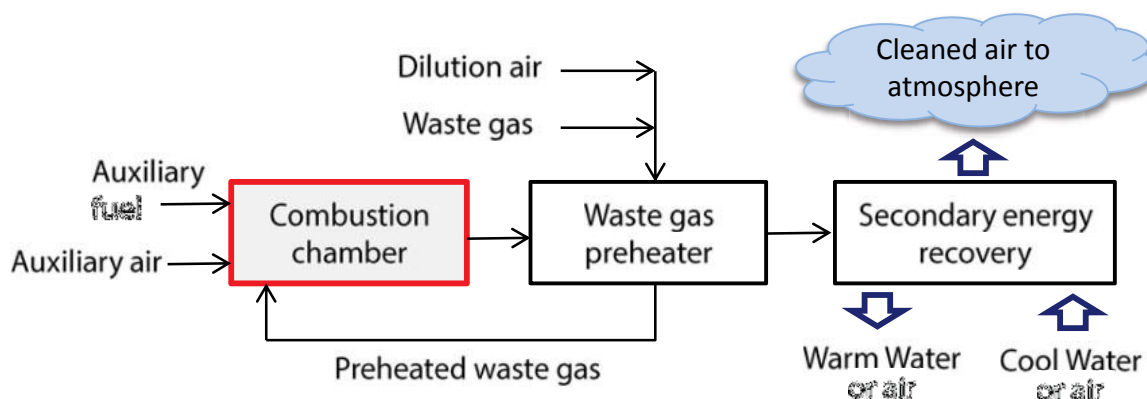


Figure 7.1 Thermal incinerator with heat recovery for treating waste gas.

The necessary residence time depends on the chemicals that are incinerated and the operating conditions (temperature, turbulence, etc.). The temperature required in the combustion chamber to get 99.99% destruction with a one second residence time is about 725°C for toluene and about 760°C for methyl chloride. Higher temperatures give faster oxidation and a higher degree of destruction. Longer residence times give a higher degree of destruction. The gas pre-heater is a heat exchanger that heats the inlet gas by recovering energy from the exiting hot gas.

The required volume of the combustion chamber and the residence time are calculated using the actual volumetric flow rate of the combustion gases.

$$V = (T_R) (F) \text{ and } T_R = V / F$$

where V = volume of the combustion chamber (m^3)

T_R = residence time (s)

F = actual volumetric flowrate of the waste gas (m^3/s)

‘Actual’ means the volume at the temperature in the combustion chamber. Since volume is proportional to the temperature, the gas volume is very large and the equipment may be surprisingly large to engineers who are accustomed to working with water.

7.4 CATALYTIC INCINERATION OF WASTE GASES

Catalytic incineration, shown in Figure 7.2, is attractive for treating gaseous emissions when the organic concentration is 50 ppmv to 10,000 ppmv and the flow is less than 3,000 m³/min (about 100,000 scfm). The catalyst increases the reaction rate, which reduces the reactor temperature and the actual volume of air passing through the incinerator.

The typical minimum temperature at the catalyst bed inlet is about 320°C (600 °F) and the temperature at the catalyst bed outlet is 550-650°C, depending on the compounds being burned and the catalyst used. Some compounds require 480°C to initiate the oxidation reactions. VOC destruction efficiency is typically 95%, but 98% is possible with larger catalyst volumes and higher temperatures. Larger catalyst volume means a longer detention time.

The waste stream is usually preheated by heat exchange with the incinerator's effluent gas or some other process stream. Auxiliary fuel may be needed.

Efficiency increases as the residence time increases and can reach 98 percent. Typical residence times are 0.05 to 0.25 second.

$$T_{R, \text{empty}} = V/F$$

$$T_{R, \text{actual}} = V_{\text{void}}/F$$

where $T_{R, \text{empty}}$ = empty-bed residence time (s)
 $T_{R, \text{actual}}$ = actual residence time (s)
 V = volume of the empty catalytic chamber (m³)
 V_{void} = void volume of the catalytic bed (m³)
 F = actual volumetric flowrate of the waste gas (m³/s)

Gases containing chlorine, sulfur, and certain other atoms can deactivate (poison) noble metal catalysts (e.g., platinum and palladium). Lead, arsenic, and phosphorous should be considered poisons for most oxidation catalysts, though if their concentration is low enough the catalyst deactivation rate is economically acceptable. Particulate matter, including aerosols, can also foul and inactivate a catalyst surface.

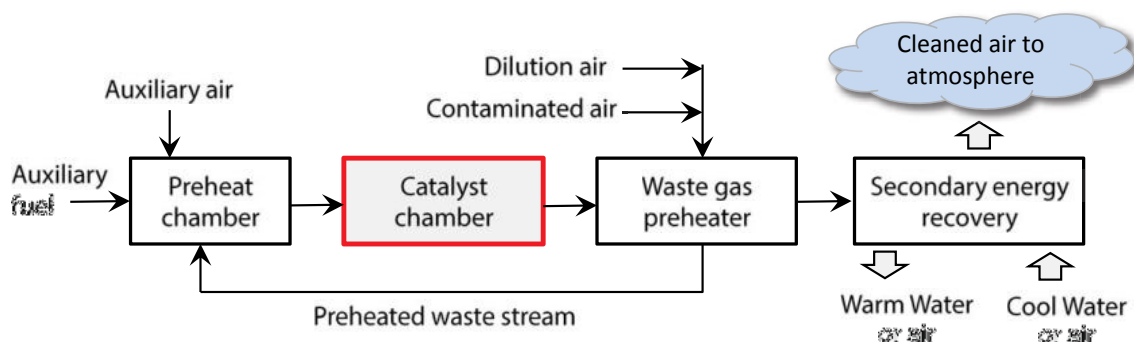


Figure 7.2 Catalytic incineration with heat recovery for treating waste gas

7.5 CASE STUDY - RECOVERY OF HEAT FROM COMBUSTION OF WASTE GASES

An industry proposes to destroy a mixture of waste gases in air and a liquid waste using thermal oxidation at 900°C with a 1 second residence time. Heat energy from the combustion will be used to generate steam (Straltz 1993).

The waste gas is 3,100 normal cubic meters per hour (Nm^3/h) that contains 2,962 Nm^3/h of air and 138 Nm^3/h (243 kg/h) of odorous sulfur compounds and methane. Normal cubic meters per hour (Nm^3/h) is the volumetric flow rate for gases at standard conditions: $T = 0^{\circ}\text{C}$ and 1 atm (101.325 kPa).

The liquid waste is 500 kg/h of pinene ($\text{C}_{10}\text{H}_{16}$), more commonly known as turpentine. The heating value of this waste is very high.

Stoichiometric Oxygen Requirement

A material balance on oxygen will determine whether the oxygen in the waste gas stream will meet the stoichiometric demand for combustion. The balanced combustion equations for the five waste gas components and the waste pinene are:

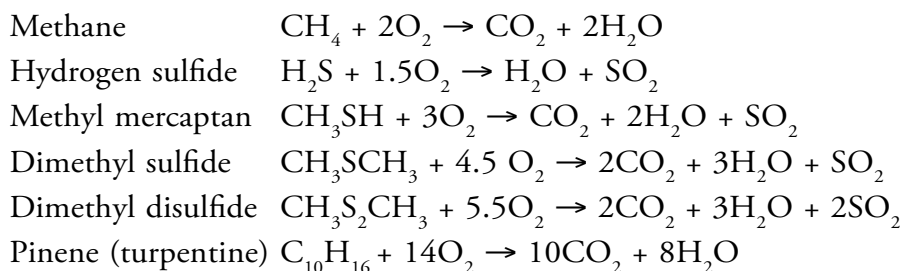


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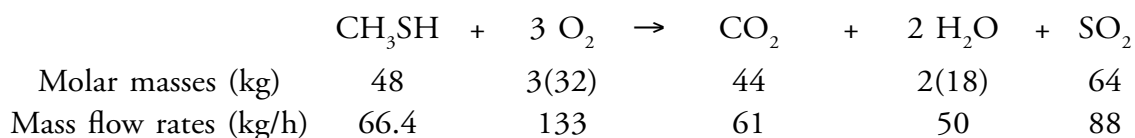
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The balanced stoichiometric reaction for the combustion of methyl mercaptan (CH_3SH) is



The reaction shows that combustion of 48 kg/h of methyl mercaptan consumes 3(32) = 96 kg/h of oxygen and produces 44 kg/h carbon dioxide, 36 kg/h water, and 64 kg/h sulfur dioxide.

The mass flow rate of methyl mercaptan in the waste gas is 66.4 kg/h and the actual quantities, obtained by simple proportion, are 133 kg/h O_2 , 61 kg/h CO_2 , 50 kg/h H_2O , and 88 kg/h SO_2 . These values are shown below the reaction equation.

Table 7.4 is a summary of the volumetric and mass flow rates for the five gases and one liquid waste inputs, the gaseous combustion products, and the stoichiometric oxygen requirements. The oxygen demands were calculated using the balanced chemical reactions is given in Table 7.4. The waste gas stream requires 598 kg/h and the pinene waste demands 1,647 kg/h, for a total oxygen requirement of 2,245 kg/h. The air in the waste gas stream contains only 888 kg/h oxygen. An additional 1,356 kg/h is needed for stoichiometric combustion of all six compounds and this must come from an auxiliary blower supply.

The calculation of the additional air requirement is also in Table 7.4. In this calculation we assume a volumetric composition of blower air of 79.1% nitrogen and 20.9 % oxygen. The corresponding mass flow rates are 76.9% nitrogen and 23.1 % oxygen. For example, blower air containing 1,361 kg/h oxygen must have a mass flow of $1,361/0.231 = 5,890$ kg/h. The stoichiometric airflow contributes no oxygen to the output gas stream because all of the oxygen it carries is consumed by the combustion. The nitrogen output equals the nitrogen input because it is inert.

Normally a small amount of excess air (typically 10%) is added to insure complete oxidation. Before adding this in, the energy balance is checked to see if a larger volume is needed to control the combustion temperature at 900°C

Inputs				Outputs		
Component	Flow rate (Nm ³ /h)	Flow rate (kg/h)	Oxygen Required (kg/h)	Component	Flow rate (Nm ³ /h)	Flow rate (kg/h)
CH ₄	60	42.8	171.4	CO ₂	1001	1965
H ₂ S	4	6.1	8.6	H ₂ O	974	782
CH ₃ SH	31	66.4	132.8	SO ₂	84	240
CH ₃ SCH ₃	37	102.3	237.8			
CH ₃ S ₂ CH ₃	6	25.2	47.1			
C ₁₀ H ₁₆ (liquid)	82 (a)	500	1647			
Air in Waste Gas	2962 (b)	3812				
O ₂ in waste gas	619	884		O ₂		0 (a)
N ₂ in waste gas	2343	2927		N ₂	2343	2927
Stoichiometric Air from auxiliary blower	4,578	5890				
O ₂ in blower air	953	1361		O ₂		0 (b)
N ₂ in blower air	3625	4530		N ₂	3625	4530
Total	7,760	10,445	2245			10,444
a) Volume if pinene existed as a vapor at Normal conditions (0°C and 101.325kPa)						
b) Calculations made assuming air is 20.9% oxygen and 79.1% nitrogen by volume.						

Table 7.4 Material balance to determine stoichiometric oxygen requirements for a thermal incineration process that will burn odorous sulfur compounds and a liquid organic waste (turpentine).

The Energy Balance

The heat energy released by the combustion, taking methyl mercaptan (CH₃SH) as an example is

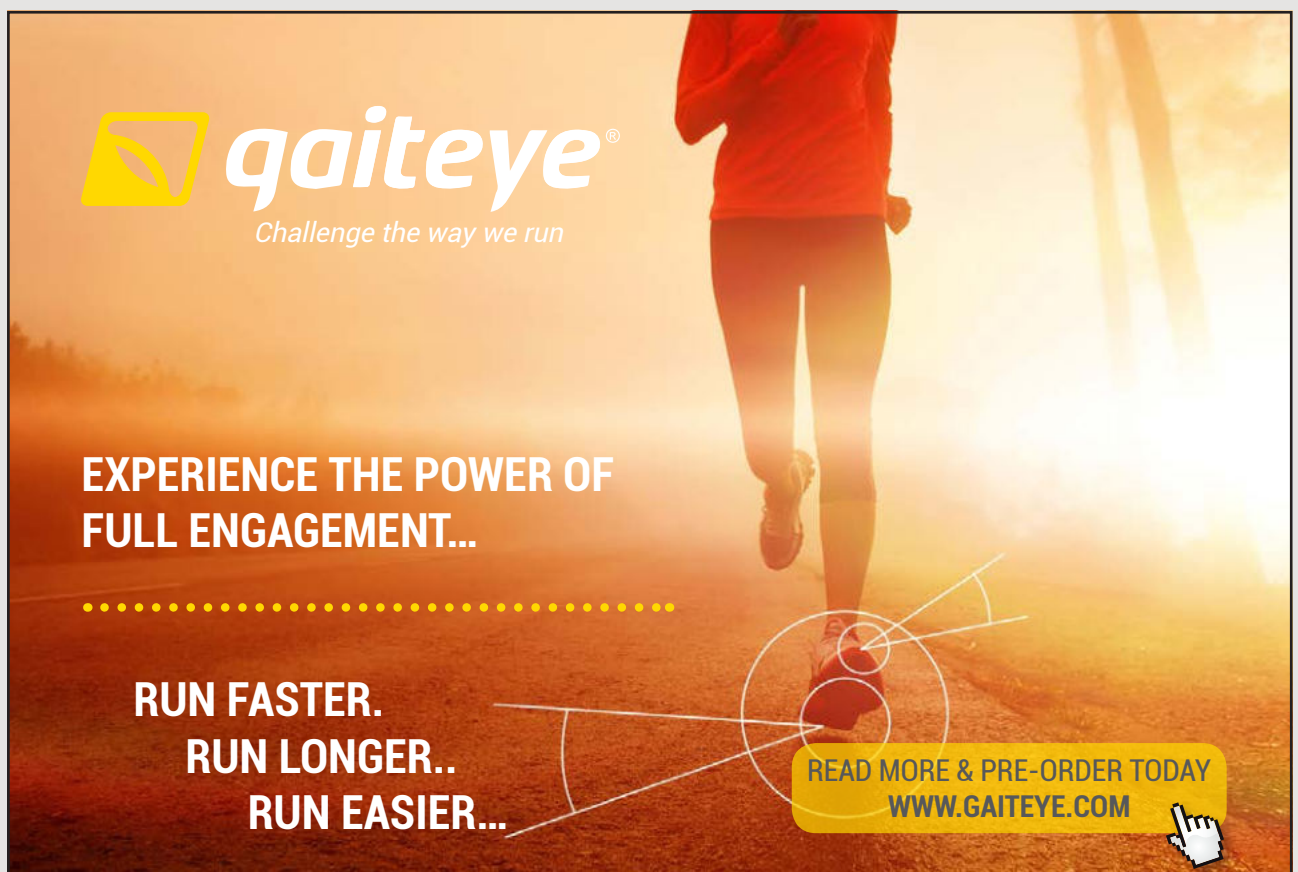
$$\text{Heat energy released (kJ/h)} = (m)(\text{HHV}) = (66.4 \text{ kg/h})(23,518 \text{ kJ/kg}) = 1,562,000 \text{ kJ/h}$$

The heat energy increase in the output gas stream, using CO₂ as an example is

$$\begin{aligned} \text{Heat energy increase} &= m c_p \Delta T \\ &= (1965 \text{ kg/h})(1.2728 \text{ kJ/kg}^\circ\text{C})(900^\circ\text{C} - 25^\circ\text{C}) \\ &= 2,189,000 \text{ kJ/h} \end{aligned}$$

The heat energy yield of the waste gas stream is 6,730,000 kJ/h. The pinene ($C_{10}H_{16}$) fuel releases an additional 22,349,000 kJ/h. The combined fuel energy of 29,079,000 kJ/h makes the process self-supporting. Auxiliary fuel will not be needed. In fact, the pinene has so much heating value that its feed rate must be carefully controlled to avoid damage to the equipment.

The very large heat energy release is summarized in Table 7.5.



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Inputs				
Component	Flow rate (Nm ³ /h)	Flow rate (kg/h)	HHV (kJ/kg)	Total heat released (kJ/h)
CH ₄	60	42.8	50,090	2,146,000
H ₂ S	4	6.1	15,231	92,000
CH ₃ SH	31	66.4	23,518	1,562,000
CH ₃ SCH ₃	37	102.4	23,267	2,382,000
CH ₃ S ₂ CH ₃	6	25.2	21,767	548,000
Waste Gas Subtotal				6,730,000
C ₁₀ H ₁₆ (liquid)	82 ^a	500	44,699	22,349,000
Waste Gas Air	2,962	3,812	NA	NA
Stoichiometric Air	4,578	5,890	NA	NA
Totals	7,760	10,445		29,079,000
Outputs ^b				
Component	Flow Rate (Nm ³ /h)	Flow Rate (kg/h)	c _p (kJ/kg°C)	Total heat increase (kJ/h)
CO ₂	1001	1,965	1.2728	2,189,000
H ₂ O	974	782	2.4032	1,645,000
SO ₂	84	240	0.6155	129,000
O ₂	0	0	1.1095	0
N ₂	5,968	7457	1.2058	7,868,000
Totals	8,027	11,014		11,831,000
a. Volume if pinene existed as a vapor at Normal conditions				
b. Inlet temperature of gases = 25°C; outlet temperature = 900°C, $\Delta T = 900 - 25 = 875^\circ\text{C}$				

Table 7.5 The preliminary energy balance for the inputs and outputs for a thermal incineration process that will burn odorous sulfur compounds and a liquid organic waste (turpentine).

Inputs				
Component	Flow rate (Nm ³ /h)	Flow rate (kg/h)	HHV (kJ/kg)	Total heat (kJ/h)
CH ₄	60	42.8	50,090	2,146,000
H ₂ S	4	6.1	15,231	92,000
CH ₃ SH	31	66.4	23,518	1,562,000
CH ₃ SCH ₃	37	102.4	23,267	2,382,000
CH ₃ S ₂ CH ₃	6	25.2	21,767	548,000
C ₁₀ H ₁₆ (liquid)	82 ^a	500	44,699	22,349,000
Waste Gas Air	2,962	3,811	NA	NA
Auxiliary (Stoichiometric) Air	4,578	5,890	NA	NA
Dilution Air	12,943	16,655	NA	NA
Totals	20,703	27,099		29,079,000
Outputs ^b				
Component	Flow Rate (Nm ³ /h)	Flow Rate (kg/h)	c _p (kJ/kg-°C)	Total Heat (kJ/h)
CO ₂	1001	1965	1.2728	2,189,000
H ₂ O	974	782	2.4032	1,645,000
SO ₂	84	240	0.6155	129,000
O ₂	2,694	3,847	1.1095	3,735,000
N ₂	17,522	20,265	1.2058	21,381,000
Totals	20,704	27,099		29,079,000
a) Volume if pinene existed as a vapor at Normal conditions b) Inlet temperature of gases = 25°C; outlet temperature = 900°C, $\Delta T = 900 - 25 = 875^\circ\text{C}$				

Table 7.6 Revised energy and material balance for the inputs and outputs for a thermal incineration process that will burn odorous sulfur compounds and a liquid organic waste (turpentine).

There is more heat energy released than is carried away by 900°C exhaust gases: (29,079,000 – 11,831,000 = 17,248,000 kJ/h). Dilution air must be added to carry away the extra heat energy. This will protect the equipment from overheating.

The mass flow of dilution air required to maintain the 900°C exhaust temperature is

$$\begin{aligned}\text{Dilution air (kg/h)} &= Q_{\text{Air}} / c_p \Delta T \\ &= (17,248,000 \text{ kJ/h}) / [(1.1836 \text{ kJ/kg} \cdot ^\circ\text{C})(900^\circ\text{C} - 25^\circ\text{C})] \\ &= 16,655 \text{ kg/h}\end{aligned}$$

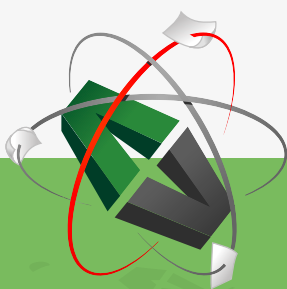
The revised energy and material balances with the excess air added are shown in Table 7.6

A fraction of the approximately 29,000MJ/h of energy generated in this combustion will be used to preheat the waste gas, as shown in Figure 7.2, and some can be used to generate steam and/or heat process flow streams for beneficial use elsewhere in the industrial plant.

Combustion Chamber Volume

The volume of the combustion chamber is calculated using the actual volumetric gas flowrate through the combustion chamber. The combustion reaction is extremely fast and the gas temperature everywhere in the combustion chamber is 900°C. Therefore the actual gas flow through the incinerator is the total rate of carbon dioxide, water vapor, sulfur dioxide gas, oxygen and nitrogen, which is $20,989 \text{ Nm}^3/\text{h} = 350 \text{ Nm}^3/\text{min} = 5.83 \text{ Nm}^3/\text{s}$

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The conversion from standard conditions ($T = 0^\circ\text{C} = 273\text{K}$ and 1 atm) to actual conditions (900°C and 1.2 atm) is

$$F_A = F_S \frac{P_S T_A}{P_A T_S} = (5.83\text{ m}^3/\text{s}) \frac{(1\text{ atm})(1173\text{ K})}{(1.2\text{ atm})(273\text{ K})} = 20.9\text{ m}^3/\text{s}$$

where F_A = actual volumetric flow rate (m^3/s)

T = temperature ($\text{K} = 273 + ^\circ\text{C}$)

P = pressure (atm)

Subscripts A and S indicate actual and standard conditions.

The required volume of the thermal oxidizer chamber for a residence time of $T_R = 1$ second is

$$V = F_A T_R = (20.9\text{ m}^3/\text{s})(1\text{ s}) = 20.9\text{ m}^3$$

7.6 CASE STUDY - ENERGY BALANCE ON A REGENERATIVE THERMAL OXIDIZER

Thermal oxidation is one way to meet a requirement of 98% volatile organic chemical (VOC) destruction of a 20,000 scfm waste gas flow. The waste gas contains two volatile organic chemicals 1000 ppmv benzene (BZ) and 1000 ppmv methyl chloride (MC).

The combustion chamber exhausts into a waste gas preheater so energy can be recovered. The desired thermal energy recovery efficiency is 70%. Figure 7.3 shows some of the known data and the calculated preheater temperature of 1150°F .

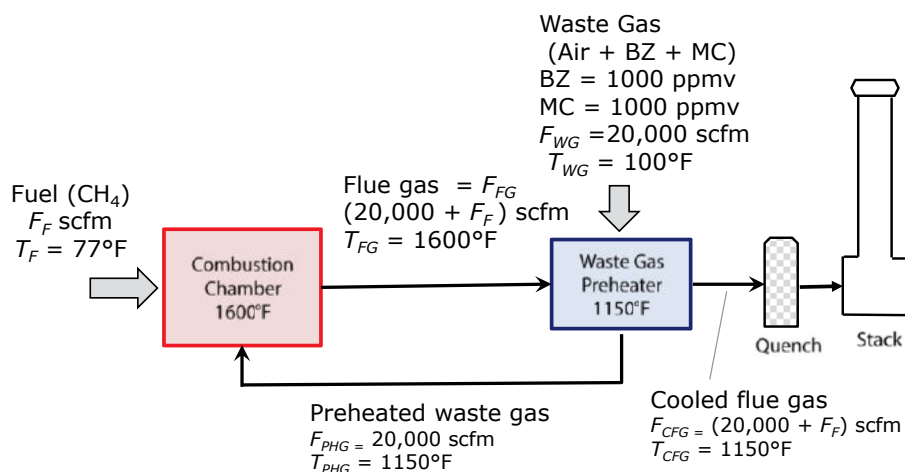
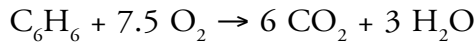
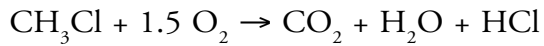


Figure 7.3 Thermal oxidation with a waste gas preheater for the destruction of benzene (BZ) and methyl chloride (MC)

Benzene is a hydrocarbon, C_6H_6 , and the combustion products are carbon dioxide and water



Methyl chloride (also known as chloromethane) contains a chloride atom and hydrochloric acid vapor will be formed during combustion.



The lower explosive limits are $\text{LEL}_{\text{BZ}} = 13,500 \text{ ppmv}$ and $\text{LEL}_{\text{MC}} = 107,000 \text{ ppmv}$. The VOC concentrations of 1000 ppmv do not present any risk of explosion or accidental fires in the ventilation system.

The concentrations of benzene and methyl chloride are both 1,000 ppmv, which is volume fractions of 0.1%. The gas composition, as volume fractions, is

0.1% benzene 0.1% methyl chloride 99.8% air

The heating values are 3,475 Btu/scf benzene and 705 Btu/scf methyl chloride. Thus the heating value of the waste gas is $(3,475 + 705)(0.001) = 4.18 \text{ Btu/scf}$.

Methane (CH_4) will be used as an auxiliary fuel. It has a heating value of 877.3 Btu/scf. The amount will be calculated by making an energy balance on the combustion chamber.

Thermal Recovery Efficiency

The energy balance shown in Figure 7.4 defines the *thermal recovery efficiency* (TRE) and shows that the preheater exit temperature of 1150°F will give the desired 70% efficiency. The temperature in the preheater cannot rise too close to the ignition temperature and it cannot drop below the acid dew point.

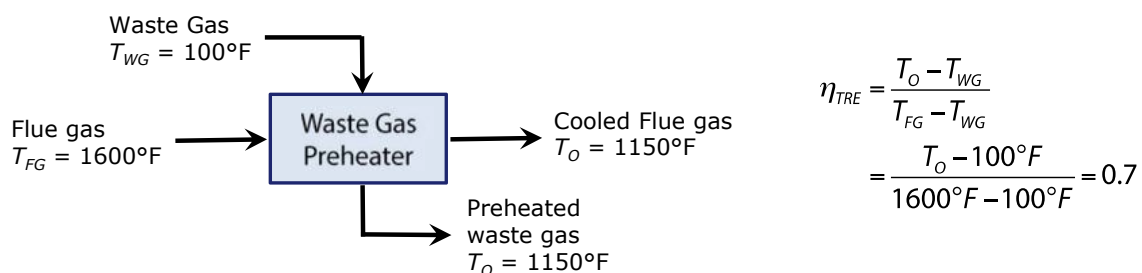


Figure 7.4 Energy balance on the waste gas heat exchanger defines the thermal recovery efficiency

The thermal recovery efficiency of the heat transfer unit (the preheater) is

$$\eta_{TRE} = \frac{\text{Actual heat transferred}}{\text{Maximum possible heat transfer @ 100\% efficiency}}$$

The maximum possible heat transfer at 100% efficiency would be when the exiting flue gas (being cooled) and the exiting waste gas (being heated) reach the same temperature. This condition is physically impossible to achieve. It is, however, a convenient simplification for making the calculation and Figure 7.5 shows the cooled flue gas and the preheated waste gas with the same temperature of 1150°F.

The actual heat transfer to the waste gas is:

$$Q_{\text{Actual}} = \rho_{WG} F_{WG} c_{p,WG} (T_O - T_{WG})$$

The maximum possible heat transfer is:

$$Q_{\text{Max}} = \rho_{WG} F_{FG} c_{p,FG} (T_{FG} - T_{WG})$$

where ρ = gas density (lb/scf)

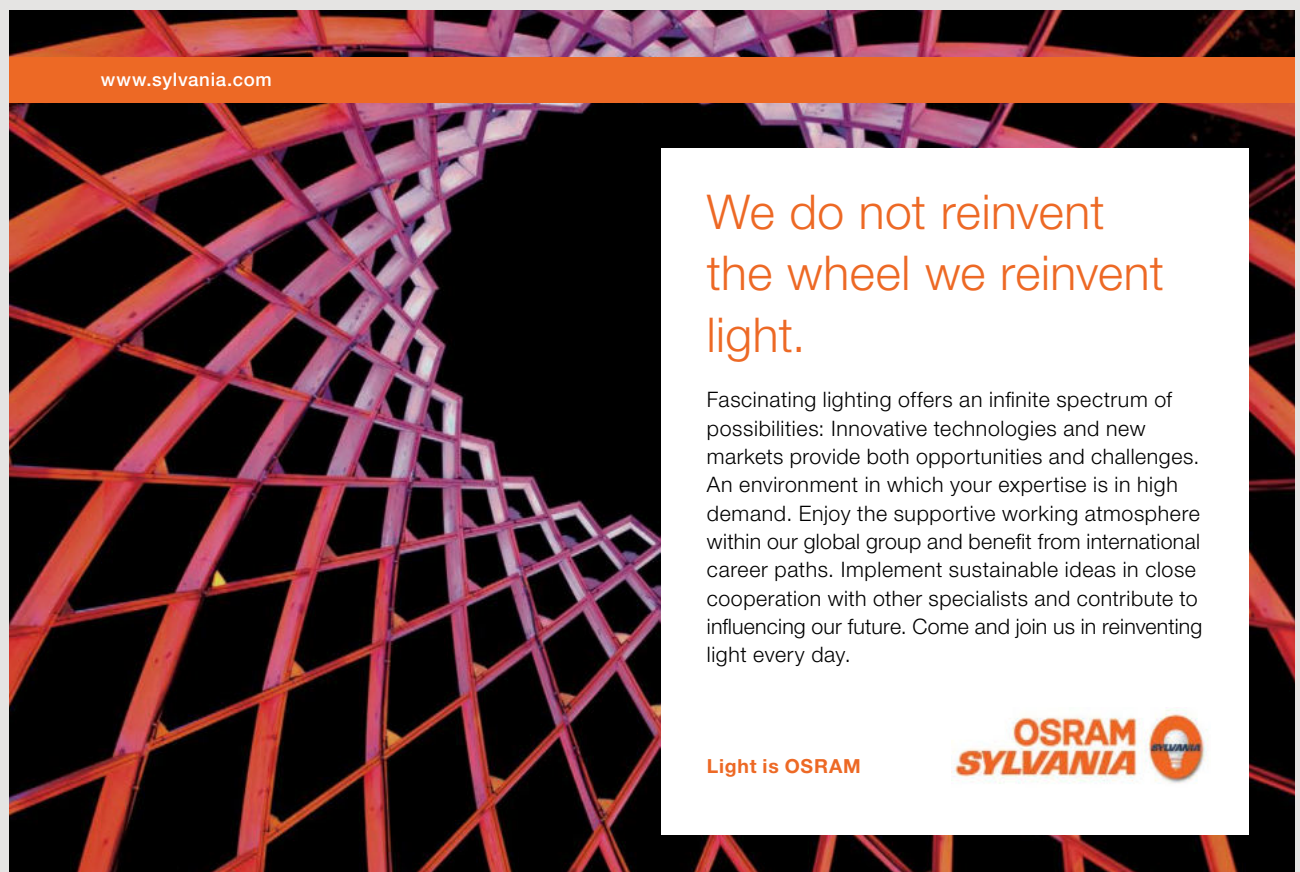
F = gas volumetric flowrate (scfm)

c_p = specific heat of the gas (Btu/lb-°F)

T_O = temperature of gas output exiting the heat exchanger (°F)

T_{WG} = gas temperature of waste gas (°F)

T_{FG} = gas temperature of flue gas (°F)



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The density, specific heat and flow rate are assumed equal for the waste gas and flue gas because both are 99.8% air (i.e. the presence of the other gases is negligible).

For the desired thermal recovery efficiency of 70% ($\eta_{TRE} = 0.7$)

$$\eta_{TRE} = \frac{Q_{Actual}}{Q_{Max}} = \frac{\rho_{WG} F_{WG} c_{P,WG} (T_O - T_{WG})}{\rho_{FG} F_{FG} c_{P,FG} (T_{FG} - T_{WG})} = \frac{(T_O - T_{WG})}{(T_{FG} - T_{WG})} = \frac{T_O - 100^\circ\text{F}}{1600^\circ\text{F} - 100^\circ\text{F}} = 0.7$$

and $T_O = 1150^\circ\text{F}$

Overall Energy Balance and the Methane Fuel Requirement

The overall energy balance for the combustion chamber is

$$H_{WG} + H_{Fuel} = Q_{FG} + H_{Loss}$$

where Q_{FG} = sensible heat flow in the flue gas (Btu/min)

H_{WG} = heat supplied by combustion of benzene and methyl chloride

H_{Fuel} = heat supplied by the methane fuel

Subscripts indicate waste gas (WG) and flue gas (FG).

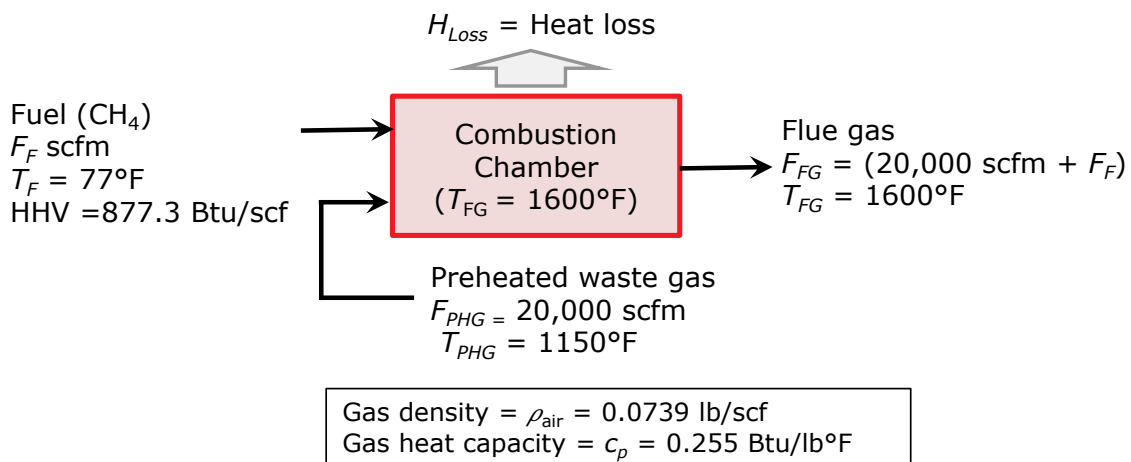


Figure 7.5 Definition diagram for the energy balance on the combustion chamber to calculate the fuel requirement. F denotes a volumetric flow rate (scfm).

The heat loss (H_{Loss}) is assumed as 10% of the heat energy in the combustion chamber flue gas, so the energy balance simplifies to

$$H_{WG} + H_{Fuel} = 1.1 Q_{FG}$$

The quantities are

$$H_{WG} = (20,000 \text{ scfm})(4.18 \text{ Btu/scf}) = 83,600 \text{ Btu/min}$$

$$H_{\text{Fuel}} = (F_{\text{Fuel}})(877.3 \text{ Btu/scf})$$

$$\begin{aligned} Q_{FG} &= (20,000 \text{ scfm})(0.0739 \text{ lb/scf})(0.255 \text{ Btu/lb}^\circ\text{F})(1600^\circ\text{F} - 1150^\circ\text{F}) + (F_{\text{Fuel}}) \\ &\quad (0.0739 \text{ lb/scf})(0.255 \text{ Btu/lb}^\circ\text{F})(1600^\circ\text{F} - 77^\circ\text{F}) \\ &= 169,600 \text{ Btu/min} + (F_{\text{Fuel}})(28.7 \text{ Btu/scf}) \end{aligned}$$

$$H_{\text{Loss}} = 0.1 Q_{FG} = 16,960 \text{ Btu/min} + (F_{\text{Fuel}})(2.87 \text{ Btu/scf})$$

where F_{Fuel} = volumetric flow rate (scfm) of methane and other variables are as defined above.

These values are used in the energy balance equation

$$H_{WG} + H_{\text{Fuel}} = Q_{FG} + H_{\text{Loss}}$$

$$83,600 \text{ Btu/min} + (F_{\text{Fuel}})(877.3 \text{ Btu/scf})$$

$$= 169,600 \text{ Btu/min} + (F_{\text{Fuel}})(28.7 \text{ Btu/scf}) + 16,960 \text{ Btu/min} + (F_{\text{Fuel}})(2.87 \text{ Btu/scf})$$

$$F_{\text{Fuel}} = (102,960 \text{ Btu/min}) / (845.7 \text{ BTU/scf}) = 122 \text{ scfm}$$

Without heat recovery about five times more methane fuel would need to be added to raise the temperature of the waste gas from 100°F to 1600°F. The reader may wish to calculate the savings as a practice exercise.

7.7 CONCLUSION

Ventilation air and waste gases from industries and a variety of manufacturing processes contain regulated organic chemicals, usually solvents. The first option is source control - change the manufacturing process to eliminate the emission or to substitute a non-toxic chemical or an unregulated solvent. When the contaminated air/gas stream cannot be eliminated at the source, the contaminant must be removed or destroyed.

Removal of organic chemicals can be accomplished by adsorption, for example onto activated carbon. This may allow for solvents to be recovered for sale or reuse.

Destruction is usually accomplished by incineration or catalytic oxidation. This removes the option of recovering solvent or some other organic chemicals, but instead heat energy can be recovered.

8 ENERGY CONSERVATIVE DESIGN

Conserving energy reduces heating and cooling costs. It also conserves water and reduces costs for water, pumping, and waste treatment. This is a win-win-win situation.

One way to reduce energy use is to use more efficient devices to transform energy, e.g., buy high efficiency motors and pumps or install more efficient lighting systems. Another is to be more efficient in the exchange and recovery of heat by designing efficient heat exchanger networks.

Energy can be recovered by transferring heat from a hot stream to a colder stream by means of a heat exchanger. The real significance of the second law is often directional, as well as quantitative. Even without calculations, the second law can give insights into the design of a process, or to the improvement of an existing design.



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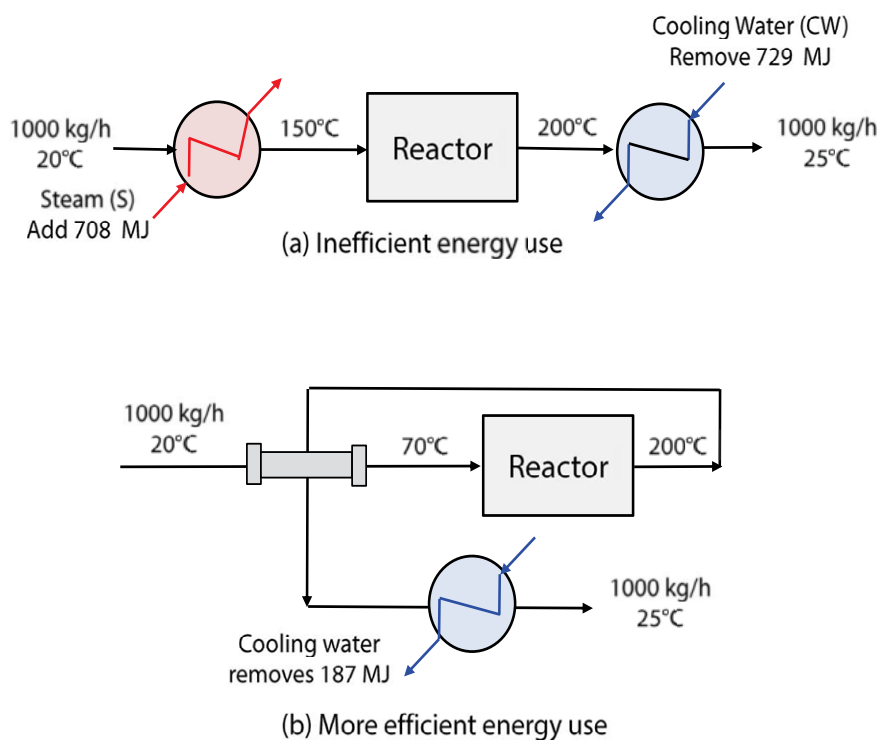


Figure 8.1 A simple energy integration system conserves energy.

Figure 8.1(a) shows a reactor that uses feedstock that is available at 20°C and a product stream that must be delivered at 25°C (to stop further reaction or to assist a following process). The mass flow through the system is 1000 kg/h. To improve the conversion in the reactor, the feed is heated from 20°C to 150°C by adding steam in the amount of 708 MJ. Chemical reactions inside the reactor release heat and the output leaves at 200°C. Cooling water is used to remove 729 MJ from the output stream. This wastes energy by dumping heat energy from the reactor into cooling water.

Figure 8.1(b) shows how the hot reactor output can be used to heat the input stream. In the heat exchange, the same 708 MJ provided by steam in the original design is transferred from the hot reactor output stream to the cold input stream. Cooling water use is four times smaller because only 187 MJ needs to be removed.

8.1 HEAT EXCHANGERS

Heat exchangers are the essential element in energy system integration. Two incoming flows, which may be gas or liquid, are separated by a metal surface and heat moves across the surface from the warmer flow to the colder one. The internal geometry of the heat transfer surface, the material (e.g. corrosion resistant steel) and other construction details exist in many variations and combinations. These details will be overlooked in order to focus on how much heat can be exchanged and how heat exchangers are networked for energy recovery efficiency.

Figure 8.2 shows *countercurrent* and *co-current* flow through an idealized heat exchanger. The cost of a heat exchanger depends on the surface area across which the heat energy moves from the hot stream to the cold stream. The greater the temperature difference between the two streams, the faster heat is transferred and the smaller the heat exchanger can be. This is accomplished by having counter-current flow of the hot and cold streams.

In a co-current flow heat exchanger, the benefit of a large initial temperature difference is quickly lost as the two streams approach a common temperature mid-way between the temperatures of the entering hot and cold streams. In countercurrent flow, the exiting hot stream can be cooled to near the entering cold stream temperature and the exiting cold stream temperature can approach the incoming hot stream temperature. This gives a larger 'average' temperature difference and makes a countercurrent heat exchanger more efficient than the co-current design.

The practical operating difference for effective heat exchange is $\Delta T = 5^{\circ}\text{C}$; working at a smaller ΔT increases the size and cost of the heat exchanger.

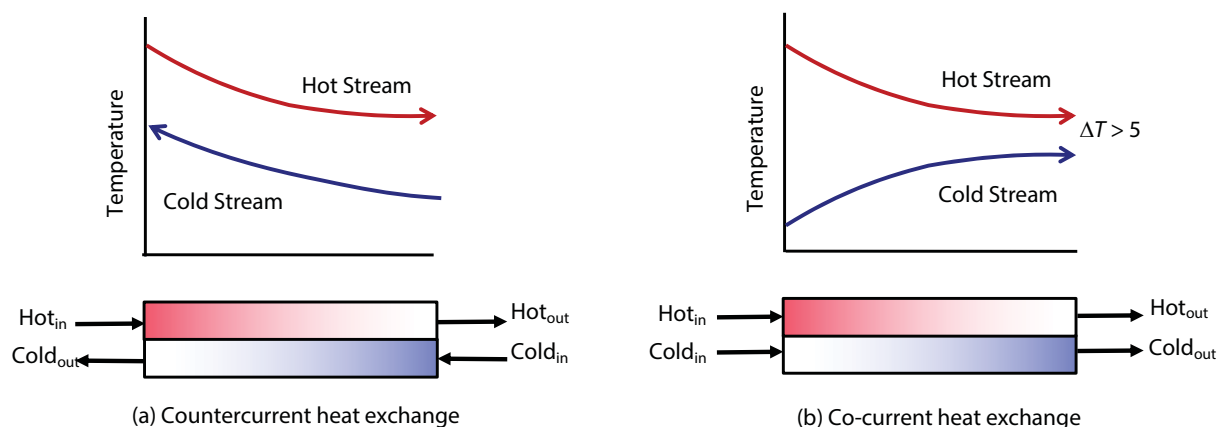


Figure 8.2 Countercurrent heat exchange is much more efficient than a co-current arrangement.

The general heat transfer equation for a heat exchanger is

$$Q = U A \Delta T_m$$

where Q is the amount of heat transferred (MJ/h, W, or Btu/h)

U is the overall heat transfer coefficient (MJ/h - m^2 - $^{\circ}\text{C}$, W/ m^2 - $^{\circ}\text{C}$, or Btu/h - ft^2 - $^{\circ}\text{F}$)

A is the area of the heat transfer surface (m^2 or ft^2), and

ΔT_m is the log mean temperature difference ($^{\circ}\text{C}$ or $^{\circ}\text{F}$) of the hot and cold streams.

The log mean temperature difference, ΔT_m , is calculated using the temperature difference at the two ends of the heat exchanger for both co-current or countercurrent flow, as shown in Figure 8.3. The larger difference is defined as T_{Max} and smaller difference is T_{Min} . The hot stream is identified by the subscript H , as in $T_{H,\text{In}}$. Likewise the cold stream is labeled $T_{C,\text{In}}$. Figure 8.4 is a graphical estimation of ΔT_m .

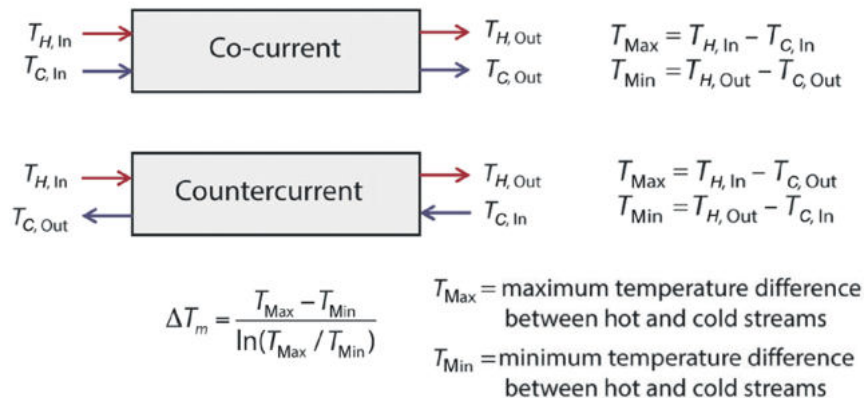


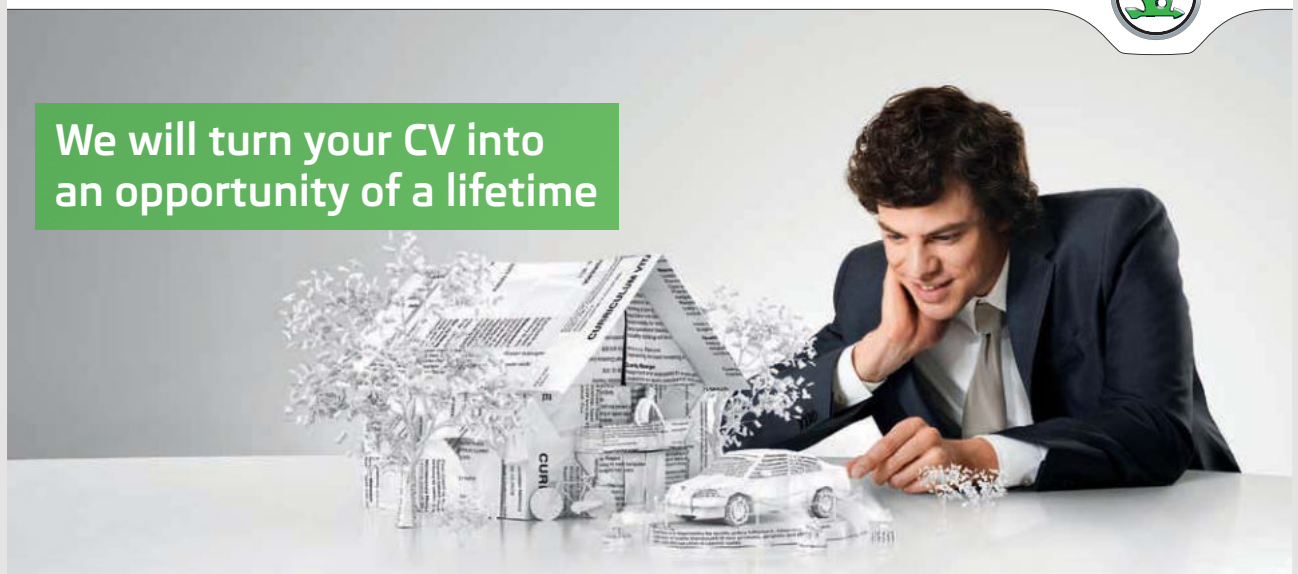
Figure 8.3 Definition of the log-mean temperature difference in heat exchangers.
 (Note: These equations are modified for condensers and boilers.)

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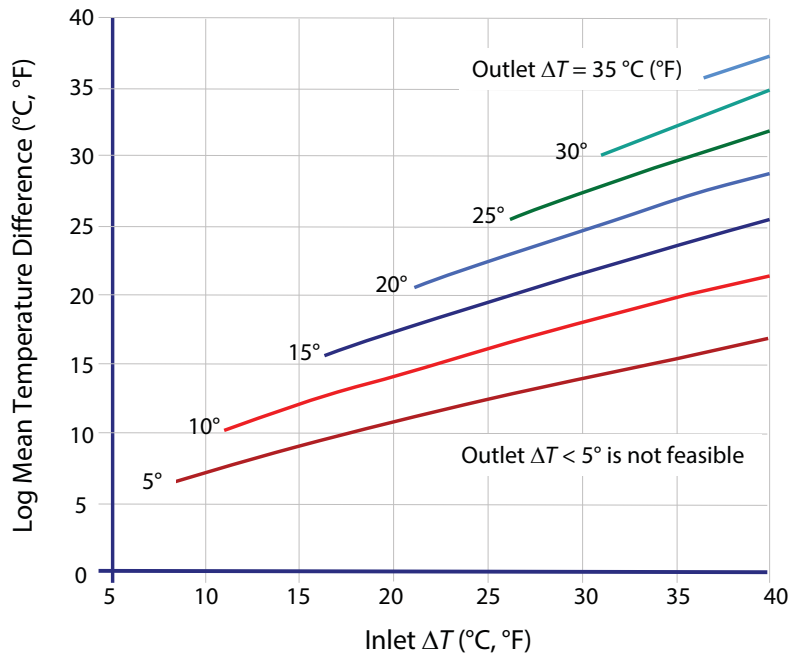


Figure 8.4 Graphical estimation of the log mean temperature difference. Horizontal axis is the temperature difference at the heat exchanger inlet. The numbers beside the curves are the temperature difference at the heat exchanger outlet. An outlet difference less than 5°C is not feasible. (adapted from The Engineering Toolbox)

The value of the *overall heat transfer coefficient* depends on the internal geometry of the equipment and the composition of the hot and cold fluids. Many times the cold stream is water and the hot stream is steam, hot water, or hot air. As a rough guide ($1 \text{ BTU/h}\cdot\text{F}\cdot\text{ft}^2 = 5.67 \text{ W/m}^2\cdot^\circ\text{C} = 20 \text{ kJ/h}\cdot\text{m}^2\cdot^\circ\text{C}$).

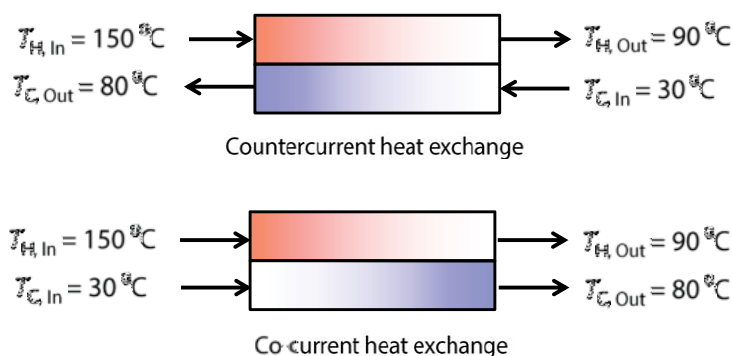
Gas-gas exchange $U = 10 - 100 \text{ W/m}^2\cdot^\circ\text{C}$
 $U = 40 - 400 \text{ kJ/h}\cdot\text{m}^2\cdot^\circ\text{C}$
 $U = 2 - 20 \text{ BTU/h}\cdot^\circ\text{F}\cdot\text{ft}^2$

Liquid-liquid exchange $U = 500 - 5000 \text{ W/m}^2\cdot^\circ\text{C}$
 $U = 2000 - 20,000 \text{ kJ/h}\cdot\text{m}^2\cdot^\circ\text{C}$
 $U = 100 - 1000 \text{ BTU/h}\cdot^\circ\text{F}\cdot\text{ft}^2$

Boiling liquids or condensing vapors $U = 1000 - 100,000 \text{ W/m}^2\cdot^\circ\text{C}$
 $U = 4000 - 400,000 \text{ kJ/h}\cdot\text{m}^2\cdot^\circ\text{C}$
 $U = 200 - 20,000 \text{ BTU/h}\cdot^\circ\text{F}\cdot\text{ft}^2$

EXAMPLE 8.1 HEAT EXCHANGER AREA CALCULATION

A heat exchanger will transfer $Q = 1,000 \text{ MJ/h}$ and operates with $U = 1.4 \text{ MJ/h}\cdot\text{m}^2\cdot^\circ\text{C}$. Make the energy balance on the heat exchangers in Figure 8.5.

**Figure 8.5** Countercurrent and Co-current heat exchanges*Countercurrent heat exchange*

$$\begin{array}{ll} T_{H, \text{In}} = 150^{\circ}\text{C} & T_{H, \text{Out}} = 90^{\circ}\text{C} \\ T_{C, \text{Out}} = 80^{\circ}\text{C} & T_{C, \text{In}} = 30^{\circ}\text{C} \end{array}$$

$$T_{\text{Max}} = T_{H, \text{In}} - T_{C, \text{Out}} = 150^{\circ}\text{C} - 80^{\circ}\text{C} = 70^{\circ}\text{C}$$

$$T_{\text{Min}} = T_{H, \text{Out}} - T_{C, \text{In}} = 90^{\circ}\text{C} - 30^{\circ}\text{C} = 60^{\circ}\text{C}$$

$$\Delta T_m = \frac{T_{\text{Max}} - T_{\text{Min}}}{\ln(T_{\text{Max}} / T_{\text{Min}})} = \frac{70^{\circ}\text{C} - 60^{\circ}\text{C}}{\ln(70^{\circ}\text{C}/60^{\circ}\text{C})} = 64.9^{\circ}\text{C}$$

$$A = Q/U\Delta T_m = (1000 \text{ MJ/h})/(1.4 \text{ MJ/h}\cdot\text{m}^2\cdot^{\circ}\text{C})(64.9^{\circ}\text{C}) = 11 \text{ m}^2$$

Co-current heat exchange

$$\begin{array}{ll} T_{H, \text{In}} = 150^{\circ}\text{C} & T_{H, \text{Out}} = 90^{\circ}\text{C} \\ T_{C, \text{In}} = 30^{\circ}\text{C} & T_{C, \text{Out}} = 80^{\circ}\text{C} \end{array}$$

$$T_{\text{Max}} = T_{H, \text{In}} - T_{C, \text{In}} = 150^{\circ}\text{C} - 30^{\circ}\text{C} = 120^{\circ}\text{C}$$

$$T_{\text{Min}} = T_{H, \text{Out}} - T_{C, \text{Out}} = 90^{\circ}\text{C} - 80^{\circ}\text{C} = 10^{\circ}\text{C}$$

$$\Delta T_m = \frac{T_{\text{Max}} - T_{\text{Min}}}{\ln(T_{\text{Max}} / T_{\text{Min}})} = \frac{120^{\circ}\text{C} - 10^{\circ}\text{C}}{\ln(120^{\circ}\text{C}/10^{\circ}\text{C})} = 44.4^{\circ}\text{C}$$

$$A = Q/U\Delta T_m = (1000 \text{ MJ/h})/(1.4 \text{ MJ/h}\cdot\text{m}^2\cdot^{\circ}\text{C})(44.4^{\circ}\text{C}) = 16 \text{ m}^2$$

The co-current heat exchanger is 45% larger than the countercurrent exchanger that does the same work.

EXAMPLE 8.2 ODOR CONTROL

A packinghouse rendering process has an odor problem caused by venting 25,000 kg/h of steam that contains unpleasant smelling compounds. The odorous compounds can be removed by condensing the

steam and cooling the condensed water below 70°C. The energy content (enthalpy) of the steam at 100°C is 2,257 kJ/kg. A condenser with $A = 420 \text{ m}^2$ heat exchange area and a heat transfer coefficient $U = 2,400 \text{ kJ/h-m}^2\text{-}^\circ\text{C}$ can operate at a log mean temperature difference $\Delta T_m = 60^\circ\text{C}$.

The heat transferred from steam to cooling water in the heat exchanger is:

$$Q = UA\Delta T_m = \left(2,400 \frac{\text{kJ}}{\text{h-m}^2\text{-}^\circ\text{C}}\right)(420 \text{ m}^2)(60^\circ\text{C}) = 60,480,000 \text{ kJ/h} = 60,500 \text{ MJ/h}$$

Condensing the steam to water (at 100°C) requires removing 2,257 kJ/kg, for a cooling requirement of $Q = mH_v = (25,000 \text{ kg})(2,257 \text{ kJ/kg}) = 56,425,000 \text{ kJ/h}$

Cooling the condensed water from 100°C water to T_{Out} is

$$\begin{aligned} \left(25,000 \frac{\text{kg}}{\text{h}}\right)\left(4.184 \frac{\text{kJ}}{\text{kg}^\circ\text{C}}\right)(100^\circ\text{C} - T_{\text{Out}}) &= 60,500 \text{ MJ/h} - 56,425 \text{ MJ/h} \\ &= 4,075 \text{ MJ/h} = 4,075,000 \text{ kJ/h} \end{aligned}$$

$$100^\circ\text{C} - T_{\text{Out}} = \frac{4,075,000 \text{ kJ/h}}{(25,000 \text{ kg/h})(4.184 \text{ kJ/kg}^\circ\text{C})} = 39^\circ\text{C}$$

$$T_{\text{Out}} = 100^\circ\text{C} - 39^\circ\text{C} = 61^\circ\text{C}$$



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8.2 HEAT EXCHANGER NETWORKS

A process may have several streams that need to be heated or cooled and the maximization of heat recovery requires that a network of heat exchangers should be used. This section shows what can be done with an intelligent arrangement, but without explaining the design strategy.

Figure 8.6 shows a *heat exchange network* (HEN) with three heat exchangers, two hot streams (H1 and H2) and two cold streams (C1 and C2). The relative mass flows are $C1 = 1$, $C2 = 4$, $H1 = 2$, and $H2 = 3$.

The objective is to achieve target temperatures for the hot and cold streams with the least use of steam or cold water. Table 8.1 shows initial and target stream temperatures, the heat to be added (+) or removed (–) and the ratio of the flow heat capacities (mc_p) between the hot and cold streams. The sum of the required heat additions and removals is $(2.5 + 33.0 - 16.5 - 22.5) = -3.5$ MW.

A practical technical limitation is that the *minimum temperature difference* between a hot stream and a cold stream is 10°C . This limit is reached in exchanger number 2: the temperature difference is 10°C ($172.5^\circ\text{C} - 162.5^\circ\text{C}$). Steam provides 5.75 MW heat and cooling water removes 9.25 MW. Furthermore, an energy balance (first law of thermodynamics) requires that any heat added from steam be accompanied by a commensurate increase in cooling water duty. The use of 5.75 MW of steam in the proposed initial HEN requires $5.75 + 3.5 = 9.25$ MW of cooling water.

Stream	Supply Temp. (°C)	Target Temp. (°C)	mc_p MW/°C	Heat to be added (+) or removed (–)	Ratio of mc_p Hot to Cold stream	
					H ₁	H ₂
H1	245	80	0.1	–16.5		
H2	180	105	0.3	–22.5		
C1	70	120	0.05	2.5	2	6
C2	50	215	0.2	33.0	0.5	1.5

Table 8.1 Data for the heat exchanger network problem (energy units are megawatts, MW).

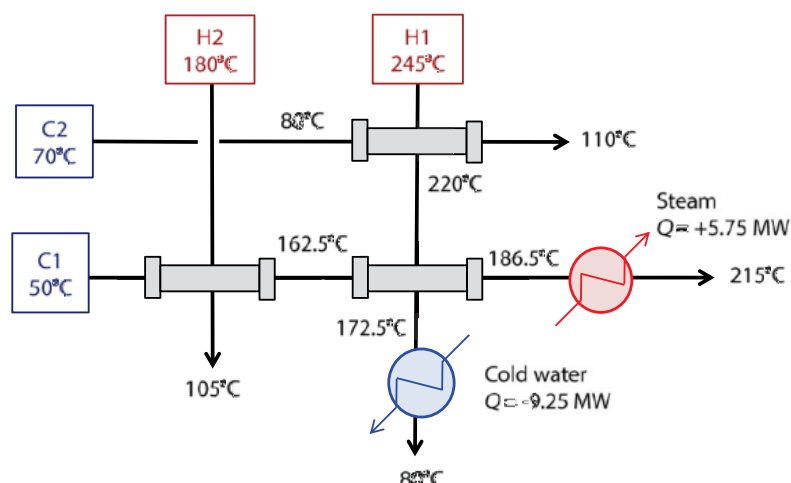
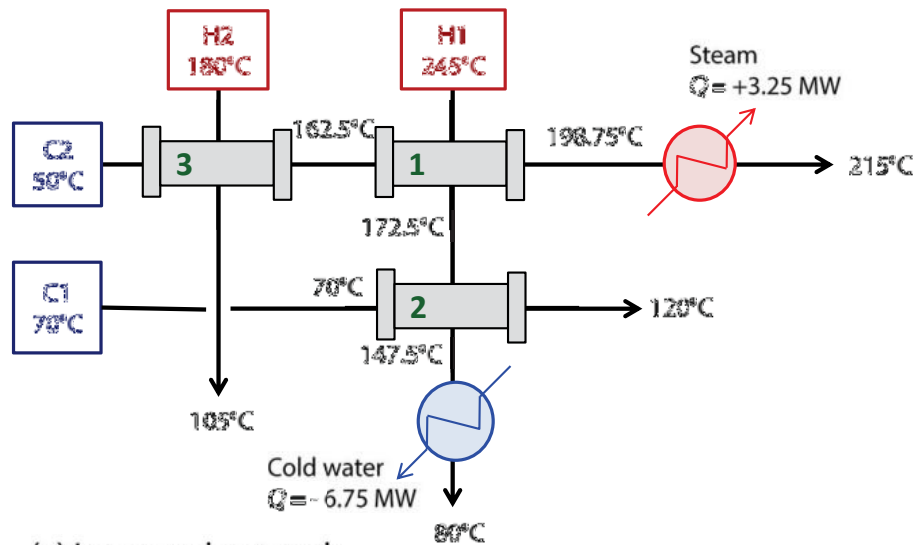


Figure 8.6 Original heat exchanger network (Sama 1995). (Relative mass flows are $C1 = 1$, $C2 = 4$, $H1 = 2$, and $H2 = 3$.)

The guidelines in Table 8.2 are instructive for a general understanding of how to improve a heat exchanger network. They may not lead to the optimal design, but they identify three important design decisions and potential improvements in the network in Figure 8.4.

1. The larger the mass flow, the larger the opportunity to save (or waste) energy.
2. Minimize the mixing of streams with differences in temperature, pressure, or chemical composition.
3. Do not discard heat at high temperatures to the ambient, or to cooling water.
4. Do not heat refrigerated streams with hot streams or with cooling water.
5. When choosing streams for heat exchange, try to match streams where the final temperature of one is close to the initial temperature of the other.
6. When exchanging heat between two streams, the exchange is more efficient if the flow heat capacities of the streams are similar. If there is a big difference between the two, consider splitting the stream with the larger flow heat capacity.
7. Minimize the use of intermediate heat transfer fluids when exchanging heat between two streams.
8. Heat (or refrigeration) is more valuable when its temperature is farther from ambient.
9. Do not use excessively large or excessively small thermodynamic driving forces in process operations.
10. The economic optimum temperature difference at a heat exchanger decreases as the temperature decreases and, vice versa.

Table 8.2 Some Common Sense Second Law Guidelines (Sama 1995)



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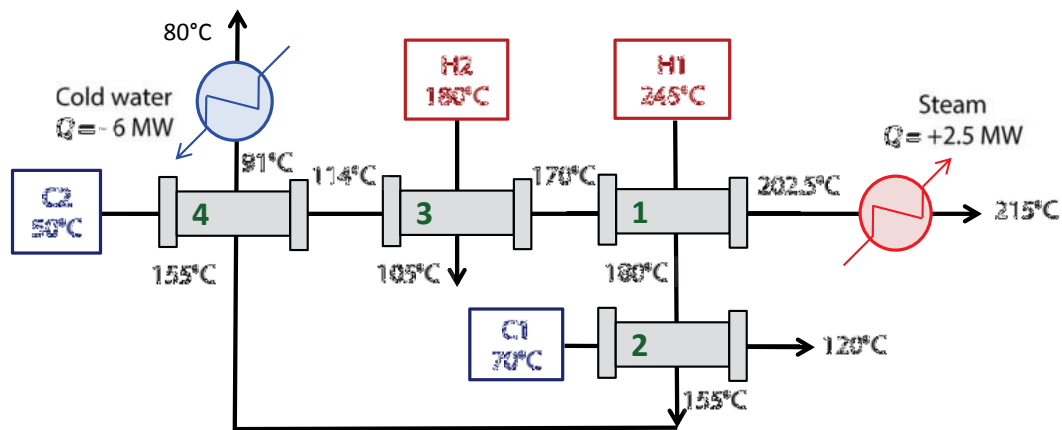
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(b) Maximum energy recovery network

Figure 8.7 (a) An improved heat exchanger network. (b) Maximum energy recovery heat exchanger network (Sama 1995). (The relative mass flows are $C1 = 1$, $C2 = 4$, $H1 = 2$, and $H2 = 3$.)

Guideline 3 says “Do not discard heat at high temperature to the ambient, or to cooling water.” In the original network, heat is discarded to cooling water from stream H1 at 172.5°C.

Guideline 5. The first heat exchange match for stream H1 (the hottest stream) at 245°C should be with C2, the cold stream that needs to be heated to the highest temperature, 215°C. For stream C2, this should be its final match before its need for heat from steam.

Guideline 6. The ratio of the flow heat capacity between streams H1 and C1 is 2, while that ratio between H2 and C1 is 6. This indicates that H1 should be used to heat C1 and that a match between stream H2 and C1 should be avoided. If a match between H2 and C1 cannot be avoided, the possibility of splitting stream H2 should be considered.

Improved networks for energy recovery are shown in Figure 8.7. The system in Figure 8.7(a) reduces steam and cooling water by 43% and 27%, respectively, from the original design. In this network, stream C2, after cooling stream H2, is used to cool stream H1 first, as suggested by guidelines 5 and 6. Using guideline 3, stream C1 then further cools H1 to a lower temperature than the original design (147.5 vs. 172.5) in heat exchanger 2.

The network in Figure 8.7(b), which is an optimal energy recovery scheme, requires only 2.5 MW of heat from steam and reduces steam and cooling water by 57% and 35%, respectively.

8.3 PINCH ANALYSIS FOR HEAT EXCHANGER NETWORK DESIGN

Pinch analysis is a method for analyzing efficient heat exchanger networks that uses an enthalpy-temperature diagram on which are plotted curves for the hot streams and the cold streams. Pinch analysis will find the most effective network, whereas the more intuitive analysis in the previous section may not, even in fairly small networks.

Heat exchanger network (HEN) design falls within the realm of chemical and mechanical engineers. Work on systematic design of networks was established years ago (Holman 1971, Linhoff & Hindmarsh 1983, Rudd et al 1973). Later work has improved the methods and provided computer-aided design software, and a vast literature can be found on-line. Freeware programs that use thermodynamic pinch analysis for heat exchanger network design are THEN (Knopf 2009) and HINT (Martin and Mato 2008). They integrate networks of heat exchangers, boilers, condensers and furnaces for best energy utilization. Grand composite curves (analogous to the mass-composite curve for water reuse system analysis) can be plotted to show the enthalpy of the process streams as a function of temperature. They also show the heat exchanger network design in a graphical form as the network as a grid diagram for visualization of the network configuration.

8.4 CONCLUSION

Recovery of heat within the processing system is essential for energy and water conservation. Heat that is removed to cooling water and dumped to the environment is wasted energy. The goal is to minimize this by designing efficient heat exchange networks.

Energy conservative design is important because heating and cooling are expensive. The pumps, heat exchangers, and cooling towers are expensive to maintain because of problems with scaling and corrosion.

Countercurrent heat exchangers are the most efficient design because they have the highest average temperature difference and this translates to the minimum exchange surface area.

9 ENERGY FOR PUMPING

A pump is a machine for raising a liquid, an incompressible fluid, to a higher level of pressure or head. This may involve actually raising the liquid to a higher elevation, or simply pushing it against the friction of pipes, valves and other hydraulic elements. More often pumping is required to do both.

Water and energy share a critical interdependency. It takes energy to treat, pump and prepare water for end-use, and water is needed to produce electricity and liquid fuels. Success in conserving water almost always conserves energy, and vice versa.

Pumping systems account for nearly 20% of the world's energy use by electric motors and 25-50% of the total electrical usage in certain industrial facilities. Energy used in pumping, treating, delivering, and preparing water for end uses accounts for about 13% of the U.S. total annual energy consumption. Each pump installation will provide an opportunity to either waste energy or save energy.



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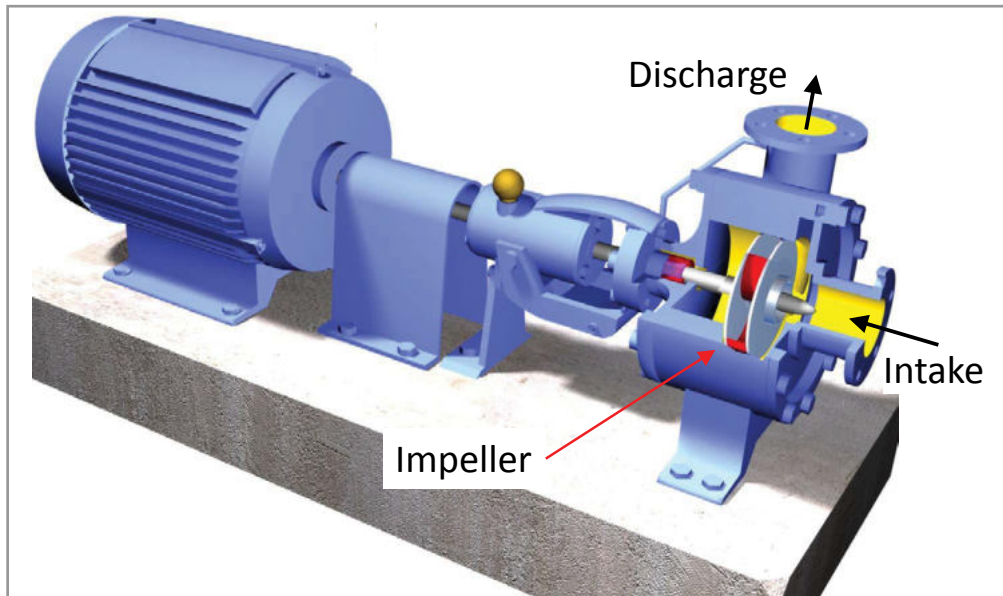


Figure 9.1 Single-stage radial centrifugal pump (Source: Wikipedia)

There are many kinds of pumps (Wikipedia has an animated catalog of pump types.) Most pumps used for water are *centrifugal pumps*, such as shown in Figure 9.1. The cutaway shows an impeller (the disk with red vanes) that is rotated by the drive motor. The quantity of water discharged depends on the diameter of the impeller, the speed of rotation, and the shape of the impeller.

Many variables influence which pump, blower, and motor is best for a particular application. The information in this chapter and the next is not sufficient to specify machinery; it is only an introduction to some basic principles that are related to energy use in pumping.

9.1 PUMP EFFICIENCY

The efficiency of a machine refers to how well it can convert one form of energy into another. The efficiency is 50% if one unit of energy supplied to a machine yields an output of one-half unit.

Pumping systems involve two extremely simple, yet efficient machines - the centrifugal pump and the AC induction motor. The centrifugal pump converts mechanical energy into hydraulic energy (flow, velocity, and pressure) and the AC motor converts electrical energy into mechanical energy. Many medium and larger centrifugal pumps operate at 75-90% efficiency and even smaller ones usually fall into the 50-70% range. Large AC motors, on the other hand, can approach an efficiency of 97% and any motor, five hp (4 kW) and above, can be designed to break the 90% barrier.

Most pumps will operate at peak efficiency intermittently because the discharge changes frequently, and the pipe networks change over time either by aging or by intentional modifications. The pumps themselves may be poorly maintained so they drift away from optimal performance.

A well-maintained centrifugal pump can provide excellent service for 30 years. A pump that is abused or poorly maintained will need replacement in a much shorter time, as suggested by Figure 9.2.

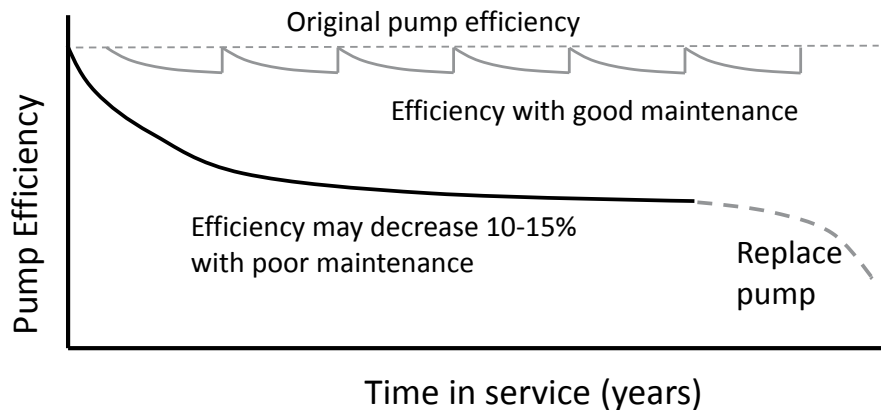


Figure 9.2 A well maintained centrifugal pump will have a long life. Poor maintenance will lead to premature replacement.

9.2 PRESSURE AND HEAD

Pressure is the resistance a pump works against. It is measured as kPa, kg/m², or lb/ft².

Another common term for this resistance is a curious term called *head* (*h*). *Pressure* and *head* measure the same thing, but in different units; they are interchangeable. *Head* is pressure expressed as an equivalent water depth and it is measured in meters or feet. This is convenient because the main job of a pump is to lift water, and it is natural to think of the lift as so many meters or feet.

Common measures of pressure are:

$$1 \text{ atmosphere (atm) of pressure} = 14.7 \text{ lb/in}^2 = 2,116 \text{ lb/ft}^2 = 10,330 \text{ kg/m}^2 = 101.35 \text{ kPa}$$

$$1 \text{ atmosphere of pressure} = 33.9 \text{ ft} = 10.33 \text{ m of water}$$

Pressure due to a depth of liquid pressing on a surface = (water density)(water depth)

$$P = \rho d$$

where ρ = density of the liquid and d = depth of the liquid. The density of water is 1000 kg/m³ and 62.4 lb/ft³.

$$1 \text{ m of head} = 1 \text{ m of water depth} = (1 \text{ m})(1000 \text{ kg/m}^3) = 1,000 \text{ kg/m}^2 = 9.807 \text{ kPa}$$

$$1 \text{ foot of head} = 1 \text{ ft of water depth} = (1 \text{ ft})(62.4 \text{ lb/ft}^3) = 62.4 \text{ lb/ft}^2 = 0.433 \text{ lb/in}^2 \text{ (psi)}$$

9.3 THE PUMP CURVE, EFFICIENCY CURVE, AND SYSTEM CURVE

Three curves, shown in Figure 9.3, define how a pumping system will operate. The *pump curve* and the *efficiency curve* are characteristics of the type of pump (single stage, multi-stage, centrifugal, etc.), the diameter of the pump impeller, and the speed of rotation. The curve shown is typical of a centrifugal pump. The pump discharge decreases as the operating head (pumping resistance) increases. There is one discharge rate that gives the maximum pumping efficiency. Higher or lower rates move the performance away from the optimum.

The *system curve*, or *system head curve*, shows is the total resistance that the pump must overcome to move the fluid. This is the sum of *static head*, *friction head*, and *pressure head*.

The duty point is at the intersection of the pump curve and the system head curves. The red circle in Figure 9.3 shows the pump operating at maximum efficiency (85%), that is operating at the *best efficiency point* with flow $Q = 15 \text{ L/s}$ and total head $h = 42 \text{ m}$.



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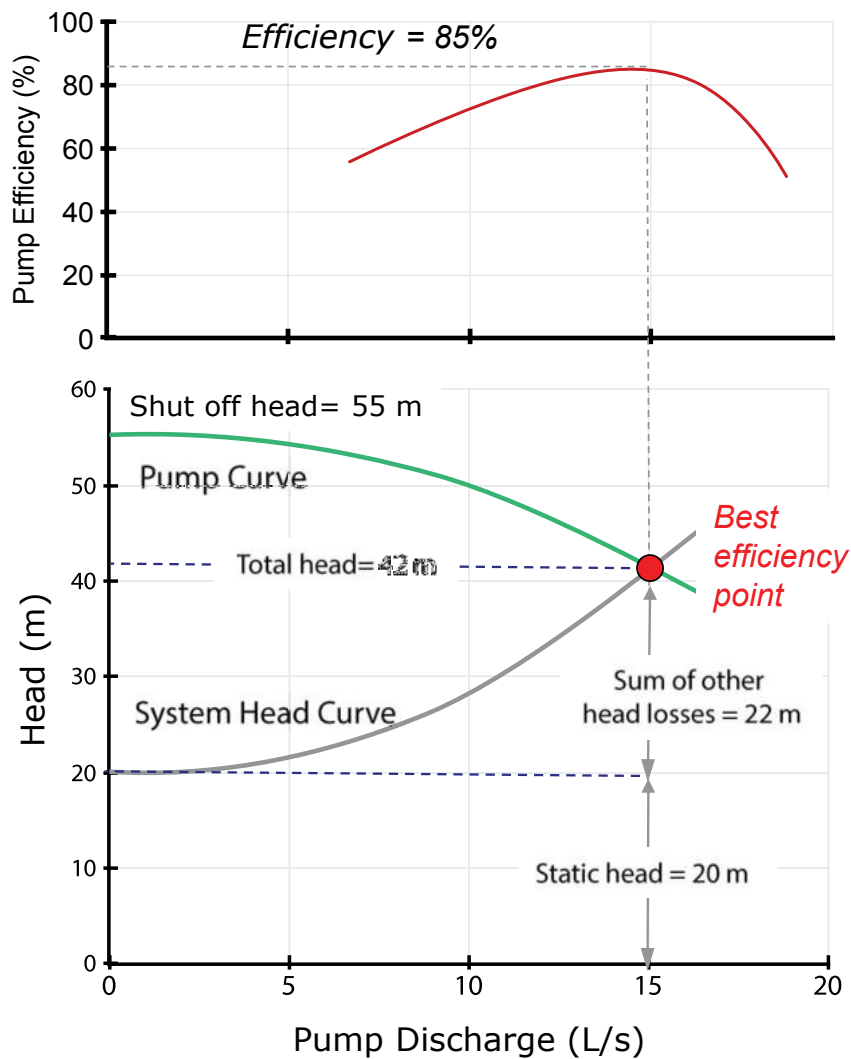


Figure 9.3 The pump head-discharge curve with the efficiency curve (top) and the system head curve. The duty point is the intersection of the pump curve and the system head curve. This curve shows a correspondence of the duty point and best efficiency point (85% efficiency) at flow $Q = 15$ L/s and total head $h = 42$ m.

The *system head curve* shows the operating head as a function of the discharge. It is defined by the system of piping, valves, filters, heat exchangers, and other devices through which the fluid must be pumped. The upward curving shape is because the friction resistance increases as the flow increases.

The height to which a fluid must be lifted is called the *static head*. This is the value of the system curve at $Q = 0$. If a pipe from the pump discharge were directed straight up the pump could lift the fluid to some height and no higher. That height is the *static*, or *shut-off head* - the head at a flow of zero.

The resistance to flow caused by the pipe, fittings, and devices is proportional to the *kinetic energy* of the fluid, $v^2/2g$. For a fixed pipe diameter (D), the velocity increases with the discharge: $Q = vA = v\pi D^2/4$ and $v = 4Q/\pi D^2$. When this quantity is expressed in meters (or feet) it is called the *velocity head*.

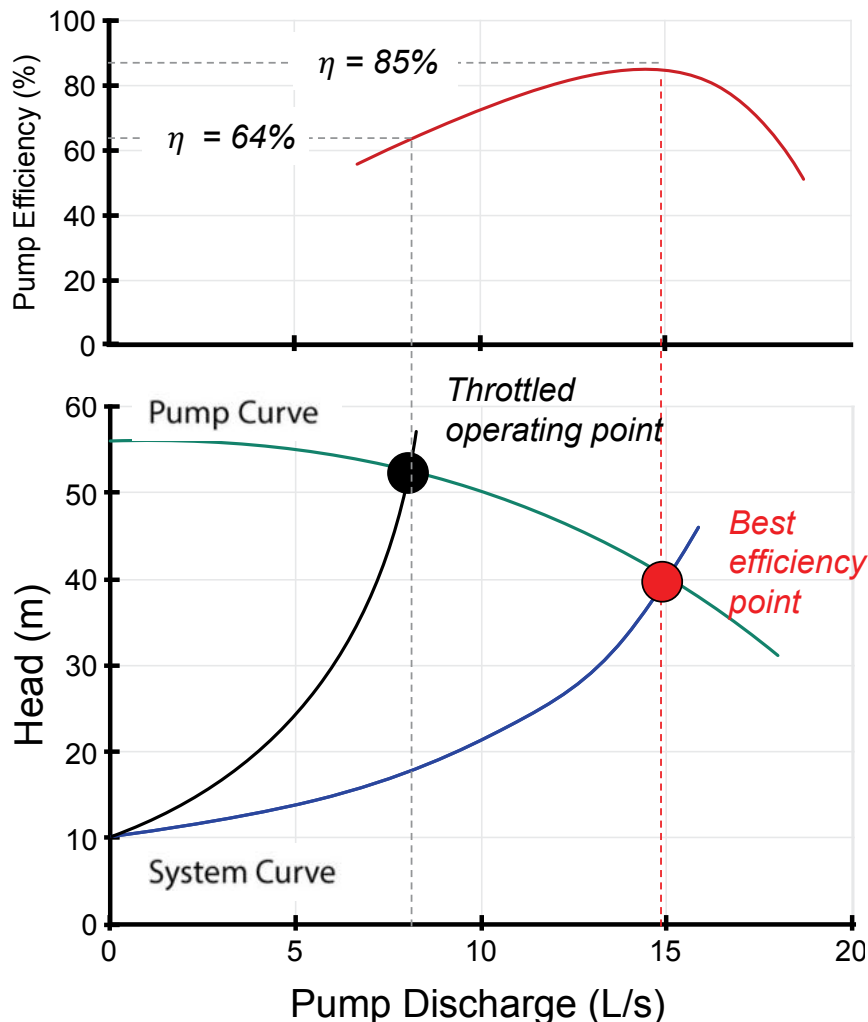


Figure 9.4 If the discharge is shifted from 15 L/s to 8 L/s the pump efficiency drops from 85% to 64%

Operating away from the best efficiency point increases operating costs and stress on the pump. Figure 9.4 shows that the pump efficiency drops from 85% to 64% when the discharge shifts from 15 L/s to 8 L/s. This shift could be caused by closing a valve to reduce the flow, or it might be caused by blockage or severe corrosion in a pipe.

If the normal operating point were to change from 15 L/s to 8 L/s, the pump, the impeller, or the operating speed should be changed. The reasons go beyond the efficiency of energy use. Continuous running below the best efficiency point will reduce the life of bearings, seals, and impellers. (Running above the best efficiency point carries similar risks to the equipment.)

A variable speed pump might be a better solution when the discharge needs to fluctuate to meet changing conditions. Changing the speed essentially produces a new pump curve with a higher efficiency. Another possibility is to alternate the pumping load between two or three pumps of different capacities so each can operate near its peak efficiency.

9.4 MOTOR EFFICIENCY

The pump industry is the largest purchaser of electric motors in the United States. According to the U.S. Department of Energy (DOE), electric motors consume more 50 percent of all electrical energy in the U.S. and more than 85 percent of industrial production electrical energy. Electric motors account for roughly \$85 out of every \$100 of a manufacturing/process plant's electrical bill. A motor that costs \$10,000 to buy may cost \$80,000 per year to operate.

The U.S Energy Policy Act of 1992 (EPACT) mandates minimum efficiencies for all electric motors up to 200 hp (150 kW) that can be purchased in the United States. Any motor made after 1997 must meet EPACT mandates. National Electrical Manufacturers Association (NEMA) created the second category called NEMA premium efficient. NEMA premium efficient electric motors range up to 500 hp (375 kW).

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The European Union also has minimum requirements for motors up to 355 kW; high efficiency is designated as IE2 and premium efficiency is IE3. There is also a super-premium category (IE4) reserved for the future. New pumps from 2017 must meet the IE3 standard. The efficiency difference between IE2 and IE3 is about two percent for large motors and 10 percent for small ones. This difference may seem small but it represents large savings in electricity use and operating cost.

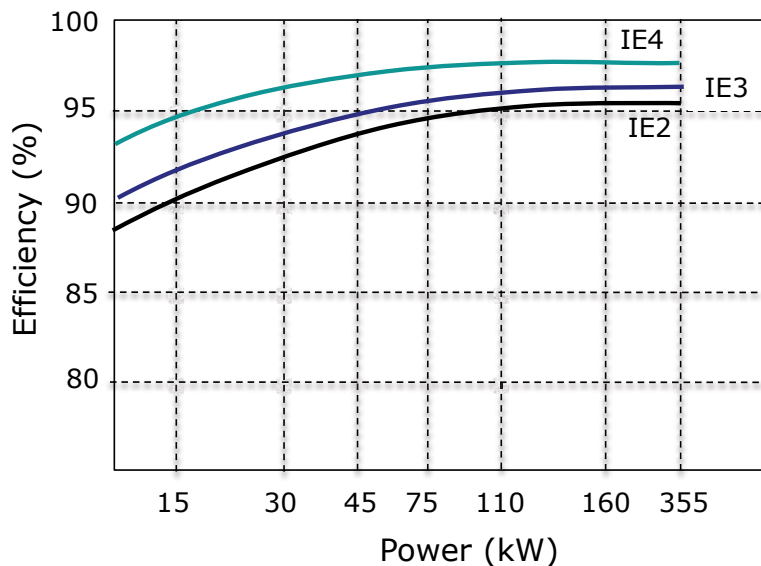


Figure 9.5 Efficiency classes for 50 Hz, 4-pole motors according to EU regulation 640/2009

9.5 POWER REQUIREMENTS

The power delivered by the pump to the fluid is P_o :

In horsepower $P_o = \gamma H Q / 550$ (1 hp = 550 ft-lb/s)

In kW $P_o = 0.0437 \gamma H Q$ (1 kW = 0.7457 hp)

where H = total head (ft)

Q = flow rate (ft³/s)

γ = specific weight of fluid (62.4 lb/ft³ for water)

In SI units

For Q in m³/h P_o (kW) = $\frac{\rho g H Q}{3.6 \times 10^6 \text{ Ws/kWh}}$

For Q in m³/s P_o (kW) = $\frac{\rho g H Q}{1,000 \text{ W/kW}}$

where ρ = density of water (kg/m^3)
 g = acceleration of gravity (9.81 m/s^2)
 H = total head (m)
 Q = flow rate (m^3/s or m^3/h)

The quantity $\rho g H Q$ has units of $\text{kg}\cdot\text{m}^2/\text{s}^3$ which is W or J/s. The denominator converts this to kW.

The power that must be supplied by the drive motor to the pump is

$$P_s = P_o / \eta_p$$

The power input that must be supplied to the motor is

$$P_i = P_s / \eta_m = P_o / \eta_p \eta_m$$

where P_s = power delivered to the pump (kW)
 η_p = pump efficiency
 P_i = input power to the pump motor (kW)
 η_m = efficiency of the motor

P_o is sometimes called the *water power*. P_s is also known as the 'shaft' or 'brake' horsepower. P_i is the power value used to estimate the pump power costs.

9.6 CALCULATING HEAD LOSSES FROM K VALUES

The pressure loss through pipes and pipe fittings is a multiple of the quantity $v^2/2g$, known as the *velocity head*, where

v = fluid velocity (m/s, ft/s)
 g = gravitational constant = 9.81 m/s^2 or 32.2 ft/s^2

This is a measure of the kinetic energy of the moving fluid and can be measured in meters (or feet) or in pressure units (kPa or lb/in²). It is called the *velocity head* when it is measured in meters (or feet).

The head loss for all hydraulic elements is

$$h_L = K \frac{v^2}{2g}$$

The head loss due to *friction* in a straight pipe is

$$K = f \frac{L}{D} \quad \text{and} \quad h_L = f \frac{L}{D} \frac{v^2}{2g}$$

where f = the pipe *friction factor*, which depends on the velocity, fluid viscosity, pipe diameter and pipe roughness. Consult any standard reference or fluid mechanics text to learn how this is calculated or extracted from charts. For simplicity, a value for f will be given in the examples that follow.

A few values of K for *head losses in pipe fittings* (elbows, valves, etc.) are

Standard 45° elbow:	$K = 0.35$		
Standard 90° elbow:	$K = 0.75$		
Gate valve: open:	$K = 0.17$	25% closed: $K = 0.9$	50% closed: $K = 4.5$

The head loss in pipes can be written in terms of flow instead of velocity. The flow in a pipe is

$$Q = vA \text{ and } v = Q/A$$

and
$$h_L = K \frac{Q^2}{2gA^2}$$

where A = cross-sectional area of the pipe (m^2)

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EXAMPLE 9.1 HEAD LOSS IN A PIPE WITH THREE ELBOWS

Figure 9.6 a hypothetical pipe that is 50 m long with three 90° elbows. The friction loss is the same if this is a plan view (looking down) or an elevation view. The calculation is for the pipe alone. The pressure is atmospheric at both ends of the pipe. There static head is zero because intake and discharge are at the same elevation.

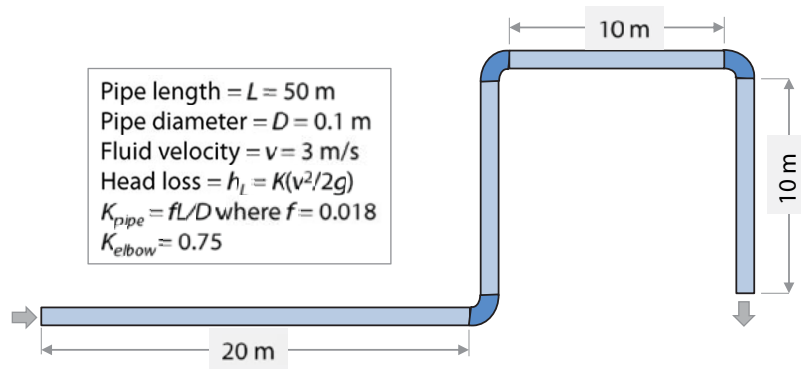


Figure 9.6 Simple pipe section with $L = 50$ m, 3 elbows, and $v = 3$ m/s has a head loss of 5.16 m.

The diameter of pipe and the elbows is $D = 0.1$ m. The velocity of water flowing in the system is $v = 3$ m/s. The gravitational constant, $g = 9.81$ m/s². The headloss of any element can be calculated using

$$\frac{v^2}{2g} = \frac{(3 \text{ m/s})^2}{2(9.81 \text{ m/s}^2)} = 0.4587 \text{ m}$$

For the pipe friction factor use $f = 0.018$. The head loss due to pipe friction is

$$h_L = f \frac{L}{D} \left(\frac{v^2}{2g} \right) = (0.018) \left(\frac{50 \text{ m}}{0.1 \text{ m}} \right) (0.4587 \text{ m}) = 4.128 \text{ m}$$

The head loss for 3 elbows, using $K = 0.75$, is

$$h_{3 \text{ elbows}} = 3K \left(\frac{v^2}{2g} \right) = 3(0.75)(0.4587) = 1.032 \text{ m}$$

Now put a gate valve in the piping system (it does not matter where in the system). The fully open gate valve has $K = 0.17$ and a half-open gate valve has $K = 4.5$. The head losses are

$$h_{\text{valve, open}} = 0.17(0.4587 \text{ m}) = 0.078 \text{ m}$$

$$h_{\text{valve, 50\% open}} = 4.5(0.4587 \text{ m}) = 2.064 \text{ m}$$

The total friction head losses for some possible systems are:

$$50 \text{ m pipe} + 3 \text{ elbows} = 4.128 + 1.032 = 5.16 \text{ m}$$

$$50 \text{ m pipe} + 3 \text{ elbows} + \text{fully open gate valve} \\ = 4.128 + 1.032 + 0.078 \text{ m} = 5.238 \text{ m}$$

$$50 \text{ m pipe} + 3 \text{ elbows} + 50\text{-closed gate valve} \\ = 4.128 + 1.032 + 2.064 \text{ m} = 7.221 \text{ m}$$

EXAMPLE 9.2 OPERATING HEAD FOR A PUMP

For the system in Figure 9.7, the water surface elevations are $z_1 = 0$ at the lower tank and $z_2 = 10$ m at the upper tank. The static head of the system is $z_2 - z_1 = 10$ m. Both tanks are open to the atmosphere so the pressures are atmospheric. The diameter of the pipe is constant, $D = 0.1$ m, so $v_1 = v_2$. A new element is head loss at the intake and discharge of the pipe. There is a loss of 0.50 velocity heads ($K = 0.50$) at the intake and one velocity head at the discharge ($K = 1.0$) because the velocity goes from v in the pipe to $v = 0$ in the tank.

The pump must provide energy to lift the water 10 m and overcome the friction losses in the pipe and the fittings.

$$h_{\text{pump}} = h_{\text{pipe}} + h_{\text{minor}} + (z_2 - z_1) = h_{\text{pipe}} + h_{\text{minor}} + h_{\text{static}}$$

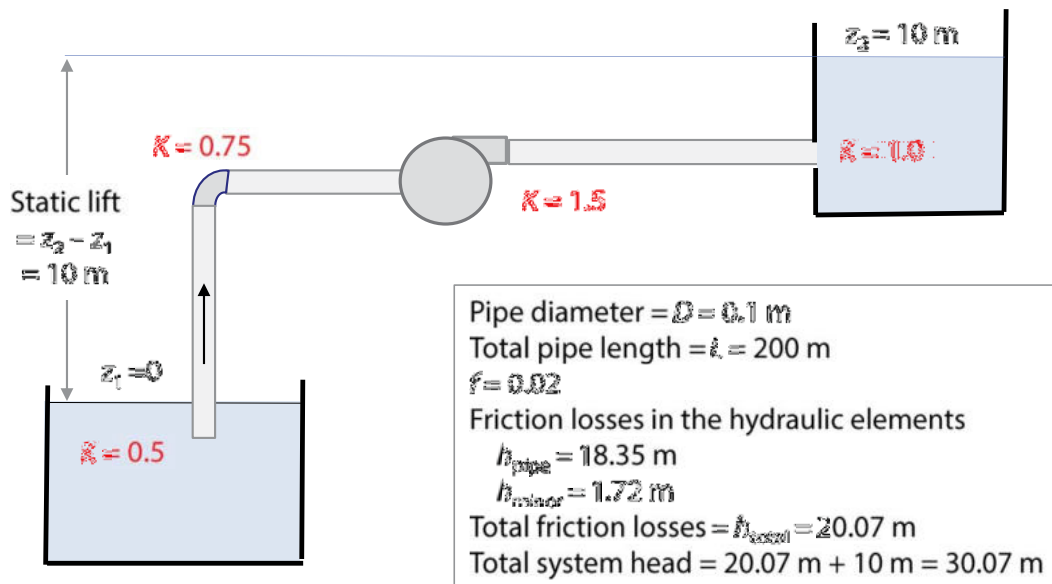


Figure 9.7 The total system head of the pump is 30.07 m, which is a static lift of 10 m plus the friction loss in 200 m of 0.1 m diameter pipe, plus four minor losses (pipe inlet, pipe outlet, elbow, and pump intake).

As in the previous example, the pipe diameter is $D = 0.1$ m and the velocity is 3 m/s, giving

$$\frac{v^2}{2g} = \frac{(3 \text{ m/s})^2}{2(9.81 \text{ m/s}^2)} = 0.4587 \text{ m}$$

The head loss in the pipe for a pipe friction factor of $f = 0.02$ is

$$h_{\text{pipe}} = f \frac{L}{D} \left(\frac{v^2}{2g} \right) = (0.02) \left(\frac{200 \text{ m}}{0.1 \text{ m}} \right) (0.4587 \text{ m}) = 18.35 \text{ m}$$

There are four minor head losses:

Inlet to the suction pipe	$K = 0.5$
Outlet to the upper tank	$K = 1.0$
Inlet to the pump	$K = 1.5$
90° elbow	$K = 0.75$

The total head loss for these four elements is

$$h_{\text{minor}} = (0.5 + 1.0 + 1.5 + 0.75)(0.4587 \text{ m}) = 3.75(0.4587 \text{ m}) = 1.72 \text{ m}$$

The total operating head is

$$h = h_{\text{pipe}} + h_{\text{minor}} + h_{\text{static}} = 18.35 \text{ m} + 1.72 \text{ m} + 10 \text{ m} = 30.07 \text{ m}$$

9.7 PIPE NETWORKS

Figure 9.8 is an example of a pipe network for a small water supply system. The nodes identify points of withdrawal. The input locations are not shown, but they could be pumping stations, elevated storage tanks, ground storage tanks, or reservoirs. Pumps and pressure reducing valves can be included in the network. This example comes from web brochure about the WatDis software (Transparent Blue 2013)



The advertisement for e-learning for kids features a large central image of a smiling woman leaning over a laptop, with two children, a boy and a girl, looking at the screen. The background is a vibrant yellow with orange and white swirling patterns. In the top left corner, there is a logo for 'e-learning for kids' consisting of a grid of colored squares. To the right of the main image, there are two circular inset photos: one showing three children looking at a book, and another showing two children working on laptops. Below these, a green oval contains three bullet points: 'The number 1 MOOC for Primary Education', 'Free Digital Learning for Children 5-12', and '15 Million Children Reached'. At the bottom, a text block provides information about the foundation, its history, and its mission.

About e-Learning for Kids Established in 2004, e-Learning for Kids is a global nonprofit foundation dedicated to fun and free learning on the Internet for children ages 5 - 12 with courses in math, science, language arts, computers, health and environmental skills. Since 2005, more than 15 million children in over 190 countries have benefitted from eLessons provided by EFK! An all-volunteer staff consists of education and e-learning experts and business professionals from around the world committed to making difference. eLearning for Kids is actively seeking funding, volunteers, sponsors and courseware developers; get involved! For more information, please visit www.e-learningforkids.org.

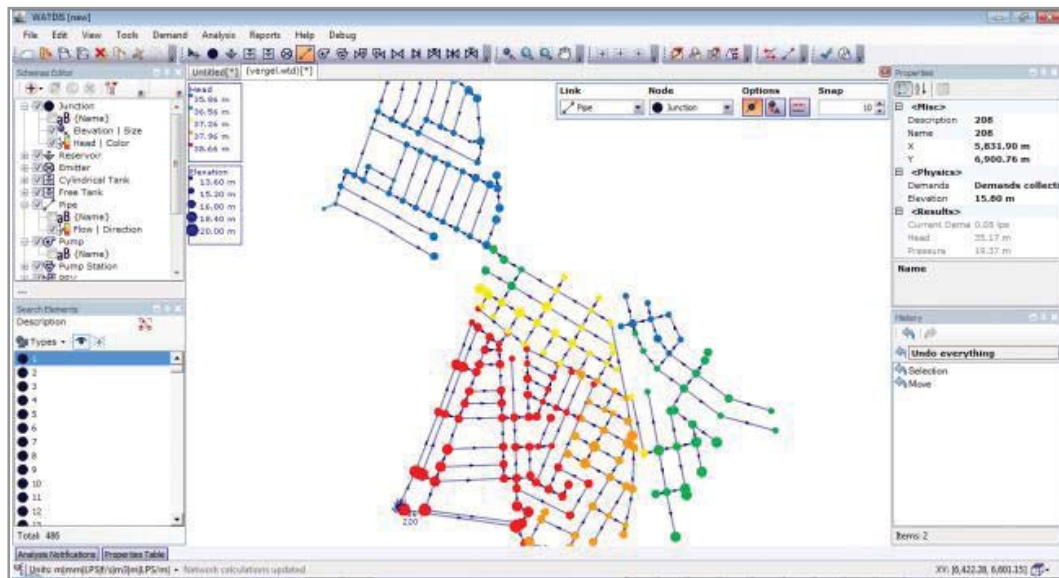


Figure 9.8 A complex looped hydraulic network being analyzed with the WatDis hydraulic network design software (Transparent Blue Consulting, 2013).

Such a network is analyzed using a system of equations that define the head loss in each pipe, a material balance equation for each loop, and an energy balance equation for each loop. The flow into each pipe junction must equal the flow out of the junction and there can only be one pressure at each junction. The flows that satisfy the conditions are found by iterative adjustments of an initial solution (Epp & Fowler 1972).

The traditional method of design was to specify the pipe system and select pumps that had the required discharge and operating head. Modern optimization technology can search simultaneously for a pipe and pump system that will minimize either the first cost or the life-cycle cost, Figure 9.9 (Hodgson & Walters, 2002). Some hydraulic design software packages are WatDis, EPANet, and WaterCAD.

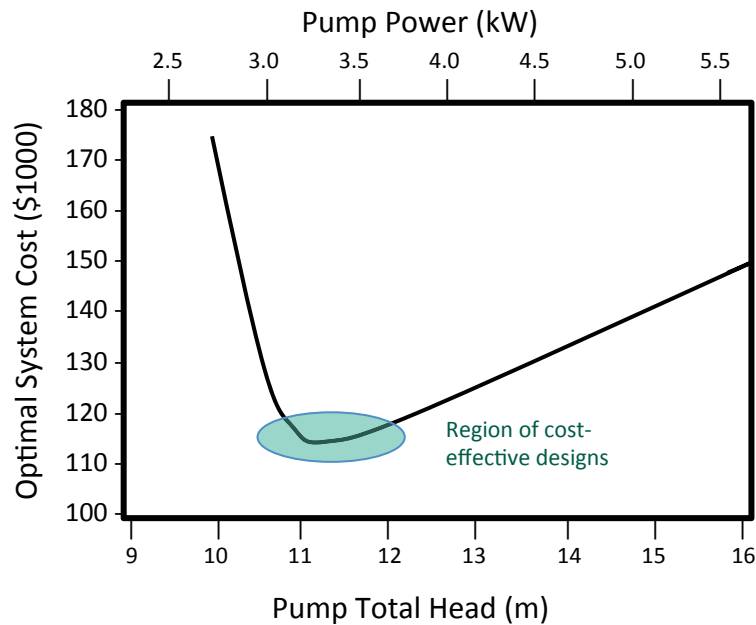


Figure 9.9 Optimized systems for pumps of different head levels and power show a region of cost effective designs. The design criterion was minimal life-cycle cost for a 20-year life.

Chapter 11 is about comparing alternate designs by taking into account the time value of money when determining project costs. The two calculations are (1) converting a series of future values to a present value and (2) converting a present value to a series of uniform future costs.

9.8 CONCLUSION

This simplified introduction of how the design of piping systems and the selection of pumps relates to energy consumption is not sufficient to do actual hydraulic design, but it should alert you to opportunities to economize.

The pipe-pump system is rarely a single long pipe with a pump at one end. Actual systems have branches and loops, changes in pipe diameter, many kinds of restrictions in the network (valves, heat exchangers, etc.) and changes in elevation.

Every pump and pumping station is an opportunity to reduce the amount of electricity used, or to increase it. Proper maintenance extends pump life and reduces lifetime operating costs.

10 ENERGY FOR BLOWERS AND COMPRESSORS

Compressors, blowers and fans are used to raise a gas, a compressible fluid, to a higher level of pressure. A compressor creates a high level of pressure. Fans move large volumes of gas with a low increase in pressure. Blowers are used in activated sludge treatment plants to move air with moderate increase of pressure. Energy consumption will account for about 75% of the total life-cycle cost for an air blower, compared with 15% for capital cost and 10% for maintenance.

The average amount of electricity used by modern wastewater treatment plants is 0.3 kWh per cubic meter of wastewater treated (1,200 kWh per million gallons) (Henze 2008). Aeration accounts for 60-70% the overall energy demand. Pumping is about 12%. If nitrification (oxidation of ammonia to nitrate) is added the air requirement increases by about 30%. A modern process that uses nitrification-denitrification with biological phosphorus removal needs less air, about 0.10-0.15 kWh/m³.



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Industries use *blowers* and *compressors* on incinerators, sprayers, dryers, etc., and they use them to convey powders and for mixing and cleaning. Most air compressors run between 60-100 hours per week at or near full capacity. Up to 10% of electricity used in industry may be to run air compressors. It takes about 8 kW of energy to produce 1 kW worth of work with compressed air. One cubic foot per minute of air (1 cfm) costs about \$100 per year for energy.

A 100-hp compressor requires about \$37,000 in annual electrical costs for a five-day, two-shift operation of about 4200 hours per year at a unit cost of \$0.1/kWh. (Note: 1 hp = 4 cfm at 100 psi). Over a 10-year period, this can be 75% or more of the total life cycle cost of the air compressor, with the purchase price representing only 12%. Because of this, even a 5% change in operating efficiency can save \$18,500 in electrical costs over 10 years.

10.1 BLOWERS

The basic concepts of air delivery systems are similar to liquid pumping systems, with the difference that air is compressible and liquids are not. Compression will raise the temperature of the air and as the temperature increases the power requirement goes up.

Centrifugal blowers are the common equipment for activated sludge aeration. The centrifugal blower, like a centrifugal pump, has an impeller on a rotating shaft. The output can be adjusted with a variable speed drive. The air flow will vary depending on the output pressure.

The blower has an operating curve, Figure 10.1, similar to a centrifugal pump, that determines the relation of air flow and discharge pressure. The air delivery piping and equipment (filters, diffusers, etc.) determine the system power curve. This also relates air flow and discharge pressure. The intersection of the operating curve and the system curve is the operating point.

The multi-stage centrifugal and the high-speed turbo blowers can maintain a relatively constant discharge pressure over a wide range of airflow.

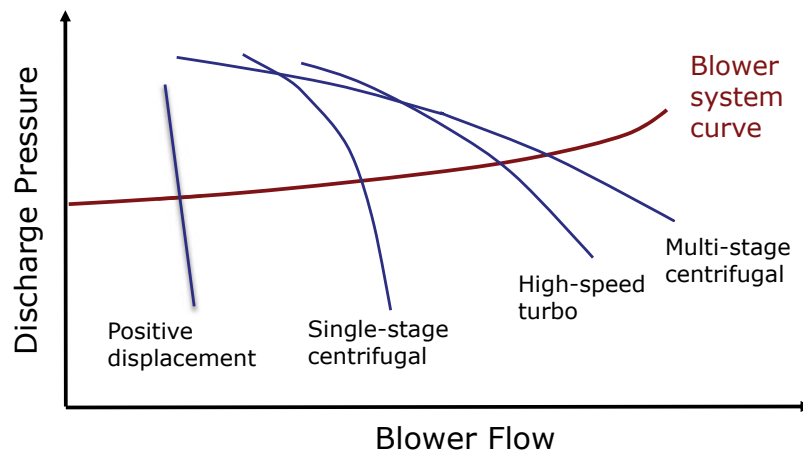


Figure 10.1 Operating curves for four types of blowers.

Figure 10.2 is the performance map for a variable speed centrifugal blower. The operating curves are shown for the maximum speed (100% rpm) and for a few other speeds. *Turndown* is the ability to reduce the blower flow rate. The lowest speed for a compressor is the *turndown limit*. Turndown can also be given as a percentage of the rated flow.

$$\text{Turndown (\%)} = 100 \left(\frac{Q_{\text{Max}} - Q_{\text{Min}}}{Q_{\text{Max}}} \right)$$

where Q_{Max} and Q_{Min} are the maximum and minimum flow rates. Values range from 45-60% but 50% is a good value for many blowers.

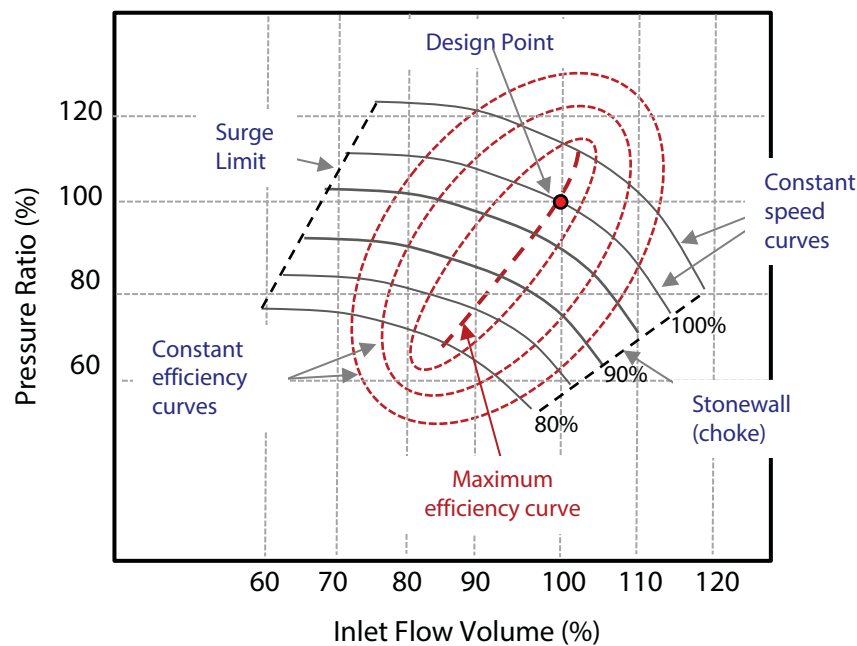


Figure 10.2 Centrifugal compressor performance map

Surge is the point at which the compressor cannot add enough energy to overcome the system resistance or backpressure. This causes a flow reversal, or surge. *Choke* occurs as the airflow increases, its velocity can approach sonic speed somewhere within the compressor stage. Blowers can operate between the surge point and the choke point.

10.2 AERATION IN WASTEWATER TREATMENT

The activated sludge process consists of aeration tanks for growing microorganisms (biomass) followed by final clarifiers that remove the biomass to produce a clear sparkling effluent. Most of the removed biomass is recycled to the aeration tank; what is not recycled is sent to sludge digestion. The organisms are aerobic; they need oxygen and they can only use oxygen that is dissolved in the wastewater. Insufficient air supply causes absolute failure to meet performance standards. This poses a design challenge because air requirements fluctuate continually due to changes in flow, organic load, temperature and atmospheric pressure. Dissolved oxygen concentration can be monitored in the aeration basins and process control computers can adjust variable speed compressors to keep pace with the demand.

The three important components are the (a) aeration tanks, (b) blowers to supply air, and (c) diffusers to disperse the air into the aeration tank contents. These are shown in Figure 10.3.

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(a) Aeration tanks at an activated sludge wastewater treatment plant



(b) Centrifugal blower



(c) Fine bubble diffusers in an aeration tank

Figure 10.3 The important parts of an activated sludge treatment system are (a) the aeration basins, (b) the centrifugal blower to supply air, and (c) the diffusers to efficiently dissolve oxygen in the wastewater.

10.3 THE DEMAND FOR OXYGEN AND AIR

The mass and volume of air required is easily obtained in most applications, such as ventilation air or incineration air supply. An important application that is not obvious is for the supply of oxygen to an activated sludge wastewater treatment process. A short explanation will be given about estimating rate of oxygen consumption and the rate and air flow that will deliver the required mass of oxygen. Efficiency of oxygen transfer are critical design steps. The details are omitted here, but they may be found in any textbook about biological wastewater treatment design (Qasim 1998, Eckenfelder 2000, Hendricks 2011, Metcalf & Eddy 2013).

An activated sludge process will fail if the oxygen supply is insufficient. Gaseous oxygen is unavailable to the microorganisms. They use *dissolved oxygen* from the liquid contents of the aeration tank. The most popular aeration systems deliver air through fine pore diffusers that cover the floor of the aeration basin. The industry standard fine bubble is about 2 mm in diameter at discharge; average bubble diameters of 0.9 mm are possible with recently developed membranes.

The fraction of oxygen in the air that dissolves in the water depends on the diffuser type, wastewater temperature, the dissolved oxygen concentration in the aeration tank, and wastewater chemistry. The oxygen transfer efficiency from the air to the liquid is 27% to 35% in clean water at 20°C and at a dissolved oxygen concentration of zero mg/L for full floor coverage of fine bubble domes. This efficiency may be as much as 30% less in wastewater than in clean water. The efficiency also decreases as temperature and dissolved oxygen concentration increase (Eckenfelder 1999).

An empirical estimate of the oxygen demand is

$$\text{kg/d O}_2 \text{ required} = a \text{ (kg/d BOD}_5 \text{ removed)} + b \text{ (kg VSS under aeration)}$$

where a and b are oxygen demand coefficients

BOD₅ = five day biochemical oxygen demand

VSS = volatile suspended solids

The first term represents the amount of oxygen consumed in the oxidation of biodegradable organic chemicals, which are measured as BOD₅; the second term represents the oxygen required for the basic metabolism of the organisms. Typical values for the coefficients are $a = 0.4 - 0.6 \text{ kg O}_2/\text{kg BOD}_5 \text{ removed/d}$ and $b = 0.05 \text{ kg O}_2/\text{kg VSS}$. These values can be obtained from simple experiments using small-scale activated sludge systems (Eckenfelder 1999).

EXAMPLE 10.1 EMPIRICAL ESTIMATE OF OXYGEN DEMAND

An activated sludge process operates at a flow of 0.1 m³/s, a hydraulic detention time of $\theta = 10 \text{ h}$, and a volatile solids concentration in the aeration tank of 1.8 kg/m³ (1800 mg/L). The BOD₅ is reduced from 0.2 kg/m³ to 0.01 kg/m³ (200 mg/L to 10 mg/L). The oxygen demand coefficients are

$$a = 0.5 \text{ kg O}_2/(\text{kg BOD}_5 \text{ removed/d})$$

$$b = 0.05 \text{ kg O}_2/\text{kg VSS under aeration}$$

$$\text{Flow} = (0.1 \text{ m}^3/\text{s})(86,400 \text{ s/d}) = 8,640 \text{ m}^3/\text{d}$$

$$\text{BOD}_5 \text{ removed per day} = (8,640 \text{ m}^3/\text{d})(0.2 \text{ kg/m}^3 - 0.01 \text{ kg/m}^3) = 1642 \text{ kg/d}$$

$$\text{Aeration tank volume} = V = Q\theta = (8,640 \text{ m}^3/\text{d})(10 \text{ h})/(24 \text{ h/d}) = 3,600 \text{ m}^3$$

VSS under aeration = $(3,600 \text{ m}^3)(1.8 \text{ kg VSS/m}^3 \text{ aeration tank volume}) = 6,480 \text{ kg VSS}$

Oxygen demand:

$$\begin{aligned}\text{kg/d O}_2 \text{ required} &= a (\text{kg BOD}_5 \text{ removed/d}) + b (\text{kg VSS under aeration}) \\ &= (0.5 \text{ kg O}_2/(\text{kg BOD}_5 \text{ removed/d})(1642 \text{ kg BOD}_5/\text{d}) + (0.05 \text{ kg O}_2/\text{kg VSS} \\ &\quad \text{under aeration})(6,480 \text{ kg VSS}) \\ &= 821 \text{ kg O}_2/\text{d} + 324 \text{ kg O}_2/\text{d} = 1,145 \text{ kg O}_2/\text{d}\end{aligned}$$

Nitrifying processes convert ammonia (NH_3) to nitrate (NO_3^-). This nitrification oxygen demand is 4.57 kg O_2 for each kilogram of ammonia that is converted to nitrate.

Dry air has a density of 1.2754 kg/m^3 at 0°C . Air is 23.2% oxygen, by mass, so one normal cubic meter of dry air at 0°C contains 0.296 kg O_2 . The best diffuser system can dissolve 25-30% of the oxygen in the air that is supplied to the aeration basin. Therefore, one Nm^3 can dissolve, at best, 0.07 to 0.09 kg of dissolved oxygen. Obviously a great volume of air must be delivered to the process.

The corresponding values in U.S. units are: density of air at $70^\circ\text{F} = 0.074887 \text{ lb/ft}^3$; 1 scfm of air contains 0.0173 lb O_2 , so one scfm might supply 0.0043 to 0.0052 lb of dissolved oxygen.



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EXAMPLE 10.2 ESTIMATING THE REQUIRED AIR FLOW (SI UNITS)

A 4,000 m³/d activated sludge process needs an oxygen supply of 1,600 kg/d. The oxygen content of 1 Nm³ of air is 0.296 kg O₂.

The selected fine bubble diffusers will dissolve 29% of this oxygen if operating in clean water at 20°C, 1 atm, and 0 mg/L dissolved oxygen (DO) concentration. This is called the *standard oxygen transfer efficiency* (SOTE) because it is obtained at the manufacturer's standard test conditions.

The SOTE can be converted to the *standard oxygen transfer rate* (SOTR) by

$$\text{SOTR} = \text{SOTE} (\text{O}_2 \text{ content of air})(\text{Air flow rate})$$

$$\text{SOTR} = 0.29(0.296 \text{ kg O}_2/\text{Nm}^3)(Q_{\text{air}}) = (0.08584 \text{ kg O}_2/\text{Nm}^3)(Q_{\text{air}})$$

The *actual oxygen transfer rate* (AOTR) is the transfer rate in wastewater at field conditions. It is less than the SOTR for a variety of reasons, such as the temperature not being 20°C, different air pressure, impurities in the wastewater, and the DO concentration being 1-2 mg/L (instead of zero). Suppose that adjusting for such factors gives an actual oxygen transfer rate (AOTR) that is one-third of the SOTR.

$$\begin{aligned}\text{AOTR} &= 0.33 (\text{SOTR}) = 1,600 \text{ kg O}_2/\text{d} \\ &= 0.33(0.08584 \text{ kg O}_2/\text{Nm}^3)Q_{\text{air}} = (0.02833 \text{ kg O}_2/\text{Nm}^3)(Q_{\text{air}})\end{aligned}$$

The required airflow rate is

$$Q_{\text{air}} = \frac{1,600 \text{ kg O}_2/\text{d}}{0.02833 \text{ kg O}_2/\text{Nm}^3} = 56,500 \text{ Nm}^3/\text{d} = 39.2 \text{ Nm}^3/\text{min}$$

EXAMPLE 10.3 ESTIMATING THE REQUIRED AIR FLOW (US UNITS)

A 1 mgd (million gallons per day) activated sludge process needs an oxygen supply of 3,150 lb O₂/d. The oxygen content of 1 scf of air is 0.0173 lb O₂.

The selected fine bubble diffusers will dissolve 29% of this oxygen if operating in clean water at 20°C, 1 atm, and 0 mg/L dissolved oxygen (DO) concentration. This is called the *standard oxygen transfer efficiency* (SOTE) because it is obtained at the manufacturer's standard test conditions.

The SOTE can be converted to the *standard oxygen transfer rate* (SOTR) by

$$\text{SOTR} = \text{SOTE} (\text{O}_2 \text{ content of air})(\text{Air flow rate})$$

$$\text{SOTR} = 0.29(0.0173 \text{ lb O}_2/\text{scf})(Q_{\text{air}}) = (0.00502 \text{ lb O}_2/\text{scf})(Q_{\text{air}})$$

The oxygen transfer rate wastewater at field conditions is less than the SOTR for a variety of reasons, such as the temperature not being 20°C, different air pressure, impurities in the wastewater, and the DO concentration being 1-2 mg/L (instead of zero). Making adjustments for such factors gives an *actual oxygen transfer rate* (AOTR) of

$$\text{AOTR} = 0.33 (\text{SOTR}) = 3,150 \text{ lb O}_2/\text{d}$$

$$\text{AOTR} = 0.33(0.00502 \text{ lb O}_2/\text{scf})(Q_{\text{air}}) = (0.00167 \text{ lb O}_2/\text{scf})(Q_{\text{air}})$$

The required airflow rate is

$$Q_{\text{air}} = \frac{O_2 \text{ required}}{\text{AOTR}} = \frac{3,150 \text{ lb } O_2/\text{d}}{0.00167 \text{ lb } O_2/\text{scf}} = 1,886,000 \text{ scfd} = 1,310 \text{ scfm}$$

The difference between SOTE and SOTR can be confusing. SOTE is the percent of oxygen in air that is transferred to water by the aeration device under standard conditions. SOTR is the mass transfer rate of oxygen to water by the aeration device at standard conditions. SOTE has units of percent. SOTR has units of mass O_2 transferred/time.

10.4 AIR DELIVERY PRESSURE

The blower power depends on the air delivery pressure. The components that determine the operating pressure and power of the air blower, shown in Figure 10.4, with typical values, are:

- 1) blower inlet filter and inlet manifold piping
- 2) header losses from blower to aeration tanks
- 3) drop-pipe and diffuser piping in basin
- 4) diffuser losses
- 5) static pressure water depth to diffuser
- 6) allowance for diffuser clogging
- 7) safety factor

Items 1, 2, 5 and 7 are the responsibility of the process designer. Items 3, 4, and 6 are the responsibility of the diffuser equipment supplier (Environmental Dynamics Int'l 2005, 20011, Siemens AG/Turblex, Inc. 2009, Mueller et al 2002).

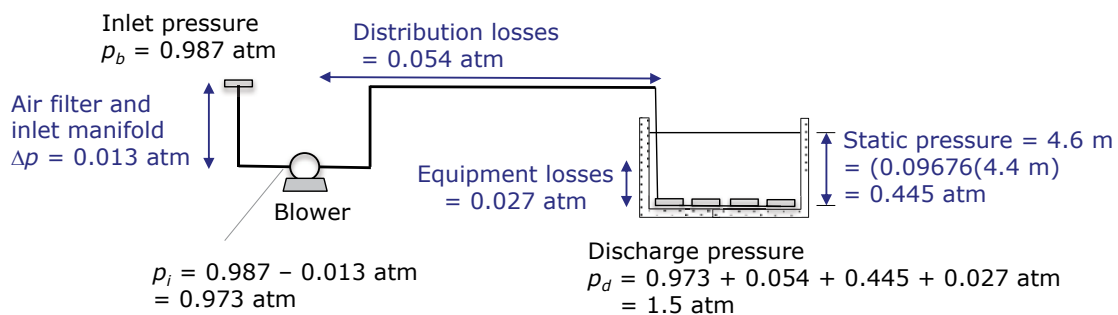


Figure 10.4 Pressures in a diffused air supply for an activated sludge process.

The static pressure (p_s) over the diffuser is proportional to the depth (d) of the water over the diffuser.

For depth in meters (m)

$$p_s \text{ (kg/m}^2\text{)} = 1000d$$

$$p_s \text{ (kP)} = 9.804d$$

$$p_s \text{ (atm)} = 0.09676d$$

$$p_s \text{ (bar)} = 0.09804 d$$

For depth in feet (ft)

$$p_s \text{ (lb/ft}^2\text{)} = 62.4d$$

$$p_s \text{ (lb/in}^2\text{)} = 0.433d$$

$$p_s \text{ (atm)} = 0.09676d$$

$$p_s \text{ (bar)} = 0.2989d$$

EXAMPLE 10.4 PRESSURE DROP IN AN ACTIVATED SLUDGE AERATION SYSTEM (SI UNITS)

An activated sludge process is supplied with oxygen by blowing air through diffusers into the aeration tank, as shown in Figure 10.4. The barometric pressure at the blower air filter inlet is 0.987 atm. The loss in the filter and the blower inlet manifold is 0.013 atm so the inlet pressure at the blower is $0.987 - 0.013 = 0.973$ atm.



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There is a pressure loss of 0.054 in the air distribution piping and a loss of 0.027 atm in the aeration tank downpipe and the fine bubble diffusers. The 4.6 m water depth above the diffusers causes a static pressure of 0.457 atm. This gives a diffuser discharge pressure of

$$p_d = 0.973 + 0.054 + 0.445 + 0.027 \text{ atm} = 1.500 \text{ atm}$$

The design pressure usually includes an allowance for diffuser clogging (0.035 atm) and perhaps a safety factor (0.035 atm), which would give a design discharge pressure of

$$\begin{aligned} p_{\text{design}} &= p_d + 0.035 \text{ atm} + 0.035 \text{ m} \\ &= 1.5 \text{ atm} + 0.07 \text{ atm} = 1.57 \text{ atm} \end{aligned}$$

EXAMPLE 10.5 PRESSURE DROP IN FINE BUBBLE DIFFUSER SYSTEM (US UNITS)

The barometric pressure at the blower air filter inlet, shown in Figure 10.5, is 14.5 psia. The loss in the filter and the blower inlet manifold is 0.2 psi so the inlet pressure at the blower is $14.5 - 0.2 = 14.3$ psia.

There is a pressure loss of 0.8 psi in the air distribution piping and a loss of 0.4 psi in the aeration tank downpipe and the fine bubble diffusers. The water depth of 15 ft causes a static pressure of 6.5 psi. This gives a diffuser discharge pressure of

$$p_d = 14.3 + 0.8 + 0.4 + 6.5 = 22 \text{ psia}$$

The design pressure is usually includes an allowance for diffuser clogging (0.5 psi) and perhaps a safety factor (0.5 psi), which would give a design discharge pressure of 23 psia.

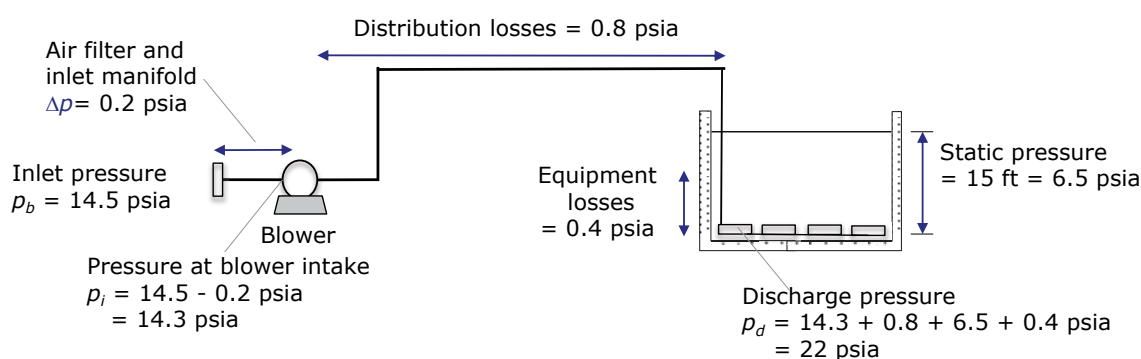


Figure 10.5 Schematic of blower piping, aeration tank, and fine bubble diffusers, with pressure losses for the system components.

10.5 BLOWER POWER

Blowers are rated in the *actual inlet airflow rate*, Q_{actual} ,

$$Q_{\text{actual}} = Q_{\text{STP}} \left(\frac{P_{\text{STP}}}{P_{\text{actual}}} \right) \left(\frac{T_{\text{actual}}}{T_{\text{STP}}} \right)$$

where P = absolute pressure (psia, kPa)

T = absolute temperature of inlet air (K, °R)

The subscripts STP and *actual* indicate standard conditions and actual inlet conditions. The conversion for flow rates is 1 scfm = 0.622 Nm³/h

The power the blower imparts to the air is

$$P = m_{\text{air}} c_{P, \text{air}} \Delta T = \rho_{\text{air}} Q_{\text{air}} c_{P, \text{air}} \Delta T$$

where m_{air} = mass flow rate of air at the inlet (kg/s, lb/s)

Q_{air} = actual volume flow rate of air at the inlet (m³/s, ft³/s)

ρ_{air} = density of inlet air (kg/m³, lb/ft³, atm)

$c_{P, \text{air}}$ = specific heat of air = 1.006 kJ/kg K (0.24 Btu/lb°R)

T_1 = inlet temperature (K, °R)

T_2 = outlet temperature (K, °R)

$T_2 - T_1$ = temperature increase across the blower (K, °R)

$c_{P, \text{air}} \Delta T$ = change in enthalpy across the blower, (kJ/kg, Btu/lb)

The outlet temperature is a function of the inlet temperature and the inlet and outlet pressures, p_1 and p_2 , respectively

$$T_2 = T_1 \left(\frac{p_2}{p_1} \right)^{0.286}$$

$$\Delta T = T_2 - T_1 = T_1 \left[\left(\frac{p_2}{p_1} \right)^{0.286} - 1 \right]$$

Combining these equations gives

$$P = m_{\text{air}} c_{P, \text{air}} T_1 \left[\left(\frac{p_2}{p_1} \right)^{0.286} - 1 \right] = \rho_{\text{air}} Q_{\text{air}} c_{P, \text{air}} T_1 \left[\left(\frac{p_2}{p_1} \right)^{0.286} - 1 \right]$$

These are absolute temperatures: K = 273 + °C, °R = 460 + °F. The pressures are absolute pressures (i.e. gauge pressure plus atmospheric pressure). The exponent on the pressure ratio is a function of the specific heat ratio (c_p/c_v) of the gas, and for dry air ranges from 0.287 at 0 °C to 0.282 at 400°C. We use a value of 0.286 which is representative for 0 – 100 °C temperatures.

Efficiencies for different types of blowers are given in Table 10.1. Wire-to-air power for the blower is the power input minus compressor losses, cooling losses, and electrical losses. For the blower system the specification also includes the motor, starter, variable speed drive, filters, gears and belts. High speed “Turbo” and other prepackaged blowers combine the motor and compressor into one unit that comes prepackaged with drive, controller and cooler. The only way to evaluate this technology’s efficiency is through a wire-to-air approach. Traditional efficiency takes into account the shaft brake horsepower, which is the power to rotate the compressor, excluding the motor and all other drive losses. As a result, comparing efficiencies of different blower technologies requires special knowledge and care.

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Blower Type	Nominal Efficiency (%)	Nominal Turndown (%)
Positive displacement	45-65	50
Multi-stage centrifugal (inlet throttled)	50-70	60
Multi-stage centrifugal (variable speed)	60-70	50
Single-stage centrifugal (integrally geared)	70-80	45
Single-stage high speed turbo	70-82	50

Table 10.1 Efficiency and turndown for types of blowers**EXAMPLE 10.6 BLOWER OUTLET TEMPERATURE**

A blower operates at atmospheric (barometric) pressure $p_b = 14.6$ psia with a pressure drop $\Delta p = 0.2$ psig through the inlet filter and inlet manifold. This makes the blower inlet pressure $p_1 = 14.6 - 0.2 = 14.4$ psia. The inlet temperature $T_1 = 50^\circ\text{F} = 510^\circ\text{R}$. The outlet pressure is 8.0 psig, which gives $p_2 = 14.4 + 8.0 = 22.4$ psia.

The outlet temperature T_2 is

$$T_2 = T_1 \left(\frac{p_2}{p_1} \right)^{0.286} = (510^\circ\text{R}) \left(\frac{22.4 \text{ psia}}{14.4 \text{ psia}} \right)^{0.286} = 578.7^\circ\text{R} = 118.7^\circ\text{F}$$

$$\Delta T = T_2 - T_1 = 118.7^\circ\text{F} - 50^\circ\text{F} = 68.7^\circ\text{F}$$

EXAMPLE 10.7 BLOWER POWER

The air requirement is 14,000 scfm and the air density is 0.075 lb/ft³. The inlet conditions are $p_1 = 14.2$ psia and $T_1 = 55^\circ\text{F} = 515^\circ\text{R}$. The outlet pressure is 14.2 psig + 8.5 psig = 22.7 psia. The specific heat for air is 0.24 Btu/lb $^\circ\text{F}$ (0.24 Btu/lb $^\circ\text{R}$).

The blower power is

$$\begin{aligned}
 P &= \rho_{\text{air}} Q_{\text{air}} c_{p, \text{air}} T_1 \left[\left(\frac{p_2}{p_1} \right)^{0.286} - 1 \right] \\
 &= \left(0.075 \frac{\text{lb}}{\text{ft}^3} \right) \left(14,000 \frac{\text{ft}^3}{\text{min}} \right) \left(0.24 \frac{\text{Btu}}{\text{lb}^\circ\text{R}} \right) (515^\circ\text{R}) \left[\left(\frac{22.7 \text{ psia}}{14.2 \text{ psia}} \right)^{0.286} - 1 \right] \\
 &= 18,630 \text{ Btu/min} = 1,118,000 \text{ Btu/h}
 \end{aligned}$$

$$P = (1,118,000 \text{ Btu/h})(1 \text{ hp}/2,544 \text{ Btu/h}) = 440 \text{ hp}$$

$$P = (1,118,000 \text{ Btu/h})(2.931 \times 10^{-4} \text{ kW/Btu/h}) = 328 \text{ kW}$$

$$P = (1,118,000 \text{ Btu/h})(1 \text{ kJ}/0.9478 \text{ Btu}) = 1,180,000 \text{ kJ/h} = 1,180 \text{ MJ/h}$$

Blower efficiency = 70%

$$\text{Input power} = 328 \text{ kW}/0.7 = 469 \text{ kW} (629 \text{ hp}; 1,686 \text{ MJ/h})$$

10.6 VARIATIONS IN AIR FLOW RATES

The hydraulic elements of treatment plants are designed to meet the peak load projected twenty years into the future. The result is that treatment plants usually operate at loads well below their peak capacity, perhaps as low as one third of design loads.

One difficulty in specifying aeration blower flow is that the process does not operate at a constant load. The most important loading variation for most plants is the diurnal (daily) fluctuation in load associated with normal human activity. As Figure 10.6 illustrates, the process load and required air flow rate shows a 2:1 range between early-morning low and daytime peak. Wastewater temperature varies seasonally, affecting microorganism metabolism, their demand for oxygen, and the oxygen transfer rate. Treatment plants served by combined sewers (domestic wastewater plus stormwater) will have more variation. Rain and snow melt can dramatically change wastewater flow, temperature, and organic loading (combined BOD and NH_3).

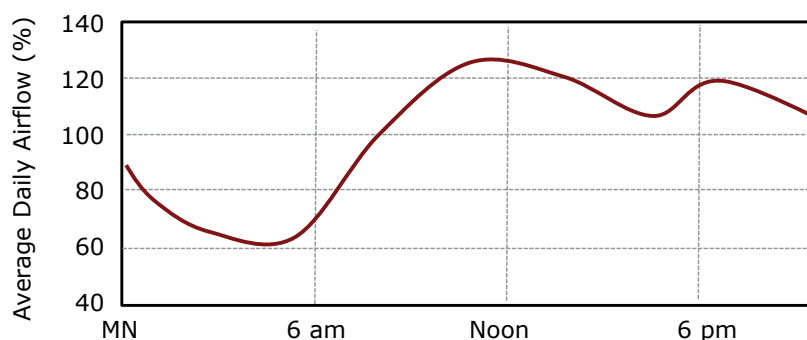


Figure 10.6 Diurnal variation in the need for air supplied to a typical municipal activated sludge process.

Blowers are essentially volumetric machines. Because the air density is lower in summer than in winter, the volumetric flow rate in summer must be higher than in winter for identical mass flow rates.

The consequence of these variations is that the blower system never operates at one specified design point. Another result is that turndown is a critical parameter in optimizing process performance and providing satisfactory blower system operation. A minimum system turndown of 80% is needed in most plants to satisfy process requirements. This requires a minimum of two blowers because most individual blowers only provide 50% turndown.

Treatment plants must have standby equipment to insure operation during failure or severe overloads of blowers, which necessitates at least two blowers sized to provide 100% of design flow. Arrangements with three or more blowers will provide greater operational flexibility. Alternate designs include three blowers sized at 50% of design flow, four blowers at 33% of design flow, and two blowers at 50% plus two blowers at 25% design flow. The impact of equipment cost for alternate arrangements is not as great as the impact of power requirements on life cycle cost.

The need for power comparisons doesn't end when equipment design is finalized. Multiple suppliers are usually competing for each project. Equipment cost is obviously a consideration, but not necessarily the most significant one. In the final selection of equipment is advisable to include an evaluation of energy in equipment selection to insure that higher energy expense doesn't negate lower first cost.

An advertisement for SKF. It features a woman with long dark hair smiling in the foreground, with a wind turbine in the background against a blue sky. The text 'Brain power' is written in large white letters. To the right, there is a block of text about wind energy and SKF's role. At the bottom left, there is a call to action to visit the SKF knowledge website. The SKF logo is in the bottom right corner.

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EXAMPLE 10.8 DIURNAL AIR FLOW VARIATION

The organic load on an activated sludge process changes continually between an early morning minimum with peaks near mid-day and early evening, as given in Table 10.2. The load is at 100% of the daily average from 0800 - 1000 hours and the average daily airflow rate for this time interval is 14,000 scfm. The discharge pressure is based on a blower inlet pressure of 14.2 psia. The inlet temperature $T_{in} = 55^{\circ}\text{F} = 515^{\circ}\text{R}$.

Sample calculations for hours 0000 – 0500:

Inlet pressure = 14.2 psia

Discharge pressure (psia) = 14.2 psia + p_g
 $= 14.2 \text{ psia} + 8.2 \text{ psig} = 22.4 \text{ psia}$

Adiabatic factor = $X_a = [(p_{Dis}/p_{In})^{0.286} - 1]$
 $= [(22.4 \text{ psia}/14.2 \text{ psia})^{0.286} - 1.0] = 0.1392$

Power (Btu/min) $P = Q_{air} \rho_{air} c_{p, air} T_{in} X_a$
 $= (9,800 \text{ scfm})(0.075 \text{ lb/ft}^3)(0.24 \text{ Btu/lb}^{\circ}\text{R})(515^{\circ}\text{R})(0.1392)$
 $= 12,650 \text{ Btu/min}$

Power (hp) = Power (Btu/min)(60 min/h)/2,544 Btu/hp
 $= (12,650 \text{ Btu/min})(60 \text{ min/h})/(2,544 \text{ Btu/hp}) = 298 \text{ hp}$

Prorated Power (hp) = (% of time/100)[Power (hp)] = (0.208)(298 hp) = 62.0 hp

Hour of day	% of time	% of avg. daily flow	Inlet airflow rate, Q (scfm)	Discharge pressure p_g (psig) (psia)		Adiabatic factor X_a	Power P (hp)	Prorated Power (hp)
0000-0500	20.8	70	9,800	8.2	22.4	0.1392	298	62
0500-0800	12.5	90	12,800	8.4	22.6	0.1421	398	50
0800-1000	8.3	100	14,000	8.5	22.7	0.1436	440	37
1000-1800	33.3	110	15,400	8.6	22.8	0.1450	488	163
1800-2400	25.0	120	16,800	8.9	23.1	0.1493	548	137
Totals	100							448

Table 10.2 Blower operation for a typical day with flow rates, discharge pressures, and power requirements.

Multiple blowers can be installed to match the demand, or variable speed drive can be used. The installed blower capacity should be sufficient to meet the full load when one blower is out of service. The aeration system is usually designed to supply 200% of the average demand for air.

Instruments and closed loop controllers can be installed so the process is given only the air it needs at any time. Controlling the process dissolved oxygen concentration the 1-2 mg/L range that is sufficient for a healthy process, instead of 4-5 mg/L, corresponds to a 35-70% reduction in airflow.

10.7 CONCLUSION

The aeration system for an activated sludge process is perhaps two percent of the total construction cost, but the energy used for aeration will be 60% or more of the total energy use in conventional wastewater treatment. The design of air blowers is in many ways similar to the design of liquid pumps. The main difference is that air is compressible and this comes into the calculation of the power requirements.

11 THE ECONOMICS OF ENERGY MANAGEMENT

Every piece of equipment has an initial cost (for purchase and installation) and a cost for operation and maintenance (O&M). The initial cost is a lump sum; O&M costs repeat month after month, and year after year.

Alternative equipment can be compared after the two kinds of costs are combined into a fair representation of the total cost. There are two methods for doing this.

- Calculate an annual cost by converting the initial cost into a series of equal annual payments that are added to the annual O&M costs. Annual revenue is subtracted from the costs. This only works when the O&M costs and revenue are equal from year to year, or are inflating in a simple way. Special maintenance costs that occur every few years, such as replacing catalyst in an incinerator or cleaning aeration diffusers, are less easy to handle by this method



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- Convert the annual O&M costs, revenues, and special maintenance cost back to an equivalent lump sum at the start of the project. The process of doing this is called discounting and the sum of the discounted values is the *present value*. This is the easiest way to handle unequal and intermittent costs and revenues.

Pumping and air supply for activated sludge wastewater treatment are a large part of O&M costs. Over a short time, their O&M costs will exceed the initial purchase price, so it is worth some time to learn how to make these evaluations.

11.1 ENERGY USE IN WATER AND WASTEWATER TREATMENT

Rich targets for conserving energy are:

- Efficient design and operation of pumping facilities.
- Efficient design and operation of blowers for aeration of wastewater treatment plants and other processes that need air for ventilation or performance (such as incinerators)
- Water conservation to reduce the volume of water and wastewater pumped and treated. This has been successful in many large cities. Figure 11.1 shows that the demand for water in Seattle, WA, has been static even though the population served has grown. Construction for expansion has been deferred and there have been savings in operating and maintenance costs.

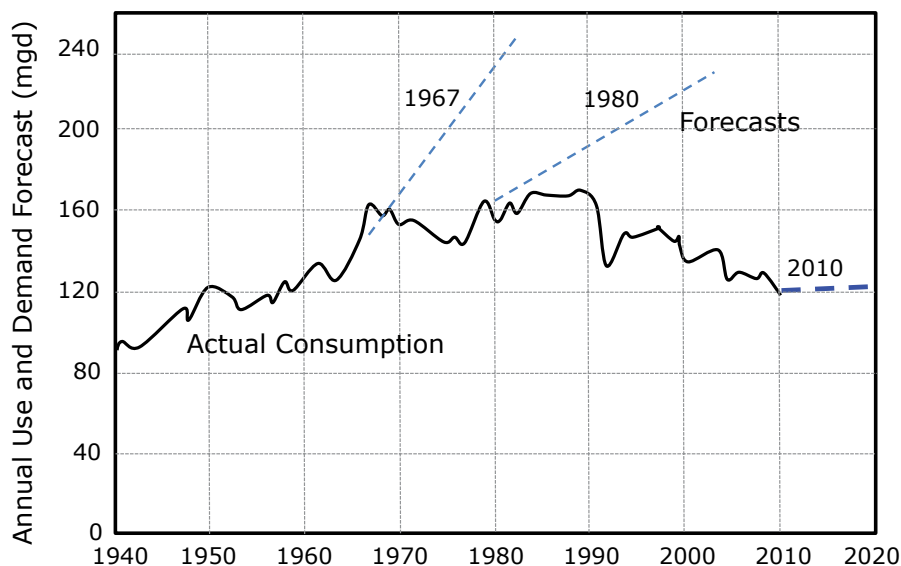


Figure 11.1 Water conservation has, in Seattle, WA, reduced per capita water demand while the population served has grown

There are 44,200 public water supply utilities in the U.S.; 39,600 (77%) are groundwater systems. Most of the population is using surface water (67%) to deliver 29.6 billion gallons per day (bgd) to 212,500,000 people; 14.6 bgd of groundwater is delivered to 86,600,000 people (Pabi et al. 2013).

There are 15,600 publicly-owned wastewater treatment plants serving 284 million people. The majority (79%) are less than one million gallons per day 18% are between 1 and 10 mgd, 3% are between 10 and 100 mgd, and 38 plants are larger than 100 mgd. The total existing flow is 32,345 mgd and the present design capacity is 44,888 mgd.

Table 11.1 and Table 11.2 give some typical values for energy use (kWh/d) in water and wastewater treatment unit processes. Some values give an unwarranted impression of precision, but these are the values reported by Pabi et al (2013). His work was intended to support planning for energy production and distribution; the data were not to be used for designing individual treatment plants.

The total energy used by any single plant will depend on the treatment technology being used. Obviously the values for any single plant could be much different than these average values, and some of the values will not apply, for example, a water treatment plant that uses ground water will not have rapid mixing, flocculation and sedimentation.

The estimated pumping costs are directly proportional to the volume of water treated. Despite this it might be expected that larger plants will find efficiencies that are not available to small plants. Chemical feed systems are independent of plant capacity.

Unit Process	Plant Production (mgd)			
	1	5	10	50
Raw surface water pumping	145	725	1,450	7,250
Raw ground water pumping	920	4,600	9,225	N/A
Finished Water Pumping	1,040	5,325	11,040	48,775
Filter backwash water pumping	15	60	125	660
Microfiltration (in lieu of sedimentation)	100	500	1,000	5,000
Ultrafiltration (contaminant removal)	800	4,000	8,000	40,000
Reverse Osmosis (brackish water)	6,000	29,800	59,500	226,600
Reverse Osmosis (ocean water)	12,000	60,000	120,000	600,000
Rapid mixing	40	175	310	1,540
Flocculation	10	50	90	450
Sedimentation	15	45	90	440
Chemical feed systems	65	65	65	65
Onsite chlorine generation for disinfection	85	420	830	4,160
Ozone disinfection	140	560	1,125	3,840
UV disinfection	62	310	625	3,120
Non-process loads (HVAC, lighting, etc.)	300	1,200	2,100	9,000

Table 11.1 Estimates of the average electric energy use (kwh/d) of public water supply unit processes (Pabi et al. 2013)

Unit Process	Average Plant Flow (mgd)			
	1	5	10	50
Wastewater pumping	220	1,100	2,200	11,000
Primary Treatment				
Odor control	150	600	1,550	12,000
Grit removal, aerated	130	160	280	800
Grit removal, forced vortex	160	200	220	430
Primary clarifiers	30	140	300	1,550
Secondary Treatment				
Trickling filters	630	2,540	5,070	25,360
Aeration without nitrification	720	3,600	7,200	33,480
Aeration with nitrification (a)	1,080	5,400	10,800	50,200
Secondary clarifiers	85	350	700	3,525
Sequencing batch reactors	1,090	5,450	10,900	50,800
Membrane bioreactors	2,700	13,530	27,060	135,300
UV disinfection	225	1,170	2,340	11,700
Sand filtration	100	350	580	2,900
Solids Handling, Treatment & Disposal				
Aerobic digestion	1,000	5,000	10,000	N/A
Anaerobic digestion	N/A	550	1,100	5,000
Gravity belt thickener	30	140	240	1,200
Dissolved air flotation	N/A	N/A	1,805	6,260
Centrifuge thickening	80	290	390	1,950
Belt filter press	230	460	690	2,550
Vacuum filtration	100	350	580	2,900
Centrifuge dewatering	260	1,300	2,610	13,040
Thermal drying	221	1,105	2,210	N/A
Disinfection				
UV disinfection	225	1,170	2,340	11,700

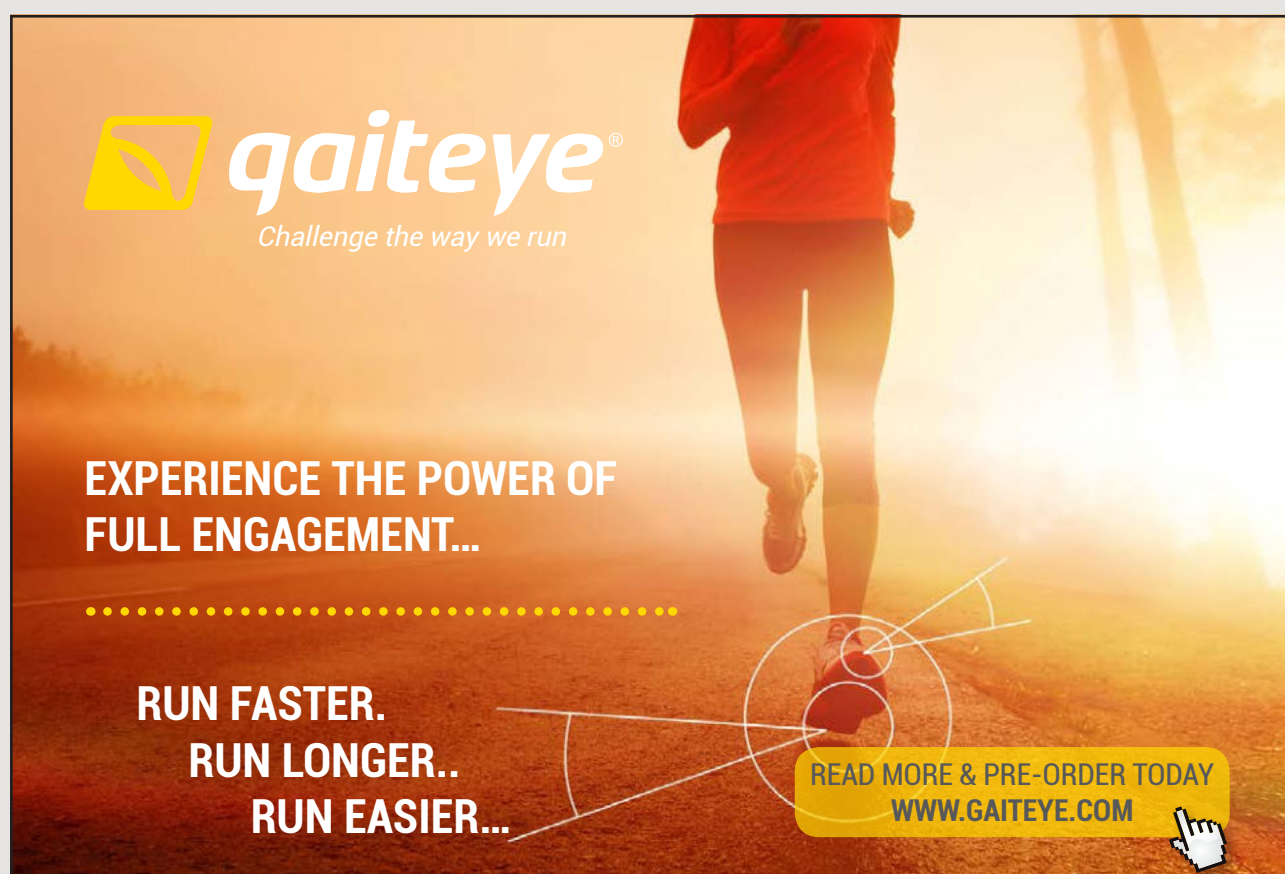
Non-Process Loads				
Pumping plant utility water	45	220	420	1,990
Non-process loads (buildings, etc.)	300	1,200	2,100	9,000
Energy recovery (biogas) (c)	N/A	(1,440)	(2,880)	(14,400)

a) Nitrification is the microbial conversion of ammonia to nitrate, which consumes 4.57 kg O₂ per kg ammonia oxidized.

b) N/A=not applicable; generally pertains to unit processes not commonly found in plants of given flow (e.g., sequencing batch reactors are not used in large plants with average flows in excess of 50 mgd. Small plants often use aerobic rather than anaerobic digestion.

c) Energy recovery values in parentheses are reductions in energy use; values are based on assumption of using conventional internal combustion engine burning biogas after treatment; alternative generation technologies may improve.

Table 11.2 Estimates of average electric energy use (kwh/d) in wastewater treatment unit processes (Pabi, et al. 2013)



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11.2 COSTS ESTIMATES

Capital costs are for tangible assets. Direct capital costs include equipment, tanks, piping, land, and site improvements. Indirect capital costs are for assembling the components into a functioning system. These include engineering, construction, and contingencies.

Operating costs pay for labor, chemicals, fuel and electric power, and all regular recurring costs. They are sometimes referred to as repeating costs in contrast to the capital cost (or first cost). The operating costs keep the enterprise, project, operation, plant, and equipment running and producing.

Operating costs can be divided into direct costs, indirect costs, and general expenses.

- *Direct operating costs* tend to be proportional to output and raw material use. These are sometimes called *variable costs*. Electricity and fuel are direct operating costs. Others are raw materials, chemicals, utilities, O&M supplies, and waste disposal.
- *Indirect costs*, sometimes called *fixed costs*, include property taxes, insurance, and depreciation. These tend to be independent of output and have to be paid even if the plant is not operating. Engineering and administration are in this category.
- *General expenses* are the overhead burden that is necessary to carry out the functions of the organization. They include management, sales, financing, and research.

Electricity and fuel are a major part of operating costs. The lifetime operating cost for pumps, blowers, heaters, and chillers will exceed the purchase price many times over.

11.3 CAPITAL COST ESTIMATES

Factor estimates start with cost estimates for the major equipment and processes, designated as E in Table 11.3. Other costs are multiples of E.

These are order-of-magnitude estimates that are helpful during a project's early development and budgeting. The accuracy is $\pm 30\%$.

Cost Component	Factor	Cost
Direct costs		
Major processes and equipment	1.0 E	\$8,400,000
Purchased equipment installation	0.3 E	\$2,520,000
Instrumentation and controls	0.1 E	\$840,000
Piping (installed)	0.3 E	\$2,520,000
Electrical (installed)	0.1 E	\$840,000
Buildings / yard improvements	0.2 E	\$1,680,000
Service facilities (installed)	0.4 E	\$3,360,000
Total direct cost	2.4 E	\$20,160,000
Indirect costs		
Engineering	0.3 E	\$2,520,000
Construction expenses	0.3 E	\$2,520,000
Total indirect costs	0.6 E	\$5,040,000
Total direct and indirect costs	3.0 E	\$25,200,000
Contractor's fee; 5%(direct + indirect)	$0.05(3.0 \text{ E}) = 0.15 \text{ E}$	\$1,260,000
Contingency; 10% (direct + indirect)	$0.1(3.0 \text{ E}) = 0.30 \text{ E}$	\$2,520,000
Fixed capital investment	3.45 E	\$29,980,000

Table 11.3 Factor estimates for waste treatment plants. Cost of land and working capital not included. E = capital cost of major processes and equipment.

This example is not a template for all projects. A solid waste handling facility will have different factor weights than a wastewater treatment plant, and so will an air pollution control facility. The purpose here is to give an idea of what goes into a capital cost estimate.

11.4 ECONOMY OF SCALE

Economy of scale is the characteristic that the unit cost of added capacity decreases as the total capacity increases. This applies to all pollution control processes and equipment. It means that doubling the size of a treatment system, process, or a machine, does not double the cost. This is easy to understand in the context of pipeline or sewer construction.

EXAMPLE 11.1 ECONOMY OF SCALE OF A PIPE INSTALLATION

The hydraulic capacity of a 24-inch pipe is almost double that of an 18-inch pipe (the ratio of the areas is 1.78). The *purchase price* of flared-end reinforced concrete pipe, in six-foot lengths, is \$385 for 18-inch pipe and \$460 for 24-in pipe. A large part of the *installed cost* to build a pipeline or sewer is excavation, sheeting, moving underground utilities, backfilling, repaving, and landscaping. There is no standard price for installed pipe, which will depend on soil type, depth of trench, pavement removal and replacement, landscaping, etc., but these costs will be almost the same for 24-inch or 18-inch pipes, and will exceed the cost of the pipe. The unit price (\$/length) increases with pipe diameter, but the installed cost will differ by a relatively small amount. Added capacity is purchased at a small incremental price. This is economy of scale.

All pollution control process and systems have a cost estimating function of the form

$$\text{Cost} = K(\text{Capacity})^M \text{ or } C = KQ^M$$

where C is cost (\$), Q is capacity, and M is the economy-of-scale exponent. Capacity may be defined in terms of air or water flow rate, surface area of a settling tank or heat exchanger, volume of a reactor, power of a blower or pump, or mass of solids handled. The value of K is specific to the type of equipment or process, and it depends on the units of Q . The dimensionless exponent M depends on the type of equipment.

The exponent M has values between 0.6 and 0.8 for many kinds of processes and equipment. Small values of M give more economy of scale. For $M = 0.6$, one unit of capacity costs \$10,000, two units costs \$15,157, and four units costs \$22,974. At $M = 0.5$, the cost doubles when the capacity is raised from one to four units, at $M = 0.8$ four units costs three times more than one unit. A few values of M are given in Table 11.4. Figure 11.2 is the cost capacity curve for a process with $M = 0.6$. M is the slope of the line on the log-log plot. K is the cost for $Q = 1.0$.

Treatment Process	Units of Capacity	M
Wastewater Treatment Processes		
Activated sludge, aeration basin	volume	0.50 – 0.70
Primary sedimentation basin	area	0.60 – 0.76
Final clarifier	area	0.57 – 0.76
Sludge Handling Processes		
Digestion (gas mixing)	volume	0.34
Vacuum filter	filter area	0.34
Centrifuge	lb dry solids/h	0.81
Wastewater Process Equipment		
Primary clarifier equipment	area	0.32
Final clarifier internal equipment	area	0.32
Surface aerators (small units)	total hp	0.96
Surface aerators (large units)	total hp	0.72
Centrifuge, solid bowl	hp	0.73
Pressure filter, plate and frame	area	0.58
Vacuum filter, rotary drum	area	0.68
Heat exchangers	area	0.58
Air compressor (125 psig)	flow (cfm)	0.28
Electric generators	kV-amp	0.71
Motors, electric, 10-100 hp	hp	0.80
Pumps, centrifugal (with motor)	(flow)(pressure)	0.51
Air pollution control equipment		
Dust collector, cyclone	air flow	0.8
Dust collector, cloth filter	air flow	0.68
Dust collector, electrostatic precipitator	air flow	0.75
Scrubbers	air flow	0.85-0.95
Incinerators for combustible gases	air flow	0.40

Table 11.4 Economy-of-scale factors

EXAMPLE 11.2 CALCULATING K AND M BY LINEAR REGRESSION

A set of cost-capacity data for similar projects is plotted in the left panel of Figure 11.2. The fitted model

$$C (\$1000) = 2.2Q^{0.55}$$

was obtained by fitting the log-transformed data shown on the right side of the diagram. Taking logarithms of the estimating model gives a linear equation

$$\log_{10}(C) = a + b \log_{10}(Q)$$

or $y = a + bx$

where $y = \log_{10}(C)$ and $x = \log_{10}(Q)$

K and M can be estimated by linear regression to find

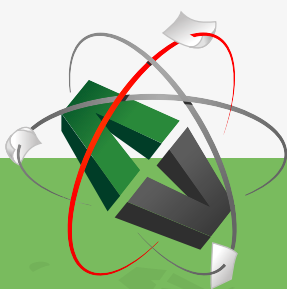
$$\log_{10}(C) = 0.5513 \log_{10}(Q) + 0.3431$$

$$M = 0.5513 \text{ and } K = 10^{0.3431} = 2.203$$

These are rounded to $M = 0.55$ and $K = 10^{0.3431} = 2.2$ giving

$$C (\$1000) = 2.2Q^{0.55}$$

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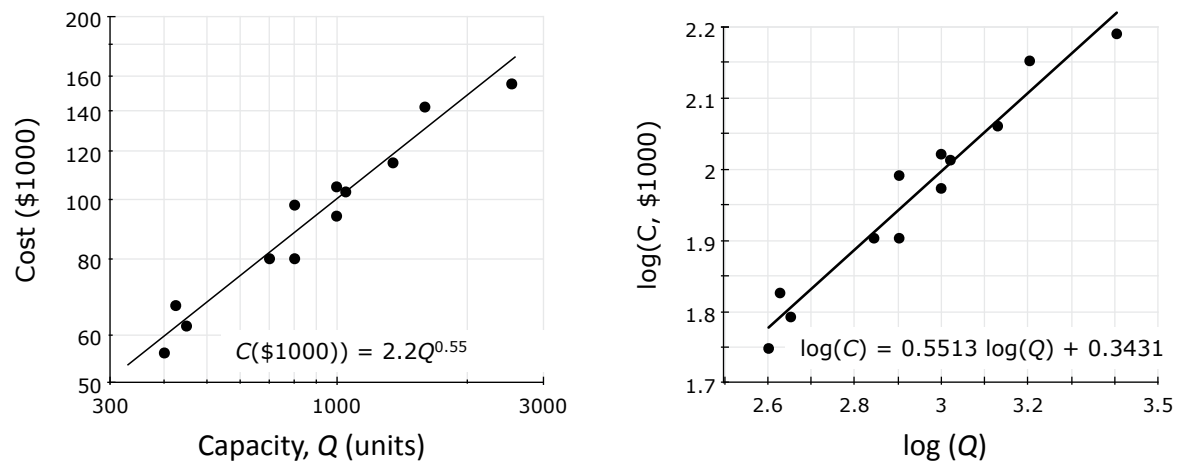


Figure 11.2 Estimating M and K by linear regression on the log-transformed costs and capacities.

Table 11.5 shows how the cost increases as the capacity increases. Doubling the capacity, from one to two units, increases the cost by 46%. Installing four units of capacity slightly more than doubles the cost.

Installed Capacity (units)	Cost for 1 machine (\$)	Cost Increase (%)
1	2200	--
2	3221	46
3	4026	83
4	4716	114
5	5332	142

Table 11.5 Cost increase as the installed capacity increases.

EXAMPLE 11.3 ECONOMY OF SCALE OF MULTIPLE UNITS

The required capacity is 500 units. System reliability is increased by installing standby equipment. Economy of scale should influence the number of machines installed. Using $M = 0.65$, and $K = \$2,200$:

The cost for one machine is

$$C = \$2,200 Q^{0.65} = \$2,200(500)^{0.65} = \$124,955$$

Installing two 500 unit machines, one as a standby, costs $2(\$124,955) = \$249,910$. That is a cost increase of 100%.

Suppose that the design is for 3 machines, each with 250 units of capacity. Two machines are used in normal operation, and the third is a standby. The cost per machine is

$$C = \$2,200(250)^{0.65} = \$79,631$$

The cost for a reliable system with three machines is \$238,894. This is less than the cost for two larger machines.

Note: If $M = 0.55$, as in Example 11.2, two 500 unit machines cost slightly less than three 250 unit machines.

11.5 PLANNING COSTS FOR PUMPS

Pumping costs are a good target for reduction. The electrical cost calculation is straight forward. It is often balanced against the capital cost of the pumping equipment. Various equations have been proposed for planning costs estimates of pumps and pump stations. *Planning* means before all details are known and before vendors are asked for bid prices. A reasonable and flexible model (Walski 2012) has the form

$$C(\$) = KQ^{M_Q}H^{M_H}$$

where Q = flow rate and H = design operating head. The coefficient K has units of dollars and it depends on the units of Q and H . It also depends of the kind of pump (variable speed vs. constant speed), material (type of steel), auxiliaries that might be included in the price (electrical gear, controls, etc.), application (water vs. wastewater pumping), and local conditions. Economy-of-scale is indicated by M_Q and M_H ; generally $M_H < M_Q < 1$.

Walski proposes

$$C(\$) = KQ^{0.85}H^{0.32}$$

Other forms that have been used for pumping stations are

$$C = aQ^b \text{ and } C = aP^c,$$

where P is power in kW or hp (Newton and Sanks 1989). Since $P = QH$ the second form would be equivalent to the three parameter model of Walski if the exponents on Q and H are equal. That is

$$C = aP^c = a(QH)^c = aQ^cH^c$$

11.6 COST OF ELECTRICITY


The cost to supply electricity actually changes minute by minute because of variations in electricity demand, availability of generation sources, fuel costs, and power plant availability. Prices are usually highest in the summer when total demand is high because more expensive generation sources are added to meet the increased demand.

The average United States residential cost for electricity in May 2018 was \$0.1315 per kWh. In 2015, the average price was \$0.1042/kWh. The average household energy consumption was about 900 kWh per year.

The average residential cost for electricity by state is given in Table 11.6. Hawaii and Alaska have the highest average retail electricity prices, \$0.32/kWh and \$0.22/kWh. (These values vary slightly from reporting agency to agency.)

Residential rates are higher than the other categories, by as much as 50% (in Nevada). Annual average prices (in 2016) by major types of utility customers were:

Residential	12.55¢ per kWh	Commercial	10.37¢ per kWh
Industrial	6.75¢ per kWh	Transportation	9.48¢ per kWh



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Prices are highest for residential and commercial consumers because it costs more to distribute electricity to them. Industrial consumers use more electricity and can receive it at higher voltages, so supplying electricity to these customers is more efficient and less expensive. The price of power to industrial customers is generally close to the wholesale price of electricity.

State	Price (¢/kWh)	State	Price (¢/kWh)	State	Price (¢/kWh)
Louisiana	9.53	Colorado	11.96	Washington, DC	13.72
Washington	9.7	Wyoming	11.99	Delaware	13.74
Arkansas	9.89	Nevada	12.05	Minnesota	13.77
Oklahoma	10.09	Mississippi	12.07	Pennsylvania	14.24
Idaho	10.55	South Dakota	12.1	Wisconsin	15.21
Kentucky	10.66	New Mexico	12.14	New Jersey	15.87
Utah	10.79	Virginia	12.28	Michigan	15.93
Tennessee	10.87	Indiana	12.3	Maine	16.27
Oregon	11.2	Missouri	12.45	Vermont	18.35
North Dakota	11.25	Alabama	12.49	New York	18.52
North Carolina	11.41	Ohio	12.81	California	18.91
Montana	11.43	zKansas	12.85	New Hampshire	19.76
Nebraska	11.43	South Carolina	12.85	Rhode Island	20.03
Florida	11.54	Illinois	13.21	Connecticut	21.54
Georgia	11.62	Iowa	13.28	Massachusetts	21.69
Texas	11.68	Arizona	13.3	Alaska	22.16
West Virginia	11.78	Maryland	13.33	Hawaii	32.03

Table 11.6 Average residential electricity prices (¢/kWh) by state for 2018 (Source: ChooseEnergy.com)

11.7 PEAK ENERGY DEMAND AND PEAK CHARGES

The cost of electricity includes an *energy charge* and a *demand charge*. The energy charge is based on the number of kilowatt hours (kWh) used during the billing cycle. The total kilowatt hours are multiplied by the energy price (\$/kWh) for the total energy billing. The energy charges can vary with the type of service, voltage, and energy consumption.

The *demand charge* compensates the electric utility for the capital investment required to serve peak loads, even if that peak load is only used for a few hours per week or month. The demand is measured in kilowatts (kW) or kilovolt amperes (kVA). These units are directly related to the amount of energy consumed in a given time interval of the billing period. The demand periods vary with the type of energy demand over a short demand period, generally 15 or 30 minutes. The period with the highest demand is the one used for billing demand charges.

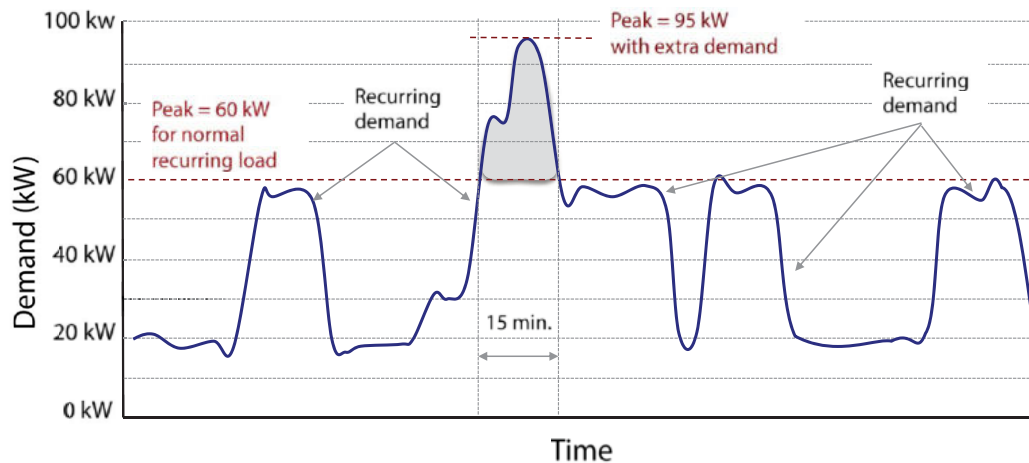


Figure 11.3 Electricity consumption and peak demand

Demand charges can be a significant portion of the total electric bill; in some cases, demand charges can amount to as much as 80 percent of the bill. The demand charge can be reduced by smoothing out the peaks in energy demand by rescheduling of work or through a demand control program to shed loads when a demand limit is approached.

EXAMPLE 11.4 DEMAND CHARGE

Suppose that the power demand for a plant is 7000 kilowatts for 15 days, followed by a demand of 140,000 kW for a 30 minute period, and then the demand drops back to 7000 kilowatts for the remainder of the billing period (one month). The *billing demand* for that month is then 140,000 kW, which is maximum energy demanded from the utility company for the month.

EXAMPLE 11.5 ON-PEAK AND OFF-PEAK CHARGES

Electric companies have on-peak and off-peak rate for certain times of the day and days of the week. Two companies are charged by time-of-use (i.e. on-peak and off-peak time of day) and by peak demand. The total electric consumption (kWh) by Company B was less than Company A, but Company B's total charges were higher because of a higher on-peak demand (kW) was much higher. This is shown in Table 11.7.

Energy Consumption	Company A 17,000 kWh	Company B 15,000 kWh
On-Peak Energy Charge	9,000 kWh x \$0.12 = \$1,080	8,000 kWh x \$0.12 = \$960
Off-Peak Energy Charge	8,000 kWh x \$0.09 = \$ 720	7,000 kWh x \$0.09 = \$630
On-Peak Demand	50 kW	100 kW
Demand Charge	50kW x \$10 = \$500	100 kW x \$10 = \$1,000
TOTAL CHARGES	\$2,300	\$2,590

Table 11.7 On-peak and off-peak demand charges



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11.8 SAVING BY SHEDDING PEAK LOADS

Electricity use can be assigned to two categories:

- *Essential loads* are essential to maintain production or safety. Unscheduled shutdowns on these loads cannot be tolerated.
- *Nonessential or sheddable loads* can be shut down temporarily without significantly affecting operations or worker comfort.

There are three general ways a facility can reduce its use of grid electricity during peak periods:

- *Load shedding* strategies include dimming or turning off lights, changing HVAC temperature set-points, and turning off non-critical equipment.
- *Load shifting* involves shifting use of equipment from on-peak to off-peak time periods (e.g., using off-peak power to pump water into storage tanks).
- *Switching to onsite generation*: Some facilities prefer meeting a portion of their on-peak loads with an onsite power generator (biogas or solar, for example) rather than shedding or shifting of loads.

EXAMPLE 11.6 SHEDDING PEAK LOADS

The size and frequency of peak demands need to be known to determine the extent of downtime necessary to achieve a given kW reduction. A sufficient number of months should be studied to develop a representative profile. Seasonal or production variations may also exist. Table 11.8 shows, in descending order, the 11 highest peak demands occurring during a typical month. Reducing the peak demand from 6,320 kW to 5,990 kW will save \$6,600 per month, based on a peak demand charge of \$20/kW, or \$79,200 annually.

Date	Time	kW	kW above 5900
May 10	10:00 am	6320	330
May 24	10:30 am	6220	230
May 14	11:00 am	6145	155
May 5	1:30 pm	6095	105
May 20	2:30 pm.	6055	65
May 15	10:30 am	6025	35
May 15	10:00 am	6010	20
May 8	2:00 pm.	6000	10
May 9	2:00 pm	5995	5
May 13	1:30 pm	5995	5
May 5	2:00 pm	5990	--

Table 11.8 Highest Demands for Hypothetical Billing Period of May

11.9 THE COST OF STEAM

There are two methods to calculate steam cost: unloaded and loaded. The *unloaded cost* is a basic comparison between the amount of steam produced and the cost of fuel required to produce it. The basic equation for calculating unloaded steam cost includes the cost of fuel, operating steam pressure, feedwater Btu, and boiler efficiency. *Loaded steam cost* captures all aspects and costs for producing steam. The largest fraction of the cost of raising steam is fuel cost so the unloaded cost is often enough information.

Fuel is sold based on its gross or higher heating value (HHV). If, at the end of the combustion process, water remains in the form of vapor, the HHV must be reduced by the latent heat of vaporization of water. This reduced value is known as the lower heating value (LHV).


Benchmarking the fuel cost of steam generation, in dollars per 1,000 pounds (\$/1,000 lb) of steam, is an effective way to assess the efficiency of a steam system. This cost depends upon fuel type, unit fuel cost, boiler efficiency, feedwater temperature, and steam pressure. This

calculation provides a good first approximation for the cost of generating steam and serves as a tracking device to allow for boiler performance monitoring. Table 11.9 shows the heat input required to produce one pound of saturated steam at different operating pressures and varying feedwater temperatures. Table 11.10 lists the typical energy content and boiler combustion efficiency for several common fuels (Sources: DOE 2002,2003, 2012).


Operating Pressure (psig)	Feedwater Temperature				
	50°F	100°F	150°F	200°F	250°F
150	1,178	1,128	1,078	1,028	977
450	1,187	1,137	1,087	1,037	986
600	1,184	1,134	1,084	1,034	984

Table 11.9. Energy Required (Btu) to Produce One Pound of Saturated Steam (U.S. DOE 2012)

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Fuel Type (sales unit)	Energy Content (Btu/sales unit)	Combustion Efficiency (%)
Natural Gas (MMBtu)	1,000,000	85.7
Natural Gas (1000 ft ³)	1,030,000	85.7
Distillate/No. 2 Oil (gallon)	138,700	88.7
Residual/No. 6 Oil (gallon)	149,700	89.6
Coal (ton)	27,000,000	90.3

Table 11.10 Energy Content and Combustion Efficiency of Fuels (U.S. DOE 2012)

EXAMPLE 11.7 STEAM COST CALCULATION

A boiler fired with natural gas costing \$8.00/MM Btu produces 450 psig saturated steam and is supplied with 230°F feedwater. Using values from Table 11.10, calculate the fuel cost of producing steam. Note: 1MM Btu = 10⁶ Btu.

$$\text{Steam cost (\$/1,000 lb)} = \frac{\left(\frac{\$8}{10^6 \text{ Btu}}\right) \left(1,006 \frac{\text{Btu}}{\text{lb}}\right) (1,000 \text{ lb})}{0.857} = \$9.39/1000 \text{ lb}$$

Complications arise in large system when process steam is produced in multiple boilers at different pressures, and with different waste heat recovery systems. Combined heat and power simulation models are used in these situations.

11.10 THE TIME VALUE OF MONEY

Rational financial decisions are based on facts and reason, not on fear or emotion, but it usually involves an element of the immediate personal financial condition.

Suppose you are offered \$10,000 today or a guaranteed \$11,000 one year from today. There are three possible rational decisions.

- Take the \$10,000 now; the money is there on the table and you have bills to pay to keep your company open. The value of money in-hand is enormous. Money one year in the future is not relevant in your financial situation.
- Take \$11,000 one year from now because you have no urgent need for \$10,000 today. A guaranteed \$11,000 is a good reward for waiting. And, you don't have to do anything or take any risk.

- Take the \$10,000 now and invest it with the *expectation* of earning 20% per year, to have \$12,000 at the end of the year. *Expect* implies risk. You might earn 20%, or only 10%, or you might lose part or all of your \$10,000 investment.

The best choice depends on how *you* judge the value (or worth) of money over time. *Value* depends on your to answer this question, “Given my current financial situation, which option is best for me?”

Sometimes the answer is clear. All the necessary information is at hand and financial calculations are not needed. More often, there are multiple options and the outcomes are unclear. Then some analysis is needed.

11.11 PAYBACK TIME

Payback time is the simplest financial evaluation that can be made. The definition is

$$\text{Payback time (y)} = \frac{\text{Gain from investment (\$)}}{\text{Annual net revenues or savings (\$/y)}}$$

EXAMPLE 11.8 ENERGY SAVINGS AT APPLETON PAPERS

Appleton Papers identified a series of energy-saving projects with annual savings of 4.8 million kWh of electricity and nearly 150 trillion Btu of natural gas. Total savings were estimated at \$3.5 million per year, with a project cost of \$2.4 million. That is an average payback time of 1.5 years. Table 11.11 shows a few of the projects and the projected payback times.

The payback time for heat recovery from paper machine vents is

$$\$1,500,000/(\$1,000,000 \text{ per year}) = 1.5 \text{ years}$$

Recommendation	Annual Savings (\$/y)	Project cost (\$)	Payback time (years)
Recover heat from paper machine vents	1,000,000	1,500,000	1.5
Reduce silo temperatures	210,000	None	Immediate
Use direct-fired water heating for de-inking	162,164	135,000	0.8
Reduce compressed air costs	37,962	8,800	0.2
Use condenser water for steam make-up	12,995	10,000	0.8
Recirculate vacuum pump seal water	51,857	25,000	0.5

Table 11.11 Payback time and return on investment for energy saving projects at Appleton Paper (U.S. DOE 2002)

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EXAMPLE 11.9 COMPOSTING PAYBACK TIME

A proposed project is to compost 4,000 tons per year of organic waste (yard, food, etc.). The city can use 2000 ton/y of compost to replace topsoil that cost \$25/ton today. The remaining 2000 ton/y will be given free to city residents. The alternate is to continue hauling this waste to a landfill at a cost of \$50/ton for hauling and \$28/ton for disposal. The costs are:

Option 1 - Dispose to landfill

\$50/ton for waste hauling + \$28/ton for disposal = \$78/ton

Annual cost = (4,000 tons/y)(\$78/ton) = \$312,000/y

Option 2 - Composting to produce 4,000 ton/y of finished product

Capital cost = \$2,800,000

Operating & maintenance costs = \$125,000/y

Compost can be used instead of topsoil that costs \$25/ton

Net annual expenses = O & M cost – Topsoil savings
 $= \$125,000/y - (2000 \text{ ton/y})(\$25/\text{ton}) = \$75,000/y$

Annual savings for composting = \$312,000/y – \$75,000/y = \$237,000/y

Payback Period = \$2,800,000/\$237,000/y = 11.8 y

There are shortcomings in the method. Time and the sequence of cash flows matter and payback time does not take this into account.

EXAMPLE 11.10 CASH FLOW FOR TWO COMPETING TECHNOLOGIES

Two technologies cost \$1,200,000 and have a guaranteed 3-year payback. Suppose the cash flows are as given in Table 11.12. Technology A returns nothing until the three years have passed. Technology B returns \$400,000 per year.

	Technology A	Technology B
Initial investment	\$ 1,200,000	\$ 1,200,000
Year 1 revenue	\$ 0	\$ 400,000
Year 2 revenue	\$ 0	\$ 400,000
Year 3 revenue	\$ 1,200,000	\$ 400,000
Payback period	3 years	3 years

Table 11.12 Cash flow for technologies A and B

Technology B is better. Invest the \$400,000 return from years 1 and 2 in some other venture and you will have considerably more than \$1,200,000 in your pocket at the end of the three years.

You also want to know what happens after the payback is accomplished. Two possibilities are shown in Table 11.13. Technology A might produce no further revenue, while B continues to profit.

Cash Flow	B surpasses A		A surpasses B	
	Technology A	Technology B	Technology A	Technology B
Initial investment	\$1,200,000	\$1,200,000	\$1,200,000	\$1,200,000
Year 1 revenue	\$0	\$400,000	\$0	\$400,000
Year 2 revenue	\$0	\$400,000	\$0	\$400,000
Year 3 revenue	\$1,200,000	\$400,000	\$1,200,000	\$400,000
Year 4 revenue	\$0	\$400,000	\$500,000	\$0
Year 5 revenue	\$0	\$400,000	\$500,000	\$0
Total revenue	\$1,200,000	\$2,000,000	\$2,200,000	\$1,200,000

Table 11.13 Cash flow beyond the payback time

Comparing Technology A and Technology B using only payback analysis hides more than it reveals. The cash flow pattern is important. A method is needed that looks at cash flow over time.

11.12 ADJUSTING COST BETWEEN FUTURE AND PRESENT TIME

The four basic time-value-of-money calculations are:

1. Convert a *present value* (PV) to a *future value* (F).
2. Convert a *future value* (F) to a *present value* (PV).
3. Convert a *present value* (PV) to an *n-year series of equal annual costs* (A).
4. Convert an *n-year series of future costs* (equal or unequal) to an *equivalent present value* (PV).

Conversion 2 is the inverse of conversion 1. Conversion 3 is the inverse of conversion 4. Conversions 3 and 4 are the most useful because expenses and revenues accrue over time.

Converting a *present value*, PV , to a *future value*, F , is the *compound interest* calculation. The future values, interest rate i per year, for one, two and n years are

$$F_{1,i} = PV(1+i)$$

$$F_{2,i} = PV(1+i)(1+i) = PV(1+i)^2$$

...

$$F_{n,i} = PV(1+i)^n$$

The quantity i , is the interest rate in ordinary loan and mortgage transactions. *Interest* implies borrowing to pay for an investment. Municipalities usually finance construction by borrowing money (selling bonds) to be repaid over 20 or 30 years and there is a specified interest rate. The loan is repaid from service charges; so are operation and maintenance costs.

A more proper definition of i is the *discount rate*. The money has a value even if the project is self-financed; there is no actual loan from a bank. In this case the company is loaning money to itself and the discount rate depends on alternate opportunities for investment. The discount rate is determined by the project owner or operator and it can be adjusted to include risk and inflation.

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Converting a future value to a present value (PV) is done by discounting the future cost. This is the inverse of the compounded interest calculation.

$$PV = \frac{F_{n,i}}{(1+i)^n}$$

The total present value for a series of n future payments is

$$PV_{n,j} = \frac{F_1}{(1+i)^1} + \frac{F_2}{(1+i)^2} + \frac{F_3}{(1+i)^3} + \dots + \frac{F_{n-1}}{(1+i)^{n-1}} + \frac{F_n}{(1+i)^n}$$

where the future values can be equal or unequal, and they can be negative or positive. A special case where they are positive and equal is shown later.

EXAMPLE 11.11 PRESENT VALUE

The present value of a future expense (or payment) of \$40,722 in 10 years, when discounted with an interest rate of 5%, is $i = 5\%$ and $n = 10$ years

$$PV = \frac{F_{n,i}}{(1+i)^n} = \frac{\$40,722}{(1+0.05)^{10}} = \$40,722(0.6139) = \$25,000$$

EXAMPLE 11.12 PRESENT VALUE OF AN UNEQUAL SERIES OF PAYMENTS

For future values of \$10,000, \$20,000 and \$30,000, due in 2 years, 4 years and 6 years, the total present value is

$$\begin{aligned} PV &= \frac{\$10,000}{(1+0.05)^2} + \frac{\$20,000}{(1+0.05)^4} + \frac{\$30,000}{(1+0.05)^6} = \frac{\$10,000}{(1.1025)} + \frac{\$20,000}{(1.2155)} + \frac{\$30,000}{(1.3401)} \\ &= \$9,070 + \$16,508 + \$22,386 = \$47,964 \end{aligned}$$

11.13 UNIFORM ANNUAL PAYMENTS

The present value of a series of equal future payments, A , over n periods at interest rate i per period is

$$\begin{aligned} PV_{n,i} &= \frac{A}{(1+i)^1} + \frac{A}{(1+i)^2} + \frac{A}{(1+i)^3} + \dots + \frac{A}{(1+i)^{n-1}} + \frac{A}{(1+i)^n} = A \sum_{j=1}^n \frac{1}{(1+i)^j} \\ PV_{n,i} &= A \left[\frac{(1+i)^n - 1}{i(1+i)^n} \right] = A \times PVF \end{aligned}$$

The term in brackets is the present value factor (*PVF*) for a series uniform payments. Think of it as the present value of a series of \$1.00 per year payments

Usually i is an annual rate and n is the number of years payments will be made. For payments every six months use $i/2\%$ and $2n$ payments. If payments are made monthly instead of annually, use $i/12\%$ per month instead of $i\%$ per year, and use $12n$ periods instead of n years.

The inverse of the present value calculation converts a present value into equal annual payments. This is a common calculation. This is how one calculates the regular payments on mortgages or bond repayments. The calculation is called *amortization*. It means, literally, kill the debt.

A is the annual payment that will pay for the present value in n years at discount rate i . The payments might pay off bonds that were sold to build a pollution control facility. The sum of the annual discount rates is *capital recovery factor*, CRF

$$A = PV \left[\frac{i(1+i)^n}{(1+i)^n - 1} \right] = PV \times CRF$$

where $CRF = 1/PVF$

Table 11.14 is an abbreviated table of CRF values for interest rates $i = 4\%$ to 10% . Larger tables of present value factors (PVF) and capital recovery factors (CRF) are in Appendix 8

Years	4%	5%	6%	7%	8%	9%	10%
5	0.2246	0.2310	0.2374	0.2439	0.2505	0.2571	0.2638
10	0.1233	0.1295	0.1359	0.1424	0.1490	0.1558	0.1627
15	0.0899	0.0963	0.1030	0.1098	0.1168	0.1241	0.1315
20	0.0736	0.0802	0.0872	0.0944	0.1019	0.1095	0.1175

Table 11.14 Capital Recovery Factors

EXAMPLE 11.13 CONVERT PV TO A SERIES OF UNIFORM ANNUAL COSTS

The amount of \$25,000 is to be repaid in 5 equal end-of-year payments. The interest rate is 5% per year. From Table 11.14, find $CRF_{5, 5\%} = 0.2310$. The uniform annual repayment amount is

$$A = (\$25,000)(0.2310) = \$5,775$$

If tabulated CRF values are not handy, just do the arithmetic term-by-term with a spreadsheet. Also use this arithmetic when the future values are not equal

Foregoing the shortcut calculation, the arithmetic goes like this;

$$\begin{aligned} \$25,000 &= \frac{A}{(1.05)} + \frac{A}{(1.05)^2} + \frac{A}{(1.05)^3} + \frac{A}{(1.05)^4} + \frac{A}{(1.05)^5} \\ &= A (0.9524 + 0.9070 + 0.8638 + 0.8227 + 0.7835) = A(4.3295) \\ A &= \frac{\$25,000}{4.3295} = (\$25,000)(0.2310) = \$5,775 \end{aligned}$$

EXAMPLE 11.14 ANNUAL PAYMENTS AND THE PRESENT VALUE CALCULATION

The present value (PV) of a series of future payments is the amount of money in a bank account on day 1 that is sufficient to meet the future payments of principal and interest. Table 11.15 shows how that works out for an initial balance of \$481,334 for an annual interest rate of 5%. This is the present value for a pump and pipe system plus the present value of annual energy costs.

A payment of \$135,312 to purchase the pump and the pipe is made at the beginning of year 1. This reduces the balance in the account to \$346,022. Assuming the energy bill is paid annually at the end of the year, the balance of \$346,022 earns \$17,301 interest (5% of \$346,022). This increases the end-of-year balance to \$363,323.



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Continue this accounting for ten years. When the final end-of-year payment for energy is made, another \$44,811, the account balance is zero. The present value of all the costs covers the initial purchase price and the annual energy bills.

Year	Financial Activity	Interest or Payment	Balance
Beginning of Year 1	Deposit equal to the Present Value		481,334
	Buy pump and piping	-135,312	346,022
End of Year 1	<i>End of Year 1</i>		
	Interest on \$346,022 for 1 y @ 5%	17,301	363,323
	Pay annual energy bill	-44,811	318,512
End of Year 2	Interest on \$318,512 for 1 y @ 5%	15,926	334,437
	Pay annual energy bill	-44,811	289,626
End of Year 3	Interest on \$289,626 for 1 y @ 5%	14,481	304,107
	Pay annual energy bill	-44,811	259,296
End of Year 4	Interest on \$259,296 for 1 y @ 5%	12,965	272,261
	Pay annual energy bill	-44,811	227,449
End of Year 5	Interest on \$227,499 for 1 y @ 5%	11,372	238,822
	Pay annual energy bill	-44,811	194,010
End of Year 6	Interest on \$194,010 for 1 y @ 5%	9,701	203,711
	Pay annual energy bill	-44,811	158,899
End of Year 7	Interest on \$158,899 for 1 y @ 5%	7,945	166,844
	Pay annual energy bill	-44,811	122,033
End of Year 8	Interest on \$122,033 for 1 y @ 5%	6,102	128,134
	Pay annual energy bill	-44,811	83,323
End of Year 9	Interest on \$83,323 for 1 y @ 5%	4,166	87,489
	Pay annual energy bill	-44,811	42,678
End of Year 10	Interest on \$42,678 for 1 y @ 5%	2,134	44,811
	Pay annual energy bill	-44,811	0

Table 11.15 Project account with a balance of \$481,334 at the beginning of year 1 and a zero balance at the end of 10 years. The Present Value of the minimum cost project is \$481,334 (10 years @ $i = 5\%$), showing the present value is the amount of money that will pay for the project.

A related calculation is for a *sinking fund*. A sinking fund is the accumulated future value of a series of n equal annual payments, A , that earn interest at a rate of $i\%$ per year. The annual savings to accumulate F is

$$A = F \left[\frac{i}{(1+i)^n - 1} \right]$$

11.14 COMPARING TWO PROJECTS

A common problem is to compare two (or more) projects, in this case two pumps, that have a ten year life but differ in purchase prices and O & M costs, as given in Table 11.16. The costs are on the same basis. Purchase price includes shipping to the point of use and installation. Annual O&M cost includes electricity, labor, minor repairs, etc.

Cost Factor	Pump A	Pump B
Purchase price (\$)	\$40,000	\$50,000
Annual O&M cost (\$/y)	\$6000	\$5000
Pump lifetime (years)	10	10
Simple Lifetime Cost	\$100,000	\$100,000
Present value O&M costs	\$40,260	\$33,550
Total Present Value	\$80,260	\$83,550

Table 11.16 Economic summary for comparison of two pumps

Assume O&M costs are paid at the end of the year. The present value of one dollar (\$1.00) for $i = 8\%$ and 10 years is

$$\begin{aligned}
 PV_{10, 8\%} &= \frac{1}{1.08^1} + \frac{1}{1.08^2} + \frac{1}{1.08^3} + \frac{1}{1.08^4} + \frac{1}{1.08^5} + \frac{1}{1.08^6} + \frac{1}{1.08^7} + \frac{1}{1.08^8} + \frac{1}{1.08^9} + \frac{1}{1.08^{10}} \\
 &= 0.926 + 0.857 + 0.794 + 0.735 + 0.681 + 0.630 + 0.583 + 0.540 + 0.500 + 0.463 \\
 &= 6.170
 \end{aligned}$$

The present values of \$6000/y O&M costs for pump A and \$5000/y O&M costs for pump B are

$$\begin{aligned}
 \text{Pump A O\&M} \quad PV &= \$6,000(6.170) = \$40,260 \\
 \text{Pump B O\&M} \quad PV &= \$5,000(6.170) = \$33,550
 \end{aligned}$$

The *total present value* is the purchase price plus the present value of the operating costs, which gives

$$\begin{aligned}
 \text{Pump A} \quad \text{Total } PV_A &= \$40,000 + \$40,260 = \$80,260 \\
 \text{Pump B} \quad \text{Total } PV_B &= \$50,000 + \$33,550 = \$83,550
 \end{aligned}$$

All other things being equal, pump A should be selected.

Changing the discount rate shifts the relative importance of the initial investment and the time stream of operating costs. A higher discount rate reduces the present value of the annual costs, and also the total present value.


11.15 LIFE CYCLE COST

The life-cycle cost of a $0.5 \text{ m}^3/\text{s}$ (22.8 mgd) pumping system will be evaluated by calculating the total present value as a function of the pipe diameter (D). The project life is ten years ($n = 10 \text{ y}$) with a discount rate $i = 5\%$, which is a present value factor of $PVF = 7.7217$.

Costs for pipe, pumps and motors are given in Table 11.18. The costs are for illustration only (i.e. the \$ is a fictitious currency). The cost of pumps and motors is reduced by operating at a lower head; i.e. a larger pipe diameter. Also:

Capital cost of pumping station = 4 (Cost of pump + Cost of motor)

Total capital cost = Cost of pumping station + Cost of pipe



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The hydraulic design factors are: pipe length $L = 200$ m, friction factor $f = 0.01$, static head $= 10$ m, and minor loss factor, $K = 3.75$

The velocity $= v = Q/A$, where $A = \pi D^2/4$.

Friction loss in the pipe is

$$h_f = f \frac{L}{D} \frac{v^2}{2g} = 0.01 \left(\frac{200 \text{ m}}{D} \right) \left(\frac{v^2}{2g} \right) = \left(\frac{2 \text{ m}}{D} \right) \left(\frac{v^2}{2g} \right)$$

Minor losses: $h_{\text{minor}} = 3.75 \left(\frac{v^2}{2g} \right)$

Total head loss: $H = h_f + h_{\text{minor}} + h_{\text{static}}$

$$H = \left(\frac{2 \text{ m}}{D} \right) \left(\frac{v^2}{2g} \right) + 3.75 \left(\frac{v^2}{2g} \right) + 10 \text{ m} = \left(\frac{2 \text{ m}}{D} + 3.75 \right) \left(\frac{v^2}{2g} \right) + 10 \text{ m}$$

The economic factors are:

Pump output power (kW)

$$P_o = \frac{\rho g Q H}{1000 \text{ W/kW}} = \frac{(1000 \text{ kg/m}^3)(9.81 \text{ m/s}^2)(0.5 \text{ m}^3/\text{s})H}{1000 \text{ W/kW}} = 4.905 H$$

Overall efficiency of the pump and motor is $\eta_{\text{P+M}} = 0.8$ and power input to pump motor (kW)

$$P_i = \frac{P_o}{\eta_{\text{P+M}}} = \frac{4.905 H}{0.8} = 6.131 H$$

The cost of electricity is \$0.15 per kWh. Continuous operation is assumed.

The velocity head and the total friction loss are calculated for each pipe diameter. The results are in Table 11.17, along with the power requirements, energy use, and annual energy cost.

A sample calculation for a pipe with $D = 0.5$ m is

Pipe area: $A = \frac{\pi D^2}{4} = \frac{3.1416 (0.5 \text{ m})^2}{4} = 0.1963 \text{ m}^2$

Velocity: $v = Q/A = (0.5 \text{ m}^3/\text{s})/(0.1963 \text{ m}^2) = 2.546 \text{ m/s}$

Velocity head: $\frac{v^2}{2g} = \frac{(2.546 \text{ m/s})^2}{2(9.81 \text{ m/s}^2)} = 0.3303 \text{ m}$

Friction losses: $h_f = f \frac{L}{D} \frac{v^2}{2g} = 0.01 \left(\frac{200 \text{ m}}{0.5 \text{ m}} \right) (0.3303 \text{ m}) = 1.321 \text{ m}$

Minor losses: $h_{\text{minor}} = 3.75 \left(\frac{v^2}{2g} \right) = 3.75(0.3303 \text{ m}) = 1.239 \text{ m}$

Static head: $h_{\text{static}} = 10 \text{ m}$

Total headloss: $H = 1.321 \text{ m} + 1.239 \text{ m} + 10 \text{ m} = 12.56 \text{ m}$

Output power:

$$P_o = \frac{\rho g Q H}{1000 \text{ W/kW}} = \frac{(1000 \text{ kg/m}^3)(9.81 \text{ m/s}^2)(0.5 \text{ m}^3/\text{s})(12.56 \text{ m})}{1000 \text{ W/kW}} = 61.61 \text{ kW}$$

$$\text{Input power: } P_i = \frac{P_o}{\eta_{p+m}} = \frac{61.61 \text{ kW}}{0.8} = 77.02 \text{ kW}$$

Annual electricity use = $(77.02 \text{ kW})(24 \text{ h/d})(365 \text{ d/y}) = 674,672 \text{ kWh/y}$

Annual electricity cost = $(674,672 \text{ kWh/y})(\$0.15/\text{kWh}) = \$101,201/\text{y}$

Pipe Diam.	Velocity	Velocity Head	Total Head Loss	Output Power	Input Power	Annual Electric Use	Annual Electric Cost
m	m/s	m	m	kW	kW	kWh/y	\$/y
0.30	7.07	2.550	36.56	179.35	224.19	1,963,884	294,583
0.35	5.20	1.377	23.03	112.95	141.19	1,236,828	185,524
0.40	3.98	0.807	17.06	83.68	104.60	916,310	137,447
0.45	3.14	0.504	14.13	69.30	86.62	758,807	113,821
0.50	2.55	0.331	12.56	61.61	77.02	674,672	101,201
0.55	2.10	0.226	11.67	57.23	71.54	626,654	93,998
0.60	1.77	0.159	11.13	54.59	68.23	597,736	89,660
0.65	1.51	0.116	10.79	52.93	66.16	579,529	86,929
0.70	1.30	0.086	10.57	51.84	64.80	567,628	85,144
0.75	1.13	0.065	10.42	51.10	63.88	559,597	83,940
0.80	0.99	0.050	10.32	50.60	63.25	554,027	83,104
0.85	0.88	0.040	10.24	50.23	62.79	550,069	82,510
0.90	0.79	0.031	10.19	49.97	62.47	547,197	82,079
0.95	0.71	0.025	10.15	49.78	62.22	545,073	81,761
1.00	0.64	0.021	10.12	49.63	62.04	543,477	81,522

Notes: Flow = $0.5 \text{ m}^3/\text{s}$, Pipe length = 200 m, Static head = 10 m

Annual energy use for combined pump and motor efficiency = 80%, assuming continuous operation.

Table 11.17 Hydraulic information, power output of the pump, and the annual energy requirement for continuous operation.

The capital cost of the pumping station and pipe and the energy cost for pumping will be integrated into the Total Present Value, using a project lifetime of 10 years and a discount rate 5% per year.

The present value of the uniform series of annual energy costs is ($PVF = 7.7217$, see Appendix 8, Table A8.1)

$$\text{Present Value of Energy Costs} = 7.7217 (\text{Annual Cost of Electricity})$$

The Total Present Value of the pumping system is the purchase prices of the pumping station and piping plus the present value of the energy costs.

$$\text{Total Present Value} = \text{Pump Station Cost} + \text{Pipe Cost} + \text{PV Electricity Cost}$$

Table 11.18 gives the costs and the present value for 10 years of operation. The present value is the amount of money needed at the beginning of the project to buy the equipment and pay the energy cost for 10 years. The calculations are as follows:

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Pipe, Pump, and Motor costs: Given

Pump station cost = 4(Pump cost plus Motor cost)

Capital cost = Pump station cost plus Pipe cost

Annual electricity cost = from Table 11.17

Present value of electricity cost = 7.7217(Annual electricity cost)

Present value for total cost = Capital cost + Present value of electricity cost

Pipe Diam.	Pipe Cost	Pump Cost	Motor Cost	Pump Station Cost	Capital Cost	Annual Electric Cost	Present Value of Electricity	Present Value for Total Cost
m	\$	\$	\$		\$	\$	\$	\$
0.30	150,000	50,000	47,000	388,000	538,000	294,583	2,274,678	2,812,678
0.35	175,000	44,000	34,000	312,000	487,000	185,524	1,432,562	1,919,562
0.40	200,000	40,000	25,000	260,000	460,000	137,447	1,061,321	1,521,321
0.45	225,000	38,000	25,000	252,000	477,000	113,821	878,892	1,355,892
0.50	250,000	36,000	25,000	244,000	494,000	101,201	781,442	1,275,442
0.55	275,000	36,000	18,500	218,000	493,000	93,998	725,825	1,218,825
0.60	300,000	36,000	18,500	218,000	518,000	89,660	692,331	1,210,331
0.65	325,000	36,000	18,500	218,000	543,000	86,929	671,242	1,214,242
0.70	350,000	36,000	18,500	218,000	568,000	85,144	657,458	1,225,458
0.75	375,000	35,000	18,500	214,000	589,000	83,940	648,156	1,237,156
0.80	400,000	35,000	18,500	214,000	614,000	83,104	641,704	1,255,704
0.85	425,000	35,000	18,500	214,000	639,000	82,510	637,120	1,276,120
0.90	450,000	35,000	18,500	214,000	664,000	82,079	633,793	1,297,793
0.95	475,000	35,000	18,500	214,000	689,000	81,761	631,334	1,320,334
1.00	500,000	35,000	18,500	214,000	714,000	81,522	629,485	1,343,485
Capital cost of pumping station = 4 (Cost of pump + Cost of motor) Total capital cost = Cost of pumping station + Cost of pipe Annual energy cost is based on continuous operation. Electric bill paid at the end of the year. Present value is for 10 years at a 5% per year cost of money.								

Table 11.18 Capital cost of the pump station and piping, annual energy cost, present value for energy and for the total cost over 10 years.

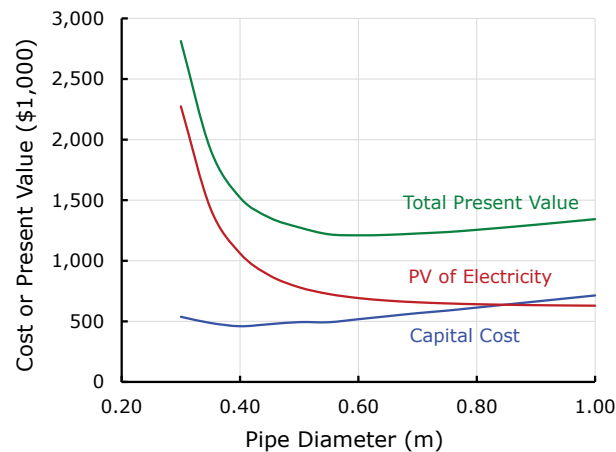


Figure 11.4 The capital cost of the pumping station and piping, the present value of the annual energy cost, and the present value of the project for a 10-y life at 5% interest per year.

The capital costs and the present values are shown in Figure 11.4. It is not easy to see where the minimum is located. The minimum cost design is a 0.60 m pipe diameter, with a capital cost of \$518,000, a 10-year present value for electricity of \$692,000, and a 10-year Present Value of \$1,210,000.

Available pipe diameters may not include 0.60 m, so the nearest available size would be installed. Pipes of 0.5 m to 0.7 m diameters have virtually the same present value.

Preliminary cost estimates, such as these, are not accurate enough to call this an *optimum design*, but it is a good design to work from. The purpose here was to demonstrate how the economic concepts can be used to find good solutions.

It is assumed that the cost of installing the pump and the piping is the same for all possible pipe diameters so this cost is omitted. Including it would simply shift the capital cost curve and the total present value curve up by a small amount. It would not change the location of the minimum present value.

11.16 INFLATION

The normal trend is for the price of everything to go up over time. This is inflation. It is impossible to predict what it will be in the future and it is not always explicitly included in the financial calculations.

To include it in the calculation of a present value requires a simple adjustment. Assume that the annual cost, A , increases by r percent per year, so that $F_2 = (1 + r)F_1$ and so on. For a discount rate of i this gives

$$PV = \frac{A(1+r)}{1+i} + \frac{A(1+r)^2}{(1+i)^2} + \frac{A(1+r)^3}{(1+i)^3} + \dots$$

As an excellent approximation $(1+r)/(1+i)$ can be replaced by $i^* = 1/(1 + i - r)$. This is how the discount rate can be adjusted to account for inflation.

EXAMPLE 11.15 ADJUSTING FOR INFLATION

Suppose that $r = 0.02$ and $i = 0.06$. The present value factors for the first five years, with inflation included, are

$$PVF_1 = \frac{1+0.02}{1+0.06} = 0.9623, \quad PVF_2 = \left(\frac{1+0.02}{1+0.06} \right)^2 = 0.9260, \quad PVF_3 = \left(\frac{1+0.02}{1+0.06} \right)^3 = 0.8920, \dots$$

Table 11.19 lists the exact discount factors for ($r = 0.02$, $i = 0.06$), the approximate discount factor $i^* = 0.06 - 0.02 = 0.04$, and a better approximation of $i^* = 0.039$. The approximation is sufficient since the future inflation rate r is unknown.

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End of year	Present Value Factor		
	$r = 0.02$ $i = 0.06$	$i^* = 0.04$	$i^* = 0.039$
1	0.9623	0.9615	0.9625
2	0.9260	0.9246	0.9263
3	0.8910	0.8890	0.8916
4	0.8574	0.8548	0.8581
5	0.8250	0.8219	0.8259

Table 11.19 Approximate discount rates for inflation

All cost factors do not inflate at the same rate. The rate of inflation for electricity has been higher than other costs. Fuel costs are highly variables and do not follow a smooth path either upward or downward.

11.17 CONCLUSION

Economic comparisons are a routine part of engineering design. The comparison may be an existing system with a proposed new system, or it may be comparing several options for new equipment. It will usually involve the purchase cost of the equipment and the costs of maintenance and operation. The purchase price is a lump sum at time zero; O & M costs run into the future for the life of the project.

The most convenient way to compare alternatives is to calculate the present value. Each future cost, or future revenue, is discounted to the present time. Tabulate the costs and revenues year by year and do the calculations with a spreadsheet.

The alternate comparison is to convert the initial purchase price to a series of uniform annual costs. This is convenient if the O&M costs are also uniform annual costs, but usually they are not. The present value is more flexible.

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13 APPENDIX 1 – ABRIDGED TABLE OF ATOMIC NUMBERS AND ATOMIC MASSES

Element	Symbol	Atomic Number	Atomic Mass	Element	Symbol	Atomic Number	Atomic Mass
Aluminum	Al	13	27.0	Mercury	Hg	80	200.6
Antimony	Sb	51	121.8	Molybdenum	Mo	42	95.9
Argon	Ar	18	40.0	Neon	Ne	10	20.2
Arsenic	As	33	74.9	Nickel	Ni	28	58.7
Barium	Ba	56	137.3	Nitrogen	N	7	14.0
Beryllium	Be	4	9.0	Oxygen	O	8	16.0
Bismuth	Bi	83	209.0	Phosphorus	P	15	31.0
Boron	B	5	10.8	Platinum	Pt	78	195.1
Bromine	Br	35	79.9	Plutonium	Pu	94	(244)
Cadmium	Cd	48	112.4	Polonium	Po	84	(209)
Calcium	Ca	20	40.1	Potassium	K	19	39.1
Carbon	C	6	12.0	Radium	Ra	88	226.0
Chlorine	Cl	17	35.5	Radon	Rn	86	(222)
Chromium	Cr	24	52.0	Selenium	Se	34	79.0
Cobalt	Co	27	58.9	Silicon	Si	14	28.1
Copper	Cu	29	63.5	Silver	Ag	47	107.9
Fluorine	F	9	19.0	Sodium	Na	11	23.0
Gallium	Ga	31	69.7	Strontium	Sr	38	87.6
Gold	Au	79	197.0	Sulfur	S	16	32.1
Helium	He	2	4.0	Tin	Sn	50	118.7
Hydrogen	H	1	1.0	Titanium	Ti	22	47.9
Iodine	I	53	126.9	Tungsten	W	74	183.8
Iron	Fe	26	55.8	Uranium	U	92	238.0
Krypton	Kr	36	83.8	Vanadium	V	23	50.9
Lead	Pb	82	207.2	Xenon	Xe	54	131.3
Lithium	Li	3	6.9	Zinc	Zn	30	65.4
Magnesium	Mg	12	24.3	Zirconium	Zr	40	91.2
Manganese	Mn	25	54.9				

Table A1.1 Atomic masses for selected elements (Rounded to one decimal place.)

14 APPENDIX 2 – CONVERSION FACTORS

Linear Measure Equivalents

meter	foot	centimeter	inch
1	3.2808	100	39.37
0.2048	1	30.48	12.0
100	0.03281	1	0.3937
0.0254	0.0833	2.54	1



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Sources: Keuzegids Master ranking 2013; Elsevier 'Beste Studies' ranking 2012; Financial Times Global Masters in Management ranking 2012

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Area Equivalents

hectare	sq. meter	acre	sq. foot
1	10,000	2.471	107,639.1
0.001	1	0.000247	10.764
0.4047	4,046.9	1	43,560
9.29x10-6	0.0929	0.000,023	1

Volume Equivalents (U.S. units)

cubic foot	U. S. gallon	acre-foot	barrel (U.S. petroleum)
1	7.48	---	0.1781
0.1337	1	0.000,0031	
43,560	325,851	1	
65.615	42.0	---	1

Volume Equivalents (Metric & U.S.)

liter	cubic meter	U.S. gallon	cubic foot
1	0.001	0.2642	0.0353
1000	1	264.172	35.315
3.785	0.00378	1	0.1337
28.317	0.02832	7.48	1

Power Equivalents

horsepower	kilowatt	ft-lb/s	Btu/s
1	0.7457	550	0.7068
1.341	1	737.56	0.9478
0.001,818	0.001,356	1	0.001285
1.415	1.055	778.16	1

Heat, Energy, or Work Equivalents

Joule (kg-m)	ft-lb	kWh	hp-h	Liter-atm	Btu
1	7.233	2.724	3.653	0.0968	0.009,296
0.1383	1	3,766	5.050	0.0134	0.000,324
367,100	2,655,000	1	1.341	35,534	3412.8
273,750	1,980,000	0.7455	1	26,494	2,545
10.33	74.73	2.815	3.774	1	0.0242
426.7	3,086	0.001,162	0.001,558	41.29	1

15 APPENDIX 3 -DENSITIES AND SPECIFIC WEIGHTS

U. S. Units			SI Units		
Temperature (°F)	Density ρ (lb/ft ³)	Specific Weight γ (lb/ft ³)	Temperature (°C)	Density ρ (kg/m ³)	Specific Weight γ (N/m ³)
-40	0.09464	0.09464	-40	1.514	14.85
-20	0.09032	0.09032	-20	1.395	13.68
0	0.08639	0.08639	0	1.293	12.67
10	0.08456	0.08456	5	1.269	12.45
20	0.08279	0.08279	10	1.247	12.23
30	0.08111	0.08111	15	1.225	12.01
32	0.08063	0.08063	20	1.204	11.81
40	0.07950	0.07950	25	1.184	11.61
50	0.07792	0.07792	30	1.165	11.43
60	0.07641	0.07641	40	1.127	11.05
70	0.07499	0.07499	50	1.109	10.88
80	0.07361	0.07361	60	1.060	10.40
90	0.07226	0.07226	70	1.029	10.09
100	0.07097	0.07097	80	0.9996	9.803
120	0.06852	0.06852	90	0.9721	9.533
140	0.06624	0.06624	100	0.9461	9.278
160	0.06408	0.06408	200	0.7461	7.317
180	0.06208	0.06208	300	0.6159	6.040
200	0.06021	0.06021	400	0.5243	5.142
300	0.05229	0.05229	500	0.4565	4.477
400	0.04621	0.04621	1000	0.2772	2.719
500	0.04138	0.04138			
750	0.03284	0.03284			
1,000	0.02721	0.02721			
1,500	0.02025	0.02025			
The correct unit for density is slugs/ft ³ ; the more convenient unit is lb/ft ³ Specific weight = (density)(acceleration of gravity)					

Table A3.1 Density and specific weight of air (at 1 atm)

Gas	Formula	Molar Mass (g/mol)	Density	
			(kg/m ³)	(lb/ft ³)
Acetylene	C ₂ H ₂	26.02	1.1708	0.0732
Air			1.2928	0.0808
Ammonia	NH ₃	17.03	0.7708	0.0482
Butane	C ₄ H ₁₀	58.08	2.5985	0.1623
Carbon dioxide	CO ₂	44.00	1.9768	0.1235
Carbon monoxide	CO	28.00	1.2501	0.0781
Chlorine	Cl ₂	70.91	3.2204	0.2011
Cyanogen	C ₂ N ₂	52.02	2.3348	0.1459
Ethane	C ₂ H ₆	30.05	2.8700	0.1793
Ethylene	C ₂ H ₄	28.03	1.2644	0.0783
Fluorine	F ₂	38.00	1.6354	0.1022
Hydrogen	H ₂	2.016	0.0898	0.0056
Hydrogen chloride	HCl	36.47	1.6394	0.1024
Hydrogen sulfide	H ₂ S	34.08	1.5992	0.0961
Methane	CH ₄	16.03	0.7167	0.0448
Methyl chloride	CH ₃ Cl	50.48	2.3044	0.1440
Natural gas		19.50	0.7-0.9	0.044-0.056
Nitrogen	N ₂	28.02	1.2507	0.0782
Oxygen	O ₂	32.00	1.4289	0.0892
Propane	C ₃ H ₈	44.09	1.8820	0.1175
Sulfur dioxide	SO ₂	64.06	2.9268	0.1828
Water vapor (steam)	H ₂ O	18.02	0.8040	0.0480

Table A3.2 Density of selected gases at standard conditions (0°C and 1 atm)

Material	Sp. gr.	Material	Sp. gr.
Metals		Various liquids	
Aluminum	2.55-2.8	Alcohol, ethyl (100%)	0.79
Bronze	7.4-8.9	Alcohol, methyl (100%)	0.80
Iron, gray cast	7.03-7.10	Acid, nitric (91%)	1.50
hematite ore	5.2	Acid, sulfuric (87%)	1.80
magnetite ore	4.9-5.2	Chloroform	1.50
Lead	11.34	Oils, vegetable	0.91-0.94
galena ore	7.3-7.6	Concrete masonry	
Steel, cold-drawn	7.83	cement, stone, sand	2.2-2.4
Various solids		slag. etc.	1.9-2.3
Cereals, corn (bulk)	0.73	cinder, etc.	1.5-1.7
Cotton, flax, hemp	1.47-1.50	Earth, etc., excavated	
Glass, common	2.4-2.8	Clay, dry	1.00
Glass, plate	2.45-2.72	damp plastic	1.76
Glass, flint	3.2-4.7	Earth, dry loose	1.20
Leather	0.86-1.02	dry, packed	1.5 95
Paper	0.70-1.15	moist, loose	1.30
Rubber, goods	1.0-2.0	moist, packed	1.60
Salt, granulated (piled)	0.77	Bituminous substances	
Sulfur	1.93-2.07	Asphalt	1.11-1.5
Timber		Refined kerosene	0.78-0.82
Fir, Douglas	0.48-0.55	Gasoline	0.70-0.75
Maple, white	0.53	Tar, bituminous	1.2
Oak, white	0.77	Coal and coke, piled	
Redwood, California	0.42	anthracite	0.75-0.93
Teak, African	0.99	bituminous	0.64-0.87
Stone, quarried & piled		charcoal	0.166-0.23
Limestone, marble, quartz	1.50	coke	0.37-0.51
Sandstone	1.30		

Table A3.3 Approximate specific gravities of miscellaneous solids and liquids.

16 APPENDIX 4 – HEATING VALUES

The heating value is the amount of heat produced by combustion of a unit quantity of a fuel. We differentiate between gross and net heating values:

The *gross or high heating value* is the amount of heat produced by the complete combustion of a unit quantity of fuel. The gross heating value is obtained when all products of the combustion are cooled down to the temperature before the combustion the water vapor formed during combustion is condensed

The *net or lower heating value* is obtained by subtracting the latent heat of vaporization of the water vapor formed by the combustion from the gross or higher heating value.

Common Units for heating values:

$$1 \text{ Btu/ft}^3 = 8.9 \text{ kcal/m}^3 = 3.73 \times 10^4 \text{ J/m}^3$$

$$1 \text{ Btu/lb} = 2,326.1 \text{ J/kg} = 0.55556 \text{ kcal/kg}$$

$$1 \text{ J/kg} = 0.00043 \text{ Btu/lb} = 2.39 \times 10^{-4} \text{ kcal/kg}$$

$$1 \text{ kcal/kg} = 1.80 \text{ Btu/lb} = 4,187 \text{ J/kg}$$



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Gas	Higher Heating Value (HHV)		Lower Heating Value (LHV)	
	(Btu/ft ³)	(Btu/lb)	(Btu/ft ³)	(Btu/lb)
Acetylene (ethyne)	1,498	21,569	1,447	20,837
Benzene	3,741	18,150	3,590	17,418
Blast Furnace gas	92	1,178	92	1,178
Butane	3,225	21,640	2,977	19,976
Butylene (butene)	3,077	20,780	2,876	19,420
Carbon to CO ₂		14,150		14,150
Carbon monoxide	323	4,368	323	4,368
Coke Oven Gas	574	17,048	514	15,266
Digester Gas (Biogas)	690	11,316	621	10,184
Ethane	1,783	22,198	1,630	20,295
Ethylene	1,631	21,884	1,530	20,525
Hexane	4,667	20,526	4,315	18,976
Hydrogen	325	61,084	275	51,628
Hydrogen Sulfide	672	7,479		
Landfill Gas	476			
Methane	1,011	23,811	910	21,433
Natural gas (typical)	950-1,150	19,500-22,500	850-1,050	17,500-22,000
Propane	2,572	21,564	2,371	19,834
Propylene (Propene)	2,332	20,990	2,181	19,630
Toluene	4,408	18,129	4,206	17,301
Sasol	500	14,550	443	13,016
Water Gas (bituminous)	261	4,881	239	4,469
Xylene	5,155	18,410		

Table A4.1 Gross (High) and Net (Low) Heating Values of Some Gases (U.S. Units)

Gas	Higher Heating Value (HHV)		Lower Heating Value (LHV)	
	(MJ/m ³)	(MJ/kg)	(MJ/m ³)	(MJ/kg)
Acetylene (ethyne)	55.88	50.17	53.97	48.47
Benzene	139.54	42.22	133.91	40.52
Blast Furnace gas	3.43	2.74	3.43	2.74
Butane	120.29	50.34	111.04	46.47
Butylene (butene)	114.77	48.34	107.27	45.17
Carbon to CO ₂		32.91		32.91
Carbon monoxide	12.05	10.16	12.05	10.16
Coke Oven Gas	21.41	39.66	19.17	35.51
Digester Gas (Biogas)	25.74	26.32	23.16	23.69
Ethane	66.51	51.63	60.80	47.21
Ethylene	60.84	50.90	57.07	47.74
Hexane	174.08	47.75	160.95	44.14
Hydrogen	12.12	142.09	10.26	120.09
Hydrogen Sulfide	25.07	17.40		
Landfill Gas	17.75			
Methane	37.71	55.39	33.94	49.86
Natural gas (typical)	35.44 - 42.90	45.56 - 52.34	31.71 - 39.20	40.71 - 51.17
Propane	95.94	50.16	88.44	46.14
Propylene (Propene)	86.98	48.82	81.35	45.66
Toluene	164.42	42.17	156.88	40.24
Sasol	18.65	33.84	16.52	30.28
Water Gas (bituminous)	9.74	11.35	8.91	10.40
Xylene	192.28	42.82		

Table A4.2 Gross (High) and Net (Low) Heating Values of Some Gases (SI Units)

Chemical Substance	Btu/lb	kJ/kg	Chemical	Btu/lb	kJ/kg
Gases			Liquids		
Ammonia (anhydrous)	8,000	18,608	Acetaldehyde	11,390	26,493
Cyanogen	8,930	20,771	Acetic acid	6,265	14,572
Formaldehyde	8,180	19,027	Acetone	13,260	30,843
Isobutane	19,610	45,613	Benzene	17,260	40,147
Methane	21,500	50,009	Butyl alcohol	15,530	36,123
Methyl chloride	5,850	13,607	Chlorobenzene	12,000	27,912
Propylene	19,700	45,822	Creosote (coal tar)	12,500	29,075
Vinyl chloride	8,140	18,934	Cresol	14,700	34,192



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Solids			Liquids		
Anthracene	17,300	40,240	Diesel fuel	18,400	42,798
Benzoic acid	11,370	26,447	Ethyl acetate	11,000	25,586
Citric acid (anhydrous)	4,390	10,211	Ethylbenzene	17,600	40,938
Hexachlorobenzene	3,220	7,490	Ethylene glycol	8,175	19,015
Menthol	17,380	40,426	Ethylene glycol diacetate	15,000	34,890
Naphthalene	17,300	40,240	Glycerine (glycerol)	7,760	18,050
Nitrobenzene	7,400	17,212	Heptane	19,380	45,078
Nitrophenol	8,900	20,701	Hexane	19,230	44,729
p-Nitrotoluene	11,665	27,133	Kerosene	15,500	36,053
Palmitic acid	16,470	38,309	Methyl acetate	9,260	21,539
Phenanthrene	17,020	39,589	Methyl alcohol	9,755	22,690
Phenol	14,000	32,564	Methyl ethyl ketone	14,600	33,960
Phtalic acid	8,350	19,422	Methyl mercaptan	11,050	25,702
Polypropylene	19,600	45,590	Motor oil	18,500	43,031
Stearic acid	17,060	39,682	Naphtha	18,000	41,868
Strychnine	14,460	33,634	Nicotine	11,510	26,772
Sucrose	7,090	16,491	Nitrobenzene	10,810	25,144
Tannic acid	9,810	22,818	o-Nitrotoluene	11,290	26,261
Trichloroacetic acid	1,020	2,373	Octanol	17,440	40,565
Trinitrobenzene	5,600	13,026	Oil, crude	18,250	42,450
Trinitrotoluene	6,500	15,119	Oil, vegetable	16,000	37,216
Urethane	8,025	18,666	Pentane	19,340	44,985
Wax (Paraffin)	18,000	41,868	Polypropylene glycol	14,200	33,029
			Propyl alcohol	14,460	33,634
			Styrene	18,100	42,101
			Tallow	18,000	41,868
			Tetraethyl lead	7,870	18,306
			Toluene	17,420	40,519
			Triethylaluminum	18,360	42,705
			Vinyl acetate	9,750	22,679
			Xylene	17,500	40,705

Table A4.3 Heat of combustion of selected chemical compounds
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17 APPENDIX 5 – ENTHALPY OF WATER AND STEAM

T (°F)	Vapor Pressure Atm	Enthalpy, Btu/lb			T (°C)	Vapor Pressure kPa	Enthalpy, kJ/kg		
		Sat. Liquid	ΔH_v	Sat. Vapor			Sat. Liquid	ΔH_v	Sat. Vapor
32	0.0060	0	1075.1	1075.1	0	0.62	0	2500.9	2500.9
40	0.0083	8.05	1070.5	1078.6	5	0.87	21.02	2489.1	2510.1
50	0.0121	18.07	1064.8	1082.9	10	1.23	42.02	2477.2	2519.2
60	0.0174	28.07	1059.1	1087.2	15	1.71	62.98	2465.4	2528.4
70	0.0247	38.05	1053.4	1091.5	20	2.34	83.92	2453.6	2537.5
80	0.0345	48.02	1047.8	1095.8	25	3.17	104.84	2441.7	2546.5
90	0.0475	58.00	1042.1	1100.1	30	4.25	125.75	2429.9	2555.6
100	0.0646	67.97	1036.4	1104.4	35	5.63	146.64	2418.0	2564.6
110	0.0867	77.94	1030.9	1108.8	40	7.38	167.54	2406.0	2573.5
120	0.115	87.91	1025.3	1113.2	45	9.59	188.44	2394.1	2582.5
130	0.151	97.89	1019.5	1117.4	50	12.35	209.34	2382.0	2591.3
140	0.196	107.88	1013.7	1121.6	55	15.67	230.24	2369.9	2600.1
150	0.253	117.87	1007.8	1125.7	60	19.95	251.15	2357.7	2608.8
160	0.332	127.87	1002.0	1129.9	65	25.04	272.08	2346.4	2618.5
170	0.408	137.89	996.1	1134.0	70	31.20	293.02	2333.1	2626.1
180	0.511	147.91	990.2	1138.1	75	38.60	313.97	2320.6	2634.6
190	0.635	157.95	984.1	1142.1	80	47.41	334.95	2308.1	2643.0
200	0.784	167.99	977.8	1145.8	85	57.87	355.95	2295.4	2651.3
210	0.961	178.06	971.5	1149.6	90	70.18	376.97	2282.5	2659.5
212	1.000	180.07	970.3	1150.4	95	84.61	398.02	2269.6	2667.6
220	1.170	188.14	965.2	1153.3	100	101.33	419.10	2256.5	2675.6
230	1.414	198.22	958.7	1156.9	110	143.38	461.36	2229.7	2691.1
240	1.699	208.34	952.1	1160.4	120	198.67	503.78	2202.1	2705.9
250	2.029	218.48	945.3	1163.8	130	270.26	546.39	2173.7	2720.1
260	2.411	228.65	938.6	1167.3	140	361.50	589.20	2144.2	2733.4
270	2.848	238.84	931.8	1170.6	150	476.10	632.25	2113.7	2745.9
280	3.348	249.06	924.6	1173.7	160	618.14	675.57	2081.8	2757.4
290	3.916	259.31	917.4	1176.7	170	792.05	719.21	2048.7	2767.9
300	4.560	269.60	910.1	1179.7	180	1002.60	763.19	2014.0	2777.2

Table A5.1 Enthalpy and latent heat of vaporization, ΔH_v , of saturated water and steam. (Complete steam tables can be found in Perry's Chemical Engineer's Handbook, 8th ed. (2006), CRC Press, Boca Rotan.

18 APPENDIX 6 – ENTHALPY OF AIR

Temp.		Saturation Pressure (N/m ²)	Moisture at Saturation (kg _{H₂O} /kg _{dry air})	Specific Volume		Specific Enthalpy	
°C	°F			Dry Air (m ³ /kg)	Saturated Mixture (m ³ /kg dry air)	Dry Air (kJ/kg)	Saturated Mixture (kJ/kg dry air)
-40	-40	12.84	0.000079	0.660	0.660	-40.2	-40.0
-30	-22	38	0.00023	0.688	0.688	-30.2	-29.6
-25	-13	63.25	0.00039	0.702	0.703	-25.2	-24.2
-20	-4	103.2	0.00064	0.716	0.717	-20.1	-18.5
-15	5	165.2	0.0010	0.731	0.732	-15.1	-12.6
-10	14	259.2	0.0016	0.745	0.747	-10.1	-6.1
-5	23	401.5	0.0025	0.759	0.762	-5.0	1.2
0	32	610.8	0.0038	0.773	0.778	0	9.5
5	41	871.9	0.0054	0.788	0.794	5.0	18.6
10	50	1227	0.0077	0.802	0.812	10.1	29.5
15	59	1704	0.011	0.816	0.830	15.1	42.9
20	68	2337	0.015	0.830	0.850	20.1	58.2
25	77	3167	0.020	0.844	0.872	25.2	76.1
30	86	4243	0.027	0.859	0.896	30.2	99.2
35	95	5623	0.037	0.873	0.924	35.2	130.1
40	104	7378	0.049	0.887	0.957	40.2	166.4
45	113	9585	0.065	0.901	0.995	45.3	213.2
50	122	12339	0.087	0.915	1.042	50.3	275.9
55	131	14745	0.12	0.929	1.1	55.3	367.6
60	140	19925	0.15	0.944	1.175	60.4	452.1
65	149	25014	0.21	0.958	1.272	65.4	615.7
70	158	31167	0.28	0.972	1.404	70.4	806.8
75	167	38554	0.38	0.986	1.592	75.5	1078
80	176	47365	0.55	1	1.879	80.5	1537
85	185	57809	0.84	1.015	2.363	85.5	2317
90	194	70112		1.03	3.340		3876

Table A6.1 Enthalpy of Air

19 APPENDIX 7 – LOWER AND UPPER EXPLOSIVE LIMITS FOR SELECTED VOLATILE COMPOUNDS

Gas	LEL (% vol)	UEL (% vol)	Gas	LEL (% vol)	UEL (% vol)
Acetaldehyde	4	60	Hexane	1.1	7.5
Acetic acid	4	19.9	Hydrogen	4	75
Acetone	2.6	12.8	Hydrogen sulfide	4.3	46
Acetylene	2.5	81	Isobutyl alcohol	2	11
Acrylonitrile	3	17	Isopropanol	2	12
Ammonia	15	28	Kerosene Jet A-1	0.7	5
Benzene	1.35	6.65	Methane	4.4	16.4
1,3-Butadiene	2	12	Methanol	6.7	36
n-Butane	1.86	8.41	Methyl Chloride	10.7	17.4
Butyl alcohol, Butanol	1	11	Methyl Ethyl Ketone	1.8	10
Butylene	1.98	9.65	Mineral spirits	0.7	6.5
Carbon Disulfide	1.3	50	Naphthalene	0.9	5.9
Carbon Monoxide	12	75	n-Heptane	1	6
Cyanogen	6	42.6	n-Hexane	1.25	7
Cyclobutane	1.8	11.1	n-Pentene	1.65	7.7
Cyclohexane	1.3	8	Nitrobenzene	2	9
1,1-Dichloroethane	6	11	n-Octane	1	7
Diethyl Ether	1.9	36	n-Pentane	1.4	7.8
Diesel fuel	0.6	7.5	Propane	2.1	10.1
Ethane	3	12.4	Propylene	2	11.1
Ethylene	2.75	28.6	Propylene oxide	2.3	36
Ethyl Alcohol, Ethanol	3.3	19	Styrene	1.1	6.1
Ethylbenzene	1	7.1	Toluene	1.27	6.75
Ethyl Chloride	3.8	15.4	Trichloroethylene	13	90
Ethylene glycol	3	22	Turpentine	0.8	-
Fuel Oil No.1	0.7	5	Vinyl acetate	2.6	13.4
Gasoline	1.4	7.6	Vinyl chloride	3.6	33
Glycerol	3	19	p-Xylene	1	6
Heptane	1	6.7			

Table A7.1 Lower Explosive Limit (LEL) and Upper Explosive Limit (UEL) for selected volatile compounds (% volume in air). Source: www.engineerstoolsbox.com

20 APPENDIX 8 – FINANCIAL TABLES

Convert a Series of Uniform Annual Costs to a Present Value

Convert a uniform annual cost, A to a Present Value, PV , using

$$PV_{n,i} = \frac{A}{(1+i)^1} + \frac{A}{(1+i)^2} + \frac{A}{(1+i)^3} + \dots + \frac{A}{(1+i)^{n-1}} + \frac{A}{(1+i)^n} = A \sum_{j=1}^n \frac{1}{(1+i)^j}$$

The summation of the $1/(1+i)^n$ terms is the Present Value Factor

$$\text{Present Value Factor} = PVF_{n,i} = F_{AP,n,i} = \sum_{j=1}^n \frac{1}{(1+i)^j} = \frac{(1+i)^n - 1}{i(1+i)^n}$$

$$PV = (A) F_{AP,n,i} = (A) PVF$$

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Year	Interest Rate (% per Year)							
	0.03	0.04	0.05	0.06	0.07	0.08	0.09	0.1
1	0.97087	0.96154	0.95238	0.94340	0.93458	0.92593	0.91743	0.90909
2	1.91347	1.88609	1.85941	1.83339	1.80802	1.78326	1.75911	1.73554
3	2.82861	2.77509	2.72325	2.67301	2.62432	2.57710	2.53129	2.48685
4	3.71710	3.62990	3.54595	3.46511	3.38721	3.31213	3.23972	3.16987
5	4.57971	4.45182	4.32948	4.21236	4.10020	3.99271	3.88965	3.79079
6	5.41719	5.24214	5.07569	4.91732	4.76654	4.62288	4.48592	4.35526
7	6.23028	6.00205	5.78637	5.58238	5.38929	5.20637	5.03295	4.86842
8	7.01969	6.73274	6.46321	6.20979	5.97130	5.74664	5.53482	5.33493
9	7.78611	7.43533	7.10782	6.80169	6.51523	6.24689	5.99525	5.75902
10	8.53020	8.11090	7.72173	7.36009	7.02358	6.71008	6.41766	6.14457
11	9.25262	8.76048	8.30641	7.88687	7.49867	7.13896	6.80519	6.49506
12	9.95400	9.38507	8.86325	8.38384	7.94269	7.53608	7.16073	6.81369
13	10.63496	9.98565	9.39357	8.85268	8.35765	7.90378	7.48690	7.10336
14	11.29607	10.56312	9.89864	9.29498	8.74547	8.24424	7.78615	7.36669
15	11.93794	11.11839	10.37966	9.71225	9.10791	8.55948	8.06069	7.60608
16	12.56110	11.65230	10.83777	10.10590	9.44665	8.85137	8.31256	7.82371
17	13.16612	12.16567	11.27407	10.47726	9.76322	9.12164	8.54363	8.02155
18	13.75351	12.65930	11.68959	10.82760	10.05909	9.37189	8.75563	8.20141
19	14.32380	13.13394	12.08532	11.15812	10.33560	9.60360	8.95011	8.36492
20	14.87747	13.59033	12.46221	11.46992	10.59401	9.81815	9.12855	8.51356
21	15.41502	14.02916	12.82115	11.76408	10.83553	10.01680	9.29224	8.64869
22	15.93692	14.45112	13.16300	12.04158	11.06124	10.20074	9.44243	8.77154
23	16.44361	14.85684	13.48857	12.30338	11.27219	10.37106	9.58021	8.88322
24	16.93554	15.24696	13.79864	12.55036	11.46933	10.52876	9.70661	8.98474
25	17.41315	15.62208	14.09394	12.78336	11.65358	10.67478	9.82258	9.07704
26	17.87684	15.98277	14.37519	13.00317	11.82578	10.80998	9.92897	9.16095
27	18.32703	16.32959	14.64303	13.21053	11.98671	10.93516	10.02658	9.23722
28	18.76411	16.66306	14.89813	13.40616	12.13711	11.05108	10.11613	9.30657
29	19.18845	16.98371	15.14107	13.59072	12.27767	11.15841	10.19828	9.36961
30	19.60044	17.29203	15.37245	13.76483	12.40904	11.25778	10.27365	9.42691

Table A8.1 Factors (*PVF*) to convert a series of uniform annual costs, *A*, to a present value, *PV*.

Convert a Present Value to a Series of Uniform Annual Costs

The factor to convert a present value, which is often the capital cost, to a series of uniform annual costs is called the Capital Recovery Factor. It is the inverse of the factors in Table A8.1

$$PV_{n,i} = \frac{A}{(1+i)^1} + \frac{A}{(1+i)^2} + \frac{A}{(1+i)^3} + \dots + \frac{A}{(1+i)^{n-1}} + \frac{A}{(1+i)^n} = A \sum_{j=1}^n \frac{1}{(1+i)^j} = A \left[\frac{(1+i)^n - 1}{i(1+i)^n} \right]$$

Solving for A gives $A = PV \left[\frac{i(1+i)^n}{(1+i)^n - 1} \right] = PV \times CRF$

$$\text{Capital Recovery Factor} = CRF_{n,i} = F_{PA,n,i} = \frac{i(1+i)^n}{(1+i)^n - 1}$$

$$A = (PV) F_{PA,n,i} = (PV) CRF$$

Year	Interest Rate (% per Year)							
	0.03	0.04	0.05	0.06	0.07	0.08	0.09	0.1
1	1.03000	1.04000	1.05000	1.06000	1.07000	1.08000	1.09000	1.10000
2	0.52261	0.53020	0.53780	0.54544	0.55309	0.56077	0.56847	0.57619
3	0.35353	0.36035	0.36721	0.37411	0.38105	0.38803	0.39505	0.40211
4	0.26903	0.27549	0.28201	0.28859	0.29523	0.30192	0.30867	0.31547
5	0.21835	0.22463	0.23097	0.23740	0.24389	0.25046	0.25709	0.26380
6	0.18460	0.19076	0.19702	0.20336	0.20980	0.21632	0.22292	0.22961
7	0.16051	0.16661	0.17282	0.17914	0.18555	0.19207	0.19869	0.20541
8	0.14246	0.14853	0.15472	0.16104	0.16747	0.17401	0.18067	0.18744
9	0.12843	0.13449	0.14069	0.14702	0.15349	0.16008	0.16680	0.17364
10	0.11723	0.12329	0.12950	0.13587	0.14238	0.14903	0.15582	0.16275
11	0.10808	0.11415	0.12039	0.12679	0.13336	0.14008	0.14695	0.15396
12	0.10046	0.10655	0.11283	0.11928	0.12590	0.13270	0.13965	0.14676
13	0.09403	0.10014	0.10646	0.11296	0.11965	0.12652	0.13357	0.14078
14	0.08853	0.09467	0.10102	0.10758	0.11434	0.12130	0.12843	0.13575
15	0.08377	0.08994	0.09634	0.10296	0.10979	0.11683	0.12406	0.13147
16	0.07961	0.08582	0.09227	0.09895	0.10586	0.11298	0.12030	0.12782
17	0.07595	0.08220	0.08870	0.09544	0.10243	0.10963	0.11705	0.12466
18	0.07271	0.07899	0.08555	0.09236	0.09941	0.10670	0.11421	0.12193
19	0.06981	0.07614	0.08275	0.08962	0.09675	0.10413	0.11173	0.11955
20	0.06722	0.07358	0.08024	0.08718	0.09439	0.10185	0.10955	0.11746
21	0.06487	0.07128	0.07800	0.08500	0.09229	0.09983	0.10762	0.11562
22	0.06275	0.06920	0.07597	0.08305	0.09041	0.09803	0.10590	0.11401
23	0.06081	0.06731	0.07414	0.08128	0.08871	0.09642	0.10438	0.11257
24	0.05905	0.06559	0.07247	0.07968	0.08719	0.09498	0.10302	0.11130
25	0.05743	0.06401	0.07095	0.07823	0.08581	0.09368	0.10181	0.11017
26	0.05594	0.06257	0.06956	0.07690	0.08456	0.09251	0.10072	0.10916
27	0.05456	0.06124	0.06829	0.07570	0.08343	0.09145	0.09973	0.10826
28	0.05329	0.06001	0.06712	0.07459	0.08239	0.09049	0.09885	0.10745
29	0.05211	0.05888	0.06605	0.07358	0.08145	0.08962	0.09806	0.10673
30	0.05102	0.05783	0.06505	0.07265	0.08059	0.08883	0.09734	0.10608

Table A8.2 Factors (*CRF*) to convert a present value, *PV*, to a series of uniform annual costs, *A*.

INDEX

A

Absorption chiller 58
Acetic acid 91
AC induction motor 143
Activated sludge process 160
Actual oxygen transfer rate (AOTR) 164
Adjusting cost between future and present time 199
Aeration tanks 160
Air classifier 79
Air diffusers 160
Air pollution control technologies 81
 Baghouse filter 81
 Pulse-jet fabric filter 83
 Selective catalytic reduction 83
 Spray dryer absorber 83
 Wet scrubber 81
Air requirement 119
American Society of Mechanical Engineers (ASME) 61
Amortization 202
Anaerobic fermentation 86
Anaerobic sludge digestion 97
Analysis 10
Approach 56
Auxiliary air 82
Auxiliary fuel 75
Avagadro's number 66

B

Benzene 111, 125
Best efficiency point 145
Betz limit 34
Biogas 86, 98
 Anaerobic sludge digestion 97
 Flaring 93
 Landfills 89
Blowdown 54
Blower air 119
Blowers 157, 160
 Air delivery pressure 165
 Centrifugal 158
 Efficiencies 169
 Operating curve 158
 Operating point 158
 Operating pressure 165
 Power 168, 170
 Temperature increase across 168
Boiler feed water 61
Boilers 50
 Blowdown 62
 Combustion efficiency 194
 Horsepower 60
British thermal unit (Btu) 24

C

Caloric value 107
Capital costs 181, 206
 Cost estimates 181
 Cost estimating function 183
Capital recovery factor (CRF) 202, 237
Carbon dioxide 29, 49, 65, 86, 98, 107, 119
Carnot engine 21
Case Study
 Energy Balance on a Regenerative Thermal Oxidizer 125
 Palm Beach Renewable Energy Facility No. 2 82
 Recovery of Heat from Combustion of Waste Gases 118
Cash flow 198
Catalyst 117
Catalytic incineration 117
Centrifugal blower 158
Centrifugal pump 143
Chloromethane 126
Choke 160
Clean energy 89
Combined heat and power (CHP) 30
 University of Wisconsin-Madison CHP plant 31
Combustion 64
Combustion chamber 80, 116
 Volume 124
Combustion equations 118
 Stoichiometry 68
Composting 198
Compound interest calculation 199
Compressors 157
Condensing steam 136
Conservation of energy 12, 16
Conservation of mass 12, 66
Cooling capacity 58
Cooling towers 54
 Effectiveness 58
 Efficiency 57
Cost estimates 181
Costs
 Comparing two projects 198, 205
Cycles of concentration 54

D

Deep cycle batteries 41
Desert Sunlight Solar Farm (Mojave Desert, CA) 37
Digester gas 101
Dilution air 123
Discount rate 200
Dissolved oxygen 162
Drift 54

Drive motor 150
Dry-bulb temperature 55
Dust 110
Duty point 145

E

Economy of scale 182, 186
 Multiple units 186
Efficiency curve 145
Electricity costs 188
 Demand charge 189
 Energy charge 189
 Residential rates 188
Electricity use 192
 Shedding peak loads 192
Empirical chemical formula 68
Empirical estimate of oxygen demand 162
Empirical relations for gas production 98
Energy 24
Energy balance 10, 16, 50, 124, 126
Energy Balance on a Regenerative Thermal Oxidizer 125
Energy conservation 176
Energy conservative design 130
Energy consumption 157
Energy conversion efficiencies 27
Energy Policy Act of 1992 (EPACT) 148

Energy system integration 131
Energy use
 Publicly-owned wastewater treatment plants 177
 Public water supply utilities 176
Enthalpy 43
 Humid air 48
 Standard state 43
Enthalpy-temperature diagram 140
Equivalence of energy units 25
Ethanol 52
EU minimum motor efficiency 149
Evaporation 54
Excess air 70, 119
Explosive limits 110, 126

F

Factor of evaporation 61
Fine bubble diffuser system 167
First law 17
Fluidized beds 80
Fly ash 82
Fossil fuel plant 28
Fresh water makeup 55
Friction factor 151
Future value (F) 199



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G

Gas extraction 91
Global warming potential 89
Greenhouse gases 89

H

Hammermill 78
Head 144
 Friction head 145
 Pressure head 145
 Shut-off head 146
 Static head 145
 Velocity head 147, 150
Head loss 150
Head losses in pipe fittings 151
Heat capacity 45
Heat energy 120
Heat engines 20
Heat exchange networks (HENs) 137
Heat exchangers 131
 Co-current flow 132
 Countercurrent flow 132
Heating value 126
 Gross energy 75
 Higher heating value (HHV) 75
 Lower heating value for fuels (Table) 76
 Lower heating value (LHV) 75
 Net heating value 75
 Tables 227
Heat loss 128
Heat of fusion 45
Heat of transition 45
Heat recovery 50, 104, 129
Heat transfer 127
Heat transfer coefficient 132
Heat transfer equation 132
Heat trap 21
High Btu fuel 106
Higher heating value (HHV) 193
Horsepower 24
Household system for photovoltaic conversion to electricity 39
Hydrochloric acid 126
Hydrogen sulfide 87, 98, 107, 119
Hydro-turbine 28

I

Ideal gas law 113
Incineration 80
Inflation 211
Insolation 36
Instruments and closed loop controllers 173
Interest rate 200
Internal combustion engine 22
International Union of Pure and Applied Chemistry (IUPAC) 111

J

Joule 24

K

Kilowatt peak 39
Kinetic energy 33, 146, 150

L

Landfill gas 89, 93
Landfill Gas Emissions Model (LandGEM) 95
Landfills 89
Latent heat of evaporation 46
Lateral gas migration 91
Le Chatelier's formula 111
Life-cycle cost 206
Linear regression 185
Liquefied natural gas (LNG) 106
Log mean temperature difference 132

M

Magnetic separation 79
Maintenance costs 175
Manure management 86, 98
Mass burn 80
Material balance 10, 124
Mechanical chiller 58
Mercaptans 98, 119
Mesophilic anaerobic sludge digester (MAD) 98
Methane 49, 66, 86, 87, 89, 98, 114, 126
Methyl chloride 111, 125
Microturbine 101, 108
Mixed waste
 Empirical stoichiometric formula 72
Molar mass 114
Mole fraction 70, 113
Motors 148
Moving grate furnace 81
Municipal refuse 64, 69
Municipal solid waste 75, 78

N

National Electrical Manufacturers Association (NEMA) 148
National Institute of Standards and Technology (NIST) 111
Natural gas 29, 88
Newton 24
Nitric acid 72
Nitrification 163
Noble metal catalyst 117
Nuclear power plant 28
n-year series of equal annual costs (A) 199
n-year series of future costs (F) 199

O

Odor control 135
O&M costs 181
 Direct operating costs 181
 Fixed costs 181
 General expenses 181
 Variable costs 181
Operating costs 147 181 205
Organosiloxanes 108
Oxygen supply to activated sludge process 161

P

Palm Beach Renewable Energy Facility No. 2 82
Payback time 101 196
Photovoltaic cells 37
Pinch analysis 140
Pinene 118
Pipe networks and software 154
Power 24 149
 Brake horsepower 150
 Shaft horsepower 150
 Water power 150
Power equivalence factors 25
ppmv 113
Preheater 125
Preliminary cost estimates 211
Present value factor (PVF) 201 235
Present value of energy costs 209
Present value (PV) 199 209
Pressure 144
Pressure drop 166
Process Integration 14 216
Pump curve 145
Pump efficiency 143
Pumping costs 187
 Planning 187 219
Pumps and pumping 142
 Operating head 145

R

Range 56
Reactors 10 131
Recovery of Heat from Combustion of Waste Gases 118
Refrigeration 58
 Tons of refrigeration (TR) 58
Refuse derived fuel (RDF) 79
Residence time 116 125
Roughness 151

S

Safety 110
Second law 20 130
Semi-volatile silicon compounds 108
Sensible heat 45 128

Separations 10
Shredder 79
Siloxane 108
Sinking fund 205
Sludge digestion 86
Sludge incineration 80
Software
 Heat exchanger networks 141
 Pipe networks 154
Solar electric cells 37
Solar energy 36
Solar heating 36
Solid waste 72
Specific enthalpy 45
Specific heat 45
Specific heat and heat of vaporization of selected liquids (Table) 47
Specific heat of some common gases 46
Standard oxygen transfer efficiency (SOTE) 164
Standard oxygen transfer rate (SOTR) 164
Standard temperature and pressure (STP) 111 125
Steady-state 12
Steam 48 50 60
 Cost of producing 193 195
Stoichiometry 64
 Balanced stoichiometric equation 64
 Stoichiometric coefficients 69
 Stoichiometric demand 118
 Stoichiometric oxygen requirement 119
 Stoichiometric ratio 70
Sulfur compounds in biogas 107
Sulfur dioxide 115
Sulfuric acid 72
Surge 160
Synthesis 10
System head curve 145

T

Tanner diagram 75
Temperature-phased anaerobic digestion (TPAD) 98
Thermal electric power plant 18
Thermal incineration 116
Thermal oxidation 125
Thermal recovery efficiency (TRE) 126
Thermodynamics
 First law 16
 Second law 16
 Third law 17
Thermophilic anaerobic sludge digester (TAD) 98
Time value of money 195
Toluene 114
Total dissolved solids (TDS) 54 62
Trommel screen 79
Turndown 159
Turpentine 119

U

- Uniform annual payments 201
- Units conversion
 - Energy and power 25

V

- Variable speed pump 148
- Variations in air flow requirements 171
- Ventilation air 110 114
- Volatile organic chemicals (VOCs) 110
- Volume fraction 70 113
- VS destruction 98

W

- Waste-to-energy power plant 82
- Wastewater treatment plant electricity use 157
- Watt 24
- Watt-peak 39
- West Virginia bituminous coal 26
- Wet-bulb temperature 56
- Wind power factor 34
- Wind turbine 33
- Work 24



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