PN-AAV-381
IAN-43924

SELECTED TOPICS
IN
ALTERNATIVE ENERGY TECHNOLOGIES
FOR
DEVELOPING COUNTRIES

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Training in Alternative Energy Technologies Program

University of Florida

Second Edition

January 1983

Sponsored by the U.S. Agency for International Development
Under Cooperative Agreement AID/DSAN/CA-0188

PREFACE

The United States Agency for International Development (USAID) sponsors a program at the University of Florida at Gainesville for the training of scientists and engineers from developing countries in the alternative energy technologies. The Training in Alternative Energy Technologies (TAET) program provides training in both the technical principles of the alternative energy technologies and the socio-economic aspects of their selection and implementation.

Through a systematic presentation and analysis of the scientific, technical, economic, and social aspects aspects of the development of the alternative sources of energy, the program participants are provided with training in the processes involved in (1) assessing the potential of each source of energy within a broad perspective of energy needs and uses at a national level, (2) selecting appropriate technological options to meet specific needs, and (3) implementing alternative energy programs.

The TAET program curriculum is designed to meet the following specific objectives:

1. To acquaint the participants with the alternative energy technologies.

2. To provide the participant with sufficient knowledge to assess the natural renewable energy resources of the participant's country and to determine the best possible technological options to utilize these resources so that the participant can provide input in establishing realistic national alternative energy programs for the participant's country.

3. To provide technically trained people with the knowledge to select among technological options and to identify their most appropriate applications.

The TAET program focuses on four major areas of alternative energy technology: biomass energy, hydropower, solar energy, and wind energy. The program consists of lectures, seminars, system demonstrations, laboratory work, and field trips, all selected to explain the theory, illustrate the practice, demonstrate the technologies, and provide a thorough training in all aspects of the renewable sources of energy and the alternative energy technologies.

This notebook presents material on only a few of the topics which are addressed during the TAET program. It is in no sense a comprehensive or exhaustive treatment of the alternative energy technologies; it is merely a part of a much larger set of written material that is made available to the program participants during the training session. Some chapters in this text provide only a brief or cursory examination of a subject area. Moreover, many topics are not discussed in these notes at all: hydropower, gasification, alcohol production, to name only a few. In due course, it is hoped that these and other important topics will be included in the text.

The opinions expressed in these notes are those of the author alone, and should not be attributed to either the U.S. Agency for International Development or the University of Florida.

SELECTED TOPICS IN ALTERNATIVE ENERGY TECHNOLOGIES FOR DEVELOPING COUNTRIES

CONTENTS

M
1

CONTENTS

b

CONTENTS

ENERGY USE IN DEVELOPING COUNTRIES

Developing countries use considerable amounts of energy but generally much less than the industrialized nations. However, both the pattern of energy supply and use, and the types of fuels utilized in the developing countries show significant and important differences when compared to the industrialized countries. There are also enormous differences between countries that are all loosely classified as developing countries. At one end of the scale there are very poor countries with little economic activity beyond subsistence agriculture and livestock tending; at the other end of the scale are countries such as Brazil, Korea, and Mexico which are industrialized to a very substantial degree. And in between lie a large number of countries with widely differing resource bases, economic activities, and energy supply and use patterns.

Despite such important inter-country differences, it is useful to compare the developing countries, both as a group and broadly disaggregated by region, with the industrialized countries. Figure 1 shows the per capita energy consumption levels for each of the developing country regions and for the group as a whole, contrasting them with the industrial countries and with aggregate global data. The wide bars distinguish between energy consumption from commercial sources (oil, gas, coal, hydroelectricity, and nuclear power) and traditional sources (firewood, charcoal, dung, and agricultural residues). Human and animal energy inputs are not included, which, in many developing countries, make significant contributions to total energy supply. In addition to the sharp contrast in energy use per capita between developing and industrial countries, there is a striking difference in the degree of reliance on traditional fuels which in Africa account for the major part of energy use. It is also notable that per capita energy consumption in Latin America is roughly twice that of Asia and nearly three times per capita consumption in Africa.

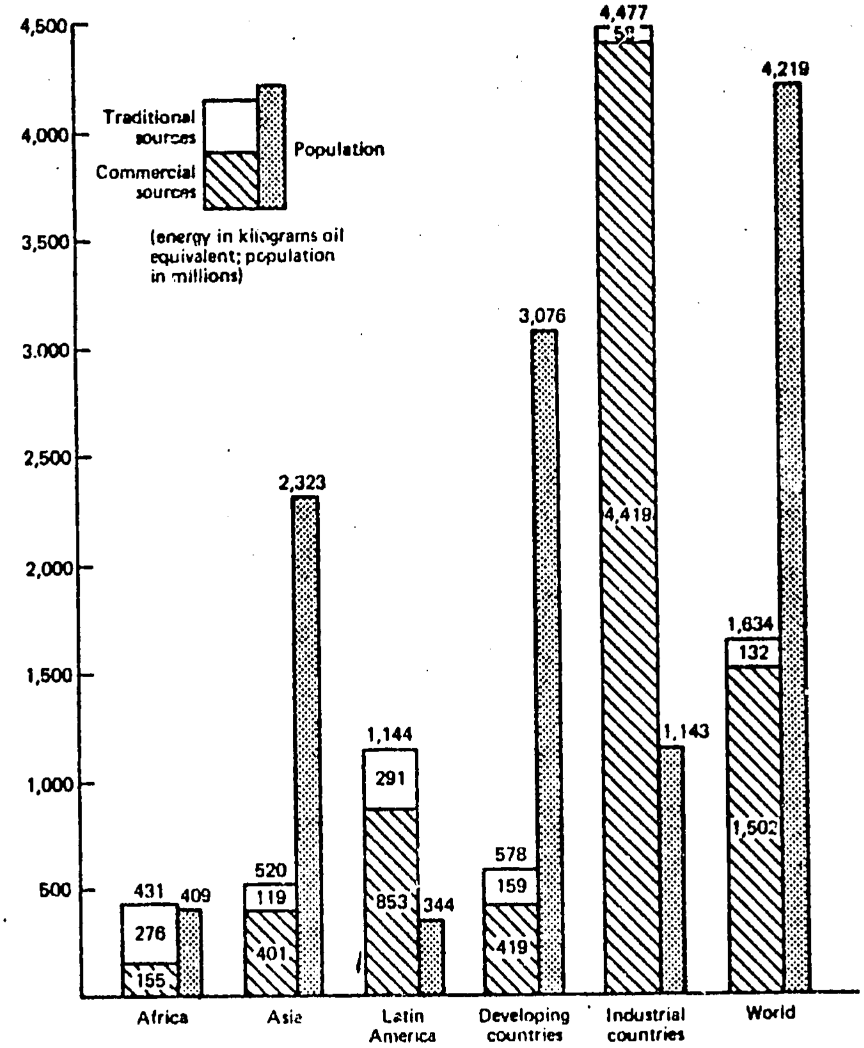

Figure 1. Per Capita Energy Consumption and Population, by Region, 1978. Traditional energy data for 1973 from Jyoti K. Parikh, "Energy and Development," World Bank Public Utilities Report No. PUN 43 (Washington, D.C., August 1978), recalculated on the assumption that per capita traditional energy consumption remained unchanged between 1973 and 1978. Commercial energy data for 1978, except for Taiwan, from United Nations, *World Energy Supplies, 1973–1978*, Series J, No. 22 (New York, 1979), with data on hydroelectricity and nuclear energy recalculated on basis of thermal generation primary energy equivalents. Data for Taiwan from an oral communication from the Coordination Council for North American Affairs to Lincoln Gordon. Population data from Population Reference Bureau, *1978 World Population Data Sheet* (Washington, D.C., 1979). [1]

COMMERCIAL ENERGY

The pattern of current energy consumption and its future demand in the developing countries depends strongly on their overall economic structure and other related socio-demographic factors. It is useful to classify the developing countries into five relatively homogeneous groups according to their degree of development as indicated by per capita income, extent of industrialization and urbanization, as well as by the characteristics of their economic and resource bases.

Table 1 shows a division of 88 developing countries into five groups classified as follows:

I Industrialized

In this group, the industrial sector accounts for the largest share of Gross Domestic Product (GDP). There is relatively more industry and less traditional handicraft production compared to the other groups. However, their modern and predominantly urban industrial sectors usually exist independently from the traditional urban and rural sectors. These countries have the highest fraction of their populations located in urban areas (60%) and, not surprisingly, the highest commercial energy consumption per capita. All are coastal nations with significant trade activities. In common with the other oil importing groups, all the industrial developing countries consume more energy than they produce [4].

II Oil Exporters

Countries in this group have a set of common choices confronting them: the rate of development and use (domestic or export) of their oil resources, and the pattern of investment of their export earnings. Most of these countries face sizeable short-term debt service and/or domestic public spending needs. This group can be expected to grow over the next few years as developing countries receive relatively more emphasis on oil exploration.

III Balanced Growth Economies

This group of countries is characterized by an industrial structure that is relatively well developed but which does not account for the largest share of GDP. Most of the members of this group have already completed the primary and intermediate phases of import substitution and are attempting to become self-sufficient in the heavy industrial sector.

All of the countries in this group are still essentially agrarian societies in that agriculture is the principal activity of a large part of the population. For example, agricultural production still absorbs well over half of the total work force in India, Turkey, and Pakistan. Apart from direct agricultural work, the rural sector provides employment opportunities in the service and traditional industries sectors, so that the vast majority of the population resides in and is dependent on the rural economy.

TABLE 1

DEVELOPING COUNTRY GROUPS

I. Industrialized

Argentina
Brazil
Chile
South Korea
Singapore
Spain
Taiwan
Uruguay
Yugoslavia

II. Oil Exporters

Angola
Bolivia
Congo
Egypt
Indonesia
Malaysia
Mexico
Oman
Syrian Arab Republic
Trinidad and Tobago
Tunisia

III. Balanced Growth Economies

Colombia
Greece
India
Pakistan
Panama
Peru
Philippines
Turkey

IV. Primary Exporters

Botswana
Guinea
Guyana
Jamaica
Liberia
Mauritania
Morocco
Sierra Leone
Surinam
Togo
Zaire
Zambia

V-a. Agricultural Exporters

Costa Rica
Dominican Republic
Gambia
Guatemala
Honduras
Ivory Coast
Senegal
Sri Lanka
Thailand

V-b. Other Agricultural

Afghanistan
Bangladesh
Benin
Burma
Burundi
Cameroon
Central African Empire
Chad
Cyprus
El Salvador
Equatorial Guinea
Ethiopia
Fiji
Ghana
Haiti
Jordan
Kenya
Lebanon
Lesotho
Madagascar
Malawi
Mali
Mauritius
Mozambique
Nepal
Nicaragua
Niger
Papua New Guinea
Paraguay
Rwanda
Somalia
Swaziland
Sudan
Tanzania
Uganda
Upper Volta
Yemen Arab Rep.

Source [2, 3]

IV Primary Exporters

As a group, the primary exporters vary considerably from the industrialized LDCs and those with balanced economies. Their mean GNP per capita figure is considerably lower than the indistrialized developing countries of group I, and they also use considerably less commercial energy than that group. Although these countries are, on the whole, not large energy consumers they still consume more than they produce.

V Agricultural

In this group, the agricultural sector clearly dominates the GDP. This group can be further sub-divided into Agricultural Exporters: countries dependent on the export of one or two agricultural commodities, and Other Agricultural: countries largely characterized by subsistence/farming. The group is generally without significant commercial quantities of mineral resources. Many of these countries are among the poorest of the world. The bulk of energy consumption by these countries is of traditional fuels, primarily wood with some crop residues and animal wastes.

From the energy use point of view, the utility of this classification lies in the relative homogeneity of the energy consumption patterns found in each group. In group I, the industrial sector, here dependent on commercial fuels, dominates the energy use structure while at the other extreme, in group V, energy use in rural households -- mainly food preparation using traditional fuels -- is the principal mode of energy consumption. Table 2 shows commercial energy use, the contribution of petroleum, and its sectoral distribution for each of the five groups. To a first approximation, the growth in per capita commercial energy consumption follows reasonably closely the rise in GNP per capita. Also apparent is the way oil consumption increases as a fraction of commercial energy use from category I to V, suggesting the extreme dependence of the poorer agricultural countries on petroleum products for their entire commercial energy supply. One should note that the figures for category III are distorted by the inclusion of India which is a major consumer of coal.

As far as oil consumption is concerned, it is instructive to note that the industrial and transportation sectors account for roughly equal amounts of this fuel but that, taken together, these two sectors only account for about 70% of total oil consumption.

Tables 3 through 9 show per capita energy consumption patterns for seven developing countries: South Korea, Indonesia, Pakistan, Turkey, Dominican Republic, Thailand, and Sudan. There is at least one representative from each of the developing country groups listed in Table 1.

Table 2

DEVELOPING COUNTRY GROUP CHARACTERISTICS

Group		Number of Countries	Population (Millions)	GNP/cap ($/cap)	Energy/cap GJ/cap(1)	Fraction Oil of Commercial	Fraction Oil by Sector (2) Industry	Fraction Oil by Sector (2) Transport
I	Industrialized	9	262	1120	43	0.67	0.35	0.24
II	Oil Exporters	10	134	740	36	0.69	0.25	0.45
III	Balanced Growth	8	784	221	21	0.41	0.25	0.45
IV	Primary Exporters	13	61	313	19	0.73	0.6	0.3
V	Agricultural	45	398	162	16	0.90	0.2	0.3-0.5

Notes: (1) Commercial energy: not including the traditional fuels.

(2) There is a great deal of variability in these numbers depending on the energy resource base and industrial structure. The data are based on Palmedo [2, 3] and are from 1975.

Table 3 SOUTH KOREA

PER CAPITA ENERGY CONSUMPTION BY SECTOR AND RESOURCE 1973 (GJ/YR)

Group I Industrialized

	Oil	Gas α Coal	Hydro*	Electricity	Non-** Commercial	TOTAL Direct Use
Residential-Commercial	1.57	8.97	----	3.65	4.34	18.53
Industry	6.06	1.11	----	1.64	0.48	9.29
Transportation	4.22	0.02	----	----	----	4.24
Agriculture	0.24	----	----	0.01	----	0.25
Electricity Generation	5.38	0.45	12.38	(5.30)	----	12.91
TOTAL	17.47	10.55	12.38	----	4.82	45.22

* Taken as fossil fuel equivalent

** Crop residues, fuelwood, charcoal, dung

% of oil consumption in total commercial energy consumption 43.2

% of oil consumption in total energy used for electricity generation 29.6

Source: Palmedo, P.F., et. al. [2]
1973 population: 34 million; 53% rural

Table 4

INDONESIA

PER CAPITA ENERGY CONSUMPTION BY SECTOR AND RESOURCE 1973 (GJ/YR)

Group II Oil Exporters

	Oil	Gas & Coal	Hydro*	Electricity	Non-** Commercial	TOTAL Direct Use
Residential-Commercial	0.97	----	----	0.05	8.00	9.02
Industry	0.48	0.28	----	0.02	----	0.78
Transportation	1.15	----	----	----	----	1.15
Agriculture	----	----	----	----	----	----
Electricity Generation	0.22	----	0.15	(0.07)	----	0.30
TOTAL	2.82	0.28	0.15	----	8.00	11.25

* Taken as fossil fuel equivalent

** Crop residues, fuelwood, charcoal, dung

% of oil consumption in total commercial energy consumption 86.7

% of oil consumption in total energy used for electricity generation 58.4

Source: Palmedo, P.F., et. al. [2]
1973 population: 130 million; 81% rural

Table 5 PAKISTAN

PER CAPITA ENERGY CONSUMPTION BY SECTOR AND RESOURCE 1974 (GJ/YR)

Group III Balanced Growth Economies

	Oil	Gas α Coal	Hydro*	Electricity	Non-** Commercial	TOTAL Direct Use
Residential-Commercial	0.67	0.11	----	0.07	4.77	5.62
Industry	0.14	1.71	----	0.16	----	2.01
Transportation	1.16	----	----	----	----	1.16
Agriculture	0.16	----	----	0.06	----	0.22
Electricity Generation	0.13	0.74	0.86	(0.29)	----	1.44
TOTAL	2.26	2.56	0.86	----	4.77	10.45

* Taken as fossil fuel equivalent

** Crop residues, fuelwood, charcoal, dung

% of oil consumption in total commercial energy consumption 39.7

% of oil consumption in total energy used for electricity generation 7.5

Source: Palmedo, P.F., et. al. [2]
1974 population: 69 million; 73% rural

Table 6 TURKEY

PER CAPITA ENERGY CONSUMPTION BY SECTOR AND RESOURCE 1975 (GJ/YR)

Group III Balanced Growth Economies

	Oil	Gas α Coal	Hydro*	Electricity	Non-** Commercial	TOTAL Direct Use
Residential-Commercial	2.63	0.90	----	0.38	6.39	10.30
Industry	2.55	0.97	----	1.00	----	4.52
Transportation	5.39	0.53	----	0.01	----	5.93
Agriculture	1.05	----	----	----	----	1.05
Electricity Generation	1.53	1.10	1.74	(1.39)	----	2.98
TOTAL	13.15	3.50	1.74	----	6.39	24.78

* Taken as fossil fuel equivalent

** Crop residues, fuelwood, charcoal, dung

% of oil consumption in total commercial energy consumption 71.5

% of oil consumption in total energy used for electricity generation 35.0

Source: Palmedo, P.F., et. al. [2]
1975 population: 40.2 million; 57% rural

Table 7 DOMINICAN REPUBLIC

PER CAPITA ENERGY CONSUMPTION BY SECTOR AND RESOURCE 1977 (GJ/YR)

Group V-a Agricultural Exporters

	Oil	Gas α Coal	Hydro*	Electricity	Non-** Commercial	TOTAL Direct Use
Residential-Commercial	0.96	----	----	0.70	2.69	4.35
Industry	6.29	----	----	0.42	5.34	12.05
Transportation	3.71	----	----	----	----	3.71
Agriculture	0.32	----	----	----	----	0.32
Electricity Generation	5.32	----	0.56	(1.12)	----	4.76
TOTAL	16.60	----	0.56	----	8.03	25.19

* Taken as fossil fuel equivalent

** Crop residues, fuelwood, charcoal, dung

% of oil consumption in total commercial energy consumption 96.7

% of oil consumption in total energy used for electricity generation 90.5

Source: Palmedo, P.F., et. al. [2]
1977 population: 4.98 million; 56% rural

Table 8 THAILAND

PER CAPITA ENERGY CONSUMPTION BY SECTOR AND RESOURCE 1976 (GJ/YR)

Group V-a Agricultural Exporters

	Oil	Gas α Coal	Hydro*	Electricity	Non-** Commercial	TOTAL Direct Use
Residential-Commercial	0.72	----	----	0.26	10.49	11.47
Industry	1.78	0.08	----	0.47	0.88	3.21
Transportation	3.92	----	----	----	----	3.92
Agriculture	0.94	----	----	----	----	0.94
Electricity Generation	1.40	0.16	1.11	(0.73)	----	1.94
TOTAL	8.76	0.24	1.11	----	11.37	21.48

* Taken as fossil fuel equivalent

** Crop residues, fuelwood, charcoal, dung

% of oil consumption in total commercial energy consumption 86.6

% of oil consumption in total energy used for electricity generation 52.3

Source: Palmedo, P.F., et. al. [2]
1976 population: 42.6 million; 83% rural

Table 9

SUDAN

PER CAPITA ENERGY CONSUMPTION BY SECTOR AND RESOURCE 1975 (GJ/YR)

Group V-b Other Agricultural

	Oil	Gas α Coal	Hydro*	Electricity	Non-** Commercial	TOTAL Direct Use
Residential-Commercial	0.26	----	----	0.07	16.18	16.51
Industry	0.35	----	----	0.08	0.15	0.58
Transportation	1.33	----	----	----	----	1.33
Agriculture	0.46	----	----	0.02	----	0.48
Electricity Generation	0.21	----	0.34	(0.17)	----	0.38
TOTAL	2.61	----	0.34	----	16.33	19.28

* Taken as fossil fuel equivalent

** Crop residues, fuelwood, charcoal, dung

% of oil consumption in total commercial energy consumption 88.5

% of oil consumption in total energy used for electricity generation 38.4

Source: Palmedo, P.F., et. al. [2]
1975 population: 15.6 million; 87% rural

THE TRADITIONAL FUELS

About 25% of the energy consumed in developing countries is provided by the traditional fuels, and approximately half the world's population rely primarily on traditional fuels for their direct energy needs. The traditional fuels include:

1. Wood fuels -- firewood and charcoal
2. Animal wastes -- dung from cattle and other animals
3. Crop residues -- such as straw and bagasse.

Wood is the principal fuel in the rural areas of developing countries. Charcoal is generally more popular in urban areas because of its convenience and ease of transportation. Crop residues and dung are usually only resorted to when wood fuels are unavailable or are too costly. In countries in the early stages of industrial development the traditional fuels may constitute 80-90% of total energy consumption.

Since most traditional fuels are not traded in commerce, estimates of their consumption are necessarily very approximate. The FAO estimates wood and charcoal consumption in all developing countries at over 1 billion cubic metres per year. The use of animal dung as a fuel amounts to about 400 million tons annually. Reliance on traditional fuels is heavier in poorer countries and in rural areas, among the urban poor relative to the urban non-poor, and geographically in Africa and Asia. Table 10 gives a general picture of the extent of national reliance on traditional fuels.

The traditional fuels are, of course, renewable sources of energy. But these fuels are not available in unlimited quantities. Deforestation is now a serious problem in many of the poorer developing countries where reliance on fuel wood and charcoal as energy sources is heaviest. Part of the problem with the traditional fuels is that they are used very inefficiently, so the amount of fuel consumed to cook food, or heat or light a dwelling is very much more than is theoretically required. The introduction of more efficient cookstoves, charcoal kilns, lamps and heating devices could substantially reduce the pressure on forest biomass resources.

Tables 11 and 12 show the contribution that traditional sources of energy make to six Central American countries and five Asian countries. The data for Bangladesh and India show the significant contribution that human and animal labour make in these countries, chiefly in agricultural production. Table 13 shows the sectoral distribution of energy consumption for one of the poorest of the developing countries -- Nepal. This country, with 95% of its population in the rural areas, is almost completely dependent on firewood as a source of energy.

Table 10. Estimated National Reliance on Traditional Fuels, 1976
(each group arranged in ascending order of per capita GNP) [6]

Modest reliance (less than half)	Medium reliance (approximately one-half to three-quarters)	Heavy reliance (three-quarters or more)
Pakistan (22)	Togo (67)	Benin (86)
Mauritius (2)	India (28)	Burundi (89)
Morocco (22)	Indonesia (62)	Cameroon (82)
Rhodesia (Zimbabwe) (36)	Sri Lanka (55)	Cape Verde (NA)
China (9)	Vietnam (55)	Central African Empire (91)
N. Korea (<1)	Gabon (44)	Chad (94)
S. Korea (8)	Liberia (53)	Ethiopia (93)
Philippines (<1)	Mauritania (63)	Gambia (73)
Ecuador (20)	Senegal (63)	Guinea (74)
Albania (24)	Zambia (45)	Guinea Bissau (87)
Algeria (4)	Thailand (34)	Kenya (74)
Tunisia (25)	Bolivia (45)	Lesotho (NA)
Iran (1)	Colombia (37)	Madagascar (80)
Lebanon (2)	El Salvador (53)	Malawi (82)
Argentina (3)	Guatemala (60)	Mali (97)
Chile (14)	Honduras (64)	Mozambique (74)
Cuba (5)	Malaysia (25)	Niger (87)
Dominican Republic (19)	Mongolia (25)	Rwanda (96)
Guadaloupe (19)	Brazil (38)	Sierra Leone (76)
Mexico (4)	Costa Rica (50)	Somalia (90)
Panama (29)	Nicaragua (47)	Sudan (81)
Peru (20)		Tanzania (94)
Uruguay (13)		Uganda (91)
Fiji (2)		Upper Volta (94)
Cyprus (NA)		Zaire (76)
Malta (NA)		Afghanistan (76)
Portugal (3)		Bangladesh (63)
Romania (2)		Bhutan (NA)
Turkey (18)		Burma (85)
Yugoslavia (4)		Cambodia (93)
Libya (5)		Laos (87)
Hong Kong (NA)		Nepal (96)
Israel (NA)		Yemen (NA)
Singapore (NA)		Haiti (92)
Bahamas (NA)		Angola (74)
Venezuela (8)		Botswana (NA)
		Congo (80)
		Eq. Guinea (86)
		Ghana (74)
		Nigeria (82)
		Swaziland (NA)
		Paraguay (74)
		Papua New Guinea (66)

Notes: Country Reliance classified according to wood fuels plus estimated dung and crop wastes as a percentage of total energy consumption. Figures in parentheses are wood fuels alone as a percentage of total energy consumption in each country. Egypt, Iraq, Syria, Bahrain, Brunei, Kuwait, Oman, Qatar, Saudi Arabia, United Arab Emirates were not classified. NA = not available.

ENERGY SOURCE	COSTA RICA 1978	NICARAGUA 1977	HONDURAS 1977	GUATEMALA 1977	EL SALVADOR 1978	PANAMA 1977
Commercial	26.0	17.4	10.2	8.6	11.5	43.1
Traditional*	9.8	8.2	7.0	7.2	6.7	7.9
TOTAL (GJ)	35.8	25.6	17.2	15.8	18.2	51.0

*Bagasse, firewood and agricultural wastes. Does not include human and animal power.

Table 11. Approximate Annual Energy Use Per Capita in Six Central American Countries (Gigajoules)

Source: adapted from reference 5.

ENERGY SOURCE	CHINA 1977	BANGLADESH 1978	INDIA 1978	SRI LANKA 1978	THAILAND 1977
	COMMERCIAL SOURCES				
Coal and lignite		0.1	3.4	0.02	0.2
Oil and gas	12.5	1.2	1.6	2.4	9.1
Hydro and nuclear		0.1	0.2	0.2	1.0
TOTAL (Commercial)	12.5	1.4	5.2	2.6	10.3
	TRADITIONAL SOURCES				
Firewood and charcoal	1.7	1.3	7.0	4.8	10.9
Other biomass	3.0	5.5		---	1.2
Human labour	1.3	1.0	1.3	---	---
Animal labour	---	0.5	3.3	---	---
TOTAL (Traditional)	6.0	8.3	11.6	4.8	12.1
TOTAL ENERGY USE (GJ)	18.5	9.7	16.8	7.4	22.4

Table 12. Approximate Annual Energy Use Per Capita in Five Asian Countries (Gigajoules)

Source: Revelle, adapted from reference 7.

Table 13 PER CAPITA ENERGY CONSUMPTION AND DISTRIBUTION IN NEPAL, 1978-79

SECTOR	PERCENTAGE OF TOTAL USE	FUEL CONSUMPTION GJ/YR						TOTAL SECTORAL USE
		FIREWOOD	CROP RESIDUES	ANIMAL DUNG	COAL AND COKE	PETROLEUM FUELS	ELECTRICITY	
TRANSPORTATION	2.2	---	---	---	0.01	0.18	---	0.19
DOMESTIC	95.0	7.92	0.16	0.06	---	0.09	0.02	8.25
AGRICULTURAL	0.2	---	---	---	---	0.01	---	0.01
COMMERCIAL/ INDUSTRIAL	2.4	0.06	---	---	0.10	0.03	0.02	0.21
OTHER/LOSSES	0.2	---	---	---	---	---	0.02	0.02
TOTAL FUEL USE		7.98	0.16	0.06	0.11	0.31	0.06	8.68

SOURCE: Country paper of Nepal, Submitted to the U.N. Conference on New and Renewable Sources of Energy, Nairobi, 1981. Figures do not include human and animate energy. The population of Nepal is 15 million; 95% of the population live in the rural areas.

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RURAL ENERGY USE

A most important distinction between the industrialized and developing countries concerns the structure of energy supply and the pattern of energy consumption in rural areas. In the industrialized nations there is very little difference in the type of energy used in rural as opposed to urban areas. All areas generally have access to the principal fuels: electricity, gasoline, diesel fuel, heating oil and fuel gas; only the intensity of use changes since industrial activity is predominantly an urban phenomenon.

This relative homogeneity is not the case in the developing nations. The structure of energy supply, the sources of energy utilized, and the tasks for which the energy is used, all show marked differences when the urban areas are compared with the rural. These regional differences have important implications for energy policy.

Most energy in rural communities is locally produced from human and animal labour, wood fuels, and animal and crop residues, with commercial fuels being used on a limited scale. Traditional fuels are usually gathered by family members, although wealthier families may purchase charcoal, dung cakes, or wood, and the poor may have to pay with services for the privilege of gathering firewood or residues on land that is privately owned. Much firewood is gathered, not from forests, but from trees scattered along roads and fields, intercropped with agricultural crops, or in gardens and yards.

Table 14 clearly illustrates the differences between the urban and rural sectors, and between the traditional and commercial fuels, for India and Bangladesh. Although per capita energy use is higher in the urban areas (as one might expect), it is in the rural areas that the greater part of the total energy consumption occurs. Furthermore, the energy sources utilized in the rural areas are dominated by the traditional fuels.

This last characteristic is illustrated further in Tables 15 and 16. In all cases, except that of Northern Mexico, the principal sources of energy used in the rural areas are the traditional fuels. In addition, Table 16 shows that most of the energy consumed is taken by the domestic sector, principally for cooking. Again, the exception to this generalization is Northern Mexico - a relatively developed region.

Finally, the most detailed analysis of energy supply and consumption in the rural areas of India is provided by Table 17. Fully 64% of total energy use is consumed by domestic activities and of this amount 98% is traditional fuels. Agriculture accounts for 22% of total energy use and more than three-quarters of this is supplied by animate energy - human and bullock work.

Commercial energy account for 10.5% of total energy; mainly for agriculture (mostly for fertilizers) and for lighting.

Table 14 ENERGY USE IN INDIA AND BANGLADESH BY SECTOR AND FUEL TYPE

SECTOR	INDIA per capita GJ/yr	INDIA total use EJ/yr	BANGLADESH per capita GJ/yr	BANGLADESH total use EJ/yr
	URBAN			
Commercial	23.2	2.55	3.9	0.02
Traditional	7.5	0.82	5.4	0.03
TOTAL	30.7	3.37	9.3	0.05
	RURAL			
Commercial	1.1	0.47	0.5	0.04
Traditional	9.1	3.99	7.0	0.48
TOTAL	10.2	4.46	7.5	0.52

SOURCE: Revelle, adapted from reference 10.

Table 15. Estimated Per Capita Use of Energy in Rural Areas of Seven Developing Countries (GJ)

	INDIA	CHINA, HUNAN	TANZANIA	NORTHERN NIGERIA	NORTHERN MEXICO	BOLIVIA	BANGLADESH
Human Labour	1.0	1.0	1.0	0.9	1.1	1.1	1.0
Animal Work	1.5	1.5	---	0.2	2.0	2.8	1.5
Fuel Wood	4.5		23.0	15.7	14.8	34.9	1.4
Crop Residues	1.8	20.9					2.5
Dung	1.0						0.9
TOTAL TRADITIONAL	9.8	23.4	24.0	16.8	17.9	38.8	7.3
Coal, Oil, Gas and Electricity	0.8	3.1	---	0.03	30.3	---	0.4
Chemical Fertilizers	0.3	0.5	---	0.08	8.2	---	0.2
TOTAL COMMERCIAL	1.1	3.6	---	0.11	38.5	---	0.6
TOTAL ALL SOURCES	10.9	27.0	24.0	16.9	56.4	38.8	7.9

Source: Revelle, adapted from reference 10.

Table 16. Characteristics of Energy Use in Rural Areas of Seven Developing Countries (GJ)

(per capita)	INDIA	CHINA, HUNAN	TANZANIA	NORTHERN NIGERIA	NORTHERN MEXICO	BOLIVIA	BANGLADESH
Total use	10.9	27.0	24.0	16.9	56.4	38.8	7.9
Domestic uses	7.4	21.3	23.4	16.0	18.1	35.3	5.5
Non-domestic uses	3.5	5.7	0.6	0.9	38.3	3.5	2.4
Domestic/non-domestic uses	2.1	3.7	37.4	18.4	0.47	10.1	2.2
Traditional/commercial	8.5	6.3	*	157.3	0.47	*	13.0

*no commercial energy used.

Source: Revelle, adapted from reference 10.

Table 17. Estimated Energy Use in Rural India
(Gigajoules per year per capita)

Source of energy	Agriculture	Domestic activities	Lighting	Pottery brick making metal work	Transportation and other uses	Total	Percentage
Traditional sources:							
Human labor	0.56	0.37	---	0.01	0.09	1.03	9.5
Bullock work	1.28	---	---	---	0.25	1.53	14.1
Firewood and charcoal	---		---		---	4.37	40.3
Cattle dung	---	6.45	---	0.71	---	1.77	16.3
Crop residues	---		---		---	1.02	9.4
Total traditional	1.84	6.82	---	0.72	0.34	9.72	89.5
Commercial sources:							
Petroleum and natural gas							
Fertilizer	0.33	---	---	---	---	0.33	3.1
Fuel	0.08	---	0.40	---	---	0.48	4.4
Soft coke	---	0.13	---	---	---	0.13	1.2
Electricity:							
Hydro	0.03	---	0.01	---	---	0.04	0.4
Thermal	0.11	---	0.05	---	---	0.16	1.5
Total commercial	0.55	0.13	0.46	---	---	1.14	10.5
TOTAL RURAL ENERGY USE	2.40	6.95	0.46	0.72	0.34	10.86	100.0
Activities as a percentage of total energy consumption	22.0	64.0	4.0	7.0	3.0	100.0	100.0

Note: Dashes = not applicable. Figures may not reconcile exactly due to rounding.
Source: Adapted from Revelle, reference 10.

ANIMAL ENERGY

In a number of countries, principally in Asia, a large part of rural energy use is in the form of animate energy - animal and human labour. This is particularly true in the agricultural sector, where the majority of people in developing countries are employed. Table 18 shows the livestock population of some developing countries in Asia [9].

Table 18 1978 LIVESTOCK POPULATION (MILLIONS)

	Cattle	Horses	Asses and Mules
India	180.3	0.9	1.1
China	64.0	6.9	12.1
Pakistan	14.4	0.4	1.9
Bangladesh	28.0	0.04	n.a
Burma	7.3	0.1	n.a
Afghanistan	3.7	0.4	1.3
TOTAL	297.7	8.7	16.4

n.a. = not available

Animals are used for plowing, lifting water, irrigation, sugarcane sugarcane crushing, chaff-cutting, oil extraction, and similar tasks. Animals represent a considerable source of power. On average, approximately 1/2 hp or 375 W can be obtained continuously over an 8 hour period from a medium-sized bullock or buffalo. Assuming about a quarter of the animals listed in Table 18 are work animals then the peak power output is of the order of 30,000 MW. Unfortunately, this significant, decentralized source of power is used very inefficiently.

An important consideration in countries with large numbers of livestock is the potential utilization of animal dung in anaerobic digestors to produce biogas. In some Asian countries, notably China and India, biogas makes a very significant contribution to rural energy supplies.

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URBAN AND INDUSTRIAL ENERGY USE

Small rural and urban industries, some large modern sector industries, and the urban poor are also important users of traditional fuels. In urban households, commercial fuels are commonly used together with traditional fuels, both of which are sold in organized markets. Charcoal is generally preferred to wood in cities because of its convenience, compactness, and cleaner burning, and surveys in Asia and Africa have found per capita consumption of wood fuels (including charcoal) in towns to be higher than in the countryside, probably because of relatively higher incomes in urban areas. In low-income urban areas, per capita demand for wood fuels can be quite high, caused by the greater use of charcoal, which usually requires a larger raw material input. As incomes rise, however, commercial fuels are generally substituted for wood fuels in urban areas.

Industrial use of traditional fuels is also quite extensive. Estimates of non-household consumption of wood for energy in surveyed areas of Africa and Asia vary from 2 to 25% of total wood consumption. As Table 19 below shows, in a number of countries industrial consumption of traditional fuels is not only large but rising both in absolute terms and also as a share of traditional fuel use. The share of traditional fuels in total industrial energy consumption, however, has generally decreased, reflecting the expansion of modern industry. Some industries, sensitive to price changes, are likely to continue to rely on or even revert to the use of traditional fuels if the price of commercial fuels increases.

Table 19. Industrial Consumption of Traditional Fuels in Selected Countries, 1967–1977
(absolute figures in thousand metric tons oil equivalent (ttoe))

	1967			1973			1976		
Countries	ttoe	Percent total traditional fuels[a]	Percent total industrial energy[b]	ttoe	Percent total traditional fuels[a]	Percent total industrial energy[b]	ttoe	Percent total traditional fuels[a]	Percent total industrial energy[b]
Argentina	1,070	51	20	1,532	69	22	3,700	80	38
Brazil	2,825	12	28	4,459	19	23	4,166	15	17
Colombia	197	3	73	267	5	55	309	6	29
Egypt	120	84	17	189	87	21	190	87	14
India	778	3	5	1,316	5	4	1,661	5	4
Indonesia	203	1	58	289	1	46	455	2	40
Iran	151	29	9	215	45	2	215	31	2
Mexico	796	26	5	927	30	4	894	30	3
Thailand	153	52	20	456	84	19	869	93	28
Venezuela	131	8	3	277	14	4	300	14	4

Source: International Energy Agency/Organisation for Economic Co-operation and Development, *Workshop on Energy Data of Developing Countries*, vol. II *Basic Energy Statistics and Energy Balances of Developing Countries, 1967–1977* (Paris, OECD, 1979). Many of these figures must be treated with caution; a relatively large proportion of consumption of many fuels is often not allocated by sector.

[a] The percentage of all traditional fuel consumption that is consumed by the industrial sector.

[b] The percentage of all industrial sector energy consumption that is traditional fuels.

Wood and charcoal are used in brick and tile making, cement and metal industries, crop drying, bread baking and fish curing. Tobacco curing appears to account for 17% of total annual energy consumption in Malawi, or 1 million cubic metres of fuelwood a year. The Ugandan tea industry and railways in Thailand are also heavy users of wood fuels. Other important industries using traditional fuels are some steel mills in Brazil, Argentina and the Philippines, which use charcoal rather than coal, and sugar mills (and in some cases, sugar refineries), which are able to be self-sufficient in energy by using bagasse to provide heat for evaporation and sometimes to produce electricity.

THE MODERN SECTOR

In contrast to the traditional sector of developing countries, the modern sector has always depended on commercial fuels for its principal sources of energy. Almost every developing country has at least a small modern sector, typically including its administrative capital, its ports, and some industrial activity in mining, plantation agriculture, food processing, and manufacture of light consumer goods. The more industrialized developing countries have large urban-industrial complexes, manufacturing both consumer and capital good and providing an array of commercial services that make their modern sectors strikingly similar to those of the industrial countries.

The rapid pace of economic development in most of the developing world since the 1950's has been accompanied by an even greater increase in the use of commercial energy. The developing countries have greatly outpaced the industrial countries in the growth of energy consumption since 1965 and especially since 1973. The vast portion of the increase, moreover, was in the form of oil. Between 1960 and 1978, the share of oil in the commercial energy supplies of developing countries rose from 24 to 42 percent, and when China and North Korea are excluded, from 56 to 62 percent [1]. Natural gas is currently a significant source of energy only for developing countries with substantial associated oil production--mainly members of OPEC. Hydroelectricity supplies only a modest share overall, but provides a substantial proportion of total electricity output and is particularly important in some major countries such as Brazil and India.

The sharp increases in commercial energy consumption are natural concomitants of the changes in economic structure involved in development. In the early stages, these changes typically include the commercialization of agriculture, the introduction of industry for processing raw materials and supplying light consumer goods, the shift of labour from agricultural to industrial and urban service occupations, the growth of urban settlements and the mechanization of transportation. Most of the activities connected with these growing sectors usually require the use of commercial as opposed to traditional fuels.

FUTURE ENERGY SUPPLY

The rapid increase in world oil prices beginning in the early 1970s marked the start of a major transition in energy supply and use patterns. This transition must be expected to culminate ultimately in the widespread use of renewable energy technologies based on hydropower, biomass, wind energy, solar energy and perhaps nuclear power. At the present time, these technologies make only a small contribution to commercial energy supply in the industrial countries. It will require several decades before new and renewable sources of energy account for a dominant portion of their energy supply structure. According to one scenario [12] during the next twenty years one can anticipate:

- Economic growth will be significantly lower than in the 1965 to 1973 period and slightly lower than the 1973 to 1979 experience. Adjusted for inflation, the world economy as a whole is expected to grow about 3 percent annually between 1979 and 2000, compared to more than 5 percent per year between 1965 and 1973.

- Real energy costs are likely to rise throughout the period as a result of a limited supply of conventional oil and the high cost of most alternate energy sources.

- World energy demand is expected to grow about 2 1/2 percent per year, less than world economic growth rate. Even at this lower growth rate, world energy demand will increase substantially by the year 2000.

- Only a modest increase in world production of conventional oil is anticipated. Volumes available for international trade are projected to show a net decline as oil-exporting countries increase domestic consumption. Consequently, neither the industrial countries, nor the developing countries, can rely on conventional oil for increases in their energy requirements.

- Most of the growth in the industrial, residential and commercial sectors, where consumers have a choice of fuels, is projected to come from coal and from nuclear energy.

- Oil use will be concentrated increasingly in specialized applications, including transportation, specialty products, such as lubricants, and some other demands for which large-scale substitution of other fuels is not yet considered practical.

- Production of synthetic fuels, especially liquids, will be needed during the late 1980s and in the 1990s to meet demands for transportation and other uses for which fuel substitution opportunities are limited.

For many years prior to 1973, oil and gas provided most of the growth in world energy supply. Oil penetrated all sectors of the world economy. Between 1965 and 1973, oil supply grew almost 8 percent per year and gas more than 7 percent, rates much higher than the rate at which total energy supply was increasing. By 1973, oil and gas accounted for two-thirds of total world energy supply.

Developments since that year have set in motion a dramatic transition in the mix of primary fuels, as illustrated in Figure 2 . Conventional oil supply is expected to grow less than 1 percent per year through 2000 resulting in a decline in oil's share of energy supply from 47 percent in 1979 to 31 percent by the year 2000. Most future energy growth will have to be supplied from other energy sources. All non-oil energy sources have projected growth rates above that for total energy supply, as shown in the table below.

ENERGY SUPPLY GROWTH, PERCENT PER YEAR

	1965-1973	1973-1979	1979-2000
Oil	7.7	2.2	0.4
Synthetics & VHO	—	—	13.8
Gas	7.3	3.6	2.6
Coal	1.0	2.4	2.8
Nuclear	27.8	20.9	10.0
Hydro & Other	3.9	4.6	3.5
Total	5.3	2.9	2.4

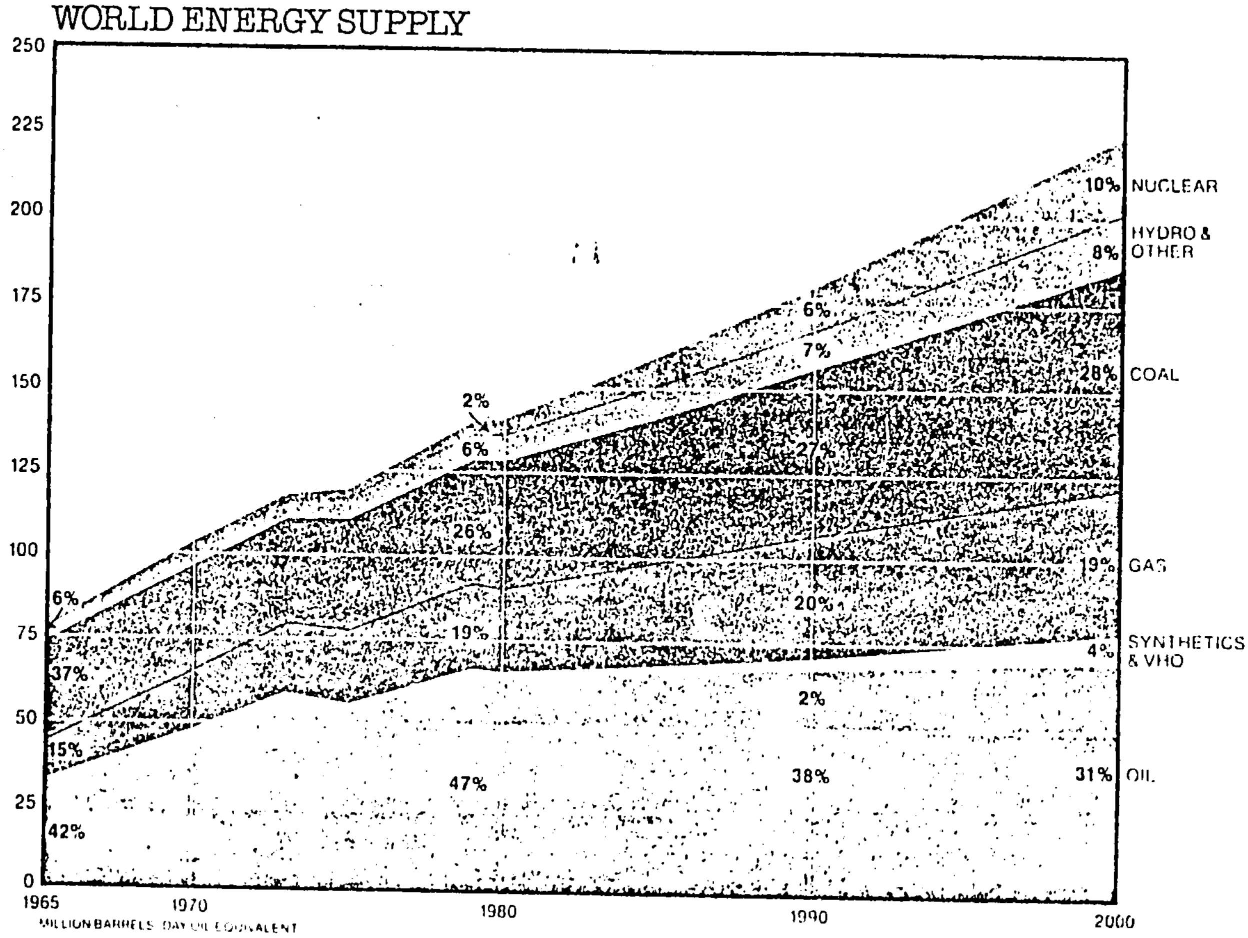
Conventional natural gas supply is projected to keep pace with overall energy use, maintaining its share of world energy supply at about 20 percent. In the 1990s, projected growth in world gas supply will require the development of reserves in remoted areas and the construction of expensive distribution systems to bring the gas to markets. Given favorable prices and supportive government policies, the necessary volumes should be available. Indeed, world gas reserves in the 1990s would probably be sufficient to support consumption above projected levels, should conditions prove to be favorable.

Coal, which grew slowly between 1965 and 1973, is projected to be a major source of energy supply growth. Coal is expected not only to meet a substantial share of new energy demand, but also to replace oil and gas in major industrial and electric utility markets. Coal use is projected to grow almost 3 percent per year, increasing its share of world energy supply from 26 percent in 1979 to 28 percent by 2000. (If the coal converted to synthetic oil and gas were included here, coal's share would increase to 30 percent by 2000.) At that level, coal will rival oil as the single largest source of energy, but world coal resources still would be large relative to production rates. Coal use is expected to be constrained by the growth in demand rather than by the availability of supply.

Most coal will continue to be consumed in the country in which it is produced, with the largest increases occurring in the United States and the centrally-planned economies. Some countries, however,-particularly in Europe and East Asia-are likely to import substantial volumes of coal. Exports from Australia, South Africa, Colombia, the U.S. and other coal producers are likely to increase rapidly. Seabourne coal trade is expected to quadruple by 2000 to over 600 million tons per year.

FIGURE 2.

WORLD ENERGY SUPPLY



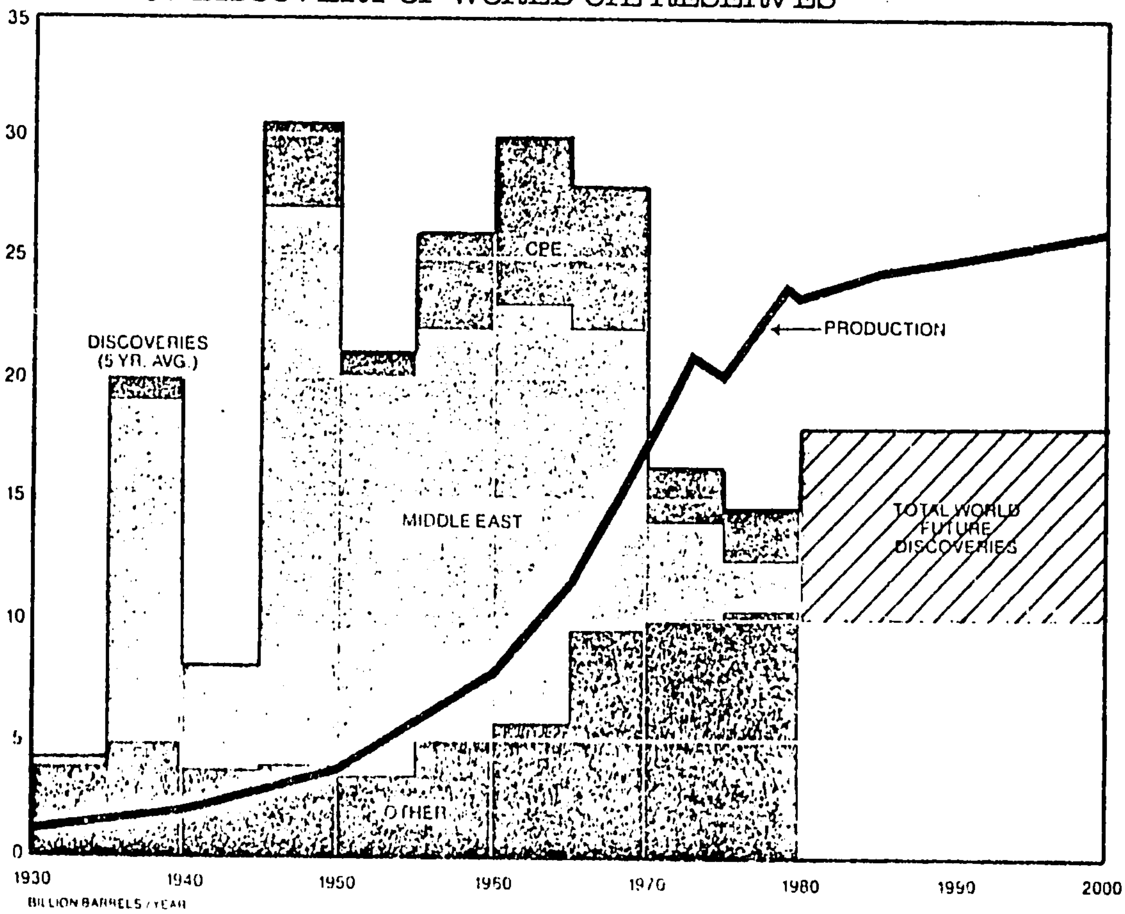
OIL— A DWINDLING RESOURCE

For some years now, the world has been consuming more oil than it has been finding. During the quarter of a century prior to 1970, substantial discoveries, principally in the Middle East, had built up the world's inventory of discovered reserves, as illustrated in Figure 3. In the early 1970s, however, the situation reversed. Smaller discoveries and a continuing rise in oil production caused the inventory of discovered reserves to decline. This pattern is expected to continue in the future, despite a much slower rate of growth in oil consumption and the increased incentives to discover oil provided by rising energy prices.

There is little doubt that finding and developing the world's as yet undiscovered oil reserves will be progressively more difficult and costly. Many prospects are in remote locations or harsh operating environments, such as the Arctic, which will be technologically demanding and will require long lead-times for development. Fields remaining to be discovered in areas where production already exists are anticipated to be somewhat smaller, on average, than past discoveries. Moreover, the number of unexplored areas is steadily diminishing.

FIGURE 3.

RATE OF DISCOVERY OF WORLD OIL RESERVES



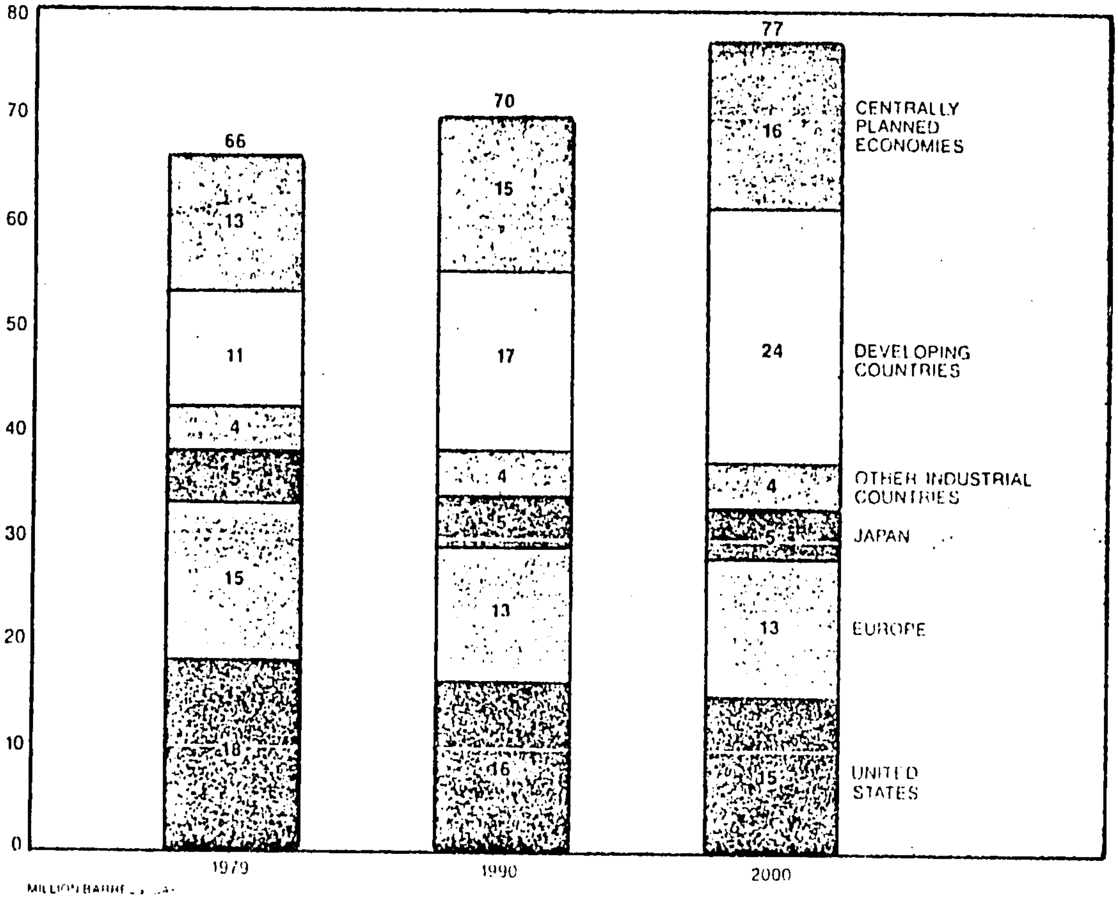
Thus, even with an active exploration effort, the average oil discovery rate for the outlook period is likely to be well below the expected production of 24 to 25 billion barrels per year and, consequently, the world's inventory of discovered oil reserves will continue to decline. Since production cannot increase indefinitely in the face of declining discovered reserves, it seems reasonable to expect that conventional oil production will reach a plateau some time shortly after the turn of the century.

Until the early 1970s, world oil demand expanded rapidly, growing at a rate substantially greater than for total energy demand. Since 1973, however, growth has averaged only about 2 percent per year and demand is expected to increase at less than one percent per year over the next twenty year period. This reversal will be most pronounced in the industrial countries which, in the past, were chiefly responsible for the rapid expansion in oil demand. As shown in Figure 4, oil demand in the major industrial countries is projected to decline as a result of conservation, efficiency improvements, and substitution of alternate liquid fuels.

However, the noteworthy feature of this projection is the significant demand for oil expected to be exerted by the developing countries.

FIGURE 4.

WORLD OIL DEMAND



PROSPECTS FOR THE TRADITIONAL FUELS

Besides the difficulty and expenses of securing adequate supplies of oil, the other problem with energy supply for the developing countries concerns the traditional fuels - principally fuel wood.

In most cases, people only burn crop or animal residues when fuel wood is unavailable or expensive. Because wood cannot usually be economically transported over long distances, large demands for wood fuels by urban people and industrial users can rapidly stress forest resources in the locality.

The fuel wood situation is already critical. Although globally, about 97 million hectares, or 2 percent of existing forests were added between 1965 and 1975, tropical forests are under much greater stress. The tropical forest areas are being lost at a rate of about 20 million hectares each year. The preliminary results of a study of fuelwood supply and needs indicate that about 100 million people in developing countries live in areas where there is already an acute shortage of fuel wood. Another 1 billion are able to meet their minimum fuel wood requirements only by cutting in excess of the sustainable yield. According to this report [13], with current trends of population growth, of fuel wood demand, and rates of depletion of tree resources, over 2 billion rural people in developing countries will need to be provided with large supplies of traditional fuels within two decades.

However, it is more likely that the overall rate of deforestation in the developing countries will decline before the turn of the century, for the simple reason that the people who are doing most of the cutting will eventually run out of forests to cut. Populations and forest resources are not evenly distributed. Some countries cleared all their forest lands accessible to them years ago (Afghanistan, for example), other densely populated nations that still have substantial forest resources will have lost most of them before the year 2000 (Indonesia and Thailand, for example), and some sparsely populated nations with very large areas of forest will still have vast forests in the year 2000 (Gabon and Congo, for example) [14].

Table 20 below summarizes the forecast for forest resources by global region for the year 2000. It should be noted that these figures are considered a "mildly optimistic" scenaria [14].

Table 20 WORLD FOREST RESOURCES

REGION	1978	2000	Change
	(million hectares)		(percent)
DEVELOPED COUNTRIES			
USSR	785	775	-1
North America	470	464	-1
Europe	140	150	+7
Australia, Japan, and New Zealand	69	68	-1
TOTAL	1464	1457	0
DEVELOPING COUNTRIES			
Latin America	550	329	-40
Asia and Pacific	361	181	-50
Africa	188	150	-20
TOTAL	1099	660	-40
WORLD TOTAL	2563	2117	-17

The depletion of fuelwood supplies has causes other than the constant demand for cooking fuel. The spread of agriculture is in fact the principal cause of deforestation. As populations grow, larger areas of forest will be cleared to create land for cultivation or grazing. The results, particularly in densely settled areas, include soil erosion, flash flooding, drying up of previously perennial streams, and eventual desertification.

The problem is further compounded by the substitution of animal dung and crop residues for scarce fuelwood supplies. Because commercial fertilizers are unavailable or too expensive for most villagers, the diversion of dung and crop residues from the land contributes to declining agricultural productivity. Yields diminish thus creating additional pressure to bring more land under cultivation for subsistence crops. This involves felling more trees and the cycle is perpetuated.

RENEWABLE ENERGY RESOURCES

All countries utilize renewable sources of energy to some extent. In the industrial countries the contribution of the renewable energy sources to total energy consumption is generally small--less than 5 percent--and is predominantly hydropower. In the developing countries the contribution made by renewable sources of energy is much higher. On the average, about 30% of the total energy supply in the developing countries is derived from renewable energy sources, principally biomass: fuelwood, charcoal, and animal and crop residues. Some countries have a very high dependence on renewable energy sources. Nepal, for example, gets fully 94¢ of its total energy supply from biomass sources.

The degree to which the renewable sources of energy will make a contribution to the energy supply in the developing countries in the future is very hard to predict. So much will depend on government policy, committment, initiative, and innovation, not to mention the price of petroleum. What we can at least do here is to ascertain, to a first approximation, whether the renewable energy resources are available, since without on adequate resource base the renewable sources of energy are hardly likely to make a significant impact on future energy supply. Table 21 indicates the potential of biomass and hydropower to supply energy in the developing countries. Also shown are data on commercial energy and traditional fuel consumption. Table 22 compares current energy consumption with the potential renewable energy supply from biomass and hydropower. Approximately 86% of the developing countries listed could provide all their current energy requirements from these two renewable energy sources alone. When one further considers the very large potential contribution of direct solar thermal energy, the increasing utilization of photovoltaic systems, and the possibility in many developing countries of exploiting the wind, the immense potential of the renewable sources of energy becomes strikingly clear.

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Table 21 Potential Annual Energy Supply From Biomass and Hydropower and Annual Energy Consumption

Country	Potential energy from forest growth (PJ/yr)	Potential energy from animal manure (PJ/yr)	Potential energy from crop residues (PJ/yr)	Total potential energy from biomass (PJ/yr)	Hydro Potential (MW)	Annual output at 50% load factor (PJ/yr)	Estimated fuelwood and charcoal consumption (PJ/yr)	Estimated animal and crop residue consumption (PJ/yr)	Commercial energy consumption (PJ/yr)	Estimated total energy consumption (PJ/yr)
CENTRAL AMERICA										
Belize	20-200	*	*	20-200	300	5	*	*	3	3
Costa Rica	20-200	31.6	16.4	68-250	9,000	142	24	*	30	54
Cuba	20-200	105.0	328.0	450-630	*	*	16	*	340	356
Dominican Republic	10-100	39.6	61.6	110-200	*	*	19	*	97	116
El Salvador	10-100	20.8	22.4	53-140	1,351	21	35	*	31	66
Guatemala	60-600	41.2	41.3	140-680	10,900	172	56	*	47	103
Haiti	2-20	30.6	22.7	55-70	*	*	42	*	4	46
Honduras	70-700	34.2	12.0	120-750	2,800	44	33	*	22	55
Jamaica	10-50	7.2	22.8	40-80	85	1	0.02	5	117	122
Mexico	400-4,000	645.0	405.0	1,500-5,100	20,334	321	86	37	2,239	2,362
Nicaragua	60-600	45.5	17.4	120-660	4,416	70	24	*	31	55
Panama	40-400	24.4	14.1	80-440	2,500	39	15	*	45	60
Puerto Rico	2-20	10.6	19.1	32-50	*	*	*	*	338	338
SOUTH AMERICA										
Argentina	600-6,000	1,128.0	414.0	2,100-7,500	48,120	759	38	155	1,360	1,553
Bolivia	470-4,700	82.7	26.7	580-4,800	18,000	284	39	*	54	93
Brazil	3,200-32,000	1,950.0	948.0	6,100-35,000	90,240	1,423	1,023	174	2,340	3,537
Chile	50-500	86.2	37.7	170-620	15,780	249	33	*	302	335
Columbia	780-7,800	408.0	163.0	1,400-8,400	50,000	788	209	13	488	710
Ecuador	180-1,800	67.8	22.4	280-1,900	21,000	331	21	*	98	119
French Guinea	90-900	*	*	90- 900	233	4	*	*	4	4
Guyana	180-1,800	6.2	28.7	210-1,800	12,000	189	0.2	*	25	25
Paraguay	210-2,100	88.5	16.6	310-2,200	6,000	95	33	*	15	48
Peru	870-8,700	142.0	85.9	1,100-8,900	12,500	197	63	*	303	366
Surinam	150-1,500	0.8	4.3	160-1,500	260	4	*	*	31	31
Uruguay	10-50	232.0	25.5	270-310	2,512	40	10	*	91	101
Venezuela	480-4,800	248.0	46.4	1,000-5,400	11,644	184	77	13	1,028	1,118
AFRICA										
Algeria	20-200	57.7	34.2	110-290	4,800	76	14	-	370	384
Angola	730-7,300	52.7	12.0	800-7400	9,664	152	73	*	32	105
Benin	90-900	18.8	6.2	120-930	1,792	28	26	*	4	30
Botswana	110-1,100	41.4	0.1	150-1,100	2,984	47	8	*	*	8
Burundi	3-30	0.5	15.8	30-45	*	*	10	*	1	11
Cameroon	300-3,000	55.2	9.9	370-3,100	22,960	362	77	*	19	96
Central African Rep.	280-2,800	9.6	3.1	290-2,800	11,040	174	22	*	2	24
Chad	150-1,500	69.1	1.1	270-1,700	3,440	54	38	*	3	41
Congo	270-2,700	1.1	4.1	280-2,700	9,040	143	20	*	6	26

Table 21 Potential Annual Energy Supply From Biomass and Hydropower and Annual Energy Consumption (Continued)

Country	Potential energy from forest growth (PJ/yr)	Potential energy from animal manure (PJ/yr)	Potential energy from crop residues (PJ/yr)	Total potential energy from biomass (PJ/yr)	Hydro Potential (MW)	Annual output at 50% load factor (PJ/yr)	Estimated fuelwood and charcoal consumption (PJ/yr)	Estimated animal and crop residue consumption (PJ/yr)	Commercial energy consumption (PJ/yr)	Estimated total energy consumption (PJ/yr)
AFRICA										
Djibouti	-	*	*	*	*	*	*	*	2	2
Egypt	-	98.3	182.0	280	3,800	60	1	8	528	537
Equatorial Guinea	10-100	0.2	0.2	10-100	2,400	38	*	*	1	1
Ethiopia	330-3,300	586.0	67.1	980-4,000	9,214	145	250	*	22	272
Gabon	250-2,500	0.5	0.4	250-2,500	17,520	276	12	*	20	32
Gambia	1-10	5.4	0.4	7-16	*	*	3	*	1	4
Ghana	120-1,200	29.9	17.1	170-1,200	1,615	25	120	*	47	167
Guinea	170-1,700	26.6	11.3	210-1,700	6,400	101	29	*	12	41
Guinea-Bissau	10-100	5.7	0.8	17-110	120	2	5	*	1	6
Ivory Coast	190-1,900	14.8	16.5	220-1,900	780	12	53	*	56	109
Kenya	20-200	149.0	37.6	200-400	13,440	212	115	3	62	180
Lesotho	-	16.9	2.6	20	490	8	*	*	*	*
Liberia	25-250	2.1	4.1	31-260	6,000	95	15	*	21	36
Libya	5-50	15.5	10.3	31-75	160	3	4	*	118	122
Madagascar	120-1,200	167.0	46.7	330-1,400	64,000	1009	55	*	16	71
Malawi	70-700	13.0	15.3	98-730	100	2	33	*	8	41
Mali	40-400	89.3	3.2	110-490	3,520	56	30	*	5	35
Mauritania	-	51.9	0.1	52	2,000	32	6	*	4	10
Morocco	50-500	27.0	5.3	82-530	975	15	30	*	142	172
Mozambique	660-6,600	26.5	22.3	710-6,600	11,290	178	90	*	37	127
Namibia	100-1,000	*	*	100-1,000	1,200	19	*	*	*	*
Niger	40-400	66.5	1.8	110-470	9,600	151	25	*	4	29
Nigeria	340-3,400	277.0	79.2	700-3,800	1,515	24	645	0.4	178	823
Rwanda	5-50	14.1	3.3	22-67	*	*	42	*	2	44
Senegal	50-500	53.0	3.4	110-560	4,400	69	25	*	23	48
Sierra Leone	3-30	5.7	9.1	18-45	3,000	47	27	*	9	36
Somalia	2-20	98.9	3.5	100-120	240	4	35	*	4	39
Sudan	420-4,200	327.0	20.5	770-4,500	16,000	252	230	*	76	306
Swaziland	-	11.2	10.8	22	700	11	5	*	*	5
Tanzania	390-3,900	217.0	96.1	700-4,200	20,800	328	400	*	31	431
Togo	40-400	8.8	4.8	53-450	480	8	10	*	6	16
Tunisia	3-30	30.5	41.6	75-100	29	0.5	19	*	76	95
Uganda	20-200	76.5	15.6	110-290	12,000	189	0.50	*	17	17
Upper Volta	35-350	39.3	1.6	76-390	12,000	189	43	*	3	46
Zaire	1,800-18,000	29.6	31.9	1,900-18,000	132,000	2,081	0.30	*	46	46
Zambia	370-3,700	1.8	9.6	380-3,700	3,834	60	40	*	82	122
Zimbabwe	280-2,800	73.8	32.1	390-2,900	5,000	79	60	*	121	181

Table 21 Potential Annual Energy Supply From Biomass and Hydropower and Annual Energy Consumption (Concluded)

Country	Potential energy from forest growth (PJ/yr)	Potential energy from animal manure (PJ/yr)	Potential energy from crop residues (PJ/yr)	Total potential energy from biomass (PJ/yr)	Hydro Potential (MW)	Annual output at 50% load factor (PJ/yr)	Estimated fuelwood and charcoal consumption (PJ/yr)	Estimated animal and crop residue consumption (PJ/yr)	Commercial energy consumption (PJ/yr)	Estimated total energy consumption (PJ/yr)
NEAR EAST										
Afghanistan	7-70	124.0	44.1	180-250	6,000	95	64.0	*	24	88
Bahrain	-	0.1	*	-	*	*	*	*	93	93
Iran	40-400	261.0	276.0	580-940	10,196	161	21	9	1,459	1,489
Iraq	15-150	98.4	48.7	160-300	1,900	30	0.1	*	245	245
Jordan	-	4.7	2.5	7	*	*	0.03	*	43	43
Kuwait	-	1.1	-	1	*	*	*	*	278	278
Lebanon	1-10	3.7	3.0	8-17	*	*	0.7	*	46	47
Oman	1-10	2.3	0.1	3-12	*	*	*	*	16	16
Qatar	-	0.4	*	-	*	*	*	*	70	70
Saudi Arabia	12-120	24.6	6.5	43-150	900	14	-	-	515	515
Syria	5-50	11.5	74.4	91-140	1,000	16	0.5	*	166	167
Turkey	180-1,800	417.0	679.0	1,300-2,900	15,200	240	*	*	874	874
U.A.E.	-	*	*	*	*	*	*	*	89	89
Yemen (AR)	-	52.5	12.9	65	*	*	*	*	8	8
Yemen (P.D.R.)	26-260	5.4	0.6	32-270	*	*	*	*	17	17
EAST ASIA										
Bangladesh	23-230	492.0	369.0	880-1,100	1,307	21	150	466	77	693
Bhutan	30-300	4.0	7.6	42-310	*	*	*	*	*	*
Brunei	4-40	0.5	0.1	5-41	*	*	*	*	74	74
Burma	450-4,500	173.0	176.0	800-4,800	75,000	1,183	210	*	44	254
China	800-8,000	3,350.0	5,096.0	9,200-16,000	330,000	5,203	1,500	2,717	17,293	21,510
India	750-7,500	4,250.0	3,043.0	8,000-15,000	70,000	1,104	1,237	3,150	3,896	8,283
Indonesia	1,250-12,500	219.0	559.0	2,000-13,000	30,000	473	1,162	19	892	2,073
Khmer Rep.	130- 1,300	47.2	*	180-1,300	*	*	44	*	*	44
Korea N.	90-900	23.3	108.0	220-1,000	2,000	32	50	*	1,462	1,512
Korea S.	70-700	40.1	206.0	320-950	5,514	87	77	-	1,072	1,149
Laos	150-1,500	33.1	16.2	200-1,500	*	*	33	*	6	39
Malaysia	240-2,400	20.7	40.2	300-2,500	1,319	21	61	*	206	267
Mongolia	150-1,500	113.0	10.9	280-1,500	*	*	15	*	51	66
Nepal	50-500	184.0	60.1	290-740	80,000	1,261	109	3	4	116
Pakistan	20-200	494.0	450.0	960-1,100	20,000	315	93	*	384	477
Philippines	160-1,600	177.0	289.0	630-2,100	7,504	118	250	*	421	671
Sri Lanka	20-200	41.9	24.3	86-270	1,180	19	46	-	42	88
Taiwan	20-200	*	*	20-200	1,632	26	*	*	*	*
Thailand	290-2,900	205.0	308.0	800-3,400	6,242	98	27	366	387	780
Vietnam	70-700	143.0	13.5	230-860	54,000	851	180	*	169	349

Notes for Table 21

* data not available
\- negligible

Potential energy from forests is based on wood production of 1-10 cubic metres per hectare and an energy content of 10 GJ/m^3. Total forest area is based on Reidar Pearson, World Forest Resources (Stockholm, Royal College of Forestry, 1974) as reported in reference 1. Energy from manure and crop residues from Survey of Biomass Energy Programs and Use in the Developing Countries (T.B. Taylor Associates; report to the Office of Technology Assessment, 1978), reported in reference 1.

Hydro potential is taken from World Energy Conference, Survey of Energy Resources, (New York, 1974), and reported in reference 1.

Estimated animal and crop residue consumption are taken from Basic Energy Statistics and Energy Balances of Developing Countries 1967-1977, (Workshop on Energy Data of Developing Countries, International Energy Data of Developing Countries, International Energy Agency, OECD, Paris, 1979). Fuelwood and charcoal consumption is taken from Prospects for Traditional and Non-Conventional Energy Sources in Developing Countries (World Bank Staff Working Paper No. 346), reported in reference 1.

Total energy consumption is taken from Statistical Yearbook (United Nations, 1978).

Additional country specific data is taken from Revelle (reference 7); Jamaica's National Energy Plan; Country Paper of Nepal, submitted to U.N. Conference on New and Renewable Sources of Energy, Nairobi, 1981. For hydro resources in Central America, see Energy and Development in Central America (MITRE, 1980) Vol. I, p21.

Table 22 Potential for Renewable Energy Sources in Developing Countries

Country	Potential energy from biomass (PJ/yr)	Potential energy from hydropower (PJ/yr)	Potential energy from biomass and hydropower (PJ/yr)	Estimated total energy consumption (PJ/yr)	Ratio of potential renewable energy supply to current energy consumption
CENTRAL AMERICA					
Belize	20-200	5	25-205	3	8.3- 68.3
Costa Rica	68-250	142	210-392	54	3.9 - 7.3
Cuba	450-630	*	450-630	356	1.3 - 1.8
Dominican Republic	110-200	*	110-200	116	0.9 - 1.7
El Salvador	53-140	21	74-161	66	1.1 - 2.4
Guatemala	140-680	172	312-852	103	3.0 - 8.3
Haiti	55-70	*	55-70	46	1.2 - 1.5
Honduras	120-750	44	164-794	55	3.0 - 14.4
Jamaica	40-80	1	41-81	122	0.3 - 0.7
Mexico	1,500-5,100	321	1821-5421	2362	0.8 - 2.3
Nicaragua	120-660	70	190-730	55	3.5 - 13.3
Panama	80-440	39	119-479	60	2.0 - 8.0
Puerto Rico	32-50	*	32-50	338	0.1
SOUTH AMERICA					
Argentina	2,100-7,500	759	2859-8259	1553	1.8 - 5.3
Bolivia	580-4,800	284	864-5084	93	9.3 - 54.7
Brazil	6,100-35,000	1,423	7523-36423	3537	2.1 - 10.3
Chile	170-620	249	419- 869	335	1.3 - 2.6
Columbia	1,400-8,400	788	2188-9188	710	3.1 - 12.9
Ecuador	280-1,900	331	611-2231	119	5.1 - 18.7
French Guinea	90- 900	4	94- 904	4	23.5 -226.0
Guyana	210-1,800	189	399-1989	25	16.0 - 79.6
Paraguay	310-2,200	95	405-2295	48	8.4 - 47.8
Peru	1,100-8,900	197	1297-9097	366	3.6 - 24.9
Surinam	160-1,500	4	164-1504	31	5.3 - 48.5
Uruguay	270-310	40	310-350	101	3.1 - 3.5
Venezuela	1,000-5,400	184	1184-5584	1118	1.1 - 5.0
AFRICA					
Algeria	110-290	76	186-366	384	0.5 - 1.0
Angola	800-7,400	152	952-7552	105	9.1 - 71.9
Benin	120-930	28	148-958	30	4.9 - 31.9
Botswana	150-1,100	47	197-1147	8	24.6 -143.4
Burundi	30-45	*	30-45	11	2.7 - 4.1
Cameroon	370-3,100	362	732-3462	96	7.6 - 36.1
Central African Rep.	290-2,800	174	464-2974	24	19.3 -123.9
Chad	270-1,700	54	324-1754	41	7.9 - 42.8
Congo	280-2,700	143	423-2843	26	16.3 -109.3
Djibouti	*	*	*	2	*
Egypt	280	60	340	537	0.6
Equatorial Guinea	10-100	38	48-138	1	48-138
Ethiopia	980-4,000	145	1125-4145	272	4.1- 15.2
Gabon	250-2,500	276	526-2776	32	16.4- 86.8
Gambia	7-16	*	7-16	4	1.8- 4.0
Ghana	170-1,200	25	195-1225	167	1.2- 7.3
Guinea	210-1,700	101	311-1801	41	7.6- 43.9
Guinea-Bissau	17-110	2	19-112	6	3.2- 18.7
Ivory Coast	220-1,900	12	232-1912	109	2.1- 17.5
Kenya	200-400	212	412-612	180	2.3- 3.4
Lesotho	20	8	28	*	*
Liberia	31-260	95	126-355	36	3.5- 9.9
Libya	31-75	3	34- 78	122	0.3- 0.6
Madagascar	330-1,400	1009	1339-2409	71	18.9- 33.9
Malawi	98-730	2	100-732	41	2.4- 17.9
Mali	110-490	56	166-546	35	4.7- 15.6
Mauritania	52	32	84	10	8.4

Table 22 Potential for Renewable Energy Sources in Developing Countries (Continued)

Country	Potential energy from biomass (PJ/yr)	Potential energy from hydropower (PJ/yr)	Potential energy from biomass and hydropower (PJ/yr)	Estimated total energy consumption (PJ/yr)	Ratio of potential renewable energy supply to current energy consumption
Morocco	82 -530	15	97-545	172	0.6 - 3.2
Mozambique	710-6600	178	888-6778	127	7.0 -53.4
Namibia	100-1000	19	119-1019	*	*
Niger	110-470	151	261-621	29	9.0 -21.4
Nigeria	700-3800	24	724-3824	823	0.9 - 4.6
Rwanda	22-67	*	22-67	44	0.5 - 1.5
Senegal	110-560	69	179-629	48	3.7 -13.1
Sierra Leone	18-45	47	65-92	36	1.8 - 2.6
Somalia	100-120	4	104-124	39	2.7 - 3.2
Sudan	770-4500	252	1022-4752	306	3.3 -15.5
Swaziland	22	11	33	5	6.6
Tanzania	700-4200	328	1028-4528	431	2.4 -10.5
Togo	53-450	8	61-458	16	3.8 -28.6
Tunisia	75-100	1	75-100	95	0.8 - 1.1
Uganda	110-290	189	299-479	17	17.6 -28.2
Upper Volta	76-390	189	265-579	46	5.8 -12.6
Zaire	1,900-18000	2081	3981-20081	46	86.5-436.5
Zambia	380-3700	60	440-3760	122	3.6 -30.8
Zimbabwe	390-2900	79	469-2979	181	2.6 -16.5
NEAR EAST					
Afghanistan	180-250	95	275-345	88	3.1 - 3.9
Bahrain	-	*	*	93	*
Iran	580-940	161	741-1101	1489	0.5 - 0.7
Iraq	160-300	30	190-330	245	0.8 - 1.3
Jordan	7	*	7	43	0.2
Kuwait	1	*	1	278	0.1
Lebanon	8-17	*	8-17	47	0.2 - 0.4
Oman	3-12	*	3-12	16	0.2 - 0.8
Qatar	-	*	*	70	*
Saudi Arabia	43-150	14	57-164	515	0.1 - 0.3
Syria	91-140	16	107-156	167	0.6 - 0.9
Turkey	1300-2900	240	1540-3140	874	1.8 - 3.6
U.A.E.	*	*	*	89	*
Yemen (AR)	65	*	65	8	8.1
Yemen (P.D.R.)	32-270	*	32-270	17	1.9 -15.9
EAST ASIA					
Bangladesh	880-1,100	21	901-1121	693	1.3 - 1.6
Bhutan	42-310	*	42-310	*	*
Brunei	5-41	*	5-41	74	0.1 - 0.6
Burma	800-4,800	1183	1983-5983	254	7.8 -23.6
China	9,200-16,000	5203	14403-21203	21510	0.7 - 1.0
India	8,000-15,000	1104	9104-16104	8283	1.1 - 1.9
Indonesia	2,000-13,000	473	2473-13473	2073	1.2 - 6.5
Khmer Republic	180-1,300	*	180-1300	44	4.1 -29.5
Korea N.	220- 1,000	32	252-1032	1512	0.2 - 0.7
Korea S.	320-950	87	407-1037	1149	0.4 - 0.9
Laos	200- 1,500	*	200-1500	39	5.1 -38.5
Malaysia	300- 2,500	21	321-2521	267	1.2 - 9.4
Mongolia	280- 1,500	*	280-1500	66	4.2 -22.7
Nepal	290- 740	1261	1551-2001	116	13.4 -17.3
Pakistan	960- 1,100	315	1275-1415	477	2.7 - 3.0
Philippines	630- 2,100	118	748-2218	671	1.1 - 3.3
Sri Lanka	86-270	19	105-289	*	1.2 - 3.3
Taiwan	20-200	26	46-226	*	*
Thailand	800-3,400	98	898-3498	780	1.2 - 4.5
Vietnam	230-860	851	1081-1711	349	3.1 - 4.9

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FUNDAMENTALS OF FLUID FLOW

It is useful to review some basic concepts:

Pressure Relationships

Pressure is force divided by area

i.e. $$P = \frac{F}{A} = \frac{\text{Newtons}}{m^2}$$

1 N/m^2 is called a Pascal (Pa)

10^5 Pa is equal to 1 bar which is about equal to atmospheric pressure. More precisely, 1 standard atmosphere is equal to

$$760 \text{ mm Hg} = 14.7 \text{ psi} = 101325 \text{ Pa} = 1.01325 \text{ bar}$$

The difference in pressure between any two points at different levels in a liquid is given by

$$p_2 - p_1 = \rho g(h_2 - h_1) \quad \text{Pa} \tag{1}$$

Pressure may also be expressed in terms of a pressure head as

$$h = \frac{p}{\rho g} \quad \text{metres} \tag{2}$$

In these expressions

h = fluid height, metres
ρ = fluid density, kg/m^3
g = acceleration due to gravity, nominally 9.81 m/s^2
p = pressure, Pa

For ideal gases

$$\frac{p_1 v_1}{T_1} = \frac{p_2 v_2}{T_2} = mR \tag{3}$$

or

$$\frac{p_1}{\rho_1 T_1} = \frac{p_2}{\rho_2 T_2} = R \tag{4}$$

where p = absolute pressure, Pa
v = volume, m^3
m = mass, kg
ρ = density, kg/m^3
T = absolute temperature in degrees Kelvin (273 + °C)

for air R = 287.1 J/kg K

Equation of Continuity

The equation of continuity follows from the principle of conservation of mass. For steady flow, the mass of fluid passing all sections per unit of time must be the same.

Therefore

$$\rho_1 A_1 V_1 = \rho_2 A_2 V_2 = \text{constant} \qquad (5)$$

For incompressible fluids and where density may be considered constant the flow rate Q is given by

$$Q = A_1 V_1 = A_2 V_2 = \text{constant} \qquad (6)$$

Bernoulli's Equation

Just as the continuity equation is a mass balance, so Bernoulli's equation follows from the principle of conservation of energy. For steady flow of an incompressible fluid in which there is negligible change in internal energy:

$$\frac{p_1}{\rho g} + \frac{V_1^2}{2g} + z_1 + H_A - H_L - H_E = \frac{p_2}{\rho g} + \frac{V_2^2}{2g} + z_2 \qquad (7)$$

where
z = elevation above any datum level, metres
ρ = fluid density, kg/m^3
g = acceleration due to gravity, m/s^2
V = average fluid velocity, m/s
H_A = energy added, metres
H_L = head losses due to friction, metres
H_E = energy extracted, metres
P = pressure, Pa

The units used in Equation (7) are J/N or metres of the fluid. It is usual to identify the elements of the Bernoulli equation as '<u>heads</u>', i.e.

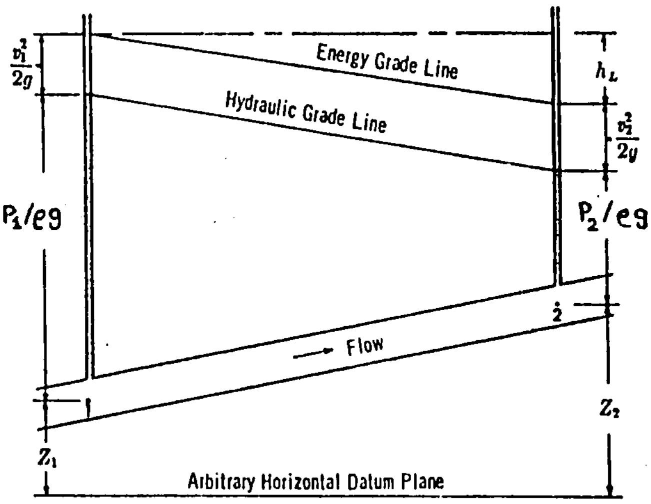
static pressure head	$p/\rho g$
velocity head	$V^2/2g$
potential head	z

Since the fluid velocity in a conduit is not uniform with respect to cross section, the velocity head should in theory be corrected by a kinetic-energy correction factor α.

for turbulent flow α = 1.02 - 1.15
for laminar flow α = 2

Generally, α may be taken as 1 without serious error since the velocity head is usually only a small part of the total head. Practically all problems dealing with flow of liquids utilize the Bernoulli equation as the basis of solution.

The energy line is a graphical representation of the energy at each section. With respect to a chosen datum, the total head in metres of fluid can be plotted at each representative section, and the line so obtained is a valuable tool in many flow problems. The energy line will slope downwards in the direction of flow except where energy is added by mechanical devices such as pumps. The hydraulic grade line lies below the energy line by an amount equal to the velocity head at the section. The two lines are parallel for all sections of equal cross section area. The ordinate between the centre of the stream and the hydraulic grade line is the pressure head at the section.



Energy Balance for Two Points in a Fluid

Power

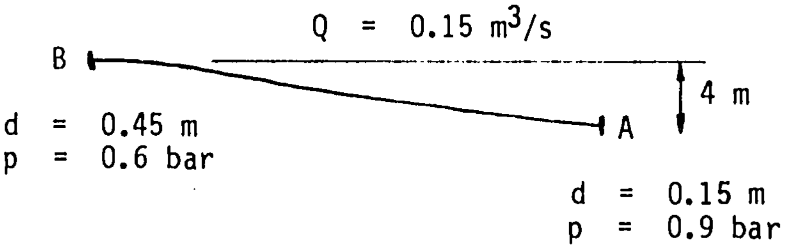
Energy may be added to or extracted from a fluid. Power, P, is calculated from

$$P = H\dot{m}g \quad \text{Watts} \tag{8}$$

where H = total head added or extracted, metres
$\dot{m}$ = mass flow, kg/s
g = acceleration due to gravity, m/s^2

Example 1

A pipe carrying oil of relative density 0.877 changes in size from 150 mm at section A to 450 mm at section B. Section A is 4 m lower than B and the pressures are 0.9 bar and 0.6 bar respectively. If the discharge is 0.15 m^3/s determine the lost head and the direction of flow.



Solution:

The velocity of the fluid at sections A and B is given by Equation (6), i.e.

$$V_A = \frac{0.15}{\pi(0.15)^2/4} = 8.5 \text{ m/s}$$

$$V_B = \frac{0.15}{\pi(0.45)^2/4} = 0.94 \text{ m/s}$$

Using point A (the lowest) as the datum plane, the total head at each section is given by Bernoulli's equation:

$$\frac{p_A}{\rho g} + \frac{V_A^2}{2g} + z_A - H_L = \frac{p_B}{\rho g} + \frac{V_B^2}{2g} + z_B$$

so $$\frac{0.9 \times 10^5}{877 \times 9.81} + \frac{(8.5)^2}{2 \times 9.81} + 0 - H_L = \frac{0.6 \times 10^5}{877 \times 9.81} + \frac{(0.94)^2}{2 \times 9.81} + 4$$

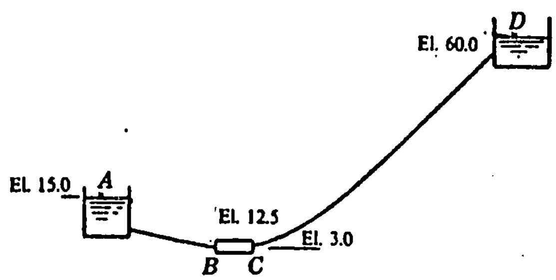
or $$14.1 - H_L = 11.0$$

or $$H_L = 3.1 \text{ metres.}$$

Note that the head at A (14.1 m) is greater than the head at B (11.0 m) by an amount equal to H_L. The oil, therefore, must flow from A to B.

Example 2

For the system shown below, pump BC must deliver 0.16 m^3/s of oil (762 kg/m^3) to reservoir D. Assuming that the head loss between A and B is 2.5 m, and from C to D is 6.5 m, (a) determine how much power the pump must supply to the system, and (b) plot the energy line.



Solution

(a) The velocity of the fluid at surfaces A and D will be very small. We neglect the velocity heads at these surfaces. Taking BC as the datum level, and applying Bernoulli's equation from A to D:

$$\frac{p_{atm}}{\rho g} + \frac{V_A^2}{2g} + z_A + H_A - H_L = \frac{p_{atm}}{\rho g} + \frac{V_D^2}{2g} + z_D$$

Most of the terms drop out leaving

$$z_A + H_A - H_L = z_D$$

or $$12 + H_A - (2.5 + 6.5) = 57$$

therefore $$H_A = 54 \text{ metres}$$

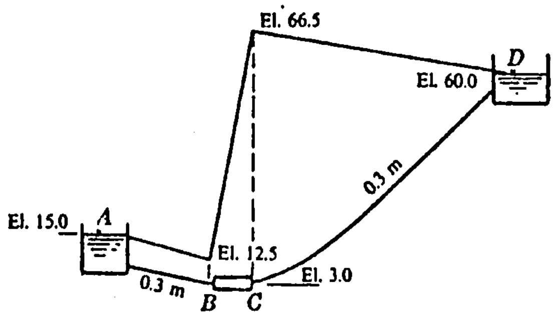
From Equation (8),

$$P = 54 \times 0.16 \times 762 \times 9.81$$

$$= 64.6 \text{ kW}$$

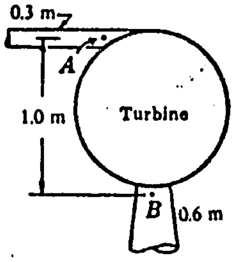
This is the power delivered to the fluid; the power delivered to the pump will depend on its efficiency.

(b) The energy line at A is at elevation 15 m above datum zero. From A to B the energy loss is 2.5 m and the energy line drops by this amount, the elevation at B being 12.5. The pump adds 54 m of head and the elevation at C is therefore 66.5 m. Finally, since the loss of energy between C and D is 6.5 m, the elevation at D is 66.5 - 6.5 = 60 m. The energy line is shown below. Note that the pump has supplied a head sufficient to raise the oil 45 m, but it has also had to overcome 9 m of losses in the piping. Therefore, 54 m is delivered to the system.



Example 3

Water flows through the turbine shown below at a rate of 0.2 m^3/s and the pressures at A and B respectively are 1.5 bar and -0.3 bar. Determine the power delivered to the turbine by the water. Neglect head losses.



Solution:

$$V_A = \frac{0.2}{\pi(0.3^2)/4} = 2.8 \text{ m/s}$$

$$V_B = \frac{0.2}{\pi(0.6)^2/4} = 0.7 \text{ m/s}$$

Applying Bernoulli's equation with point B as the datum level:

$$\frac{p_A}{\rho g} + \frac{V_A^2}{2g} + z_A - H_E = \frac{p_B}{\rho g} + \frac{V_B^2}{2g} + z_B$$

$$\frac{1.5 \times 10^5}{1000 \times 9.81} + \frac{(2.8)^2}{2 \times 9.81} + 1 - H_E = \frac{-0.3 \times 10^5}{1000 \times 9.81} + \frac{(0.7)^2}{2 \times 9.81} + 0$$

or H_E = 19.7 metres

Power $= H\dot{m}g = 19.7 \times 1000 \times 0.2 \times 9.81$

= <u>38.7 kW to turbine</u>

Flow in Pipes

Flow in pipes is generally laminar or turbulent. If the flow is laminar the viscosity of the fluid is dominant and suppresses any tendency to turbulent conditions. The Reynolds number, Re, is the ratio of inertial forces to viscous forces.

Laminar flow: $Re < 2100$
Turbulent flow: $Re > 6000$

where
$$Re = \frac{VD\rho}{\mu} \tag{9}$$

V = fluid velocity, m/s
D = pipe diameter, metres
ρ = fluid density, kg/m^3
μ = viscosity, Pa s

For non-circular cross-sections an equivalent diameter (or hydraulic diameter) is used where

$$D = \frac{4 \text{ X cross-section area}}{\text{wetted perimeter}} \tag{10}$$

The Darcy-Weisbach formula is the basis for evaluating the lost head for fluid in pipes and conduits:

$$H_L = f\frac{L}{D} \cdot \frac{V^2}{2g} \quad \text{metres} \tag{11}$$

where
L = length of pipe
D = hydraulic diameter
f = friction factor

For laminar flow f is a simple function of the Reynolds number:

$$f = 64/Re \tag{12}$$

For turbulent flow the situation is more complex. Graphs are available which show the relationship between friction factor f, Reynolds number Re, and the relative roughness of the pipe, ε/D. A typical chart is shown overleaf in Diagram 1.

Other Losses of Head

It is common practice to express all losses in terms of a velocity head $V^2/2g$. That is, Equation 11 is written as

$$H_L = K\frac{V^2}{2g} \tag{13}$$

and K is evaluated from tables according to the actual structure of the flow system. Tables 1 and 2 give values of K for common pipeline items.

Kind of Pipe or Lining (New)	Values of ε in mm	
	Range	Design Value
Brass	0.0015	0.0015
Copper	0.0015	0.0015
Concrete	0.3-3.0	1.20
Cast Iron – uncoated	0.12-0.61	0.24
" " – asphalt dipped	0.061-0.183	0.12
" " – cement lined	0.0024	0.0024
" " – bituminous lined	0.0024	0.0024
" " – centrifugally spun	0.003	0.003
Galvanized Iron	0.061-0.24	0.150
Wrought Iron	0.030-0.091	0.061
Comm. & Welded Steel	0.030-0.091	0.061
Riveted Steel	0.91-9.1	1.83
Transite	0.0024	0.0024
Wood Stave	0.18-0.91	0.61

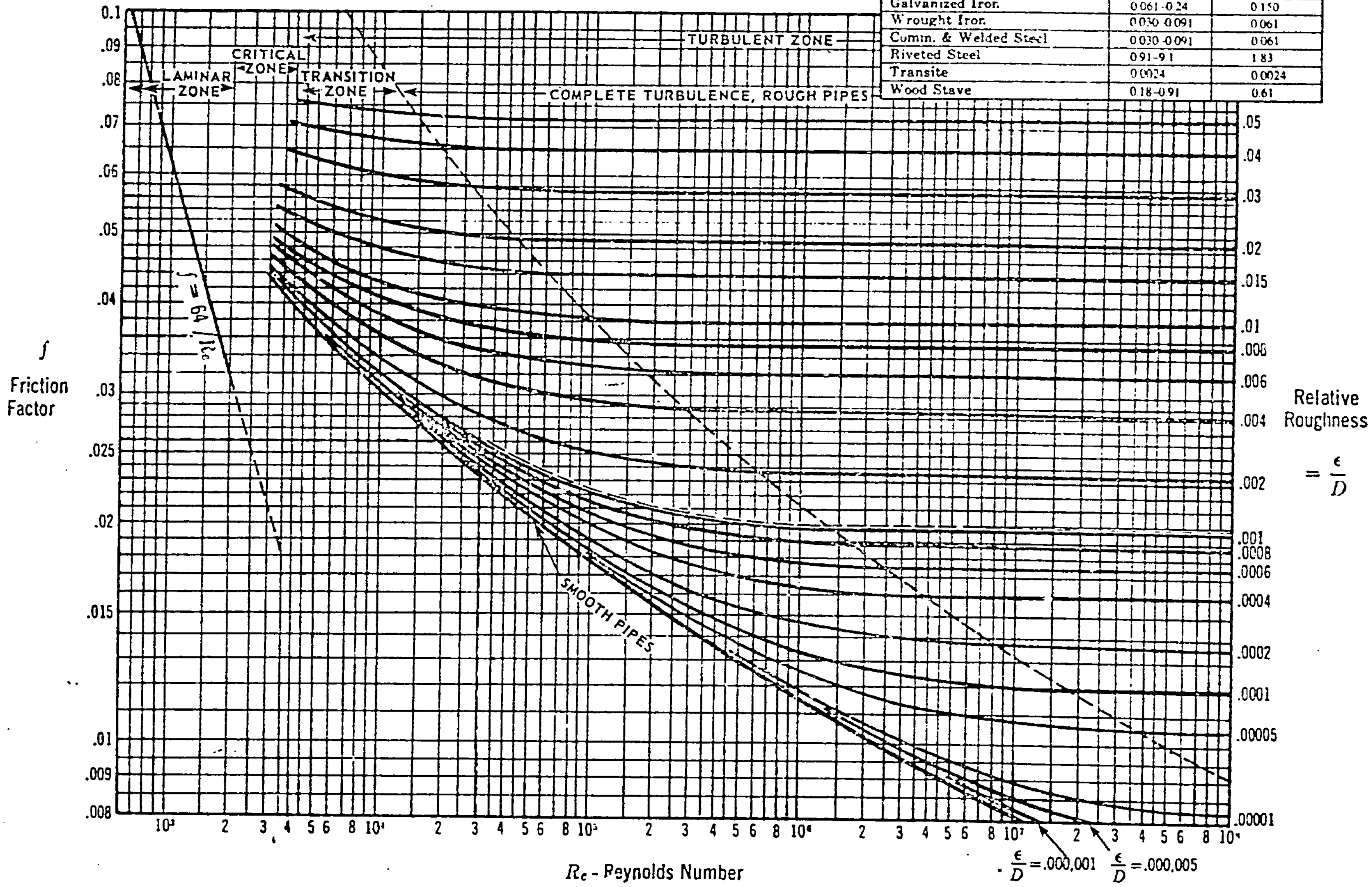


TABLE 1

TYPICAL LOSS OF HEAD ITEMS

(Subscript 1 = Upstream and Subscript 2 = Downstream)

Item	Average Lost Head
1. From Tank to Pipe – flush connection (entrance loss)	$0.50\frac{V_2^2}{2g}$
– projecting connection	$1.00\frac{V_2^2}{2g}$
– rounded connection	$0.05\frac{V_2^2}{2g}$
2. From Pipe to Tank (exit loss)	$1.00\frac{V_1^2}{2g}$
3. Sudden Enlargement	$\frac{(V_1 - V_2)^2}{2g}$
4. Gradual Enlargement (see Table 2)	$K\frac{(V_1 - V_2)^2}{2g}$
5. Venturi Meters, Nozzles and Orifices	$\left(\frac{1}{c_v^2} - 1\right)\frac{V_2^2}{2g}$
6. Sudden Contraction (see Table 2)	$K_c\frac{V_2^2}{2g}$
7. Elbows, Fittings, Valves Some typical values of K are: 45° Bend .... 0.35 to 0.45 90° Bend .... 0.50 to 0.75 Tees .... 1.50 to 2.00 Gate Valves (open) .... about 0.25 Check Valves (open) .... about 3.0	$K\frac{V^2}{2g}$

TABLE 2

VALUES OF K*

Contractions and Enlargements

Sudden Contraction		Gradual Enlargement for Total Angles of Cone						
d_1/d_2	K_c	4°	10°	15°	20°	30°	50°	60°
1.2	0.08	0.02	0.04	0.09	0.16	0.25	0.35	0.37
1.4	0.17	0.03	0.06	0.12	0.23	0.36	0.50	0.53
1.6	0.26	0.03	0.07	0.14	0.26	0.42	0.57	0.61
1.8	0.34	0.04	0.07	0.15	0.28	0.44	0.61	0.65
2.0	0.37	0.04	0.07	0.16	0.29	0.46	0.63	0.68
2.5	0.41	0.04	0.08	0.16	0.30	0.48	0.65	0.70
3.0	0.43	0.04	0.08	0.16	0.31	0.48	0.66	0.71
4.0	0.45	0.04	0.08	0.16	0.31	0.49	0.67	0.72
5.0	0.46	0.04	0.08	0.16	0.31	0.50	0.67	0.72

*Values from King's "Handbook of Hydraulics" - McGraw-Hill Book Company

Sizing Pipes

In designing any fluid flow system one generally seeks the lowest cost system that will operate reliably over a projected lifetime. A reduction in pipe diameter will mean a lower system capital cost but the pressure drop will increase thus raising pumping costs. The economic pipe diameter is therefore the diameter which minimizes total costs, i.e., amortized capital plus operating costs.

Note also from Equation 11 that head loss may be written as

$$H_L = \frac{fL}{2Dg}\left(\frac{Q}{A}\right)^2 = \frac{8fL}{g\pi^2}\frac{Q^2}{D^5} \tag{14}$$

and that therefore the pressure drop varies inversely with the fifth power of pipe diameter. Clearly, pumping requirements may become excessive if conduits are undersized.

In the absence of any firm economic data, pipes may well be sized on the basis of a maximum permissible system pressure drop.

Example 4

Water at 60°C is to flow at a rate of 10 litres/s through a galvanized iron pipe 20 metres long with a head loss not to exceed 0.5 metres. The density and viscosity for water at this temperature are 983.3 kg/m^3 and 4.71×10^{-4} Pa.s respectively. Determine the minimum acceptable pipe diameter.

Solution

A trial-and-error approach is required. One way to start is to guess the friction factor, f, which is usually quite close to 0.03. This allows us to calculate a first estimate of the pipe diameter.

From Equation 14 $$D^5 = \frac{8fLQ^2}{H_L g \pi^2}$$

so $$D^5 = \frac{8 \times 0.03 \times 20 \times (0.01)^2}{0.5 \times 9.81 \times \pi^2}$$

or $$D = 0.10 \text{ metres}$$

To check our guess at the friction factor we need to calculate the Reynolds number.

Since $$Re = \frac{\rho V D}{\mu} = \frac{4\rho Q}{\mu \pi D}$$

We have $$Re = \frac{4 \times 983.3 \times 0.01}{4.71 \times 10^{-4} \times \pi \times 0.1}$$

$$= 2.66 \times 10^5$$

From Diagram 1 $\varepsilon = 0.15$ mm (G.I. pipe)

so $\varepsilon/D = 0.00015/0.1 = 0.0015$

So from Diagram 1 $f = 0.022$

Recalculating the pipe diameter:

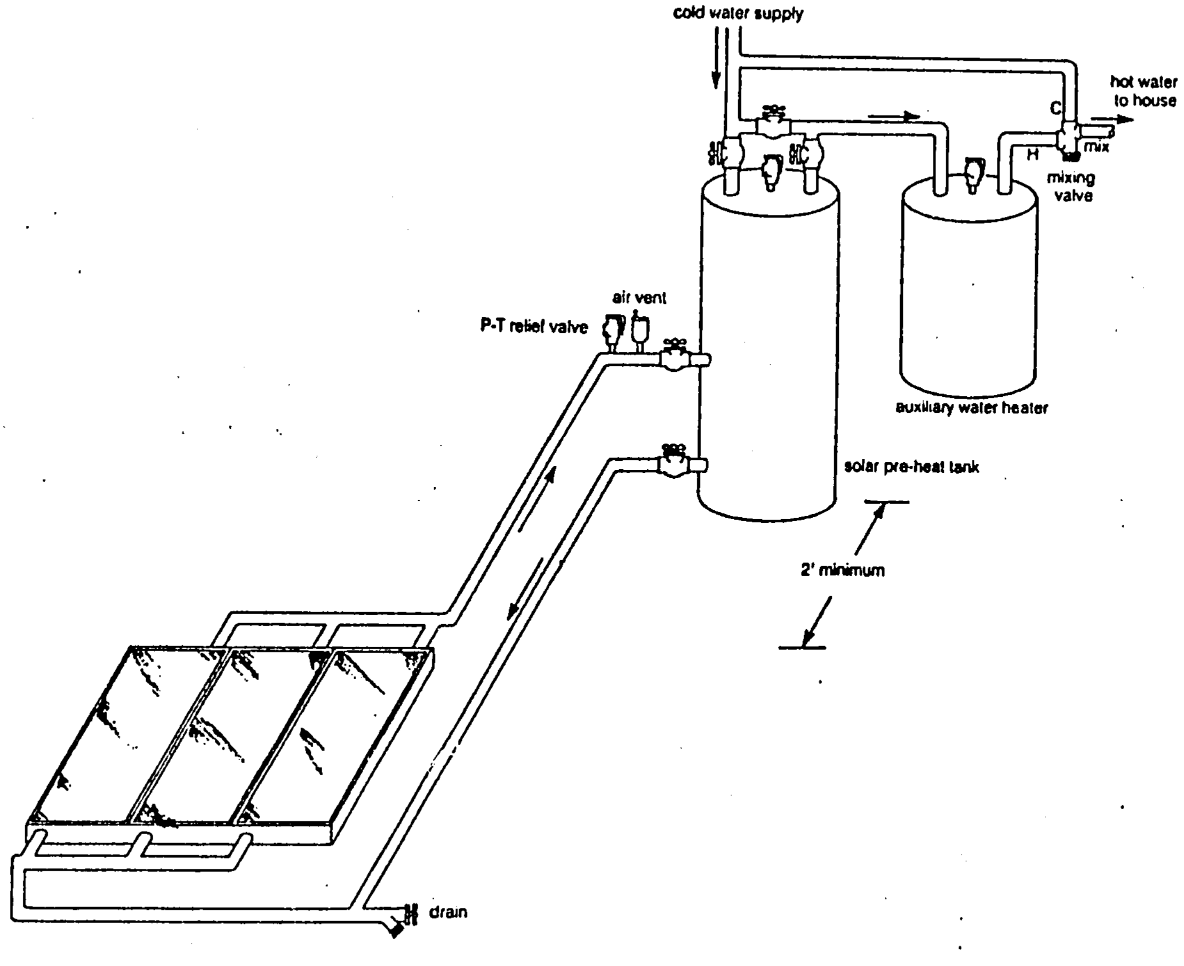
$$D^5 = \frac{8 \times 0.022 \times 20 \times (0.01)^2}{0.5 \times 9.81 \times \pi^2}$$

gives $\underline{D = 9.4 \text{ cm}}$

So we would choose an available pipe equal or larger than this diameter.

Thermosyphon Systems

The tendency of a less dense fluid to rise above a more dense fluid can be exploited in a simple, natural circulation solar water-heating system called a thermosyphon. The sketch below shows the arrangement of a typical system.

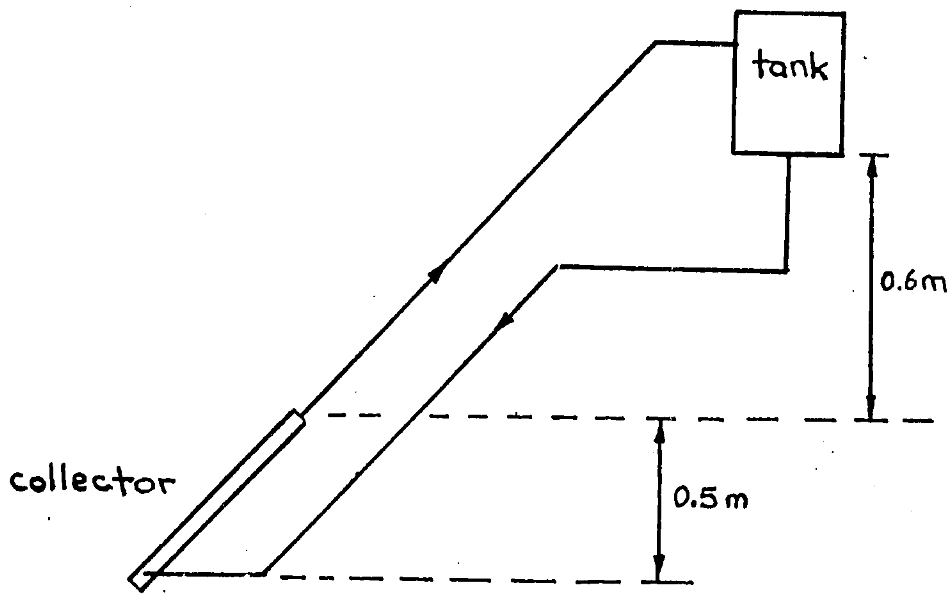


When the sun shines on the collector, it heats the water present in the tubes. This water becomes less dense than the colder water in the tank and downcomer and therefore rises to the highest point in the assembly, the top of the storage tank. A natural circulation flow pattern thus becomes established. Since the driving force in a thermosyphon system is due to small density differences and not the presence of a pump, friction losses through the system must be kept to a minimum. In general, one pipe size larger than would be used with a pump system is satisfactory. Most commercial systems use 1-inch inside diameter pipe. The flow rate through a thermosyphon system is about 1 gal/ft^2.hr (40 litres/m^2.h) in bright sun.

After sunset, a thermosyphon system can reverse its flow direction and lose heat to the environment during the night. To avoid reverse flow, the top of the collector should be at least 2 feet (0.6 m) below the bottom of the storage tank.

Example 5

A simple thermosyphon system is shown below. The water in the tank is at 140°F (density 983.3 kg/m^3), and leaves the collector at 180°F (density 970.2 kg/m^3). The pipes are smooth copper, I.D. = 2.5 cm. The pressure drop through the collector is estimated at 5 velocity heads. The total length of piping is approximately 8 metres. The viscosity of water may be assumed constant at 4×10^{-4} Pa.s. Determine the flow rate of the water.



Solution

Estimate the pressure drop through the system from Table 1.

	$V^2/2g$
3 x 45° bends (K = 0.45)	1.35
1 90° bend (K = 0.75)	0.75
tank to pipe	0.5
pipe to tank	1.0
collector (given)	5.0
	8.6

From Equation 11 the head loss is therefore:

$$H_L = (\frac{fL}{D} + 8.6)\frac{V^2}{2g}$$

We will have to guess the friction factor, f, since we don't know the velocity V. A good rule-of-thumb is that f is about 0.03.

so $$H_L = \left(\frac{0.03 \times 8}{0.025} + 8.6\right) \frac{V^2}{2g} = 0.928\ V^2 \text{ metres}$$

This gives us the head loss as a function of velocity. As the water in the solar collector heats up the fluid begins to circulate: the warmer fluid, being less dense, tends to rise. The fluid velocity will increase until the head losses resulting from the fluid motion become equal to the driving force caused by the density differences in the hot and cold legs of the system. We now calculate the driving force.

Cold Leg

Taking the temperature of the cold leg as 140°F and the height as 1.1 metre we have

$$P = \rho gh = 983.3 \times 9.81 \times 1.1$$
$$= 10610.8 \text{ Pa}$$

Hot Leg

We assume that the temperature of the fluid in the collector is the mean of the inlet and outlet temperatures, i.e. 160°F. At this temperature the density of water is 977.3 kg/m^3. So the pressure at the base of the hot leg is given by:

$$P = 977.3 \times 9.81 \times 0.5 + 970.2 \times 9.81 \times 0.6$$
$$= 10504.3 \text{ Pa}$$

The driving force is 10610.8 - 10504.3

= 106.5 Pa.

The head loss of 0.928 V^2 metres of water, is equivalent to a pressure loss of

$$P_L = \rho gh = 977.3 \times 9.81 \times 0.928\ V^2$$
$$= 8897.0\ V^2 \text{ Pa.}$$

The fluid velocity will be such that the forces balance, i.e.

$$8897\ V^2 = 106.5$$

or $$V = 0.109 \text{ m/s}$$

However, we must now check that the estimated friction factor of $f = 0.03$ was reasonable.

We therefore compute the Reynolds number.

$$Re = \frac{\rho V D}{\mu} = \frac{977.3 \times 0.109 \times 0.025}{4 \times 10^{-4}}$$

$$= 6658$$

So the flow is barely turbulent.

from Diagram 1 we find $f = 0.034$
(smooth pipe)

Recalculating the head loss gives

$$H_L = \left(\frac{0.034 \times 8}{0.025} + 8.6\right)\frac{V^2}{2g} = 0.993\ V^2$$

equivalent to a pressure loss of

$$P_L = 977.3 \times 9.81 \times 0.993\ V^2$$
$$= 9520.2\ V^2 \text{ Pa.}$$

At steady state

$$9520.2\ V^2 = 106.5$$

or

$$V = 0.106 \text{ m/s}$$

The flowrate is given by

$$Q = V \times A \quad m^3/s$$
$$Q = 0.106 \times \pi \times (0.025)^2/4$$
$$\underline{\underline{= 0.052 \text{ litres/sec.}}}$$

Problems

1. Air flows in a 0.15 m pipe at a pressure of 2.06 bar gauge and a temperature of 37°C. If the barometric pressure is 1.03 bar and the velocity is 4 m/s, calculate the mass flow rate.

2. Carbon dioxide passes point A in a 75 mm pipe at a velocity of 5 m/s. The pressure at A is 2 bar and the temperature is 20°C. At a point B downstream the pressure is 1.4 bar and the temperature is 30°C. For a barometric reading of 1.03 bar, calculate the velocity at B and compare the volumetric flows at A and B. The gas constant, R, for carbon dioxide is 187.8 J/kgK.

3. A horizontal air duct reduces in cross section area from 0.75 m^2 to 0.20 m^2. Assuming no losses, what pressure change will occur when 6 kg/s of air flows? Take the density of air as 3.2 kg/m^3 under these conditions.

4. Water at 32.2°C is to be lifted from a sump at a velocity of 2 m/s through the suction pipe of a pump. Determine the theoretical maximum height of the pump setting under the following conditions: atmospheric pressure = 0.983 bar, friction losses in the suction pipe equal to 3 velocity heads.

5. For the turbine sketched in Example 3, if 50 kW is extracted while the pressure gauges at A and B read 1.4 bar and -0.3 bar respectively, what is the water volumetric flow?

6. Heavy fuel oil flows from A to B through 104.4 m of horizontal 153 mm steel pipe. The pressure at A is 1.069 MPa and at B is 34.48 kPa. The kinematic viscosity is 412.5 x 10^{-6} m^2/s and the relative density is 0.918. What is the volumetric flow?

7. Points A and B are 1224 m apart along a new 153 mm I.D. steel pipe. Point B is 15.39 m higher than A and the pressures at A and B are 848 kPa and 335 kPa respectively. How much medium fuel oil will flow from A to B if the relative density is 0.854 and the kinematic viscosity is 3.83 x 10^{-6} m^2/s?

HEAT TRANSFER

The design and analysis of solar energy conversion systems requires an understanding of the principal modes of heat transfer: conduction, convection, and radiation. In this set of notes we will examine these mechanisms and see how they may be combined to facilitate the thermal analysis of solar collectors and heat storage systems.

Conduction

Conduction is the only mode of heat transfer in opaque solid media. The rate of heat transfer is given by Fourier's law:

$$Q = -kA \frac{dT}{dx} \tag{1}$$

where k is the thermal conductivity of the material, A is the area available for heat transfer, and dT/dx is the temperature gradient. The negative sign is required because dT/dx is itself negative since heat is transferred in the direction of decreasing temperature. If the thermal conductivity is independent of temperature, equation 1 may be integrated directly to give

$$Q = kA \frac{\Delta T}{\Delta x} \tag{2}$$

where ΔT is the temperature difference and Δx is the thickness of the material through which heat is being conducted.

The units of thermal conductivity are Btu/hr ft°F or W/m K in S.I. units. The rate of heat transfer, Q, will then have units of Btu/hr or Watts (W). The following conversion factors apply.

1 Btu/hr = 0.2931 Watts
1 Btu/hr ft °F = 1.731 W/m K

Example 1

The glass cover of a solar collector has an area of 30 square feet and a thickness of 5/16 inches. The thermal conductivity of the glass is 0.5 Btu/hr ft°F. Determine the rate of heat transfer through the glass if the outside surface temperature is 50°F and the inside surface temperature is at 75°F.

Solution: The temperature difference, ΔT, is 25°F

also Δx = 5/16 inch
k = 0.5 Btu/hr ft °F
A = 32 ft^2

Hence from equation 2

$$Q = 0.5 \times 32 \times \frac{(75 - 50) \times 12}{5/16} = 15{,}360 \text{ Btu/hr}$$

For many substances k is, in fact, a linear function of temperature in which case the thermal conductivity should be evaluated at the mean temperature, i.e. at $(T_1 + T_2)/2$.

It is usual to determine conduction heat transfer rates by working in terms of the total <u>resistance</u> to the transfer of heat. One may then write:

$$Q = \frac{\Delta T_{overall}}{\Sigma R} \tag{3}$$

where $\Delta T_{overall}$ is the temperature difference between inner and outer surfaces and ΣR is the sum of the resistances to the transfer of heat. From equation 2 it is clear that the resistances may be evaluated as

$$\Sigma R = \Sigma(\Delta x/kA) \tag{4}$$

<u>Example 2</u>

The walls of a house are constructed as follows:

	material	thickness	conductivity
outside	brick	0.1 m	0.7 W/m K
	insulation	0.2 m	0.065 W/m K
inside	plaster board	0.03 m	0.48 W/m K

If the temperature difference across the inside and outside surfaces is 20°C, determine the rate of heat transfer due to conduction if the total wall area is 80 m^2.

The resistances are found as follows:

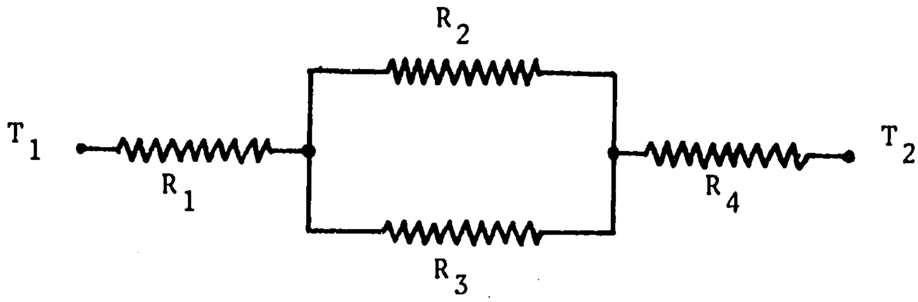
brick	R = 0.1/0.7 X 80	= 0.00179 K/W
insulation	R = 0.2/0.065 X 80	= 0.03846 K/W
plaster board	R = 0.03/0.48 X 80	= 0.00078 K/W

so $\Sigma R = 0.04103$ K/W

The heat transfer is therefore

$$Q = 20/0.04103 = 487.4 \text{ Watts}$$

The concept of a resistance to heat transfer is analogous to electrical resistance in an electric circuit. The wall structure of example 2 is an example of resistances in series. For some composite structures resistances may be also in parallel. For example, where there are two conductive routes between two surfaces the network may be presented as:



The overall resistance to heat transfer would then be calculated as

$$\Sigma R = R_1 + \frac{R_2 R_3}{R_2 + R_3} + R_4$$

This form of analysis may also be extended to cylinders or pipes where the resistance to conductive heat transfer is given by

$$R = \frac{\ln (r_o/r_i)}{2\pi kL} \qquad (5)$$

where r_i and r_o are the inner and outer radii of the relevant surface and L is the length of the pipe. The total conduction heat transfer is then found, as before, from equation 3.

Two further terms appear in the literature. The first is conductance, C, which is simply the reciprocal of resistance, i.e.

$$C = 1/R$$

but which may be given for a specified thickness of material on a unit area basis. The second is the overall heat transfer coefficient, U, which is simply the reciprocal of the sum of the resistances, i.e.

$$U = 1/\Sigma R$$

but which may also be given on a unit area basis in which case equation 3 is written as

$$Q = UA\Delta T \tag{6}$$

In all calculations it is important to closely examine the units of the relevant data to ensure that the appropriate equation is being used and that the calculation is dimensionally consistent.

Design Values of Various Building and Insulation Materials[a]

Table 1 Conductivities (k), Conductances (C), and Resistances (R) of Various Construction Materials

Material	Description	Density (Lb per Cu Ft)	Mean Temp F	Conductivity (k)	Conductance (C)	Resistance (R) Per inch thickness (1/k)	Resistance (R) For thickness listed (1/C)	Specific Heat, Btu per (lb) (F deg)
BUILDING BOARD Boards, Panels, Subflooring, Sheathing, Woodbased Panel Products	Asbestos-cement board	120	75	4.0	—	0.25	—	
	Asbestos-cement board ⅛ in.	120	75	—	33.00	—	0.035	
	Asbestos-cement board ¼ in.	120	75	—	16.50	—	0.07	
	Gypsum or plaster board ⅜ in.	50	75	—	3.10	—	0.32	
	Gypsum or plaster board ½ in.	50	75	—	2.25	—	0.45	
	Plywood	34	75	0.80	—	1.25	—	0.29
	Plywood ¼ in.	34	75	—	3.20	—	0.31	0.29
	Plywood ⅜ in.	34	75	—	2.13	—	0.47	0.29
	Plywood ½ in.	34	75	—	1.60	—	0.62	0.29
	Plywood or wood panels ¾ in.	34	75	—	1.07	—	0.93	0.29
	Insulating board							
	Sheathing, regular density ½ in.	18	75	—	0.76	—	1.32	0.31
	25/32 in.	18	75	—	0.49	—	2.06	0.31
	Sheathing intermediate density ½ in.	22	75	—	0.82	—	1.22	0.31
	Nail-base sheathing ½ in.	25	75	—	0.88	—	1.14	0.31
	Shingle backer ⅜ in.	18	75	—	1.06	—	0.94	0.31
	Shingle backer 5/16 in.	18	75	—	1.28	—	0.78	0.31
	Sound deadening board ½ in.	15		—	0.74	—	1.35	0.30
	Tile and lay-in panels, plain or acoustic	18	75	0.40	—	2.50	—	0.32
	½ in.	18	75	—	0.80	—	1.25	0.32
	¾ in.	18	75	—	0.53	—	1.89	0.32
	Laminated paperboard	30	75	0.50	—	2.00	—	
	Homogeneous board from repulped paper	30	75	0.50	—	2.00	—	0.28
	Hardboard							
	Medium density siding 7/16 in.	40	75	—	1.49	—	0.67	0.28
	Other medium density	50	75	0.73	—	1.37	—	0.31
	High density, service temp. service, underlay	55	75	0.82	—	1.22	—	0.33
	High density, std. tempered	63	75	1.00	—	1.00	—	0.33
	Particleboard							
	Low density	37	75	0.54	—	1.85	—	0.31
	Medium density	50	75	0.94	—	1.06	—	0.31
	High density	62.5	75	1.18	—	0.85	—	0.31
	Underlayment ⅝ in.	40	75	—	1.22	—	0.82	0.29
	Wood subfloor ¾ in.		75	—	1.06	—	0.94	0.34
BUILDING PAPER	Vapor—permeable felt	—	75	—	16.70	—	0.06	
	Vapor—seal, 2 layers of mopped 15 lb felt	—	75	—	8.35	—	0.12	
	Vapor—seal, plastic film	—	75	—	—	—	Negl.	
FINISH FLOORING MATERIALS	Carpet and fibrous pad	—	75	—	0.48	—	2.08	
	Carpet and rubber pad	—	75	—	0.81	—	1.23	0.34
	Cork tile ⅛ in.	—	75	—	3.60	—	0.28	
	Terrazzo 1 in.	—	75	—	12.50	—	0.08	
	Tile—asphalt, linoleum, vinyl, rubber	—	75	—	20.00	—	0.05	0.30
INSULATING MATERIALS Blanket and Katt	Mineral Fiber, fibrous form processed from rock, slag, or glass							
	approx. 2–2¾ in.	—	75	—	—	—	7	0.18
	approx. 3–3½ in.	—	75	—	—	—	11	0.18
	approx. 5¼–6½ in.	—	75	—	—	—	19	0.18
Board and Slabs	Cellular glass	9	75	0.40	—	2.50	—	0.24
	Glass fiber, organic bonded	4–9	75	0.25	—	4.00	—	0.19
	Expanded rubber (rigid)	4.5	75	0.22	—	4.55	—	
	Expanded polystyrene extruded, plain	1.8	75	0.25	—	4.00	—	0.29
	Expanded polystyrene extruded, (R-12 exp.)	2.2	75	0.20	—	5.00	—	0.29
	Expanded polystyrene extruded, (R-12 exp.) (Thickness 1 in. and greater)	3.5	75	0.19	—	5.26	—	0.29
	Expanded polystyrene, molded beads	1.0	75	0.28	—	3.57	—	0.29
	Expanded polyurethane (R-11 exp.)	1.5	75	0.16	—	6.25	—	0.38
	(Thickness 1 in. or greater)	2.5						0.38
	Mineral fiber with resin binder	15	75	0.29	—	3.45	—	0.17
	Mineral fiberboard, wet felted Core or roof insulation	16–17	75	0.34	—	2.94	—	

Notes: a. From ASHRAE *Handbook of Fundamentals*.

Table 1 *Continued*

Material	Description	Density (Lb per Cu Ft)	Mean Temp F	Conductivity (k)	Conductance (C)	Resistance (R) Per inch thickness (1/k)	Resistance (R) For thickness listed (1/C)	Specific Heat, Btu per (lb)(F deg)
BOARD AND SLABS (*Continued*)	Acoustical tile	18	75	0.35	—	2.86	—	
	Acoustical tile	21	75	0.37	—	2.73	—	
	Mineral fiberboard, wet molded Acoustical tile	23	75	0.42	—	2.38	—	
	Wood or cane fiberboard							
	Acoustical tile ½ in.	—	75	—	0.80	—	1.25	0.30
	Acoustical tile ¾ in.	—	75	—	0.53	—	1.89	0.30
	Interior finish (plank, tile)	15	75	0.35	—	2.86	—	0.32
	Insulating roof deck							
	Approximately 1½ in.	—	75	—	0.24	—	4.17	
	Approximately 2 in.	—	75	—	0.18	—	5.56	
	Approximately 3 in.	—	75	—	0.12	—	8.33	
	Wood shredded (cemented in preformed slabs)	22	75	0.60	—	1.67	—	0.38
LOOSE FILL	Cellulose insulation (milled paper or wood pulp)	2.5–3	75	0.27	—	3.70	—	0.33
	Sandust or shavings	0.8–1.5	75	0.45	—	2.22	—	0.33
	Wood fiber, softwoods	2.0–3.5	75	0.30	—	3.33	—	0.33
	Perlite, expanded	5.0–8.0	75	0.37	—	2.70	—	
	Mineral fiber (rock, slag or glass)							
	approx. 3 in.	—	75	—	—	9	—	0.18
	approx. 4½ in.	—	75	—	—	13	—	0.18
	approx. 6¼ in.	—	75	—	—	19	—	0.18
	approx. 7½ in.	—	75	—	—	24	—	0.18
	Silica aerogel	7.6	75	0.17	—	5.88	—	
	Vermiculite (expanded)	7.0–8.2	75	0.47	—	2.13	—	
		4.0–6.0	75	0.44	—	2.27	—	
ROOF INSULATION	Preformed, for use above deck							
	Approximately ½ in.	—	75	—	0.72	—	1.39	
	Approximately 1 in.	—	75	—	0.36	—	2.78	
	Approximately 1½ in.	—	75	—	0.24	—	4.17	
	Approximately 2 in.	—	75	—	0.19	—	5.56	
	Approximately 2½ in.	—	75	—	0.15	—	6.67	
	Approximately 3 in.	—	75	—	0.12	—	8.33	
	Cellular glass	9	75	0.40	—	2.50	—	0.24
MASONRY MATERIALS CONCRETES	Cement mortar	116		5.0	—	0.20	—	
	Gypsum-fiber concrete 87½% gypsum, 12½% wood chips	51		1.66	—	0.60	—	
	Lightweight aggregates including expanded shale, clay or slate; expanded slags; cinders; pumice; vermiculite; also cellular concretes	120		5.2	—	0.19	—	
		100		3.6	—	0.28	—	
		80		2.5	—	0.40	—	
		60		1.7	—	0.59	—	
		40		1.15	—	0.86	—	
		30		0.90	—	1.11	—	
		20		0.70		1.43		
	Sand and gravel or stone aggregate (oven dried)	140		9.0	—	0.11	—	
	Sand and gravel or stone aggregate (not dried)	140		12.0	—	0.08	—	
	Stucco	116		5.0	—	0.20	—	
MASONRY UNITS	Brick, common	120	75	5.0	—	0.20	—	
	Brick, face	130	75	9.0	—	0.11	—	
	Clay tile, hollow:							
	1 cell deep 3 in.	—	75	—	1.25	—	0.80	
	1 cell deep 4 in.	—	75	—	0.90	—	1.11	
	2 cells deep 6 in.	—	75	—	0.66	—	1.52	
	2 cells deep 8 in.	—	75	—	0.54	—	1.85	
	2 cells deep 10 in.	—	75	—	0.45	—	2.22	
	3 cells deep 12 in.	—	75	—	0.40	—	2.50	
	Concrete blocks, three oval core:							
	Sand and gravel aggregate 4 in.	—	75	—	1.40	—	0.71	
	8 in.	—	75	—	0.90	—	1.11	
	12 in.	—	75	—	0.78	—	1.28	
	Cinder aggregate 3 in.	—	75	—	1.16	—	0.86	
	4 in.	—	75	—	0.90	—	1.11	
	8 in.	—	75	—	0.58	—	1.72	
	12 in.	—	75	—	0.53	—	1.89	
	Lightweight aggregate (expanded shale, clay, slate or slag; pumice) 3 in.	—	75	—	0.79	—	1.27	
	4 in.	—	75	—	0.67	—	1.50	
	8 in.	—	75	—	0.50	—	2.00	
	12 in.	—	75	—	0.44	—	2.27	

Table 1 *Concluded*

Material	*Description*	*Density (Lb per Cu Ft)*	*Mean Temp F*	*Conductivity (k)*	*Conductance (C)*	*Resistance (R)*		*Specific Heat Btu per (lb) (F deg)*
						Per inch thickness (1/k)	*For thickness listed (1/C)*	
	Concrete blocks, rectangular core.							
	Sand and gravel aggregate							
	2 core, 8 in. 36 lb.	—	45	—	0.96	—	*1.04*	
	Same with filled cores	—	45	—	0.52	—	*1.93*	
	Lightweight aggregate (expanded shale, clay, slate or slag, pumice):							
	3 core, 6 in. 19 lb.	—	45	—	0.61	—	*1.65*	
	Same with filled cores	—	45	—	0.33	—	*2.99*	
	2 core, 8 in. 24 lb.	—	45	—	0.46	—	*2.18*	
	Same with filled cores	—	45	—	0.20	—	*5.03*	
	3 core, 12 in. 38 lb.	—	45	—	0.40	—	*2.48*	
	Same with filled cores	—	45	—	0.17	—	*5.82*	
	Stone, lime or sand	—	75	12.50	—	*0.08*	—	
	Gypsum partition tile:							
	3 × 12 × 30 in. solid	—	75	—	0.79	—	*1.26*	
	3 × 12 × 30 in. 4-ce[illegible]	—	75	—	0.74	—	*1.35*	
	4 × 12 × 30 in. 3-c[illegible]	—	75	—	0.60	—	*1.67*	
PLASTERING MATERIALS	Cement plaster, sand aggregate	116	75	5.0	—	*0.20*	—	
	Sand aggregate ⅜ in.	—	75	—	13.3	—	*0.08*	
	Sand aggregate ¾ in.	—	75	—	6.66	—	*0.15*	
	Gypsum plaster:							
	Lightweight aggregate ½ in.	45	75	—	3.12	—	*0.32*	
	Lightweight aggregate ⅝ in.	45	75	—	2.67	—	*0.39*	
	Lightweight agg. on metal lath ¾ in.	—	75	—	2.13	—	*0.47*	
	Perlite aggregate	45	75	1.5	—	*0.67*	—	
	Sand aggregate	105	75	5.6	—	*0.18*	—	
	Sand aggregate ½ in.	105	75	—	11.10	—	*0.09*	
	Sand aggregate ⅝ in.	105	75	—	9.10	—	*0.11*	
	Sand aggregate on metal lath ¾ in.	—	75	—	7.70	—	*0.1*	
	Vermiculite aggregate	45	75	1.7	—	*0.59*	—	
ROOFING	Asbestos-cement shingles	120	75	—	4.76	—	*0.21*	
	Asphalt roll roofing	70	75	—	6.50	—	*0.15*	
	Asphalt shingles	70	75	—	2.27	—	*0.44*	
	Built-up roofing ⅜ in.	70	75	—	3.00	—	*0.33*	0.35
	Slate ½ in.	—	75	—	20.00	—	*0.05*	
	Wood shingles, plain a plastic film faced	—	75	—	1.06	—	*0.94*	0.31
SIDING MATERIALS (ON FLAT SURFACE)	Shingles							
	Asbestos-cement	120	75	—	4.76	—	*0.21*	
	Wood, 16 in., 7½ exposure	—	75	—	1.15	—	*0.87*	0.31
	Wood, double, 16-in., 12-in. exposure	—	75	—	0.84	—	*1.19*	0.31
	Wood, plus insul. backer board, 5/16 in.	—	75	—	0.71	—	*1.40*	0.31
	Siding							
	Asbestos-cement, ¼ in., lapped	—	75	—	4.76	—	*0.21*	
	Asphalt roll siding	—	75	—	6.50	—	*0.15*	
	Asphalt insulating siding (½ in. bd.)	—	75	—	0.69	—	*1.46*	
	Wood, drop, 1 × 8 in.	—	75	—	1.27	—	*0.79*	0.31
	Wood, bevel, ½ × 8 in., lapped	—	75	—	1.23	—	*0.81*	0.31
	Wood, bevel, ¾ × 10 in., lapped	—	75	—	0.95	—	*1.05*	0.31
	Wood, plywood, ¾ in., lapped	—	75	—	1.59	—	*0.59*	0.29
	Aluminum or Steel, over sheathing							
	Hollow-backed	—	—	—	1.61	—	*0.61*	
	Insulating-board backed nominal ⅜ in.	—	—	—	0.55	—	*1.82*	
	Insulating-board backed nominal ⅜ in. foil backed	—	—	—	0.34	—	*2.[illegible]6*	
	Architectural glass	—	75	—	10.00	—	*0.10*	
WOODS	Maple, oak, and similar hardwoods	45	75	1.10	—	*0.91*	—	0.30
	Fir, pine, and similar softwoods	32	75	0.80	—	*1.25*	—	0.33
	Fir, pine, and similar softwoods ¾ in.	32	75	—	1.06	—	*0.94*	0.33
	1½ in.	32	75	—	0.53	—	*1.89*	0.33
	2½ in.	32	75	—	0.32	—	*3.12*	0.33
	3½ in.	32	75	—	0.23	—	*4.35*	0.33

U-Values of Windows, Skylights, and Light-transmitting Partitions[a]

PART A—VERTICAL PANELS (EXTERIOR WINDOWS AND PARTITIONS)—FLAT GLASS, GLASS BLOCK, AND PLASTIC SHEET

Description	Exterior Winter	Exterior Summer	Interior
Flat Glass			
single glass	1.13	1.06	0.73
insulating glass—double			
3/16 in. air space	0.69	0.64	0.51
1/4 in. air space	0.65	0.61	0.49
1/2 in. air space	0.58	0.56	0.46
insulating glass—triple			
1/4 in. air spaces	0.47	0.45	0.38
1/2 in. air spaces	0.36	0.35	0.30
storm windows			
1 in.–4 in. air space	0.56	0.54	0.44
Glass Block			
6 × 6 × 4 in. thick	0.60	0.57	0.46
8 × 8 × 4 in. thick	0.56	0.54	0.44
—with cavity divider	0.48	0.46	0.38
12 × 12 × 4 in. thick	0.52	0.50	0.41
—with cavity divider	0.44	0.42	0.36
12 × 12 × 2 in. thick	0.60	0.57	0.46
Single Plastic Sheet,	1.09	1.00	0.70

PART B—HORIZONTAL PANELS (SKYLIGHTS)—FLAT GLASS, GLASS BLOCK, AND PLASTIC BUBBLES

Description	Exterior Winter	Exterior Summer	Interior
Flat Glass			
single glass	1.22	0.83	0.96
insulating glass—double —			
3/16 in. air space	0.75	0.49	0.62
1/4 in. air space	0.70	0.46	0.59
1/2 in. air space	0.66	0.44	0.56
Glass Block			
11 × 11 × 3 in. thick with cavity divider	0.53	0.35	0.44
12 × 12 × 4 in. thick with cavity divider	0.51	0.34	0.42
Plastic Bubbles			
single walled	1.15	0.80	—
double walled	0.70	0.46	—

PART C—ADJUSTMENT FACTORS FOR VARIOUS WINDOW TYPES (MULTIPLY U VALUES IN PARTS A AND B BY THESE FACTORS)

Window Description	Single Glass	Double or Triple Glass	Storm Windows
All Glass	1.00	1.00	1.00
Wood Sash—80% Glass	0.90	0.95	0.90
Wood Sash—60% Glass	0.80	0.85	0.80
Metal Sash—80% Glass	1.00	1.20	1.20

Notes: a. From ASHRAE *Handbook of Fundamentals*; units are Btu/hr-ft²-°F.

U-Values of Solid Wood Doors[a]

Thickness[b]	Winter No Storm Door	Winter Storm Door[c] Wood	Winter Storm Door[c] Metal	Summer No Storm Door
1 in.	0.64	0.30	0.39	0.61
1¼ in.	0.55	0.28	0.34	0.53
1½ in.	0.49	0.27	0.33	0.47
2 in.	0.43	0.24	0.29	0.42

Notes: a. Units are Btu/hr-ft²-°F; from ASHRAE *Handbook of Fundamentals*.
b. Nominal thickness.
c. Values for wood storm doors are for approximately 50 percent glass; for metal storm doors values apply for any percent of glass.

Air-Space Resistances (*R*) for 50°F Mean Temperature[a]

Position of Air Space	Direction of Heat Flow	Air Space Bounded by Ordinary Materials 0.75-inch *R*	Air Space Bounded by Ordinary Materials 4-inch *R*	Air Space Bounded by Aluminum Foil 0.75-inch *R*	Air Space Bounded by Aluminum Foil 4-inch *R*
Horizontal	Upward	0.78	0.85	1.67	2.06
Horizontal	Downward	1.02	1.23	3.55	8.94
Vertical	Horizontal	0.96	0.94	2.80	2.62

Notes: a. From ASHRAE *Handbook of Fundamentals*; units of hr-ft²-°F/Btu.

Thermal Conductivity (*k*) of Various Construction Materials

Form	Material (Composition)	Accepted Max Temp for Use, F	Typical Density (lb/cu ft)	Typical Conductivity k at Mean Temp F													
				−100	−75	−50	−25	0	25	50	75	100	200	300	500	700	900
BLANKETS & FELTS	MINERAL FIBER (Rock, Slag, or Glass) Blanket, Metal Reinforced	1200	6–12									0.26	0.32	0.39	0.64		
		1000	2.5–6									0.24	0.31	0.40	0.61		
	Mineral Fiber, Glass Blanket, Flexible, Fine-Fiber Organic Bonded	350	0.65				0.25	0.26	0.28	0.30	0.33	0.36	0.53				
			0.75				0.24	0.25	0.27	0.29	0.32	0.34	0.48				
			1.0				0.23	0.24	0.25	0.27	0.29	0.32	0.43				
			1.5				0.21	0.22	0.23	0.25	0.27	0.28	0.37				
			2.0				0.20	0.21	0.22	0.23	0.25	0.26	0.33				
			3.0				0.19	0.20	0.21	0.22	0.23	0.24	0.31				
	Blanket, Flexible, Textile-Fiber Organic Bonded	350	0.65				0.27	0.28	0.29	0.30	0.31	0.32	0.50	0.68			
			0.75				0.26	0.27	0.28	0.29	0.31	0.32	0.48	0.66			
			1.0				0.24	0.25	0.26	0.27	0.29	0.31	0.45	0.60			
			1.5				0.22	0.23	0.24	0.25	0.27	0.29	0.39	0.51			
			3.0				0.20	0.21	0.22	0.23	0.24	0.25	0.32	0.41			
	Felt, Semi-Rigid Organic Bonded	400	3–8						0.24	0.25	0.26	0.27	0.35	0.44			
	Laminated & Felted	850	3	0.16	0.17	0.18	0.19	0.20	0.21	0.22	0.23	0.24	0.35	0.55			
	Without Binder	1200	7.5											0.35	0.45	0.60	
	VEGETABLE & ANIMAL FIBER Hair Felt or Hair Felt plus Jute	180	10						0.26	0.28	0.29	0.30					
BLOCKS, BOARDS & PIPE INSULATION	ASBESTOS Laminated Asbestos Paper	700	30									0.40	0.45	0.50	0.60		
	Corrugated & Laminated Asbestos Paper 4-ply	300	11–13								0.54	0.57	0.68				
	6-ply	300	15–17								0.49	0.51	0.59				
	8-ply	300	18–20								0.47	0.49	0.57				
	MOLDED AMOSITE & BINDER	1500	15–18									0.32	0.37	0.42	0.52	0.62	0.72
	85% MAGNESIA	600	11–12									0.35	0.38	0.42			
	CALCIUM SILICATE	1200	11–13									0.38	0.41	0.44	0.52	0.62	0.72
		1800	12–15												0.63	0.74	0.95
	CELLULAR GLASS	800	9			0.32	0.33	0.35	0.36	0.38	0.40	0.42	0.48	0.55			
	DIATOMACEOUS SILICA	1600	21–22												0.64	0.68	0.72
		1900	23–25												0.70	0.75	0.80
	MINERAL FIBER Glass, Organic Bonded, Block and Boards	400	3–10	0.16	0.17	0.18	0.19	0.20	0.22	0.24	0.25	0.26	0.33	0.40			
	Non-Punking Binder	1000	3–10									0.26	0.31	0.38	0.52		
	Pipe Insulation, slag or glass	350	3–4					0.20	0.21	0.22	0.23	0.24	0.29				
		500	3–10					0.20	0.22	0.24	0.25	0.26	0.33	0.40			
	Inorganic Bonded-Block	1000	10–15									0.33	0.38	0.45	0.55		
		1800	15–24									0.32	0.37	0.42	0.52	0.62	0.74
	Pipe Insulation slag or glass	1000	10–15									0.33	0.38	0.45	0.55		
	MINERAL FIBER Resin Binder		15			0.23	0.24	0.25	0.26	0.28	0.29						
	Rigid Polystryrene Extruded, R-12 exp	170	3.5	0.16	0.16	0.15	0.16	0.16	0.17	0.18	0.19	0.20					
	Extruded, R-12 exp	170	2.2	0.16	0.16	0.17	0.16	0.17	0.18	0.19	0.20						
	Extruded	170	1.8	0.17	0.18	0.19	0.20	0.21	0.23	0.24	0.25	0.27					
	Molded Beads	170	1	0.18	0.20	0.21	0.23	0.24	0.25	0.26	0.28						
	Polyurethane R-11 exp	210	1.5-2.5	0.16	0.17	0.18	0.18	0.18	0.17	0.16	0.16	0.17					
	RUBBER, Rigid Foamed	150	4.5						0.20	0.21	0.22	0.23					
	VEGETABLE & ANIMAL FIBER Wool Felt (Pipe Insulation)	180	20						0.28	0.30	0.31	0.33					
INSULATING CEMENTS	MINERAL FIBER (Rock, Slag, or Glass) With Colloidal Clay Binder	1800	24–30									0.49	0.55	0.61	0.73	0.86	
	With Hydraulic Setting Binder	1200	30–40									0.75	0.80	0.85	0.95		
LOOSE FILL	Cellulose insulation (Milled pulverized paper or wood pulp)		2.5–3							0.26	0.27	0.29					
	Mineral fiber, slag, rock or glass		2–5			0.19	0.21	0.23	0.25	0.26	0.28	0.31					
	Perlite (expanded)		5–8	0.25	0.27	0.29	0.30	0.32	0.34	0.35	0.37	0.39					
	Silica aerogel		7.6			0.13	0.14	0.15	0.15	0.16	0.17	0.18					
	Vermiculite (expanded)		7–8.2			0.39	0.40	0.42	0.44	0.45	0.47	0.49					
			4–6			0.34	0.35	0.38	0.40	0.42	0.44	0.46					

Convection

Heat transfer takes place between a solid surface and a fluid whenever a temperature difference exists. If the fluid is in laminar motion then heat transfer is considered to take place largely by conduction. Even when the bulk of the fluid is in turbulent motion, the layer immediately adjacent to the wall is in laminar motion. Where mixing of the fluid particles occurs, the heat is transferred by convection. Convection may be either forced or natural (free) convection depending on whether the fluid motion is imposed or whether it occurs because of differences in density caused by temperature changes. A buffer layer exists between the laminar layer and the turbulent bulk. In this intermediate region the heat transfer is characterized by both conduction and convection.

Since the laminar layer presents a much greater resistance to heat transfer than either the buffer region or the turbulent bulk, most of the temperature resistance occurs across the laminar layer. The entire resistance to heat transfer is, for practical purposes, regarded as being concentrated in this thin layer. Thus the conductance term for a fluid is generally referred to as the film heat transfer coefficient.

The heat transfer brought about by convection is generally computed in a manner analogous to heat transfer by conduction. That is, one may write

$$Q = h_c A \Delta T \tag{7}$$

where h_c = convective heat transfer coefficient, often called a film coefficient

A = area available for convective heat transfer

ΔT = temperature difference between the surface and the bulk of the fluid.

Convective heat transfer may also be treated within the framework of a thermal resistance network in a manner analogous to conduction. The thermal resistance to convection is given by

$$R_c = \frac{1}{h_c A} \tag{8}$$

As an example, consider the heat transfer from the interior of a room at T_i through a wall to the air outside at temperature T_o. Heat is first transferred by free convection to the interior surface of the wall, then by conduction through the wall to the exterior surface, and finally from the exterior surface to the air outside. There are, therefore, three resistances to the transfer of heat. The total resistance R is given by:

$$R = \frac{1}{h_{ci}A} + \frac{x}{kA} + \frac{1}{h_{co}A}$$

where h_{ci}, h_{co} are the inner and outer convective film coefficients, x is the wall thickness, and k is the thermal conductivity of the wall material. The overall heat transfer is simply:

$$Q = \frac{T_i - T_o}{R}$$

Film Coefficients

Convective heat transfer film coefficients are generally determined experimentally and many correlations have been reported in the literature. The data are generally structured in terms of five dimensionless numbers. These are:

Nusselt number (Nu) = hL/k
Reynolds number (Re) = $\rho uL/\mu$
Prandtl number (Pr) = $\mu C_p/k$
Grasshof number (Gr) = $g\beta\Delta T\ L^3\rho^2/\mu^2$
Rayleigh number (Ra) = $g\beta\Delta T\ L^3\rho^2 C_p/\mu k$

where h = heat transfer coefficient
L = characteristic dimension
k = thermal conductivity
u = fluid velocity
ρ = fluid density
μ = viscosity
C_p = specific heat (constant pressure)
β = coefficient of expansion of the fluid
ΔT = temperature difference
g = acceleration due to gravity (9.81 m/s^2 or 32.2 ft/s^2)

All these terms are well defined except for the characteristic dimension L. This term will depend on the configuration of the system being examined. For ideal gases β is equal to the reciprocal of absolute temperature, i.e. $\beta \approx 1/T$. This is a good enough approximation for air.

In general, the Nusselt number, for convection, can be related to the other dimensionless numbers by equations of the form:

$Nu = C(Re^n Pr^m)$ forced convection
and $Nu = C(Gr^n Pr^m)$ free convection

where C, n, m are empirical constants which must be determined experimentally. Once the Nusselt number has been determined for the system under consideration, the film coefficient follows directly from

$$h = (Nu)k/L \tag{9}$$

The following correlations are applicable for common system configurations found in solar energy systems.

1. Laminar Flow in Pipes and Ducts

$$Nu = 1.86 \left(Re.Pr.\frac{D_h}{L}\right)^{1/3} \left(\frac{\mu_b}{\mu_w}\right)^{0.14} \quad (10)$$

applicable for $Re.Pr.D_h/L > 10$
and $Re < 2100$
D_h is the hydraulic diameter of the pipe or duct, given by

$$D_h = \frac{4 \times \text{flow area}}{\text{wetted perimeter}} \quad (11)$$

μ_b is the viscosity at the bulk (mean) temperature of the fluid (use this temperature for Pr also); μ_w is the viscosity of the fluid at the wall temperature.

also $Nu = hD_h/k$
$Re = \rho u D_h/\mu_b$

and L is the length of the pipe or duct.

If the conduit is short, i.e. $L/D_h < 60$, Nu may be multiplied by a factor equal to

$$1 + (D_h/L)^{0.7}$$

2. Turbulent Flow in Pipes and Ducts

When the Reynolds number is above 6000 then fluid flow is fully turbulent and heat transfer is enhanced. The Nusselt number may be estimated as

$$Nu = 0.023\, Re^{0.8} Pr^{1/3} \left(\frac{\mu_b}{\mu_w}\right)^{0.14} \quad (12)$$

applicable for $Re > 10{,}000$
$0.7 < Pr < 7000$

and properties based on bulk temperatures.

If the tube is short increase Nu by

$$1 + (D_h/L)^{0.7}$$

3. Turbulent Flow Between Flat Plates

One side heated:

$$Nu = 0.0196\, Re^{0.8} Pr^{1/3} \quad (13)$$

where Re and Nu are based on the hydraulic diameter.

4. Flow in a Helical Coil

For flow in a helical coil the value of the heat transfer coefficient calculated for a straight tube should be multiplied by

$$1 + 3.5 \left(\frac{\text{tube diameter}}{\text{coil diameter}}\right)$$

5. Free Convection from Surfaces

a) Vertical surfaces, L = vertical dimension, < 3 ft

$$\begin{array}{rll} Ra < 10^4, & Nu = 1.36\ Ra^{0.2} & \\ 10^4 < Ra < 10^9, & Nu = 0.59\ Ra^{0.25} & (14) \\ Ra > 10^9, & Nu = 0.13\ Ra^{1/3} & \end{array}$$

b) Horizontal Cylinder, L = diameter, < 8 ins

$$\begin{array}{rll} 1 < Ra < 10^4, & Nu = 1.09\ Ra^{0.2} & \\ 10^4 < Ra < 10^9, & Nu = 0.53\ Ra^{0.25} & (15) \\ Ra > 10^9, & Nu = 0.13\ Ra^{1/3} & \end{array}$$

c) Horizontal Flat Surfaces

$$\begin{array}{rll} 10^4 < Ra < 10^7, & Nu = 0.76\ Ra^{0.25} & (16) \\ 10^7 < Ra < 10^{10}, & Nu = 0.15\ Ra^{1/3} & \end{array}$$

The characteristic length, L, is four times the area divided by the perimeter.

d) Sphere, L = diameter

$$Nu = 2 + 0.45\ Ra^{0.25} \qquad (17)$$

In all the above correlations fluid properties are to be evaluated at temperature, T_f, where

$$T_f = 1/2\ (\text{surface temperature} + \text{ambient temperature})$$

6. Free Convection Between Two Parallel Surfaces

For air, the Nusselt number may be found as:

$$Nu = 1 + 1.44\ [1 - 1708/B]^+ \{1 - \frac{1708}{B} (\sin 1.8\beta)^{1.6}\}$$

$$+ [(B/5830)^{1/3} - 1]^+ \qquad (18)$$

where the meaning of the + exponent is that only the positive values of the term in the square brackets are to be used (i.e. use zero if the term is negative).

In equation (18) $B = Ra \cos \beta$ where Ra is the Rayleigh number and β is the angle between the surfaces and the horizontal. Ra is based on $L = d$, the distance between the plates. Equation 18 is valid for β between zero and 75°.

For inclinations between 75° and 90° the recommended relation for air is

$$Nu = \max \left[1,\ 0.288\ (A\ Ra \sin \beta)^{1/4},\ 0.039\ (Ra \sin \beta)^{1/3} \right] \qquad (19)$$

The constant A in equation 19 is the aspect ratio of the air layer, defined as the ratio of the thickness to the length along the layer measured along either surface in the upslope direction.

7. Air Flow over a Flat Surface

The calculation of heat transfer coefficients for flat heated surfaces exposed to wind does not appear to be well established. For smooth surfaces a rough approximation is given by the dimensional equations:

$$h = 4.5 + 2.9u \ ; \qquad h = W/m^2\,K\ , \qquad u = m/s$$
$$\text{or}\ h = 0.8 + 0.23u \ ; \qquad h = Btu/hr\ ft^2\,{}^\circ F\ , \qquad u = mph$$

Radiation

All heated bodies emit thermal electromagnetic radiation whose wavelengths and intensities are dependent upon the temperature of the body and its optical characteristics.

Thermal radiation is usually considered to lie within that part of the electromagnetic wave spectrum with a wavelength between 0.1 to 100 μm (microns). Solar radiation has most of its energy in the range between 0.1 and 3 μm. The visible part of the spectrum is between about 0.4 - 0.7 μm.

It can be shown that the energy density at a given wavelength is related to the monochromatic radiation emitted by a perfect radiator, usually called a black body, according to the relation.

$$E_{b\lambda} = \frac{C_1}{(e^{C2/\lambda T} - 1)\,\lambda^5} \; W/m^2 \cdot \mu m \tag{21}$$

where $C_1 = 3.7405 \times 10^8 \quad W.\ \mu m^4/m^2$

$C_2 = 1.43879 \times 10^4 \quad \mu m.K$

E_b is the monochromatic emissive power of a blackbody, defined as the energy emitted by a perfect radiator per unit wavelength, at the specified wavelength λ, per unit area and per unit time at the specified temperature T (in degrees Kelvin).

The total energy emitted by a blackbody can be obtained by integration over all wavelengths:

$$E_b = \int_0^\infty E_{b\lambda} d\lambda = \sigma T^4 \quad W/m^2 \tag{22}$$

where σ is called the Stefan-Boltzmann constant and is equal to 5.67×10^{-8} $W/m^2\ K^4$.

It is also of interest to know the wavelength corresponding to the maximum intensity of blackbody radiation. This may be determined from Wien's displacement law:

$$\lambda_{max}\, T = 2897.8 \ \mu m \tag{23}$$

For example, we can estimate the wavelength of the maximum intensity of the radiation emitted from the human body.

Taking the body temperature as 98.4°F or 37°C, we have

$$\lambda_{max} = \frac{2897.8}{37 + 273} = 9.34 \ \mu m$$

Table 4 Fraction of Blackbody Radiant Energy Between Zero and λT for even increments of λT

λT, μm K	$f_{0-\lambda T}$	λT, μm K	$f_{0-\lambda T}$
1000	0.0003	6200	0.7541
1100	0.0009	6300	0.7618
1200	0.0021	6400	0.7692
1300	0.0043	6500	0.7763
1400	0.0077	6600	0.7831
1500	0.0128	6700	0.7897
1600	0.0197	6800	0.7961
1700	0.0285	6900	0.8022
1800	0.0393	7000	0.8080
1900	0.0521	7100	0.8137
2000	0.0667	7200	0.8191
2100	0.0830	7300	0.8244
2200	0.1009	7400	0.8295
2300	0.1200	7500	0.8343
2400	0.1402	7600	0.8390
2500	0.1613	7700	0.8436
2500	0.1831	7800	0.8479
2700	0.2053	7900	0.8521
2800	0.2279	8000	0.8562
2900	0.2506	8100	0.8601
3000	0.2732	8200	0.8639
3100	0.2958	8300	0.8676
3200	0.3181	8400	0.8711
3300	0.3401	8500	0.8745
3400	0.3617	8600	0.8778
3500	0.3829	8700	0.8810
3600	0.4036	8800	0.8841
3700	0.4238	8900	0.8871
3800	0.4434	9000	0.8899
3900	0.4624	9100	0.8927
4000	0.4829	9200	0.8954
4100	0.4987	9300	0.8980
4200	0.5160	9400	0.9005
4300	0.5327	9500	0.9030
4400	0.5488	9600	0.9054
4500	0.5643	9700	0.9076
4600	0.5793	9800	0.9099
4700	0.5937	9900	0.9120
4800	0.6075	10000	0.9141
4900	0.6209	11000	0.9318
5000	0.6337	12000	0.9450
5100	0.6461	13000	0.9550
5200	0.6579	14000	0.9628
5300	0.6693	15000	0.9689
5400	0.6803	16000	0.9737
5500	0.6909	17000	0.9776
5600	0.7010	18000	0.9807
5700	0.7107	19000	0.9833
5800	0.7201	20000	0.9855
5900	0.7291	30000	0.9952
6000	0.7378	40000	0.9978
6100	0.7461	50000	0.9988

It is also useful to know what fraction of the total radiated energy is being emitted over a range of wavelengths. Table 4 shows the fraction of blackbody radiant energy emitted between zero and λT for increments of λT.

For example, we can determine the fraction of the sun's radiative energy output that lies within the visible part of the electromagnetic spectrum. The temperature of the surface of the sun is about 6000 K. The visible part of the EM spectrum lies approximately between 0.4 and 0.7 microns.

From Table 4 we have:

$$\lambda T = 0.7 \times 6000 = 4200 \qquad f\,(<4200) = 0.516$$
$$\lambda T = 0.4 \times 6000 = 2400 \qquad f\,(<2400) = 0.140$$

Fraction between = 0.376

so about 38% of the sun's output is visible.

Absorptance, Emittance and Reflectance

The absorptance, α, is the fraction of incident light of a given wavelength that is absorbed when light strikes an absorbing surface. The absorptance of a surface is therefore a function of the wavelength intensity distribution of the incident light.

The emittance, ε, is the fraction of the emittance of a perfect blackbody at a given wavelength emitted by a heated surface.

When radiation strikes a body some is reflected, some absorbed, and if the material is translucent, some is transmitted. It is clear that

$$\alpha + \tau + r = 1 \tag{24}$$

α = fraction absorbed
τ = fraction transmitted
r = fraction reflected

If a body is opaque then $\tau = 0$

The reflection of radiation can be specular or diffuse. When the angle of incidence is equal to the angle of reflection, the reflection is called specular. If the reflected radiation is uniformly distributed in all direction it is said to be diffuse. A real surface exhibits both kinds of reflection. A highly polished surface approaches specular reflection, a rough surface generally reflects diffusely.

At a particular wavelength, absorptance is equal to emittance. This relationship is essentially Kirchhoff's law:

$$\alpha_\lambda = \varepsilon_\lambda$$

For an opaque surface therefore

$$\varepsilon_\lambda = 1 - r_\lambda$$

and

$$\alpha_\lambda = 1 - r_\lambda$$

The subscript λ is important to note because, for most materials, α, ε, and r vary significantly with wavelength over the range of interest in solar energy systems. The few materials for which they do not vary with λ are termed gray bodies, and those with $\alpha = \varepsilon = 1$ for all wavelengths are termed blackbodies.

Infrared Radiation Heat Transfer Between Gray Surfaces

The majority of heat-transfer problems in solar energy applications involve radiation between two surfaces. For this situation and assuming:

1. The surfaces are gray and reflection is diffuse.
2. Surface temperatures are uniform.

The radiative heat transfer between the surfaces is given by

$$Q = \frac{\sigma(T_2^4 - T_1^4)}{\frac{1-\varepsilon_1}{\varepsilon_1 A_1} + \frac{1}{A_1 F_{12}} + \frac{1-\varepsilon_2}{\varepsilon_2 A_2}} \tag{25}$$

where subscripts 1 and 2 refer to the two surfaces, ε is the emittance, T is absolute temperature (Kelvin), A is area, and F_{12} is the view factor.

For the special case of radiation between two large parallel plates (i.e. as in flat-plate collectors) the areas A_1 and A_2 are equal, and F_{12} is unity. Equation 25 therefore reduces to:

$$Q = \frac{A\,\sigma(T_2^4 - T_1^4)}{\frac{1}{\varepsilon_1} + \frac{1}{\varepsilon_2} - 1} \tag{26}$$

Equation 26 also applies to radiation between two concentric long cylinders forming an annulus when the diameter ratio approaches unity.

The second special case is for a small body (surface 1) surrounded by a large enclosure (surface 2). Under these conditions, the area ratio A_1/A_2 approaches zero, F_{12} is again unity, and equation 25 becomes

$$Q = \varepsilon_1 A_1 \sigma(T_2^4 - T_1^4) \tag{27}$$

The result is independent of the surface properties of the large enclosure since virtually none of the radiation leaving the small object is reflected back from the large enclosure. The large enclosure effectively absorbs all radiation from the small body and thus acts like a black body. Equation 27 applies in the case of a flat plate radiating to the sky.

The sky can be considered as a black body at some equivalent sky temperature, T_s. The net radiation to a surface with emittance ε and temperature T is therefore found from

$$Q = \varepsilon A \sigma (T_s^4 - T^4) \tag{28}$$

Several relations have been proposed to relate T_s, for clear skies, to other measured meteorological variables. One simple relation is:

$$T_s = 0.0552\, T_a^{1.5} \tag{29}$$

where T_a is the local air temperature in degrees Kelvin.

It is possible to define heat transfer coefficients such that equations 26 and 27 reduce to simple form of equation 7. That is, we have

$$h = \frac{\sigma}{1/\varepsilon_1 + 1/\varepsilon_2 - 1} (T_2 + T_1)(T_2^2 + T_1^2) \tag{30}$$

or $$h = \varepsilon_1 \sigma (T_2 + T_1)(T_2^2 + T_1^2) \tag{31}$$

derived from equations 26 and 27 respectively.

Radiation Transmission Through Covers

The transmittance, reflectance, and absorption of solar radiation by translucent solar collector covers are functions of the incoming radiation, and the thickness, refractive index, and extinction coefficient of the material. Generally, the refractive index, n, and the extinction coefficient, K are functions of the wavelength of the radiation. However, for glass these properties may be taken as independent of wavelength.

Reflectance

For smooth surfaces the reflection of unpolarized radiation on passing from a medium 1 with a refractive index n_1 to medium 2 with refractive index n_2 is given by

$$r_{\perp} = \frac{\sin^2 (\Theta_2 - \Theta_1)}{\sin^2 (\Theta_2 + \Theta_1)} \tag{32}$$

$$r_{\parallel} = \frac{\tan^2 (\Theta_2 - \Theta_1)}{\tan^2 (\Theta_2 + \Theta_1)} \tag{33}$$

$$r(\Theta_1) = \frac{I_r}{I_i} = 1/2\ (r_{\perp} + r_{\parallel}) \tag{34}$$

where Θ_1 and Θ_2 are the angles of incidence and refraction as shown in Figure 1.

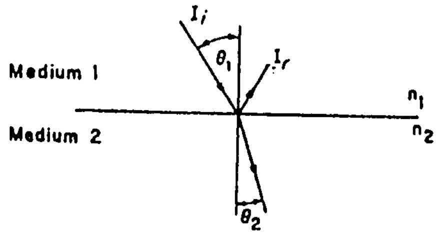


Figure 1 Angles of incidence and refraction in media having refractive indices n_1 and n_2.

Equation 30 represents the perpendicular component of unpolarized radiation $r_{\perp}$ and equation 31 represents the parallel component of unpolarized radiation, $r_{\parallel}$. Equation 32 then gives the reflection of unpolarized radiation as the average of the two components. The angles θ_1 and θ_2 are related to the indices of refraction by Snell's law

$$\frac{n_1}{n_2} = \frac{\sin \Theta_2}{\sin \Theta_1} \tag{35}$$

Thus if the angle of incidence and refractive indices are known, equations 32 through 35 are sufficient to calculate the reflectance of the single interface.

For radiation at normal incidence ($\Theta_1 = \Theta_2 = 0$) equations 34 and 35 may be combined to yield

$$r(0) = \frac{I_r}{I_i} = \left[\frac{(n_1 - n_2)}{(n_1 + n_2)}\right]^2 \tag{36}$$

Refractive indices for some common translucent materials are given below:

TABLE 5 Refractive Index for Various Substances in the Visible Range Based on Air

Material	Index of refraction
Air	1.000
Clean polycarbonate (PCO)	1.59
Diamond	2.42
Glass (solar collector type)	1.50–1.52
Plexiglass (polymethyl methacrylate, PMMA)	1.49
Mylar (polyethylene terephthalate, PET)	1.64
Quartz	1.54
Tedlar (polyvinyl fluoride, PVF)	1.45
Teflon (polyfluoroethylenepropylene, FEP)	1.34
Water–liquid	1.33
solid	1.31

Example 3

Calculate the reflectance of one surface of glass at normal incidence and at 60°. The average index of refraction of glass for the solar spectrum is 1.526 (for air $n \approx 1$).

At normal incidence, equation 36 may be written for $n_1 = 1$ as

$$r(0) = \left(\frac{n - 1}{n + 1}\right)^2$$

or

$$r(0) = \left(\frac{1.526 - 1}{1.526 + 1}\right)^2 = 0.0434$$

At an incidence angle of 60°, equation 35 gives the refraction angle Θ_2 as

$$\Theta_2 = \sin^{-1}\left(\frac{\sin 60}{1.526}\right) = 34.58°$$

Then from equation 34 the reflectance is

$$r(60) = \frac{1}{2}\left[\frac{\sin^2(34.58 - 60)}{\sin^2(34.58 + 60)} + \frac{\tan^2(34.58 - 60)}{\tan^2(34.58 + 60)}\right]$$

$$= \frac{1}{2}(0.185 + 0.001) = 0.093$$

Transmittance

For a single cover the average transmittance after reflection losses is given by

$$\tau_r = \frac{1}{2}\left[\frac{1 - r_{\parallel}}{1 + r_{\parallel}} + \frac{1 - r_{\perp}}{1 + r_{\perp}}\right] \tag{37}$$

For a system of N covers, all of the same material, the average transmittance after reflection losses are accounted for is given by

$$\tau_r = \frac{1}{2}\left[\frac{1 - r_{\perp}}{1 + (2N - 1)r_{\perp}} + \frac{1 - r_{\parallel}}{1 + (2N - 1)r_{\parallel}}\right] \tag{38}$$

Example 4

Calculate the transmittance of two covers of nonabsorbing glass at normal incidence and at 60°.

At normal incidence the reflectance of one interface r(0) = 0.0434 (see example 3). From equation 38 with $r_{\perp} = r_{\parallel}$ we have

$$\tau_r(0) = \frac{1 - r(0)}{1 + 3r(0)}$$

$$= \frac{1 - 0.0434}{1 + 3(0.0434)} = 0.85$$

At a 60° incidence angle equations 32 and 33 give

$$r_{\perp} = 0.185$$

$$r_{\parallel} = 0.001$$

and from equation 38 we then have

$$\tau_r(60) = \frac{1}{2}\left[\frac{1 - 0.001}{1 + 3(0.001)} + \frac{1 - 0.185}{1 + 3(0.185)}\right]$$

$$= \underline{0.76}$$

Figure 2 below shows the effect of multiple glass nonabsorbing covers on overall transmittance. Table 6 lists the average refractive indices of some common cover materials.

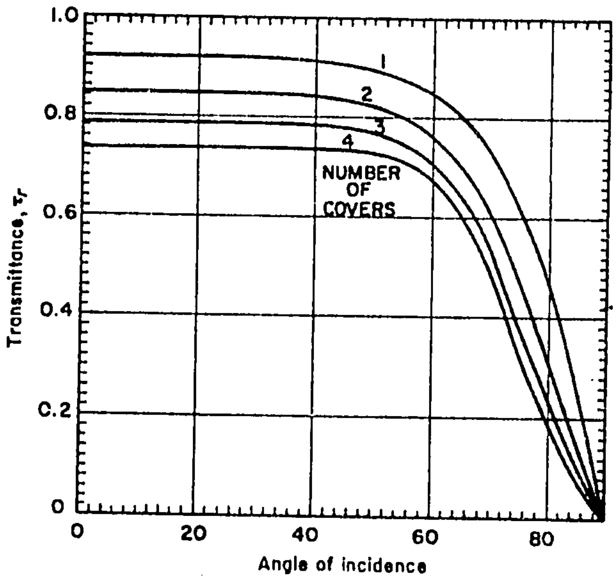


Figure 2. Transmittance of 1, 2, 3, and 4 nonabsorbing covers having an index of refraction of 1.526.

Table 6. Average Refractive Index in Solar Spectrum of Some Cover Materials

Cover Material	Average Refractive Index
Glass	1.526
Polymethyl methacrylate	1.49
Polyvinylfluoride	1.45
Polyfluorinated ethylene propylene	1.34
Polytetrafluoroethylene	1.37
Polycarbonate	1.60

Absorptance

The absorption of radiation in translucent media is described by Bouguer's law, which leads to an estimate of absorptance as

$$\alpha = 1 - \exp^{-KL} \tag{39}$$

where K is the extinction coefficient and L is the distance that the radiation travels, i.e.

$$L = \frac{\text{cover thickness}}{\cos \Theta_2}$$

The overall transmittance of a single cover is then given by

$$\tau = (1 - \alpha)\tau_r \tag{40}$$

and the reflectance r from the simple identity:

$$r = 1 - \alpha - \tau \tag{41}$$

The extinction coefficients for some common transparent materials are listed below.

TABLE 7. Extinction Coefficients for Transparent Materials

Polyvinyl fluoride (Tedlar)	1.4 cm^{-1}
Fluorinated ethylene propylene (Teflon)	0.59
Polyethylene terephthalate (Mylar)	2.05
Polyethylene	1.65
Ordinary window glass	~0.3
White glass (<0.01% Fe_2O_3)	~0.04
Heat-absorbing glass	1.3–2.7

Example 5

Calculate the transmittance, reflectance, and absorptance of a single glass cover 2.3 mm thick at an angle of 60°. The extinction coefficient of the glass is 32 m^{-1}.

Assuming for this glass n = 1.526 then from Example 3 we have

$$\Theta_2 = 34.58°$$
$$r_\perp(60) = 0.185$$
$$r_\parallel(60) = 0.001$$

then from equation 39

$$\begin{aligned}\alpha &= 1 - \exp(-32 \times 0.0023/\cos 34.58)\\ &= 0.085\end{aligned}$$

From equation 37 we have

$$\tau_r = 1/2\left[\frac{1 - 0.001}{1 + 0.001} + \frac{1 - 0.185}{1 + 0.185}\right]$$

$$= 0.843$$

It follows then that

$$\tau = (1 - 0.085) \times 0.843 = 0.771$$

and $$r = 1 - 0.085 - 0.771 = 0.144$$

Although equations 39, 40 and 41 were derived for a single cover they also apply to identical multiple covers, except that τ_r should now be evaluated using equation 38 and the value of L used in euqation 39 should be equal to the total cover system thickness.

Wavelength Variation of Transmission

Most transparent media transmit selectively. Transmittance is a function of the wavelength of the incident radiation. Glass, the material most commonly used as a cover material in solar collectors, may absorb little of the solar energy spectrum if its Fe_2O_3 (iron oxide) content is low. If the Fe_2O_3 content is high, it will absorb in the infrared portion of the solar spectrum. The transmittance of several glasses of varying iron content is shown in Figure 3.

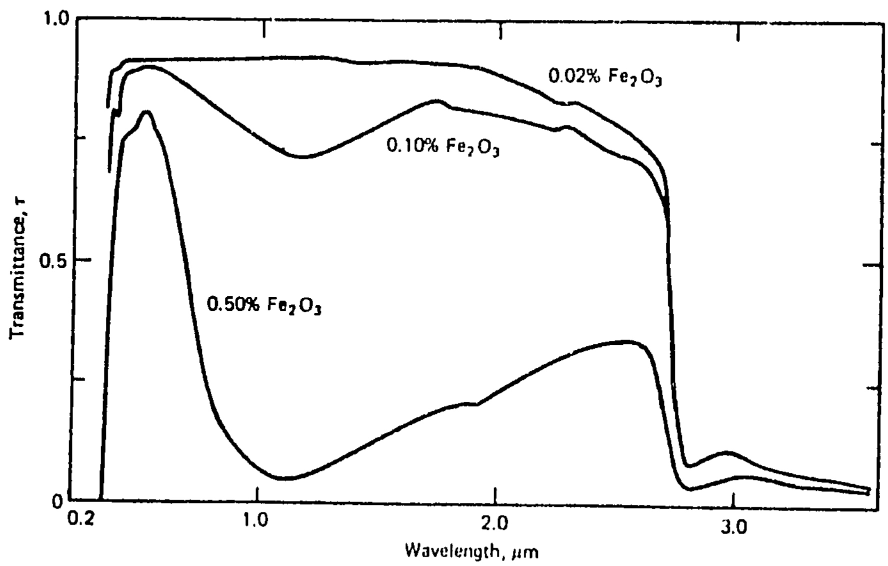
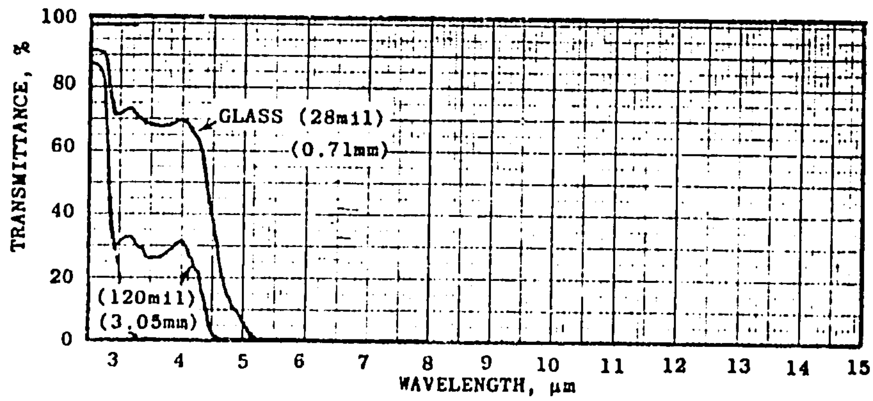


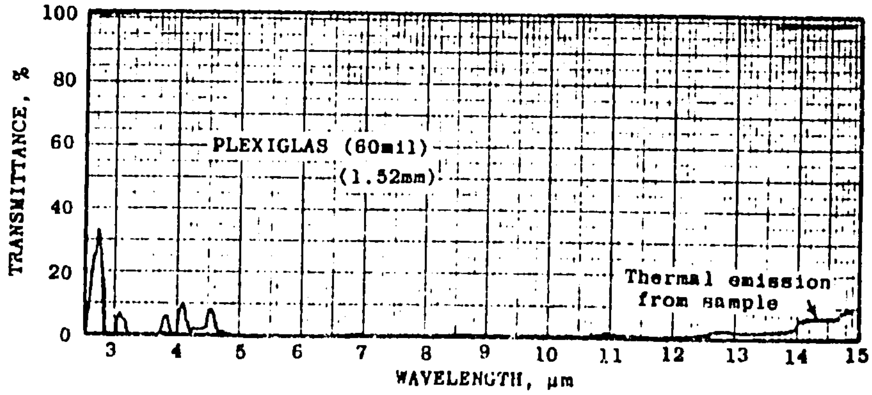
Figure 3 Spectral transmittance of 6 mm thick glass with various iron oxide contents.

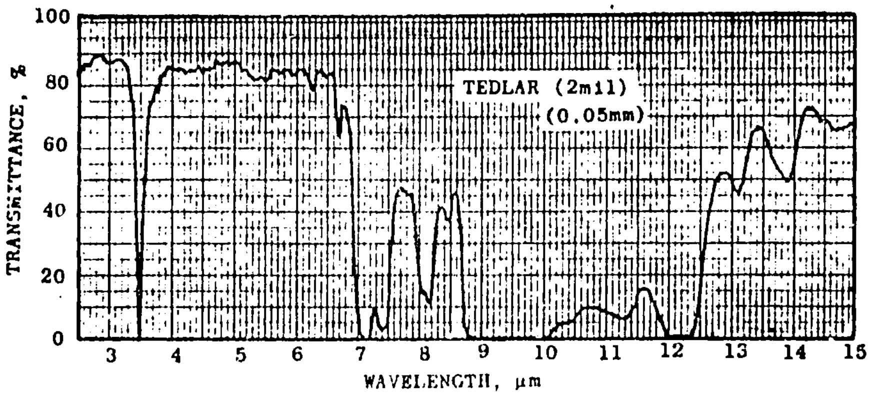
It is apparent that "water glass" (low iron) glass has the best transmission; glasses with high Fe_2O_3 content have a greenish appearance and are relatively poor transmitters. Note that the transmission is not a strong function of wavelength in the solar spectrum except for the high iron content glass. Glass becomes substantially opaque at wavelength longer than 3 µm and can be considered as opaque to longwave radiation (i.e. thermal infrared). This useful characteristic is the principal reason that glass is such an attractive material for covering flat-plate solar collectors.

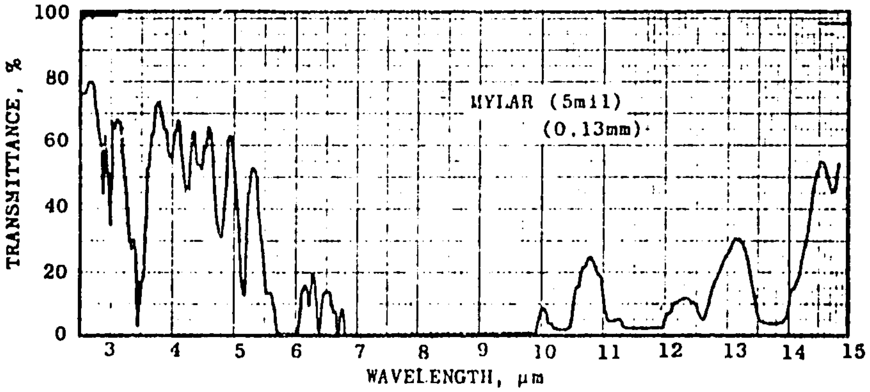
Plastics are generally more transparent than glass. Like glass, they absorb in the untraviolet but they have variable transmittance in the infrared depending on the thickness and the molecular bonds present in the particular plastic. Simple plastics like polyethylene have few absorption bands at certain wavelengths.

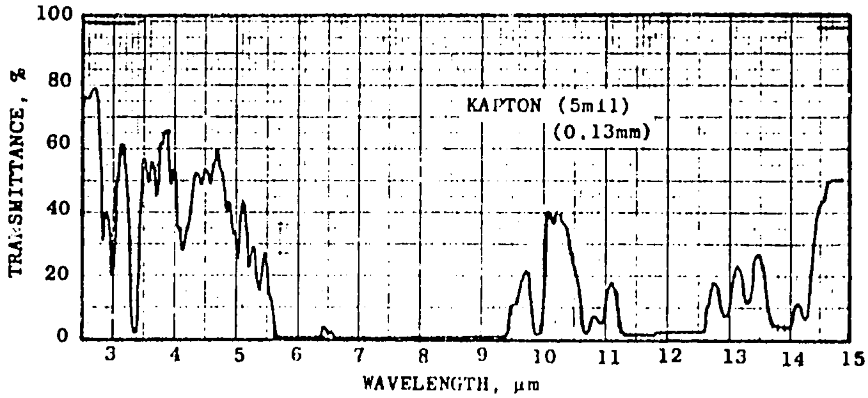
The infrared absorption of plastics is important in collector behavior. Glass being opaque to the thermal infrared, traps heat radiation. Some plastics, being relatively transparent, allow thermal radiation to escape. If plastic windows are used, the plastic must either be thick enough to absorb the thermal radiation or be instrinsically opaque to it. The transmittance - wavelength curves of a number of plastics of importance for solar energy collectors are shown below and overleaf.

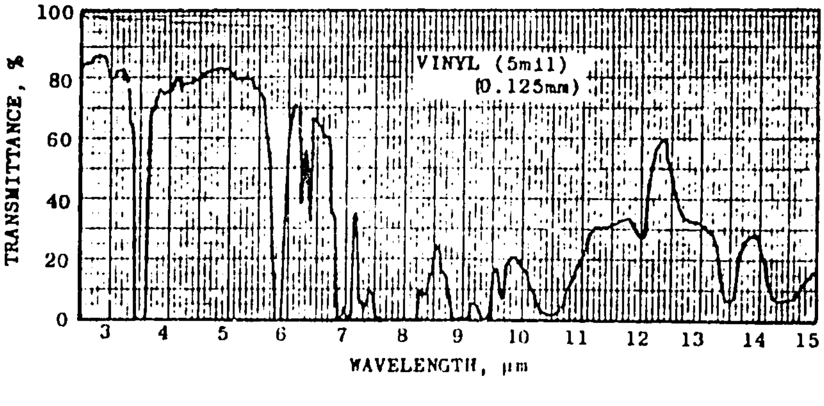


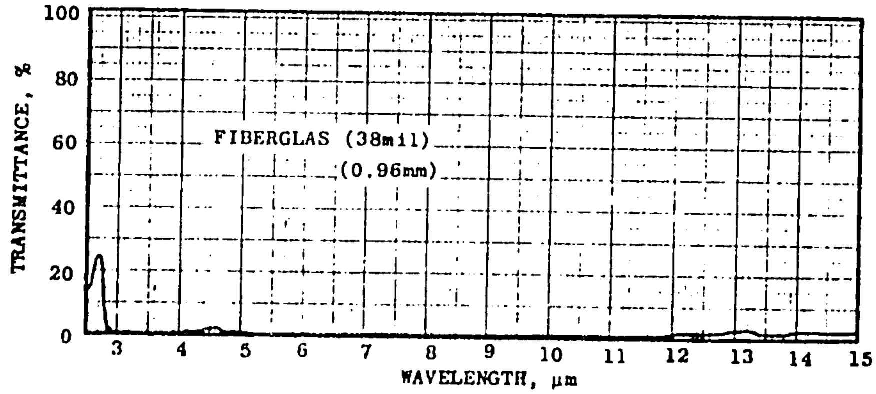



TEDLAR (2mil)
(0.05mm)
TRANSMITTANCE, %
100
80
60
40
20
0
3
4
5
6
7
8
9
10
11
12
13
14
15
WAVELENGTH, µm

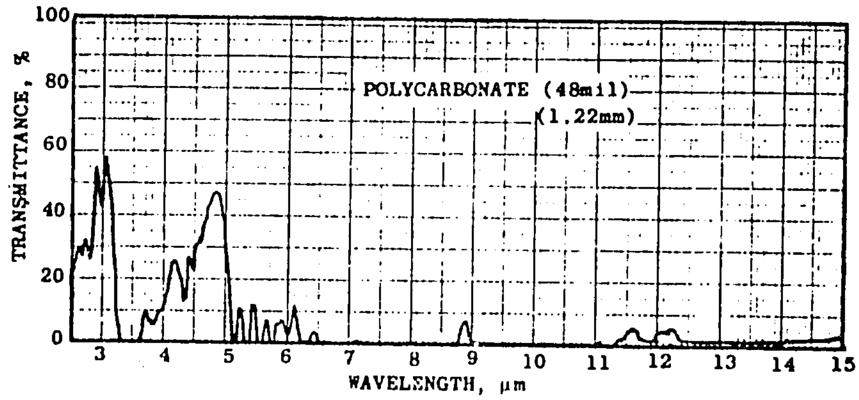

MYLAR (5mil)
(0.13mm)
TRANSMITTANCE, %
100
80
60
40
20
0
3
4
5
6
7
8
9
10
11
12
13
14
15
WAVELENGTH, µm


KAPTON (5mil)
(0.13mm)
TRANSMITTANCE, %
100
80
60
40
20
0
3
4
5
6
7
8
9
10
11
12
13
14
15
WAVELENGTH, µm

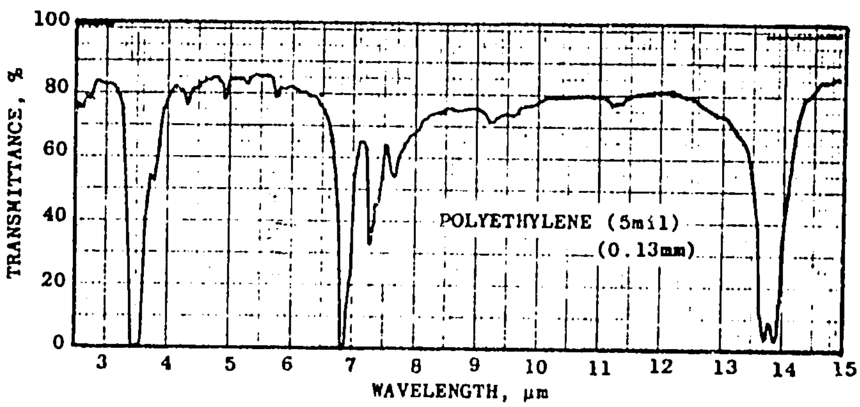

VINYL (5mil)
(0.125mm)
TRANSMITTANCE, %
100
80
60
40
20
0
3
4
5
6
7
8
9
10
11
12
13
14
15
WAVELENGTH, µm



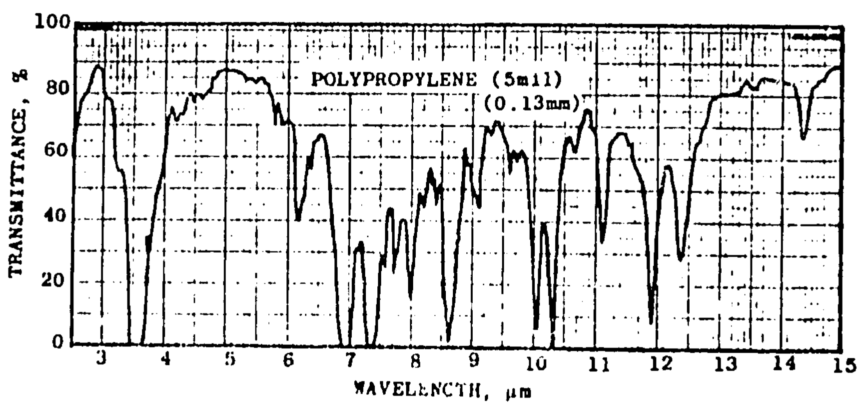
100
80
60
40
20
0
TRANSMITTANCE, %
FIBERGLAS (38mil)
(0.96mm)
3
4
5
6
7
8
9
10
11
12
13
14
15
WAVELENGTH, μm



100
80
60
40
20
0
TRANSMITTANCE, %
POLYCARBONATE (48mil)
(1.22mm)
3
4
5
6
7
8
9
10
11
12
13
14
15
WAVELENGTH, μm



100
80
60
40
20
0
TRANSMITTANCE, %
POLYETHYLENE (5mil)
(0.13mm)
3
4
5
6
7
8
9
10
11
12
13
14
15
WAVELENGTH, μm



100
80
60
40
20
0
TRANSMITTANCE, %
POLYPROPYLENE (5mil)
(0.13mm)
3
4
5
6
7
8
9
10
11
12
13
14
15
WAVELENGTH, μm

In the transmittance curves, the thickness shown are typical for solar collector systems. Plastic films are very thin and are used in tension for window coverings. Their thinness tends to make them transparent, whereas the thicker plastics used for rigid window coverings are thick enough to be almost totally opaque in the thermal infrared. Plexiglas and Fiberglas are more opaque than glass, but polycarbonate shows some transmission out to 6 microns.

Selective Surfaces

The problem of minimizing heat losses from a solar collector brings us to an examination of the optical properties of the absorber surface and the transparent windows. It is clear that we want as much radiant energy from the sun as possible to reach the absorber, while at the same time we wish to reduce to a minimum the thermal infrared energy radiating from the hot parts of the collector. Optical properties which vary widely from one spectral region to another produce what is termed selectivity. Figure 4 below illustrates the essential characteristics of selective surfaces.

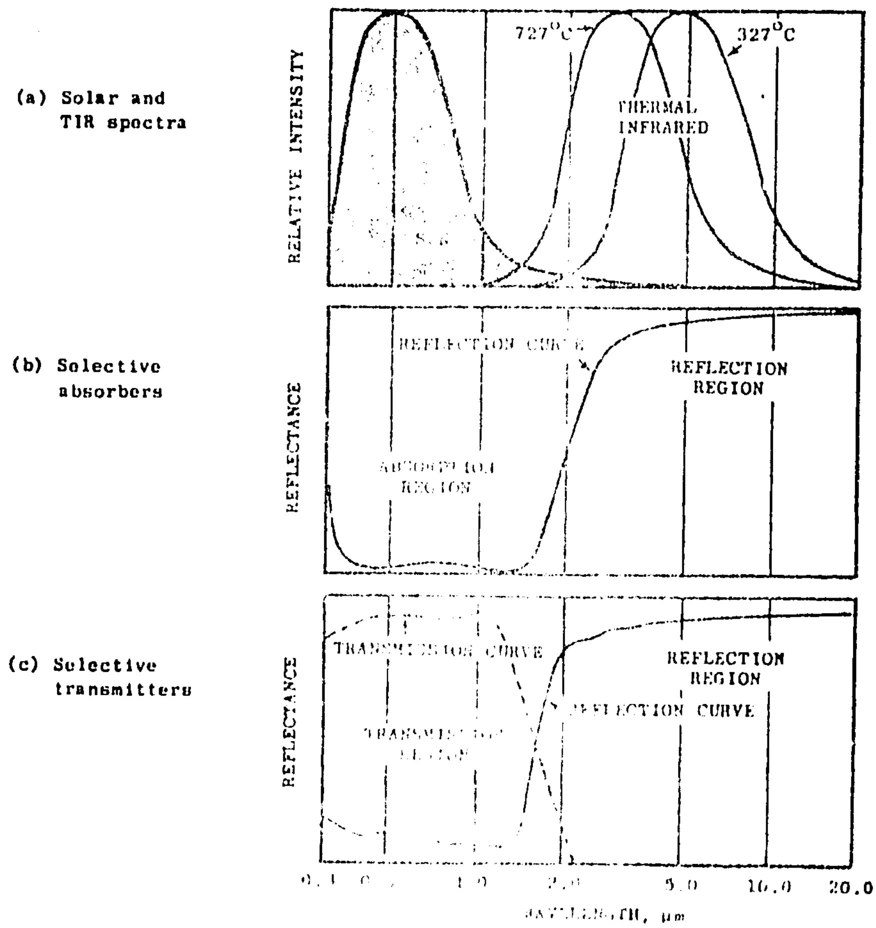


Fig. 4. Three diagrams illustrating the basic physics of selective surfaces. The top diagram shows radiant energy curves for the sun and for a hot surface radiating mainly in the thermal infrared; the middle diagram shows a typical curve for a selective absorber, the bottom diagram shows a typical curve for a selective transmitting surface

There are basically two types of selective surface of use in solar collectors:

1) Selective absorbing surfaces, where the surface is black to sunlight, making the transition from absorptive to reflective behavior in the region between 1.5 and 3 microns. By Kirchoff's law a reflective surface is a poor emitter, the value of the emittance being $\varepsilon = 1 - r$, where r is reflectivity of the surface. A highly selective surface is therefore one that has the highest possible reflectance in the thermal infrared. The measure of selectivity is the ratio of the absorptance for sunlight divided by the emittance for thermal infrared at the temperature of the projected use of the selective surface. This ratio, α/ε, can therefore vary with temperature, depending on the exact variations of both absorptance and emittance with wavelength.

2) Selective transmitting surfaces, where the surface is transparent to sunlight, making the transition from transmissive to reflective behavior in the region between 1.5 and 3 microns. The function of such surfaces is to let sunlight into a collector but to inhibit the loss of thermal infrared from the absorber.

TABLE 8. Properties of Some Selected Plated Coating Systems[a]

Coating[b]	Substrate	$\bar{\alpha}_s$	$\bar{\epsilon}_l$	Durability: Breakdown temperature (°F)	Durability: Humidity-degradation MIL STD 810B	Estimated manufactured cost per ft² (U.S.)
Black nickel on nickel	Steel	0.95	0.07	>550	Variable	0.30
Black chrome on nickel	Steel	0.95	0.09	>800	No effect	0.35–0.15
Black chrome	Steel	0.91	0.07	>800	Completely rusted	0.10
	Copper	0.95	0.14	600	Little effect	0.10
	Galvanized steel	0.95	0.16	>800	Complete removal	0.10
Black copper	Copper	0.88	0.15	600	Complete removal	0.10
Iron oxide	Steel	0.85	0.08	800	Little effect	0.05
Manganese oxide	Aluminum	0.70	0.08			0.10
Organic overcoat on iron oxide	Steel	0.90	0.16		Little effect	0.15
Organic overcoat on black chrome	Steel	0.94	0.20		Little effect	0.15

[a]From U.S. Dept. of Commerce, "Optical Coatings for Flat Plate Solar Collectors," NTIS No. PB-252-383, Honeywell, Inc., 1975.
[b]Black nickel coating plated over a nickel-steel substrate has the best selective properties ($\bar{\alpha}_s = 0.95$, $\bar{\epsilon}_l = 0.07$) but degraded significantly during humidity tests. Black chrome plated on a nickel-steel substrate also had very good selective properties ($\bar{\alpha}_s = 0.95$, $\bar{\epsilon}_l = 0.09$) and also showed high resistance to humidity.

M
15

Problems

1. The wall of a building consists of a 10 cm thick layer of common brick (k = 0.7 W/mK) followed by a 3.75 cm layer of plaster (k = 0.5 W/mK). Estimate what thickness of loosely packed rock-wool insulation (k = 0.6 W/mK) is necessary to reduce the rate of heat transfer through the wall by 75 percent.

2. An air collector has a rectangular flow sectional area of 0.05 by 1 metre. If the velocity of the air passing through the collector is 2 m/s, calculate the heat transfer coefficient between the collector and the air assuming one side of the collector is at 330K, the air is at 320 K, and the other side of the collector is insulated. Heat losses on the side may be neglected.

3. Insolation falls on a metal plate at a rate of 700 W/m^2. The solar absorptance is 0.9 and the plate is well insulated on the back. If the convective heat transfer on the upper side of the plate is 10 W/m^{2}K and the ambient temperature is 300K, estimate the temperature of the plate at thermal equilibrium.

4. Repeat problem 3 but assume that the metal plate is treated with a selective coating so that its solar absorptance is 0.9 and its emittance in the thermal infrared is 0.2.

5. Two large parallel plates having surface properties approximating those of a black body, are maintained at temperatures of 300 K and 400 K respectively. Calculate the rate of heat transfer by radiation between the plates.

6. A 1 metre diameter sheet metal duct is carrying air from a solar collector to a rock bed storage system. The duct is covered with 2.5 cm of fibreglass (k = 0.038 W/m K). If the overall heat transfer coefficient at the interior surface is 10 W/m^{2}K and at the outside surface is 5 W/m^{2}K and at the outside surface is 5 W/m^{2}K is 5 W/m^{2}K determine the rate of heat loss if the air temperature inside the duct is 310K and the ambient temperature is 290K.

7. An electric resistance heater with an outside surface area of 0.1 m^2 is immersed in a water storage tank at 413 K. The convective heat transfer coefficient between the heater and the fluid is estimated as 60 W/m^{2}K. If the electrical power supplied to the heater is 1 kW, calculate the average outside surface temperature of the heater under steady-state conditions.

8. What length of copper tubing (1 inch I.D., 1.25 inch O.D), immersed in a tank of hot water at 200°F, is required to heat water flowing in the tubing at 1 ft/s from 80°F to 130°F? Take the thermal conductivity of copper to be 400 W/mK.

9. Determine the top loss coefficient for the collector specified below:

Plate to cover spacing	1.5 inch
Absorber temperature	93.3°C
emittance	0.96
Glazing temperature	48.9°C
emittance	0.94
Glazing thickness	5/16 inch
Thermal conductivity of glass	1.02 W/mK
Ambient temperature	27° C
Collector size	8 x 4 ft.
Windspeed	4 m/s
Collector tilt	30°

10. For the collector specified above, determine the reduction in heat loss that would result from the deposition of a selective surface on the absorber plate. Assume the plate emittance falls to 0.1 and that all temperatures remain the same.

ANALYSIS OF FLAT PLATE COLLECTORS

Although solar energy is sometimes portrayed as a 'simple' technology, the thermal analysis of a solar collector is, in fact, quite complex. Flat plate collectors can be designed for applications requiring energy at moderate temperatures, up to about 100°C. They absorb both beam and diffuse solar radiation, do not need to track the sun, and generally require little maintenance.

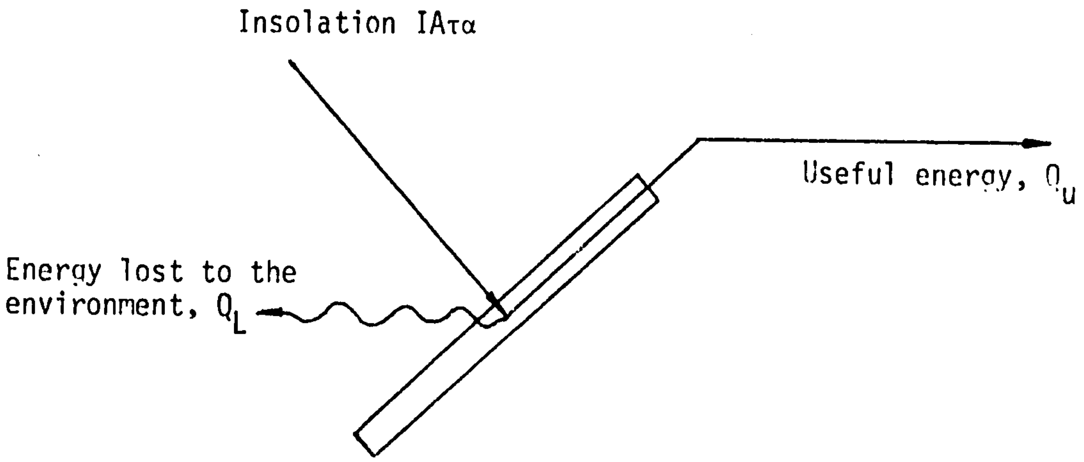


Figure 1. Energy balance over collector.

In the steady state, the heat balance over the collector may be written

$$Q_u = IA\tau\alpha - Q_L \tag{1}$$

where Q_u = useful energy transferred from the absorber plate to the working fluid.

Q_L = heat losses from the collector.

I = incident solar radiation.

A = area of the collector.

τ = overall transmittance of the collector covers.

α = absorptance of the absorber surface.

The instantaneous efficiency of the collector, η , would then be defined as

$$\eta = \frac{Qu}{IA} \qquad (2)$$

In practice, this is not a useful parameter since it varies continually with time. The average efficiency $\bar{\eta}$ is then:

$$\bar{\eta} = \frac{\int Q_u dt}{\int AIdt} \qquad (3)$$

In Equation (1) the heat losses from the collector Q_L can be written as a function of the overall heat loss coefficient U_L as follows:

$$Q_L = U_L A(T_p - T_a) \qquad (4)$$

where T_p is the mean plate temperature and T_a is the ambient temperature. Equation (1) becomes

$$Q_u = A[I\tau\alpha - U_L(T_p - T_a)] \qquad (5)$$

The problem here is that the temperature of the absorber plate T_p is difficult to calculate or measure since it is a function of the collector design, the incident solar radiation, and the entering fluid conditions.

To help in the thermal analysis of flat plate collectors, and to get around the fact that the absorber plate temperature T_p in Equation (5) is not known, it is conventional practice to introduce two new variables into the analysis. These variables are the Collector Efficiency Factor and the Heat Removal Factor.

Collector Efficiency Factor

The collector efficiency factor F' is given by the following expression

$$F' = \frac{1/U_L}{W\left[\frac{1}{U_L[D + (W - D)F]} + \frac{1}{C_B} + \frac{1}{\pi D_i h_f}\right]} \tag{6}$$

where
U_L = the collector overall heat loss coefficient.
W = the distance between tubes centres on the absorber plate.
D = the outside diameter of the tubes.
F = the fin efficiency.
C_B = the bond conductance.
D_i = the inside diameter of the tubes.
h_f = the inside convective film coefficient for the fluid.

Figure 2 below may be used to estimate the fin efficiency, F, or it may be calculated directly from

$$F = \frac{\tanh\ [m(W - D)/2]}{m\ (W - D)/2} \tag{7}$$

where $m = \sqrt{U_L/k\,\delta}$ (8)

where
δ = absorber plate thickness.
k = thermal conductivity of the plate.

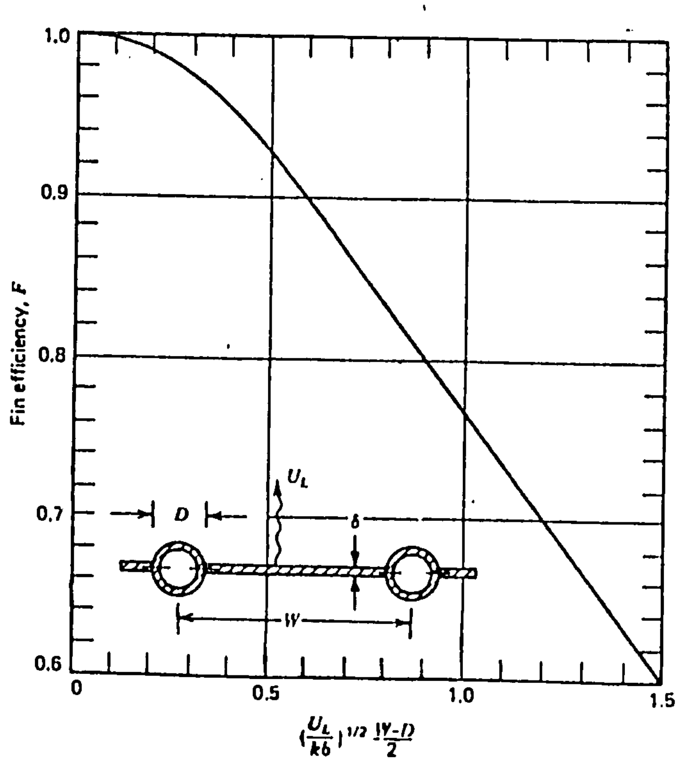


Figure 2 Fin efficiency for tube and sheet solar collectors.

The bond conductance, C_B, can be estimated from a knowledge of the bond thermal conductivity, k, the bond average thickness, , and the bond width, B. On a per unit length basis

$$C_B = \frac{kB}{Y} \tag{9}$$

The bond conductance can be very important in accurately describing collector performance. Simple wiring or clamping of the tubes to the absorber plate may result in a significant loss of performance.

The collector efficiency factor is essentially a constant for any collector design and fluid flow rate.

Collector Heat Removal Factor

The collector heat removal factor, F_R, may be determined from the following expression.

$$F_R = \frac{\dot{m}C_p}{U_L} \left(1 - e^{-1/C} \right) \tag{10}$$

where C is a dimensionless collector capacitance equal to

$$C = \frac{\dot{m}C_p}{U_L F'} \tag{11}$$

$\dot{m}$ = fluid mass flow rate, per unit area kg/m^2s
C_p = specific heat of the fluid, J/kg K
U_L = overall heat loss coefficient, W/m^2 K
F' = collector efficiency factor

It now becomes possible to write a simple expression for the useful energy collected by a flat plate collector.

$$Q_u = F_R A \left[I\tau\alpha - U_L(T_{in} - T_a) \right] \tag{12}$$

This is a much more useful expression than Equation 5, since both T_{in}, the inlet temperature of the fluid, and the ambient temperature, T_a, are usually known. The heat removal factor, F_R, may be computed once U_L has been determined, and $I\tau\alpha$, the radiation striking the absorber plate, will also be available.

Example 1

Calculate the collector efficiency factor, F', and the collector heat removal factor, F_R, for the following system:

Overall loss coefficient	8 W/m^2 K
Tube spacing	150 mm
Tube I.D.	10 mm
Plate thickness	0.5 mm
Plate conductivity	385 W/m K
Heat transfer coefficient inside tubes	300 W/m^2 K
Bond resistance	0
Flow rate	0.03 kg/s
Specific heat of water	4190 J/kg K
Dimension	1 X 2 m

Solution

Determine the fin efficiency, F, from Equations 7 and 8.

$$m = \left(\frac{8}{385 \times 5 \times 10^{-4}}\right)^{1/2} = 6.45$$

$$F = \frac{\tan h \ [6.45(0.15 - 0.01)/2]}{6.45(0.15 - 0.01)/2}$$

$$= 0.937$$

The collector efficiency factor, F', is then given by Equation 6.

$$F' = \frac{1/8}{0.15 \ \frac{1}{8(0.01 + 0.14 \times 0.937)} + \frac{1}{\pi \times 0.01 \times 300}}$$

$$= 0.84$$

To find the heat removal factor, F_R, we first determine the dimensionless capacitance, C, from Equation 11.

$$C = \frac{0.03 \times 4190}{2 \times 8 \times 0.84} = 9.35$$

so from Equation 10

$$F_R = \frac{0.015 \times 4190}{8} \ [1 - \exp \ (-1/9.35)]$$

$$= \underline{\underline{0.797}}$$

The Calculation of the Overall Loss Coefficient U_L

A basic calculation is to determine the overall collector heat transfer coefficient U_L. The thermal network for a two-cover flat plate collector is shown overleaf in Figure 3. It is clear that

$$R_1 = \frac{1}{h_{c2} + h_{r2}} \tag{13}$$

$$R_2 = \frac{1}{h_{c1} + h_{r1}} \tag{14}$$

$$R_3 = \frac{1}{h_{cp} + h_{rp}} \tag{15}$$

$$R_4 = \Delta x/k \tag{16}$$

$$R_5 = \frac{1}{h_{cb} + h_{rb}} \tag{17}$$

$$\text{and } U_L = \frac{1}{R_1 + R_2 + R_3} + \frac{1}{R_4 + R_5} \tag{18}$$

In some texts, a 'top loss' coefficient, U_t, and a 'back loss' coefficient U_b are specified, where

$$U_t = \frac{1}{R_1 + R_2 + R_3} \tag{19}$$

$$U_b = \frac{1}{R_4 + R_5} \tag{20}$$

In general, it is possible to assume R_5 is zero and that all resistance to heat flow is due to the insulation. However, it may also be necessary to consider edge losses. In a well designed system the edge loss should be small. It is recommended that edge insulation should be about the same thickness as that on the back of the collector. In this case edge losses can be included with the back loss to give

$$U_b = \frac{k}{x}(1 + A_e/A_c) \tag{21}$$

where A_e is area of the edge. This formulation assumes R_5 is zero and that the back and edges are insulated in a similar manner. Edge losses for well constructed large collector arrays are usually negligible, but for small collectors the edge losses may be significant.

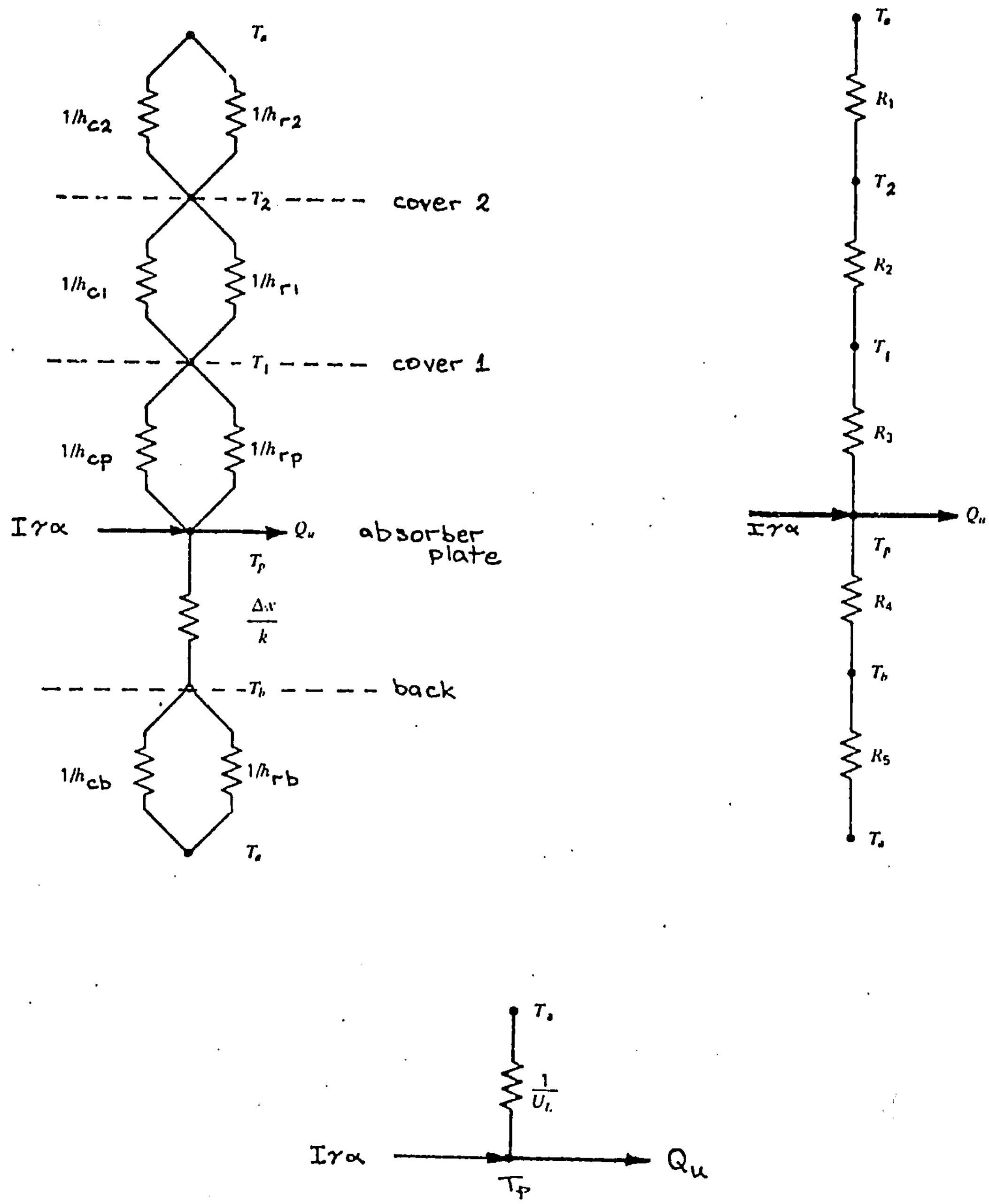


Figure 3. Thermal network for a two-cover flat-plate collector, (a) in terms of conduction, convection, and radiation resistances, (b) in terms of resistances between plates, (c) in terms of an overall heat transfer coefficient.

h_c = convective heat transfer coefficient
h_r = radiative heat transfer coefficient

The procedure for determining the loss coefficient U_L is an iterative process. First, a guess is made of the unknown absorber plate and cover temperatures. This permits the calculation of the heat transfer coefficients and therefore the resistances to heat transfer. The value of U_L then follows from Equation 18. The absorber plate temperature is then recalculated from

$$T_p = T_{in} + \frac{Q_u/A}{U_L F_R}(1 - F_R) \qquad (22)$$

A new temperature is then calculated for the first cover. This cover temperature is used to find the next cover temperature and so on. For any two adjacent covers, the new temperature of cover 2 can be expressed in terms of cover 1 as

$$T_2 = T_1 - \frac{U_t(T_p - T_a)}{h_{c1} + h_{r1}} \qquad (23)$$

When the absorber plate temperature and the cover temperatures have been recalculated, the overall loss coefficient, U_L, is calculated once again. This iterative procedure continues until calculated and estimated plate and cover temperatures remain the same.

However, the calculation of U_L depends on estimating the radiative and convective heat transfer coefficients (h_r and h_c respectively) for the heat transfer between the absorber plate and the first cover, between the covers if there is more than one, and between the outer cover and the environment. The equations used to determine these coefficients are given below.

A) PLATE TO COVER

Radiation: $$h_r = \frac{\sigma(T_p^2 + T_1^2)\cdot(T_p + T_1)}{1/\varepsilon_p + 1/\varepsilon_1 - 1} \quad \text{W/m}^2\text{K} \qquad (24)$$

where T_p = absorber plate temperature, K
T_1 = innermost cover temperature, K
ε_p = absorber plate emittance
ε_1 = cover emittance
σ = Boltzmann's constant
= 5.67×10^{-8} W/m^{2}K^4

Convection: $$h_c = \frac{k}{d} N \quad \text{W/m}^2\text{K} \qquad (25)$$

where k = thermal conductivity of air, Wm/K
d = distance between the surfaces
N = a dimensionless number (the Nusselt number) which may be determined here as

$$N = 1 + 1.44\,[1 - z]^+[1 - z(\sin 1.8\beta)^{1.6}] + [0.664z^{-1/3} - 1]^+ \qquad (26)$$

In this equation the meaning of the + exponent is that only the positive values of the term in the square brackets are to be used, (i.e. a value of zero is used if the term is negative).

Also $z = 1708/R \cos\beta$ (27)

where β is the angle between the collector and the horizontal; R is another dimensionless number, the Rayleigh number and is given by

$$R = g\Delta T\, d^3\rho^2\, C_p/\mu k T \qquad (28)$$

and here
- g = acceleration due to gravity, 9.81 m/s^2
- ΔT = temperature difference between the surfaces, K
- d = distance between the surfaces, m
- ρ = density of air, kg/m^3
- C_p = specific heat of air at constant pressure, J/kg K
- μ = viscosity of air, kg/m.s
- k = thermal conductivity of air, Wm/K
- T = the average temperature of the air between the surfaces, K

B) COVER TO COVER

Radiation: Same as Equation (24) except that the equation is now applied to the two cover surfaces.

Convection: Same as for the plate-to-cover situation.

C) OUTER COVER TO SKY

Radiation: $$h_r = \varepsilon\sigma(T_2^2 + T_s^2)(T_2 + T_s) \qquad (29)$$

where
- ε = emittance of outer cover
- T_2 = cover temperature, K
- T_s = sky temperature, K
- σ = Boltzmann's constant = 5.67×10^{-8} W/m^2K^4

Convection: $$h_c = 4.5 + 2.9\,u \quad W/m^2K \qquad (30)$$

The calculation of heat transfer coefficients for flat heated surfaces exposed to wind is not yet well established. For smooth surfaces Equation (30) is a reasonable approximation. The average wind speed, u, must be in metres per second.

Example 2

Calculate the overall loss coefficient for a collector (single cover) with the following specifications:

Plate to cover spacing	25 mm
Plate emittance	0.95
Ambient air and sky temperature	10°C (283 K)
Wind heat transfer coefficient	10 W/m^2 K
Mean plate temperature	100°C (373 K)
Collector tilt	45°
Glass emittance	0.88
Back insulation thickness	50 mm
Insulation conductivity	0.045 W/m.K
Collector array dimensions	10 X 3 X 0.075 m

Solution

Estimate the cover temperature as 35°C (308 K). In this example the absorber plate temperature has been specified.

A) PLATE TO COVER

Radiation: From Equation (24),

$$h_{rp} = \sigma \frac{(T_p{}^2 + T_1{}^2)(T_p + T_1)}{1/\varepsilon_p + 1/\varepsilon_1 - 1}$$

$$= 5.67 \times 10^{-8} \times \frac{(373^2 + 308^2)(373 + 308)}{1/0.95 + 1/0.88 - 1}$$

$$= \underline{7.60 \text{ W/m}^2 \text{ K}}$$

Convection: $h_{cp} = kN/d$ where Equations 26, 27 and 28 are to be used.

$$R = \frac{g\ \Delta T\ d^3 \rho^2 C_p}{\mu k T}$$

from Table 1 at $T = \frac{100 + 35}{2} = 67.5°C$

$= 340.5$ K

$\rho = 1.032 \text{ kg/m}^3$
$C_p = 1.0084 \times 10^3$ J/kg K
$\mu = 2.0575 \times 10^{-5}$ kg/ms
$k = 0.02931$ W/m K
$\Delta T = 100 - 35 = 65$ K
$d = 0.025$ m

$$\text{so} \quad R = \frac{9.81 \times 65 \times 0.025^3 \times 1.032^2 \times 1008.4}{2.0575 \times 10^{-5} \times 0.02931 \times 340.5}$$

$$= 52110$$

From Equation 27, $\quad Z = 1708/52110 \times \cos 45° = 0.0464$

and $\quad (\sin 1.8\beta)^{1.6} = 0.98$

$$\text{so} \quad N = 1 + 1.44\,[1 - 0.0464][1 - 0.0464(0.98)] + [0.664(0.0464)^{-1/3} - 1]$$

$$= 3.159$$

$$\text{hence} \quad h_{cp} = 3.159 \times \frac{0.02931}{0.025} = \underline{3.70 \text{ W/m}^2 \text{ K}}$$

B) <u>COVER TO SKY</u>

<u>Radiation</u>: $\quad h_{r1} = \varepsilon\sigma(T_1^2 + T_s^2)(T_1 + T_s)$

$$= 0.88 \times 5.67 \times 10^{-8}(308^2 + 283^2)(308 + 283)$$

$$= \underline{5.16 \text{ W/m}^2 \text{ K}}$$

<u>Convection</u>: $\quad \underline{h_{c1} = 10 \text{ W/m}^2 \text{ K}}$ $\quad$ (given)

so resistance, plate to cover $= \dfrac{1}{7.60 + 3.70}$

$$= 0.0885 \text{ m}^2 \text{ K/W}$$

and resistance, cover to sky $= \dfrac{1}{5.16 + 10}$

$$= 0.0660 \text{ m}^2 \text{ K/W}$$

$$\text{so} \quad U_t = \frac{1}{0.0885 + 0.0660} = 6.47 \text{ W/m}^2 \text{ K}$$

This is the first estimate of the top loss coefficient. We now check the first estimate of the cover temperature. From Equation 23

$$T_1 = T_p - \frac{U_t(T_p - T_a)}{h_{cp} + h_{rp}}$$

$$= 100 - \frac{6.47(100 - 10)}{7.6 + 3.70}$$

$$= 48.5°\text{C}$$

The procedure now is to recompute all the film coefficients using this new estimate of the cover temperature. We do not repeat the calculations here, but the results are:

$$h_{rp} = 8.03 \text{ W/m}^2 \text{ K}$$
$$h_{cp} = 3.52 \text{ W/m}^2 \text{ K}$$
$$h_{r1} = 5.53 \text{ W/m}^2 \text{ K}$$
$$h_{c1} = 10 \quad \text{W/m}^2 \text{ K} \qquad \text{as before}$$

so $$U_t = \left(\frac{1}{8.03 + 3.52} + \frac{1}{5.53 + 10}\right)^{-1}$$
$$= \underline{6.62 \text{ W/m}^2\text{K}}$$

The third estimate of the cover temperature, T_1, is therefore

$$T_1 = 100 - \frac{6.62(100 - 10)}{8.03 + 3.52}$$
$$= \underline{48.4°\text{C}}$$

so the calculation is acceptable.

Once the top loss coefficient has been determined, the back loss coefficient can be quickly found.

from Equation 21 $$U_b = \frac{k}{x}\left(1 + \frac{A_e}{A_c}\right)$$

so $$U_b = \frac{0.045}{0.05}\left[1 + \frac{2(10 + 3) \times 0.075}{10 \times 3}\right]$$
$$= 0.96 \text{ W/m}^2 \text{ K}$$

so $$U_L = U_t + U_b = 6.62 + 0.96$$
$$= \underline{7.58 \text{ W/m}^2 \text{ K}}$$

Minimizing Thermal Losses

Assuming the collector to be adequately insulated (including the edges), there remain two principal modifications to further reduce heat losses from the collector. The first is to add additional covers or glazings, the second is to incorporate a selective absorber plate. Figure 4 below shows the effect on thermal losses of double glazing and selective surfaces. The cover temperatures and the heat flux by convection and radiation are shown for one and two glass covers and for selective and non-selective absorber plates. Note that radiation between the inside surfaces is the dominant mode of heat transfer in the absence of a selective surface. When a selective surface having an emittance of 0.10 is used, convection is the dominant heat transfer mode between the selective surface and the cover, but radiation is still the largest term between the two covers in the double-glazed system.

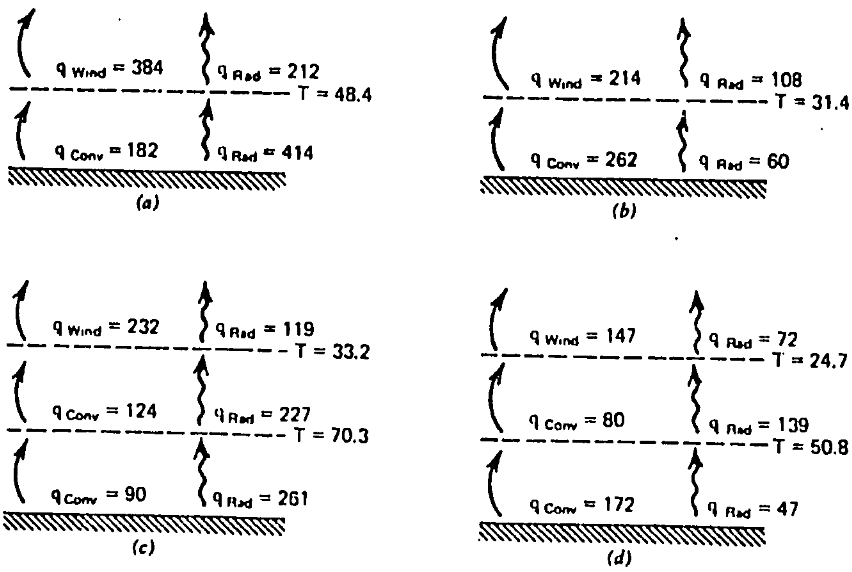


Figure 4 Cover temperature and upward heat loss for flat-plate collectors operating at 100 C with ambient and sky temperatures of 10 C, plate spacing of 25 mm, tilt of 45°, and wind heat transfer coefficient of 10 W/m² °C. (All heat flux terms in W/m².) (*a*) one cover, plate emittance = 0.95, U_t = 6.6 W/m²C; (*b*) one cover, plate emittance = 0.10, U_t = 3.6 W/m² C; (*c*) two covers, plate emittance = 0.95, U_t = 3.9 W/m²C; (*d*) two covers, plate emittance = 0.10; U_t = 2.4 W/m²C.

Energy Gain from Flat Plate Collectors

It is now possible to evaluate all the terms necessary to compute the amount of useful energy delivered by a flat plate collector. This quantity, Q_u watts, is found from Equation 12, after F_R and U_L have been determined in the manner illustrated by Examples 1 and 2. However, equation 12 is time-dependent since I, the incident solar radiation, obviously varies through the day. In order to determine, therefore, the useful energy delivered by the collector and its mean efficiency it is necessary to compute Q_u for short time increments over the period of a day. The procedure is illustrated by the following example.

Example 3

Calculate the daily useful gain and efficiency of a bank of 10 solar collectors installed in parallel. The hourly radiation on the plane of the collector, I, and the hourly ambient temperature, T_a, are given in the table below. Assume that the combined $\tau\alpha$ coefficient is 0.85, the overall loss coefficient, U_L, is 6.6 W/m^2K, and the heat removal factor is 0.8. Each collector is 2 m^2 in area. If the fluid inlet temperature is 40°C and the flow rate through each collector is 0.03 kg/s, what is the fluid temperature rise and how does it vary during the day?

Time	T_a °C	I W/m^2
7 - 8	20	5.6
8 - 9	24	119.4
9 - 10	25	275.0
10 - 11	28	788.9
11 - 12	31	833.3
12 - 1	33	913.8
1 - 2	31	866.7
2 - 3	30	644.4
3 - 4	29	336.1
4 - 5	26	13.9
		4797.1 W hr/m^2

Solution

We wish to calculate the useful energy delivered from Equation 12 and the mean efficiency from Equation 3. For each time increment we have:

Time	$I\tau\alpha$ W/m^2	$U_L(T_{in} - T_a)$ W/m^2	Q_u/A W/m^2
7 - 8	4.5	132.0	0
8 - 9	95.5	105.6	0
9 - 10	220.0	99.0	96.8
10 - 11	631.1	79.2	441.5
11 - 12	666.6	59.4	485.8
12 - 1	731.0	46.2	547.8
1 - 2	693.4	59.4	507.2
2 - 3	515.5	66.0	359.6
3 - 4	268.9	72.6	157.0
4 - 5	11.1	92.4	0
			2595.7 W hr/m^2

$$\text{the mean efficiency } \bar{\eta} = \frac{\Sigma Q_u/A}{\Sigma I}$$

$$= \frac{2595.7}{4797.1} = 0.54$$

The energy delivery by the 20 m^2 array over the day is

$$2595.7 \times 20 \times 3600 = \underline{\underline{186.9 \text{ MJ}}}$$

The temperature rise for the water will vary according to the period. The smallest positive temperature rise is between 9 and 10; the highest between 12 and 1.

taking $C_p = 4195$ J/kg K

and $\dot{m} = 0.03$ kg/s for each 2 m^2 collector.

then $\Delta T = \dfrac{Q_u}{\dot{m}C_p}$

so from 9 - 10: $\Delta T = \dfrac{96.8 \times 2}{0.03 \times 4195} = 1.5°C$

and from 12 - 1: $\Delta T = \dfrac{547.8 \times 2}{0.03 \times 4195} = 8.7°C$

Performance Characteristics

The performance characteristics of flat-plate collectors are often presented graphically.

Since the instantaneous efficiency is given by

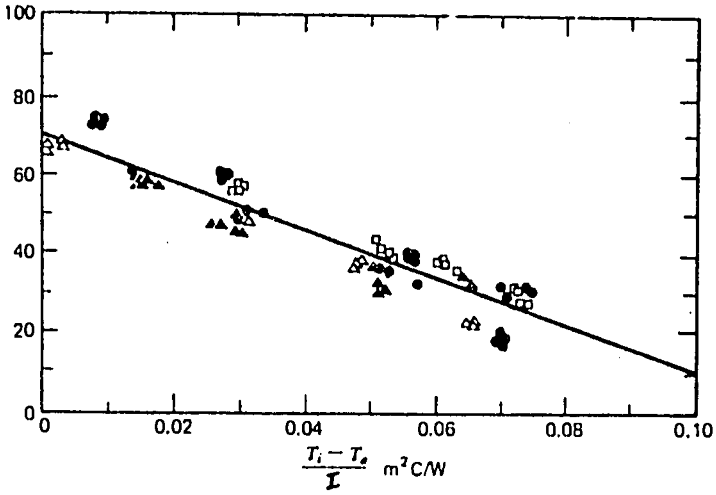
$$\eta = \frac{Q_u}{IA}$$

and since $Q_u = F_R A[I\tau\alpha - U_L(T_{in} - T_a)]$

the efficiency may be expressed as a function of the fluid inlet temperature, T_{in}, as

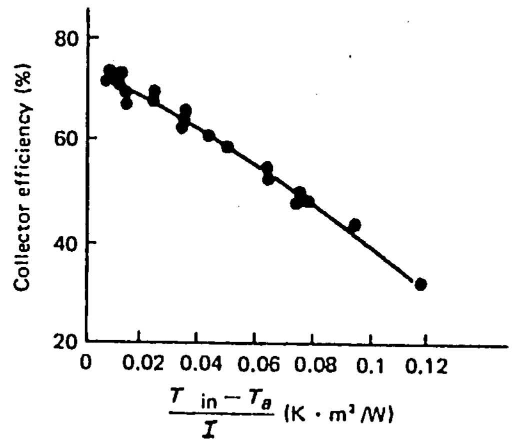
$$\eta = -F_R U_L\left(\frac{T_{in} - T_a}{I}\right) + F_R\tau\alpha \qquad (31)$$

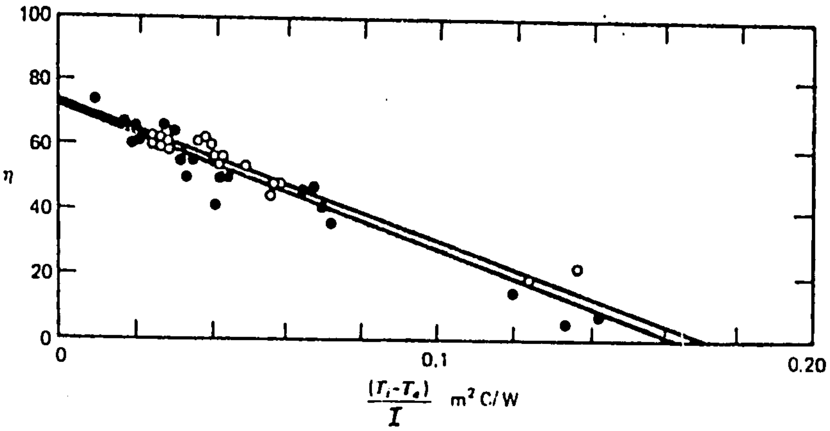
If η is plotted against $(T_{in} - T_a)/I$ then a straight line results with a negative slope of $F_R U_L$. The intercept on the abscissa is equal to $F_R\tau\alpha$. A number of typical plots are shown overleaf. It is clear that, in practice, there is considerable data scatter and that, moreover, the plots are slightly non-linear. However, a straight line drawn through the data points and intercepting the abscissa presents a very convenient indication of collector performance. It will be necessary to calculate or estimate the transmittance of the covers, τ, and the absorptance of the collector plate surface α. The intercept divided by the product, $\tau\alpha$, gives the value of F_R, the collector heat removal factor. The slope of the line divided by F_R then gives U_L, the overall heat loss coefficient.



Experimental collector efficiency data measured for a type of liquid heating collector with one cover and a selective absorber. Sixteen points are shown for each of five test sites. The curve represents the theoretical characteristic derived from points calculated for the test conditions.

Efficiency curve for a double-glazed flat-plate liquid-heating solar collector with a selective coating on the absorber. $\dot{m}$ = 0.0136 kg/sec · m²; T_a = 29°C; T_{in} = 38-101°C; I = 590-977 W/cm²; wind = 3.1 m/sec. (The tests were run indoors using a solar simulator.)



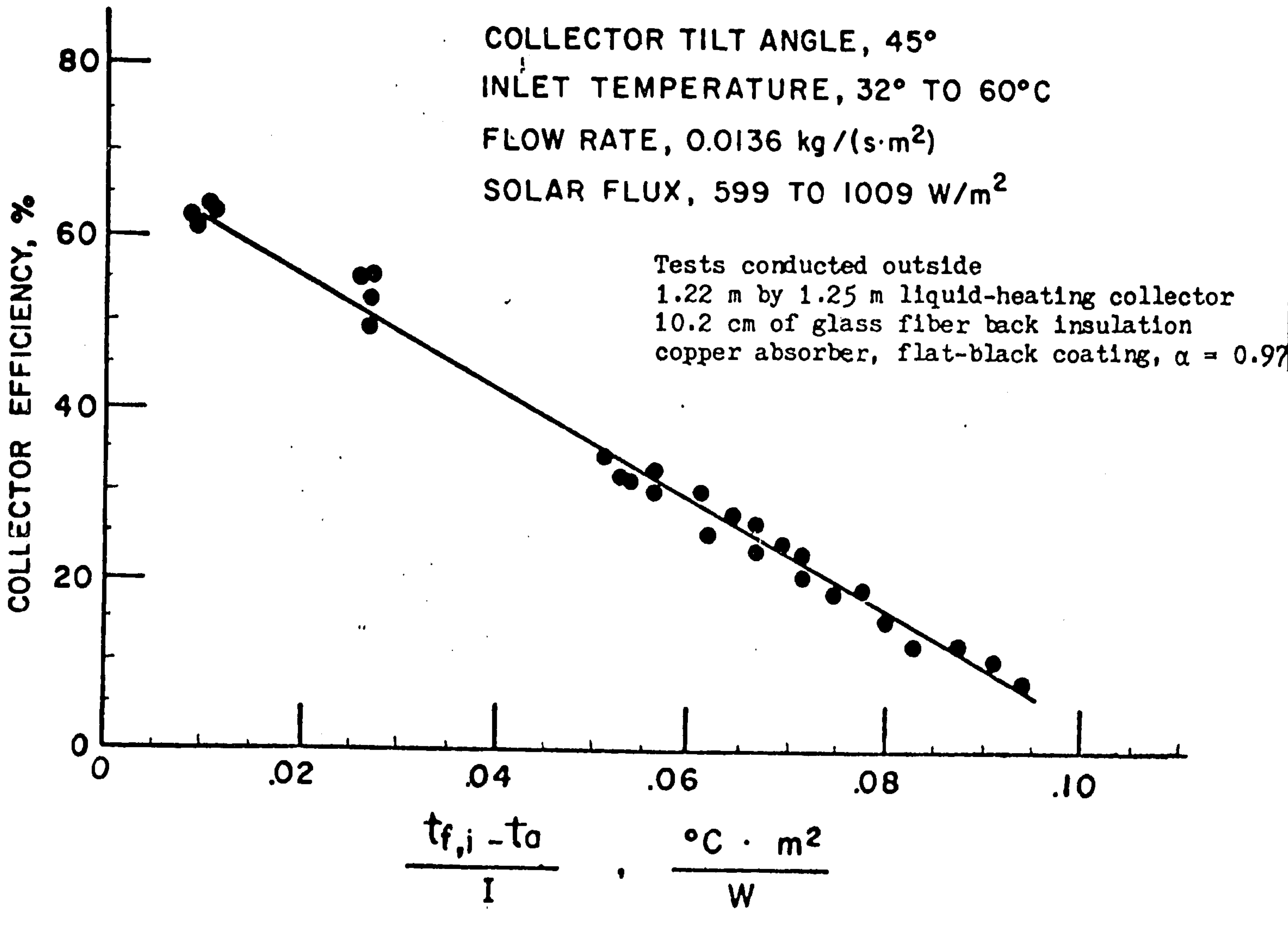


Experimental thermal efficiency curves for two air heaters operated outdoors. Absorbing surface was flat black paint.

Problems

1. The performance curve shown is for a single glazed flat plate collector. The transmittance is estimated at 0.88. Determine:

i) The collector heat removal factor, F_R
ii) The overall loss coefficient, U_L
iii) The energy collected when the level of insolation on the plane of the collector is 700 W/m_2, and the temperature of the water leaving the collector at the above insolation level if the inlet temperature of the water is 50°C and the ambient temperature is 29°C.
iv) The collector performance if the fluid flow rate is raised to 0.05 kg/m^2s.



Thermal Efficiency Curve for a Double-Glazed Flat-Plate Liquid-Heating Solar Collector

SOLAR THERMAL SYSTEMS

Solar thermal systems designed to utilize solar radiation for heating or cooling are generally composed of a number of basic subsystems. These are:

1. Solar collector system
2. Thermal energy storage system
3. Fluid circulation system
4. Heat exchanger system
5. Control system

The solar collection part of the system has been examined in a previous section. In this set of notes we want to look at the remaining components of a solar thermal system: thermal energy storage, fluid circulation, heat exhangers and control.

Thermal Energy Storage

Solar energy is a time-dependent and variable resource. This characteristic makes thermal energy storage virtually a mandatory component of any solar thermal system. Thermal storage is used to dampen out diurnal and meteorological variations in the level of insolation and provide a more nearly constant heat source for the system load. The optimal size of the storage system depends on a number of considerations: the insolation and meteorological characteristics of the area where the system is located, the nature of the system load, and the economics of the total system.

Water is by far the most common thermal storage medium for solar powered thermal systems requiring a temperature of less than 100°C. Water has a number of very attractive properties as a thermal storage medium. It is cheap, non-toxic and non-flammable; it has a high specific heat, high density and excellent transport properties.

For some applications heat storage in solid media is a possibility, particularly when air is used to collect and transport thermal energy. Solid-phase storage has a number of advantages: they generally permit larger temperature variations and higher operating temperatures, they stratify thermally, and they are durable and reliable systems. Table 1 shows thermal properties for a number of liquids and solids.

It is clear that on both a mass and a volumetric basis, water is an excellent thermal storage medium.

An alternative method of thermal energy storage takes advantage of the fact that a considerable amount of heat is absorbed or evolved during a change of phase. A thermal energy storage system that uses solid-liquid phase changes in the storage medium to store heat is called a phase-change storage system.

There are two kinds of solid-liquid phase change storage systems. The first is simple melting and freezing; the second involves the chemical reaction between water and salt hydrates. When the hydrate is heated, the salt dissolves in its water of crystallization and absorbs heat. On cooling the anhydrate becomes hydrated and crystallizes with the evolution of heat. Table 2 gives the melting point and heat of fusion for a number of phase change materials.

Although phase-change storage has great potential, it suffers from a lack of reliability and durability, particularly with salt hydrate systems. After many cycles, the rehydration (crystallization) phase change requires progressively more subcooling before it takes place. The isothermal behavior, therefore, deteriorates. In addition, since the phase change is not a true melting, density differences between component compounds may occur which exacerbates the subcooling problem.

Table 1. Properties of Heat Storage Materials

Material	Specific heat kJ/kg K	Density kg/m^3	Heat capacity kJ/m^3 K
Water	4.2	1000	4200
Isobutanol	3.0	808	2420
Propanol	2.5	800	2000
Scrap iron	0.50	7850	2748
Magnetite	0.75	5126	2691
Scrap aluminum	0.96	2723	1830
Concrete	1.13	2240	1772
Stone	0.88	2720	1676
Brick	0.84	2240	1317

Note: The volumetric heat capacity for the solid materials assumes a 30% void fraction.

Table 2. Properties of Phase Change Materials

Material	Melting point °C	Heat of fusion kJ/kg
Calcium chloride hexahydrate	30	168
Sodium carbonate decahydrate	33	267
Disodium phosphate dodecahydrate	40	279
Calcium nitrate tetrahydrate	47	153
Sodium sulfate decahydrate (Glauber's salt)	32	241
Sodium thiosulphate pentahydrate (STP)	45	95
Naphthalene	80	149
Naphthol	95	163
Paraffin	74	230
P-116 wax	47	209

Water Storage Systems

Water is a good storage medium. A typical system is shown in Figure 1.

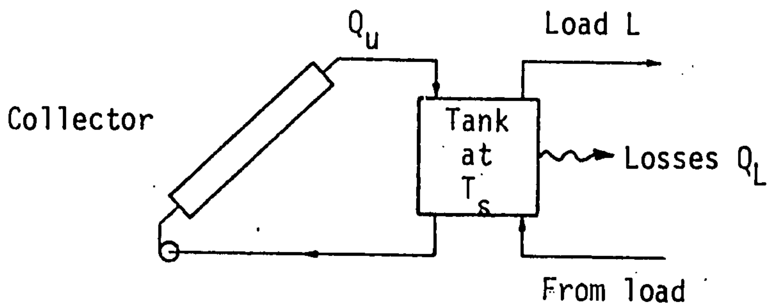


Figure 1. Typical collector and storage system

For a well mixed tank of water it is not too difficult to accurately model the system. An energy balance over the tank yields the equation

$$M_s c_p \frac{dT_s}{dt} = Q_u - Q_L - L \tag{1}$$

where M_s = mass of water in storage, kg,
c_p = specific heat of water, J/kgK,
T_s = temperature of stored water,
t = time,
Q_u = heat added to the storage, Watts,
Q_L = heat lost from the system, Watts,
L = heat extracted by the load, Watts.

Since the thermal losses, Q_L, can be expressed as a linear function of temperature difference, $T_s - T_a{}^*$, where $T_a{}^*$ is the ambient temperature close to the storage system, Equation 1 can be rewritten as

$$M_s c_p \frac{dT_s}{dt} = Q_u - (UA)_s(T_s - T_a{}^*) - L \tag{2}$$

where $(UA)_s$ is the loss coefficient for the storage system. This term can be readily estimated by determining conduction, convection, and radiation losses from the storage tank. The example below shows how this equation can be used to estimate the storage temperature as a function of time by the use of a simple numerical integration procedure.

Example 1

A well-mixed water tank storage system containing 1,500 kg of water has a loss coefficient of 11.1 W/K. The tank commences a 24 hour day at a temperature of 45°C and is in a room at 20°C. Energy Q_u is added and energy L is extracted as indicated on an hourly basis below. Calculate the temperature of the tank over the 24-hour period using numerical integration.

Hour	Energy Added Q_u, MJ/hr	Energy Extracted L, MJ/hr
1	-	12
2	-	12
3	-	11
4	-	11
5	-	13
6	-	14
7	-	18
8	-	21
9	21	20
10	41	20
11	60	18
12	75	16
13	77	14
14	68	14
15	48	13
16	25	18
17	2	22
18	-	24
19	-	18
20	-	20
21	-	15
22	-	11
23	-	10
24	-	9

Solution

The differential equation of Equation 2 can be approximately solved by rewriting it as:

$$T_s^+ = T_s + \frac{\Delta t}{M_s c_p} \left[Q_u - (UA)_s (T_s - T_a^*) - L\right]$$

The starting temperature (T_s) of 20°C is used to estimate the tank temperature after 1 hr, T_s^+. This calculated temperature then becomes the tank temperature, T_s, and a new temperature T_a^+ is estimated by recursively applying the above equation. Substituting the given constants reduces the equation to:

$$T_s^+ = T_s + \frac{\Delta t}{1500 \times 4190} \left[Q_u - 11.1 \times 3600 (T_s - 20) - L\right]$$

$$\text{or } T_s^+ = T_s + \frac{\Delta t}{6.285 \times 10^6} [Q_u - 39{,}960\,(T_s - 20) - L]$$

The temperature profile may now be estimated as follows. After one hour:

$$T_s = 45 + \frac{1}{6.285 \times 10^6} [0 - 39{,}960\,(45-20) - 12 \times 10^6]$$

or $T_s^+ = 42.9°C$

After 2 hours:

$$T_s^+ = 42.9 + \frac{1}{6.285 \times 10^6} [0 - 39{,}960\,(42.9 - 20) - 12 \times 10^6]$$

or $T_s^+ = 40.9°C$

This recursive procedure is continued through the 24 hour period to produce the approximate time variation of the storage temperature. The calculated temperatures are shown below.

Hour	Q_u MJ/hr	L MJ/hr	T_s °C
0	-	-	45
1	-	12	42.9
2	-	12	40.9
3	-	11	39.0
4	-	11	37.1
5	-	13	35.0
6	-	14	32.6
7	-	18	29.7
8	-	21	26.3
9	21	20	26.4
10	41	20	29.7
11	60	18	36.3
12	75	16	45.6
13	77	14	55.5
14	68	14	63.8
15	48	13	69.1
16	25	18	69.9
17	2	22	66.4
18	-	24	62.3
19	-	18	59.2
20	-	20	55.7
21	-	15	53.1
22	-	11	51.2
23	-	10	49.4
24	-	9	47.8

Equation 2 may be extended by introducing the relationship between Q_u, the energy gain from the solar collector, the temperature of the storage system T_s, the insolation characteristics, and the collector overall loss coefficient, U_L. We can write

$$Q_u = F_R A \, [I\tau\alpha - U_L \, (T_s - T_a)] \qquad \text{Watts} \qquad (3)$$

where F_r = collector heat removal factor
A = collector area, m^2
T_a = ambient temperature, °C
τ = transmissivity of the collector covers
α = absorptance of the absorber.

If Q_u is to be calculated from Equation 3 from hourly insolation and temperature data, it should be recalled that it is the function of the system control devices to pump fluid through the collector loop only when there is an energy gain. This occurs when the insolation has increased to the point where $I\tau\alpha$ is greater than $U_L \, (T_s - T_a)$.

Example 2

Determine the hourly performance of the large solar heating installation indicated below

Area of collectors	= 100 m^2
Storage volume	= 7.5 m^3
Collector overall loss coefficient	= 5.2 W/m^2K
Storage losses	= negligible

The heat removal factor, F_R, is estimated as 0.8, $\tau\alpha$ as 0.85. The load over the period is constant at 25 kW. The insolation levels and ambient temperatures are shown below. Assume the initial storage temperature is 70°C.

Time	I (W/m^2)	T_a (°C)
8	157.6	20
9	516.9	24
10	740.7	25
11	870.0	28
12	914.1	31
13	870.0	33
14	740.7	32
15	516.9	31
16	157.6	29

Solution

The heat balance over the storage system, Equation 2, reduces to

$$7500 \times 4190 \frac{dT_s}{dt} = Q_u - 25{,}000 \qquad \text{Watts}$$

where $Q_u = 80 \, [0.85I - 5.2 \, (T_s - T_a)\,]$ Watts (4)

The differential equation can be solved numerically by rewriting it as

$$T_s^+ = T_s + \frac{\Delta t}{31.43 \times 10^6} [Q_u - 25{,}000]$$

or for $\Delta t = 1$ hr = 3,600s

$$T_s^+ = T_s + 1.145 \times 10^{-4} [Q_u - 25{,}000] \qquad (5)$$

The calculation proceeds as follows:

Time = 0800 hrs, $I = 157.6$ W/m^2, $T_a = 20°C$, $T_s = 70°C$

From Equation 4 $Q_u = 80 [0.85 \times 157.6 - 5.2 \times 50]$
$= - 10{,}083$ Watts

Therefore there is no energy gain from the collectors at this point. With $Q_u = 0$, (no flow through the collectors) Equation 5 becomes

$$T_s^+ = T_s + 1.145 \times 10^{-4} [-25{,}000]$$

or

$$T_s^+ = 70 - 2.9 = \underline{67.1°C}$$

Time = 0900 hrs, $I = 516.9$ W/m^2, $T_a = 24°C$, $T_s = 67.1°C$

From Equation 4 $Q_u = 80 [0.85 \times 516.9 - 5.2 \times 43.1]$
$= 17{,}220$ Watts

From Equation 5 $T_s^+ = 67.1 + 1.145 \times 10^{-4} [17{,}220 - 25{,}000]$

$T_s^+ = 66.2°C$

This calculation procedure can be continued in this manner to produce the table shown below.

Time	I W/m^2	T_a °C	Q_u KW	L KW	T_s °C
8	157.6	20	0	25	67.1
9	516.9	24	17.2	25	66.2
10	740.7	25	33.2	25	67.1
11	870.0	28	42.9	25	69.1
12	914.1	31	46.3	25	71.5
13	870.0	33	43.1	25	73.6
14	740.7	33	33.4	25	74.6
15	516.9	31	17.0	25	73.7
16	157.6	29	0	25	70.8
	5.48 kWh/m^2		233.1 KWh	225 KWh	

The data used in this example are realistic and show some interesting characteristics. The mean collector efficiency can be estimated for the $100m^2$ array as

$$\frac{233.1}{5.48 \times 100} = 43\%$$

which is on the low side. The low collector efficiency is due to the high fluid temperatures at which the collector operates.

The energy collected over the course of the day is:

233.1 kWh = 839.2 MJ

equivalent to 8.39 MJ/m^2 day
or 739 Btu/ft^2 day

These figures are close to the convenient solar 'rule of thumb' for a flat plate collector:

Energy collected = 700 - 1000 Btu/ft^2 day
= 8 - 11 MJ/m^2 day

The other thermal characteristic indicated here is temperature swing. In this example the storage temperature fluctuates between about 66°C and 75°C. These variations can be important depending on the load characteristics. For example, if the solar thermal system is driving an absorption chiller, temperatures of at least 60°C will be required in the generator of the chiller. The coefficient of performance of the cooling system will drop off sharply if such a temperature is not maintained. It is therefore important that the thermal storage system is designed with the load thermodynamic requirements in mind.

Design Guidelines

Computer simulations, experimental investigations, and a great deal of practical experience have yielded some general guidelines for sizing flat plate collectors and thermal storage systems.

The optimum storage size for an active solar thermal system will fall in the range of 0.2 to 0.4 MJ/°C m^2, equivalent to 10-20 Btu/°F ft^2. These ranges are for thermal capacity per unit area of collector, i.e.

$$0.2 < M_s c_p/A < 0.4 \text{ MJ/°C m}^2$$

For water as the storage medium (C_p = 4190 J/kg K) these figures produce the rule of thumb that thermal storage should be 45-90 litres/m^2 or 1.2 - 2.4 gal/ft^2.

Flow rates through forced-circulation systems are generally based on 0.02 - 0.04 gpm/ft^2 of collector, equivalent to 0.015 - 0.03 litre/m^2s.

As we have previously mentioned, the collector array can be roughly sized based on its ability to collect about 8-11 MJ/m^2 day or 700-1000 Btu/ft^2 day. These design guidelines permit the rapid sizing of solar thermal systems to drive a specified load. An approximate design based on these rules of thumb provides a convenient and often quite accurate starting point for more detailed analyses and simulations of system performance. For example, consider again Example 2. Suppose we were told the system load and asked to roughly size the solar collectors and thermal storage requirements for the system. The load extracts an amount of energy equal to

$$9 \times 25 \text{ kWh} \times 3.6 \text{ MJ/kWh} = 810 \text{ MJ/day}$$

knowing the relatively high storage temperature requirements we would estimate collector energy gain at the lower end of the 8-11 MJ/m^2 day scale. Estimating collector area as

$$810 \frac{\text{MJ}}{\text{day}} \cdot \frac{\text{m}^2 \text{ day}}{8 \text{ MJ}} \approx 100 \text{ m}^2 \text{ collector}$$

Storage requirements would be about 75 $litres/m^2$ which gives us an estimated storage volume of 7.5 cubic metres.

Stratified Storage

When heated fluid from the collector does not mix with the bulk of the fluid in the storage system, the storage system is said to be stratified. This phenomena is made possible because most fluids become less dense as they become hotter. The incoming fluid therefore has a tendency to remain at the top of the tank above the colder fluid below. If the fluid flowing to the collector is taken from the bottom of the tank in a perfectly stratified system the collector inlet temperature will remain constant until one tank volume of fluid has passed through the collectors, at which time the collector inlet temperature will show a step rise in temperature.

In practice, perfect thermal stratification is impossible since the inlet fluid velocity always causes some local fluid mixing and also because in mid to late afternoon, as insolation levels decrease, the tank inlet water temperature may actually be less than the highest fluid temperatures at the top of the tank. The fluid therefore descends into the body of the tank again producing a degree of mixing.

With a well-mixed storage tank, on the other hand, the temperature of the tank is uniform and will rise slowly throughout the day as the solar collectors absorb solar energy. As the storage temperature rises, the efficiency of the collectors decreases. However, compensating for this disadvantage with well-mixed storage systems is the rather higher heat transfer attained by using a more rapid flow rate than is consistent with an attempt to maintain thermal stratification in the storage tank.

In practice, there is little to choose between the two design concepts for liquid systems. It is useful and convenient to take advantage of the degree of stratification that always exists in any liquid thermal storage tank in the absence of a stirring device or excessively high inlet flow rates. Hot fluid is added or withdrawn from the top of the tank, cold fluid cold fluid added or withdrawn from the bottom.

Rock Bed Storage

Rock bed storage units have been successfuly used to store heat for many years. Hot air from solar collectors flows through a bed of pebbles and heat is transferred from the air to the rocks. The air leaves the storage unit at a temperature very close to the temperature of the pebbles adjacent to the outlet plenum. This is the charging mode. To remove heat from the storage system, the flow of air is reversed: cold air enters the cold end of the unit and is heated as it passes through the bed. Since there can be no mixing of the storage media in a rock storage unit the system is always thermally stratified when charged. This means that the collectors always operate with the coolest available incoming air, which helps to increase their efficiency. However, it should be noted that in rock bed storage systems, unlike liquid storage systems, heat cannot be added and removed from storage simultaneously. A typical rock bed thermal storage unit is shown in Figure 2. In general, the air flow is downward during charging, and upward during discharging. Although it is not essential that this flow pattern is adopted, such a design minimizes heat losses to the ground since the bottom of the storage is at the lowest temperature.

Rock bed storage units have several advantages. They are inexpensive, stable, and easy to construct; they will not freeze or boil; they are virtually maintenance free, and they will last a long time. Their disadvantages include their incompatibility with hot fluid systems, their relative large volume (approximately three times the volume of a water system of equal thermal capacity), and possible problems with dust and particle entrainment.

Thermal Stratification

One of the principal advantages of rock bed storage units is that, if properly designed, they exhibit a high degree of thermal stratification. The small size of the pebbles provides a large surface area for heat transfer so that the average temperature of a particular rock is close to local air temperature. When hot air is blown into the bed a well-defined thermal wave moves through the bed in the direction of the air flow. Figure 3 shows typical bed temperature profiles during the charging mode. The rock bed in this figure was initially at 21^{o}C; the incoming air was at 65.5^{o}C. Note that the outlet temperature of the air remains constant at the initial bed temperature of 21^{o}C until, at about 9 hours after charging begins, the leading edge of the thermal wave reaches the bottom of the bed.

In a real system charged with hot air from a solar collector, the temperature of the air entering the rock bed varies during the day, typically reaching a maximum temperature around noon or soon after depending on the location and configuration of the collector (and assuming a clear day). During the afternoon the temperature of the air entering the rock bed will be decreasing. The effect of this daily temperature variation is to drive a temperature peak through the bed, a peak that continually flattens as it moves down through the layers of rock. This phenomenon can clearly be seen in Figure 4 which shows measured time-lapsed cross-sectional temperature profiles through a rock bed storage system over a 22-hour period.

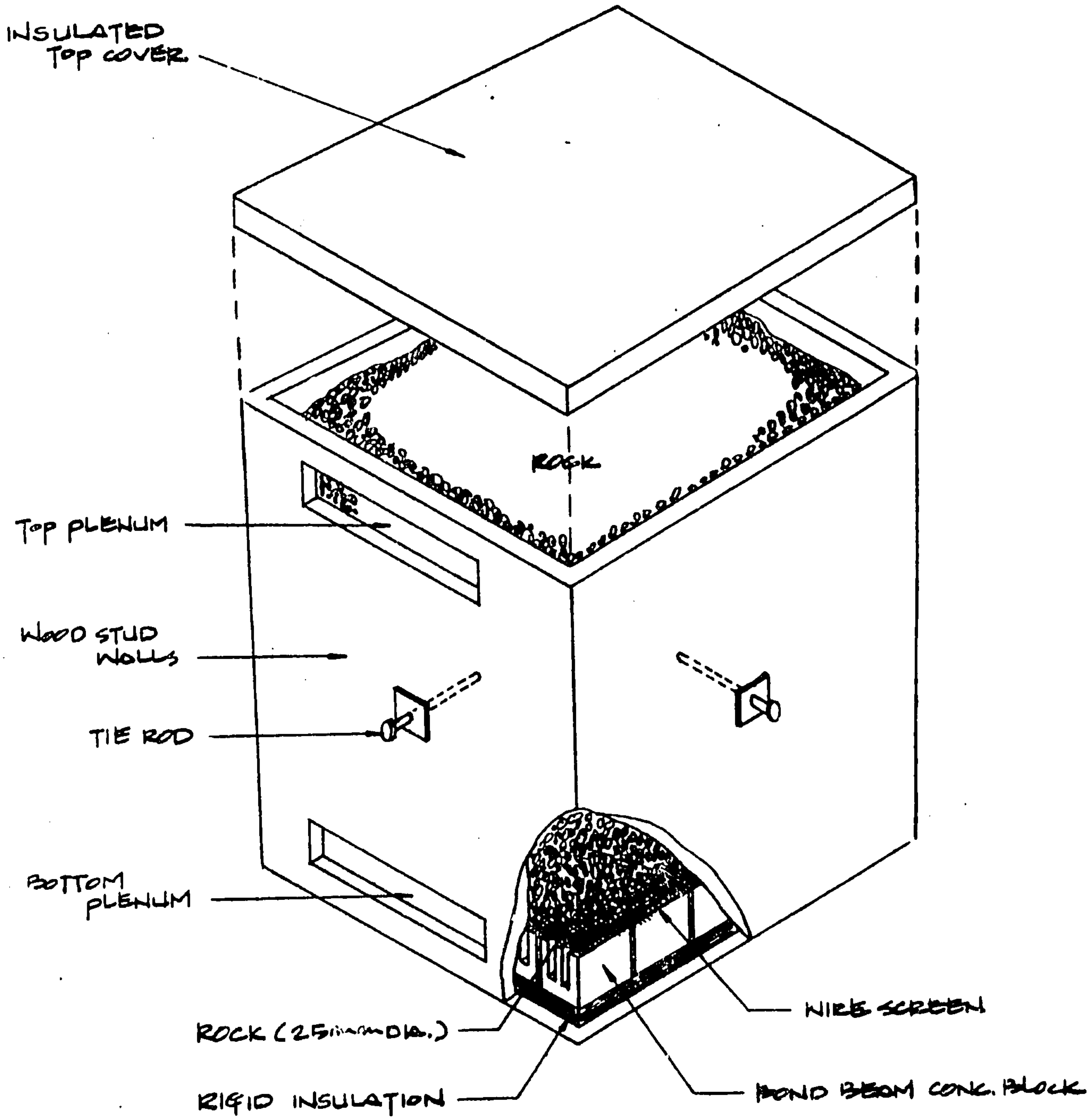


Figure 2. Typical Rock Bed Thermal Storage Unit

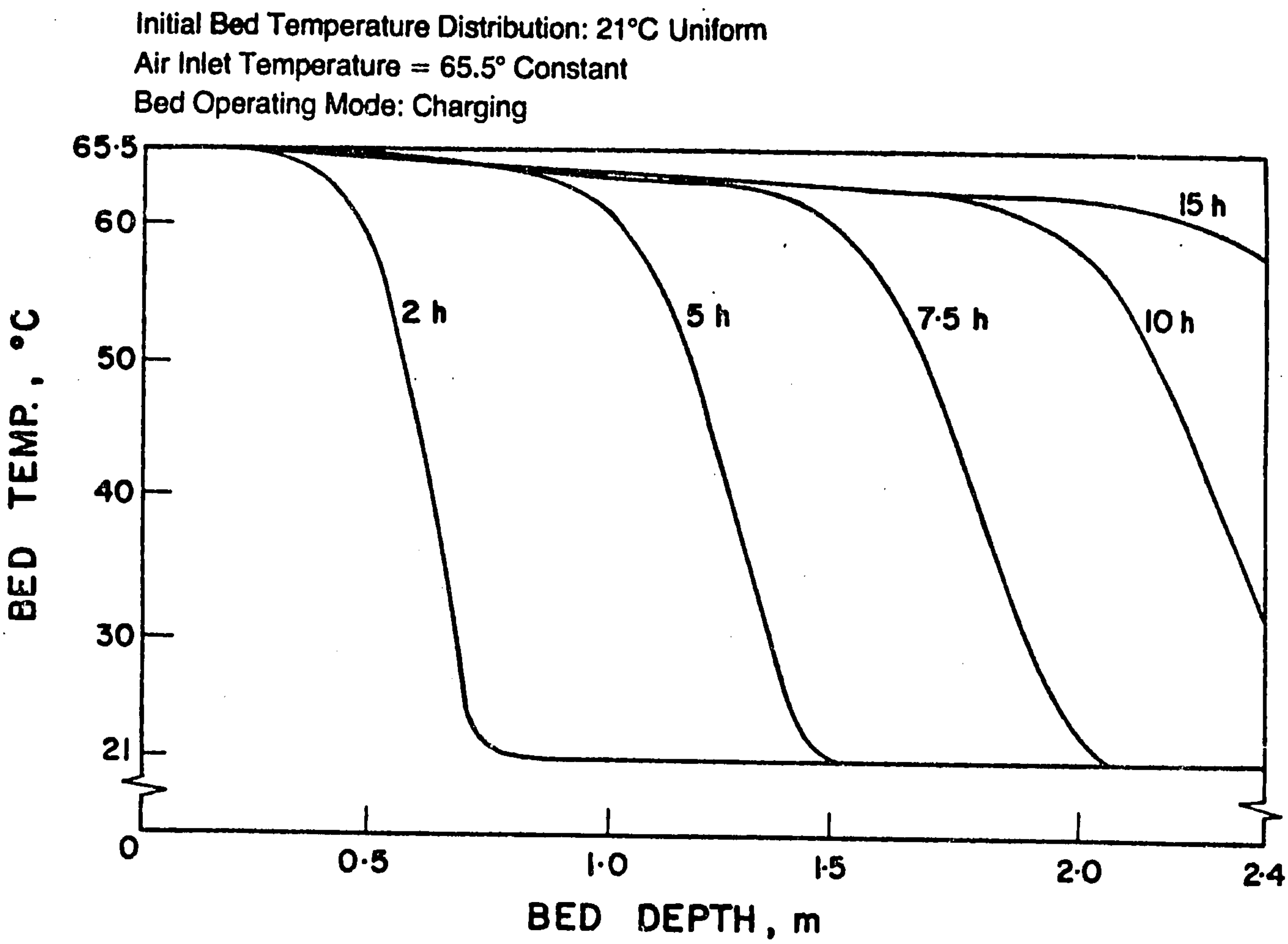


Figure 3. Rock Bed Temperature Profiles During Charging.

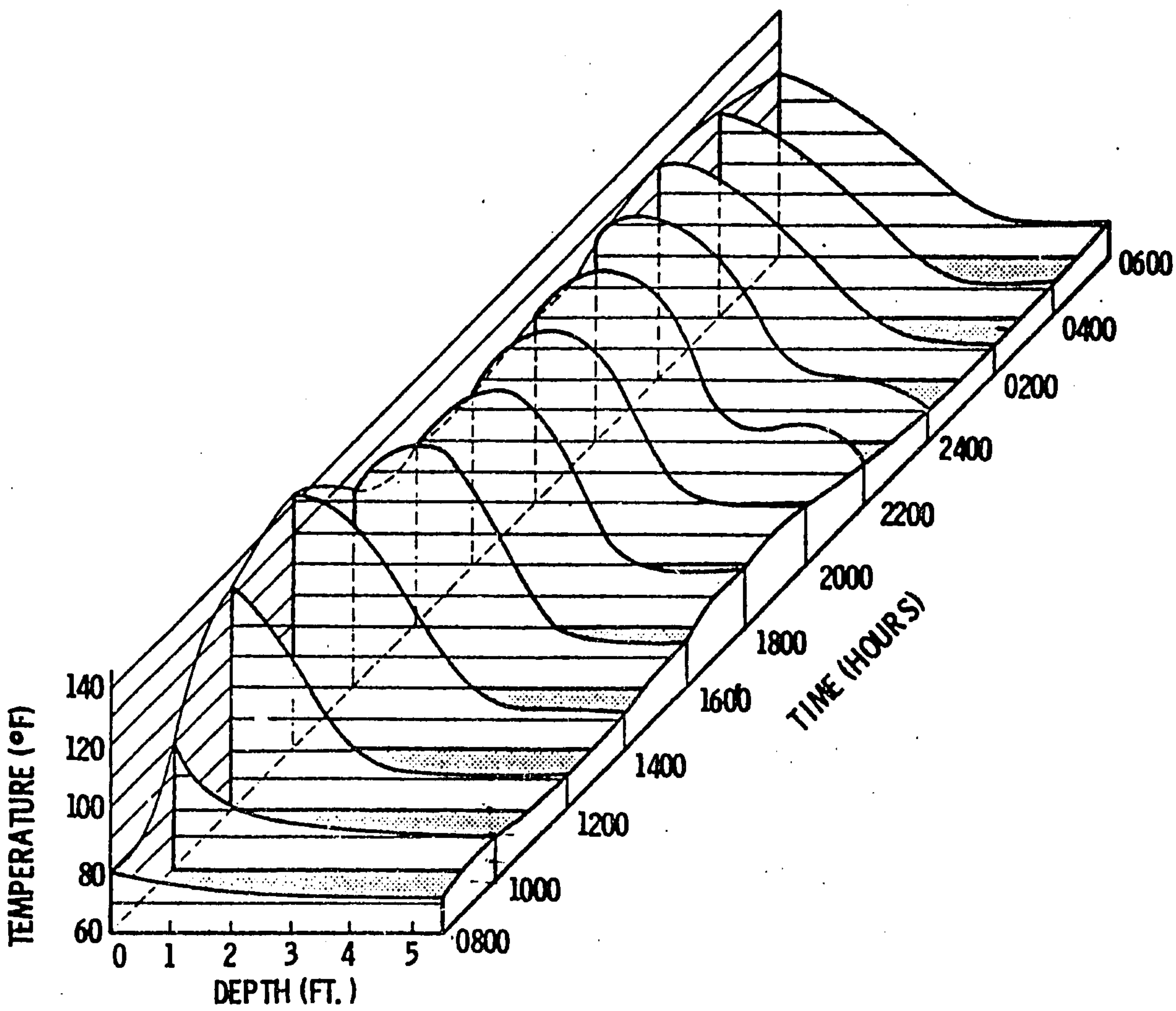


Figure 4. **Propagation of Pebble-Bed Temperature Profile (During Heat Storage Charge-Discharge Cycle),**

The time required for a thermal wave to traverse a rock bed is important to the performance of solar heating systems. The rock bed size and air flowrate should be selected so that the air returned to the collector remains at the minimum temperature for most of the heat collection period, thus maximizing the efficiency of the solar collector. The time taken for the thermal wave to traverse the rock bed should not be longer than the heat collection period since this would mean that a part of the bed would remain unused. The time taken, t, for the thermal wave to traverse the rock bed can be estimated from

$$t = \frac{Lc_s}{Vc_a} \quad \text{seconds} \tag{6}$$

where L = rock bed length in direction of flow, metres
c_s = rock storage heat capacity, MJ/m^3 K.
c_a = heat capacity of air, MJ/m^3 K
V = rock bed face velocity, m/s

The heat capacity of a rock bed storage system is generally about 1.4 MJ/m^3 K. This figure assumes a typical 40% void fraction. The kind of rock used does not markedly affect the heat capacitance of the storage system.

If left undisturbed, a thermally stratified rock bed will progressively destratify until the bed reaches a uniform temperature. This decay of the thermocline is undesirable since it both raises the temperature of the air flowing to the collectors during charging, thereby reducing their efficiency, and lowers the temperature of the air going to load during the discharge cycle. However, the rate of thermal destratification is typically quite slow. It takes several days for an undisturbed rock bed to reach a uniform temperature. Figure 5 shows rock bed temperature profiles as a function of time. Reasonably good thermal stratification was maintained for at least three days in this experiment. Thus, decay of thermal stratification in rock bed storage units has only a minor effect on system performance. Charging the bed from the bottom as opposed to the top also has little impact on performance.

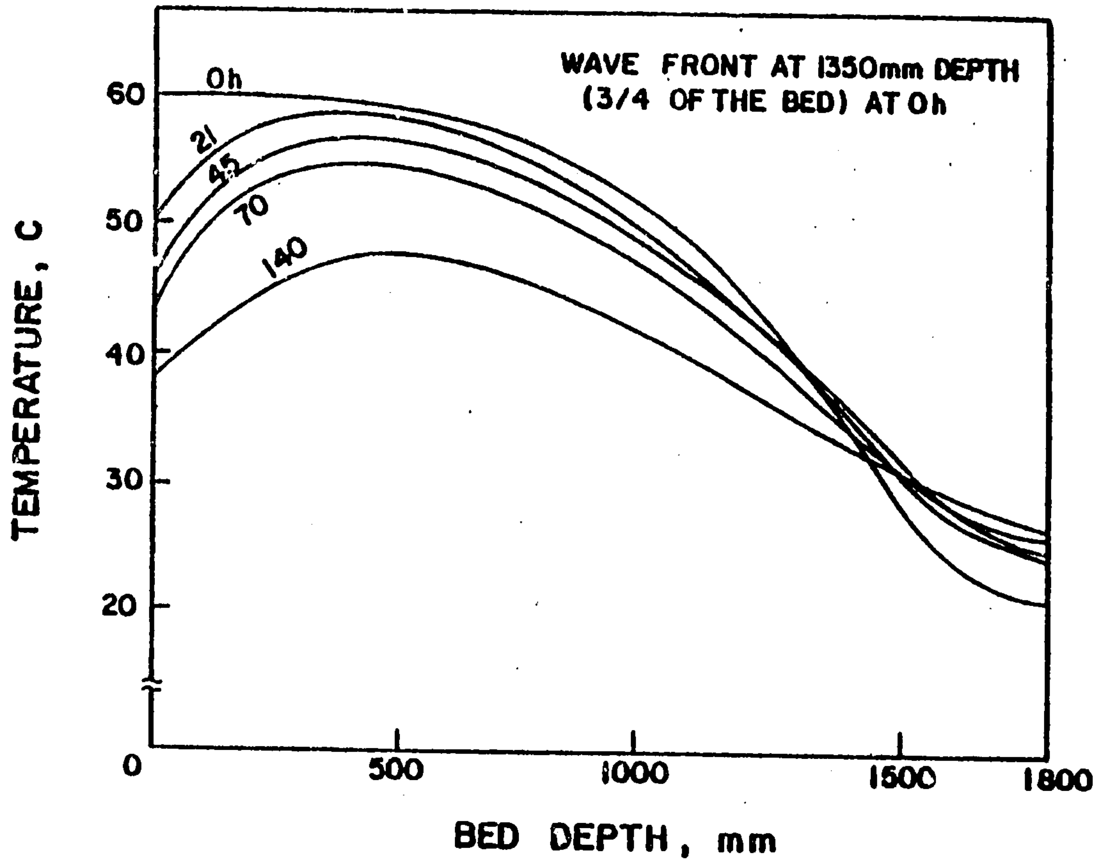


Figure 5. Stratified Rock Bed Temperature Profiles
- Variation with Time in Hours.

Sizing the Rock Bed

Studies suggest [5] that system performance improves as the rock bed storage volume increases up to about 0.2 cubic metres of storage volume for for each square metre of flat plate collector (m^3/m^2). Above this value the performance of the system does not change markedly with additional storage volume. The volume of storage recommended for air systems is 0.15 to 0.3 m^3/m^2 (0.5 - 1.0 ft^3/ft^2 collector).

If storage sizes greater than this range are used, the performance of the system may actually decrease because heat losses from the storage system will increase, and the temperature of the air going to the system load will decrease.

To minimize heat losses and material costs, the shape of the rock bed should be such that its surface area is minimal. However, in practice, rock bed storage units are usually constructed as square or rectangular bins with the air passing vertically through the pebbles as shown in Figure 2. A maximum depth of about 2.5 m is recommended to limit the pressure drop through the bed. For active solar systems, the flow rate of air through the rock bed is determined by the collector area. The design value is typically 0.01 m^3/s per square metre of collector. Combined with the recommended storage volume of 0.15 - 0.3 m^3/m^2, these figures indicate a flowrate of 0.03 - 0.07 m^3/s per cubic metre of rock storage, with 0.05 m^3/s a good rule-of-thumb. For a bed depth of 1.5 to 2.5 metres, the bed face velocity should be approximately 0.075 to 0.1 m/s regardless of the size of the storage system.

Rock bed storage systems with horizontal flow can be constructed if vertical space is limited. However, horizontal flow rock beds are not recommended because of problems with air flow distribution and channeling. When the bed is loaded, smaller pebbles tend to move to the bottom of the bed creating greater resistance to the horizontal flow of air through the lower portion of the bed. The air therefore has a tendency to channel through the upper part of the bed causing a loss in storage capacity and a reduction in system performance. Attempts have been made to ameliorate these problems as shown in Figures 6 and 7. Baffles can be installed perpendicular to the air flow or, alternatively, horizontal sheets of impervious materials such as plastic sheets can be used to encourage uniform horizontal flow.

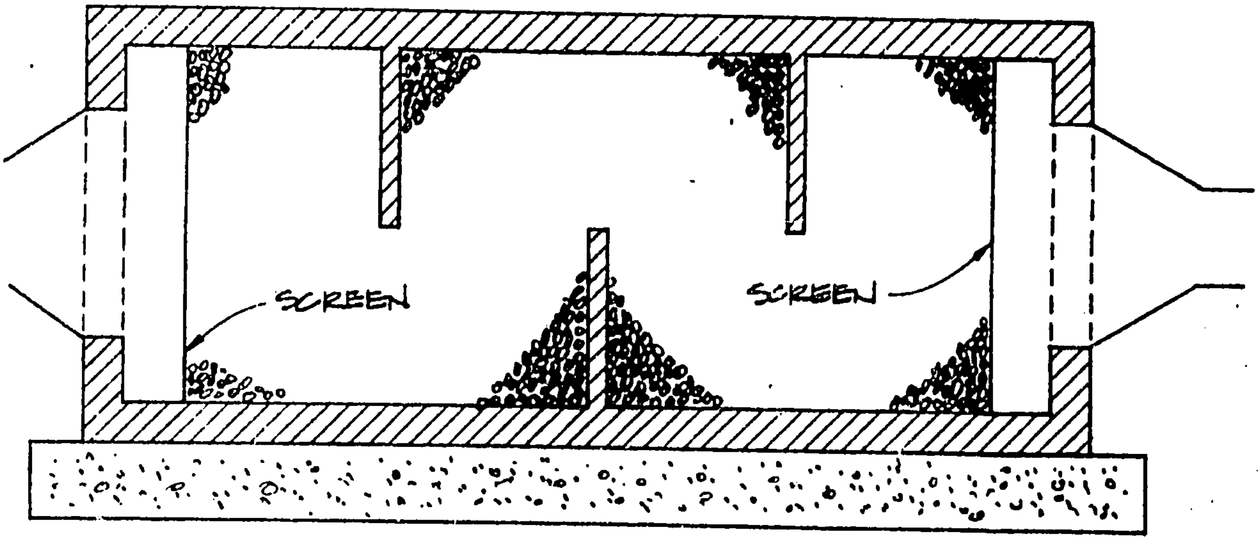


Figure 6. Horizontal Flow Rock Bed (Alternative 1)

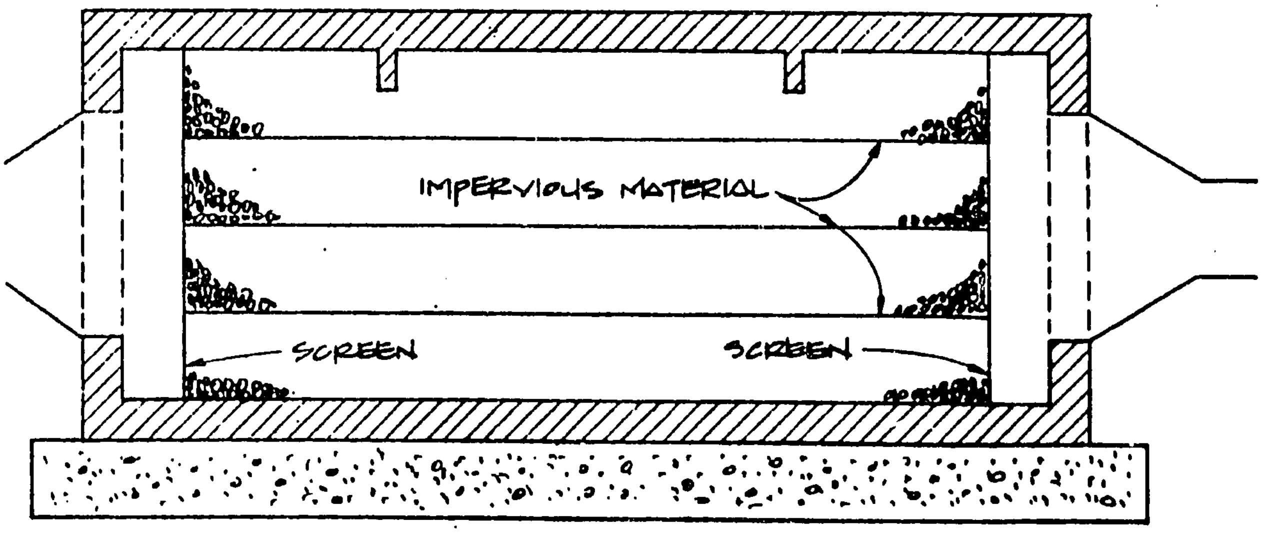


Figure 7. Horizontal Flow Rock Bed (Alternative 2)

Rock Bed Pressure Drop

The size of the pebbles or rocks used in the bed affects two important design parameters. Decreasing the rock size increases the rate of heat transfer but also increases the pressure drop across the bed. The rock diameter should be small enough to ensure that heat is conducted into the centre of the rock as fast as heat is transferred to the rock from the air flow. For typical flow rates a rock diameter of 25 mm or less is recommended.

Too high a pressure drop across the bed means high system electrical energy consumption, whereas too low a pressure drop makes uniform flow distribution difficult. A pressure drop of 30 to 50 Pa across the bed is considered acceptable (0.0044-0.0073 psi, or 0.12-0.2 inches water). Accurate prediction of rock bed pressure drop, however, is never very precise since it varies according to how the storage container is filled. Differences in pressure drop of up to 20% have been measured in laboratory tests of the same rocks in the same container [9]. The shape of the rocks also has an effect on the bed pressure drop. Generally, one can distinguish three varieties of rock bed fill:

(1) Round stones; obtained from gravel pits, no sharp edges.
(2) Crushed stones; obtained by crushing round stones, sharp and round edges.
(3) Crushed rock; obtained from quarries, no round edges - all sharp.

These varieties are screened. The grade classified as 'clear' is the appropriate type for rock bed storage units since the clear grade has only a narrow range of particle sizes [5].

Nine samples of different kinds of rocks and stones were tested for their pressure drop characteristics [9]. The results are shown in Figure 8. The curves shown in this figure are for randomly packed beds and for clean, washed rocks. Unwashed rocks can have twice the pressure drop of clean rocks.

Fan Requirements

In order to select the size of fan required for the system, the designer must determine the overall pressure drop through the rock bed-collector loop. The total static pressure drop is the sum of the pressure drops through the rock bed, plenums, ducting and solar collectors. The pressure drop through the solar collectors can usually be obtained from the manufacturer. As a rule-of-thumb, fan power requirements will be approximately 2.5 Watts per square metre of collector area.

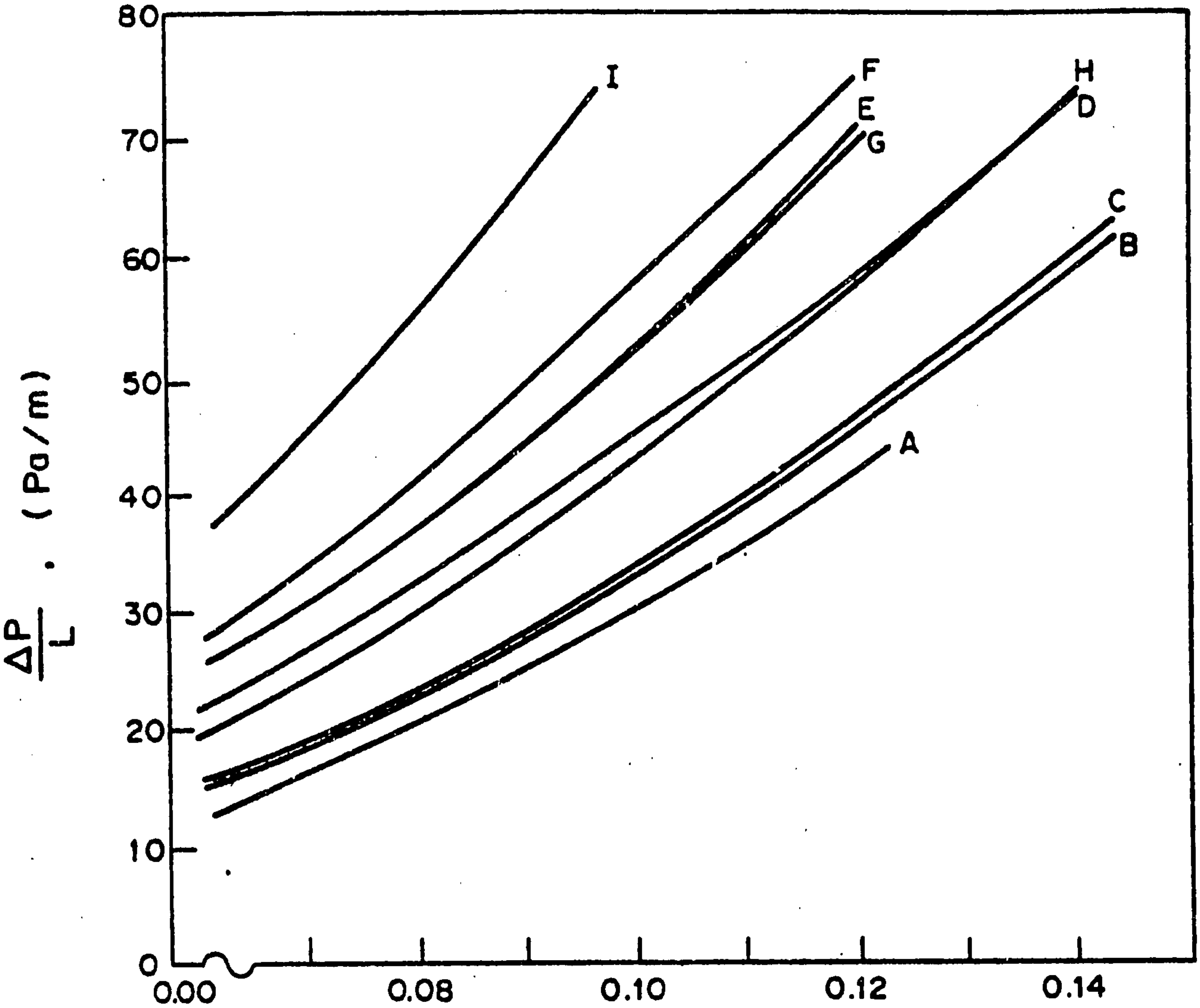


Figure 8. Plot of Rock Bed Pressure Drop Versus Face Velocity.

where P is static pressure loss (Pa)
L is rock bed length in the flow direction (m)
V is rock bed face velocity (m/s) (total flow rate divided by bed cross-sectional area)

A 19 mm Rounds (clear)
B 19 mm Crushed Rock (clear)
C 19 mm Crushed Stone (clear)
D HL4 Crushed Stone
E HL1 Crushed Stone
F 12.7 mm Rounds (clear)
G HL6 Crushed Rock
H HL1 Crushed Rock
I 9.5 mm Rounds (clear)

Note: Although this figure is an approximation based on a very limited sampling, it is entirely adequate for the purpose of ensuring a pressure drop that leads to a uniform flow.

Pump Selection

In an active solar hot water system a pump will be required to circulate a fluid within the system. It is important that the pump selected for this task is carefully chosen. Pumps are selected by matching their performance curves with the pressure drop-flowrat characteristics of the systems itself. A typical performance curve for a small centrifugal pump is shown below in Figure 9. The curve indicates that this pump will lift water to a maximum height of 8 feet, but at this point there is no flow. As the head against which the pump is working falls, so the flowrate increases up to a maximum of about 1300-1700 gallons per hour depending on the size of the fittings. In actual operation, the pump will operate somewhere between these two extremes at a point which depends on the system curve.

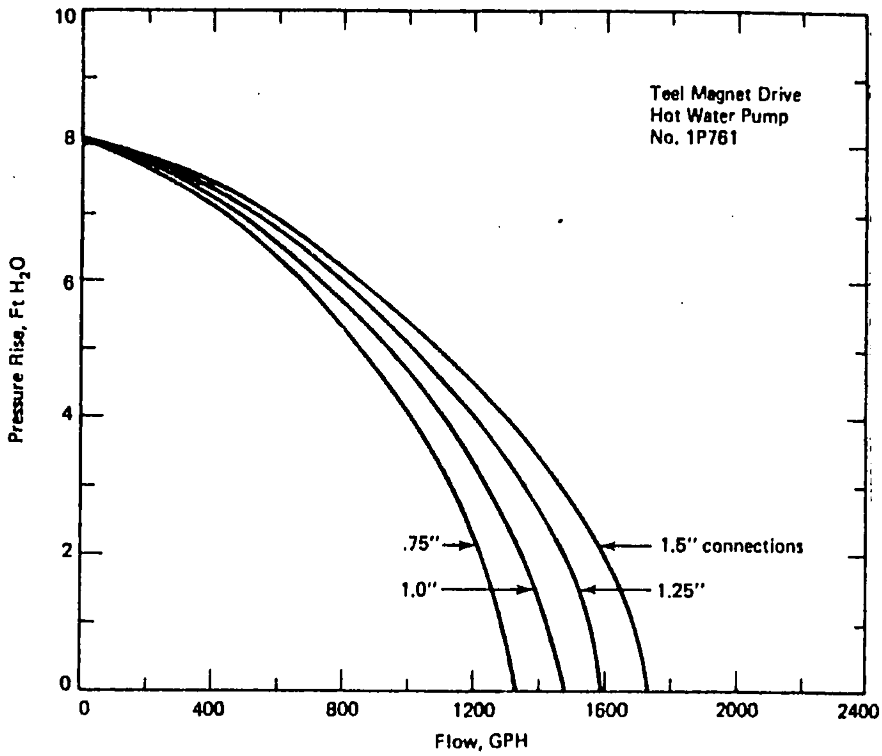


FIGURE 9. Pressure-flow characteristic curve for a magnetic drive centrifugal pump.

To construct the system curve it is necessary to calculate the system pressure drop at the design flowrate. The system curve then follows from the relationship between flowrate and system pressure drop:

$$\frac{H_1}{H_2} = \left(\frac{Q_1}{Q_2}\right)^2 \qquad (7)$$

where:

H = pressure drop or head,
Q = volumetric flow rate.

A system curve is shown in Figure 10. Where the system curve intersects the pump performance curve defines the operating point of the system. A pump is therefore selected so the estimated operating point is at a system flow rate close to the design value.

It should be noted that the system curve shown is for a closed-loop system where the pump is required only to overcome fluid friction at the design flowrate. In open-loop systems, the pump must also lift the fluid to an elevation which will depend on the system configuration. The system curve will be displaced upwards on the graph by a head equal to the lift required of the pump.

Figure 11 shows performance curves for several small pumps frequently used in solar heating systems. This figure illustrates how the shape of the performance curve can vary. Centrifugal pumps used in closed loop piping circuits should be selected for a mid-curve operating condition, and should have relatively flat performance curves. Pumps with a steep- curve characteristic should not be used because they tend to limit system flow rates.

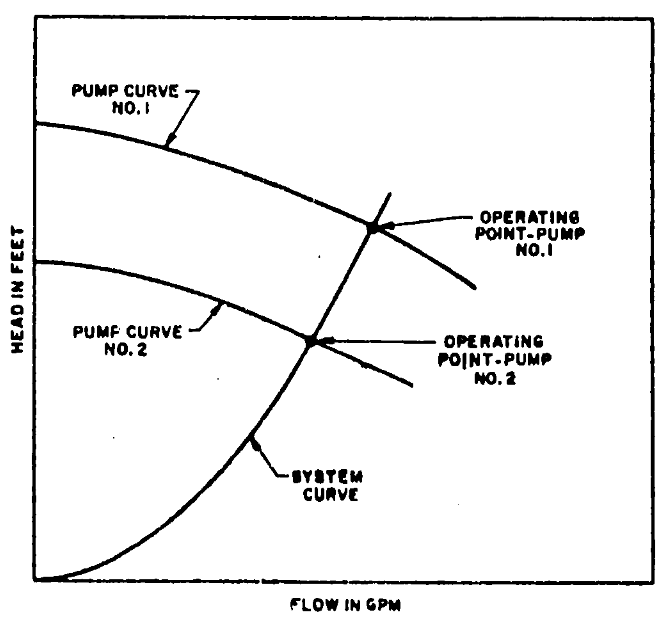


Fig 10. Typical System Curve for a Closed-Loop System.

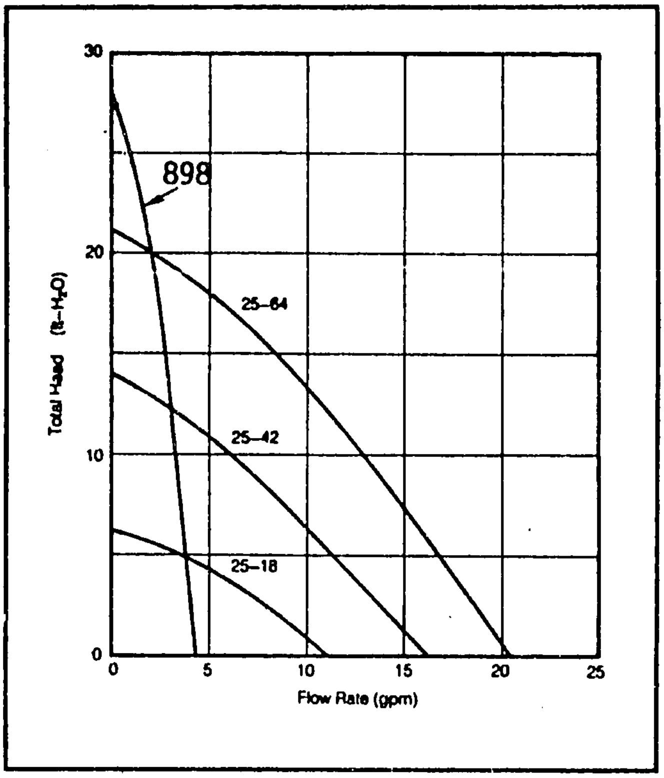


Figure 11 Example pump curves (25-64, 25-42, and 25-18 are Grundfos; 898 is a Richdel pump).

Parallel Pumping

Pumps are often used in parallel. For parallel operation, each pump operates at the same head and provides one half the system flowrate. The pump curve for parallel operation can be established by doubling the flow rate of the single pump curve as shown in Figure 12. The operating point for both single and parallel operation can be determined by drawing in the system curve as indicated in Figure 13. When only a single pump is in operation the system flow rate is reduced, not by half, but by an amount dependent on both the characteristics of the pumps and the system. When only a single pump is operating, the flowrate is higher than that through each pump when they are operated together in parallel. When possible, the pumps should be selected to permit single-pump operation. This allows single-pump operation in the event one of the pumps fails.

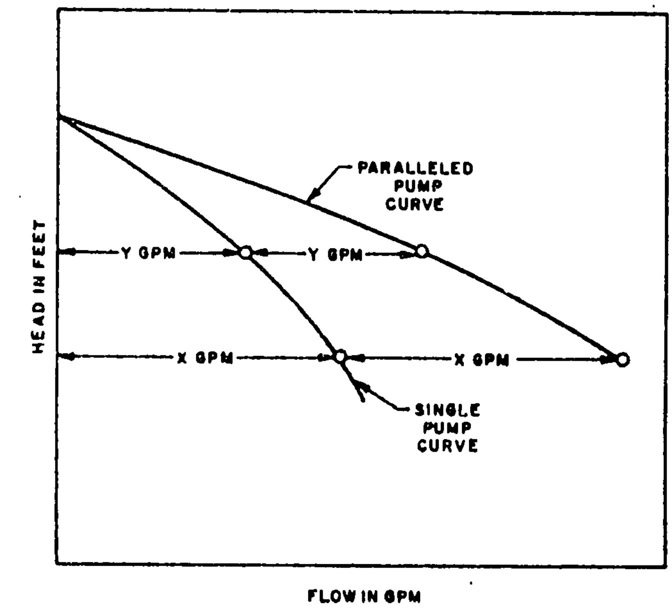


Fig. 12. Pump Curve for Parallel Operation

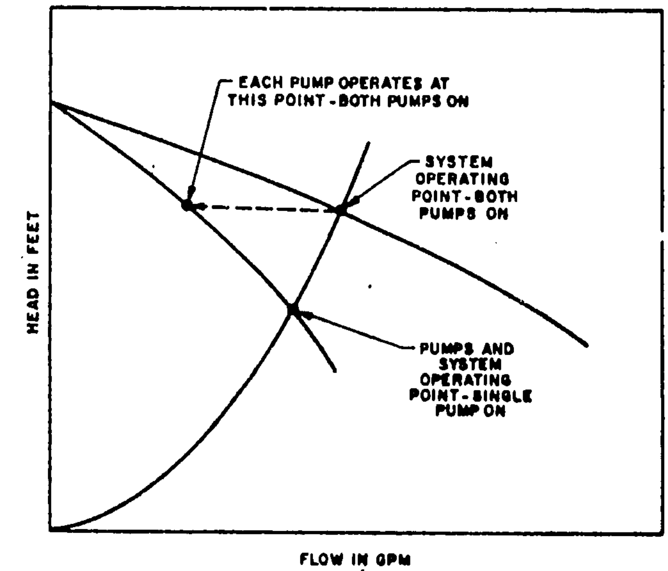


Fig. 13 Operating Conditions for Parallel Pump Installation

Pumps in series

When pumps are placed in series, each pump provides one half the total system pumping head at equal pumping rates. The pump curve for pumps in series can be constructed from the performance curve of a single pump by doubling the head at each point of the curve as shown in Figure 14. Figure 15 shows the system curve drawn in and the system operating point. It should be noted that each pump draws maximum power during the series operation. During single-pump operation, the pump draws minimum power.

It should be emphasized that both parallel and series configurations require close attention to pump and system characteristics in order to accurately determine the expected operating points. The use of safety factors, the use of improper or inaccurate pressure drop charts, inadequate pressure drop calculation, etc., may lead to inappropriate pump selection and consequent operational difficulties.

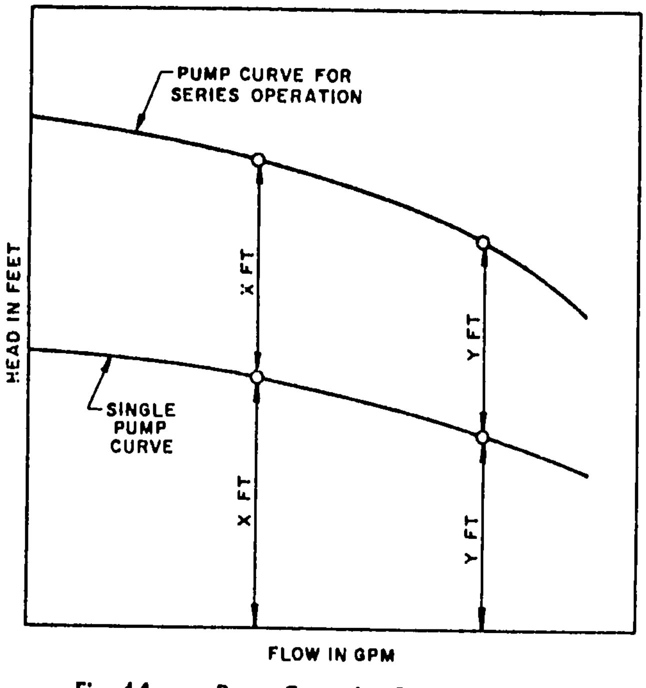


Fig. 14 Pump Curve for Series Operation

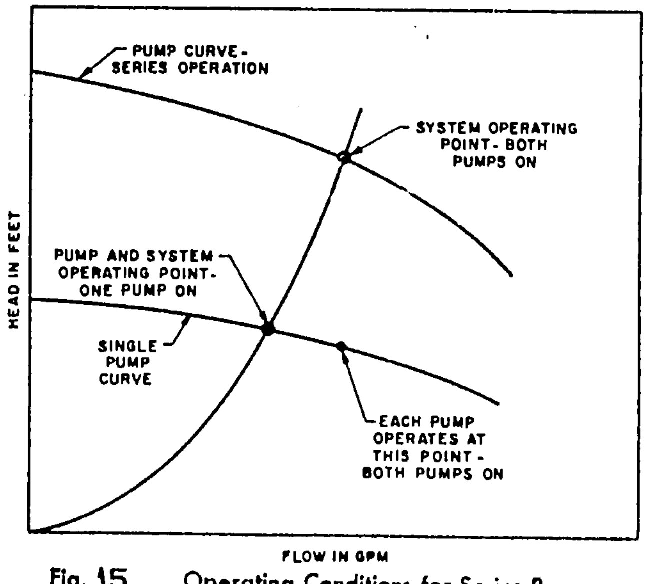


Fig. 15 Operating Conditions for Series Pump Installation

System Pressure Drop

In order to establish the system curve it is necessary to determine the system pressure drop at the design flowrate. The volumetric flowrate is determined from heat transfer considerations, but the system pressure drop will depend on fixing the diameter of the pipework and accounting for all fittings, bends, and other equipment in the piping system. The pipe diameter can be determined by observing some simple design criteria. The general range of pipe friction losses in hydronic systems is between 1 and 4 ft. of head per 100 ft. of pipe length. A value of 2.5 ft/100 ft. is a mean value to which many systems are designed [10]. A further constraint is velocity. Closed loop hydronic systems are generally sized so that the fluid velocity does not exceed 4 fps (1.2 m/s). Above this velocity the system becomes noisy and erosion can start to become a problem especially at elbows.

It is also recommended that fluid velocity should not fall below 2 fps (0.6 m/s). These limits on fluid velocity and pressure drop closely constrain the selection of an appropriate pipe diameter. The central area of Figure 16 shows the region of permissible pipe sizes for a specified fluid flow. One would generally select the smallest available pipe diameter that falls within this central region.

For valves, bends, functions, and other elements, pressure drop is usually listed in elbow equivalents. The elbow equivalent is then used to estimate an equivalent pipe length from Table 3. Elbow equivalents for valves and fittings for iron and copper pipes are shown in Table 4.

Table 3. Equivalent Length in Feet of Pipe for 90-Deg Elbows

Vel. Fps	Pipe Size														
	½	¾	1	1¼	1½	2	2½	3	3½	4	5	6	8	10	12
1	1.2	1.7	2.2	3.0	3.5	4.5	5.4	6.7	7.7	8.6	10.5	12.2	15.4	18.7	22.2
2	1.4	1.9	2.5	3.3	3.9	5.1	6.0	7.5	8.6	9.5	11.7	13.7	17.3	20.8	24.8
3	1.5	2.0	2.7	3.6	4.2	5.4	6.4	8.0	9.2	10.2	12.5	14.6	18.4	22.3	26.5
4	1.5	2.1	2.8	3.7	4.4	5.6	6.7	8.3	9.6	10.6	13.1	15.2	19.2	23.2	27.6
5	1.6	2.2	2.9	3.9	4.5	5.9	7.0	8.7	10.0	11.1	13.6	15.8	19.8	24.2	28.8
6	1.7	2.3	3.0	4.0	4.7	6.0	7.2	8.9	10.3	11.4	14.0	16.3	20.5	24.9	29.6
7	1.7	2.3	3.0	4.1	4.8	6.2	7.4	9.1	10.5	11.7	14.3	16.7	21.0	25.5	30.3
8	1.7	2.4	3.1	4.2	4.9	6.3	7.5	9.3	10.8	11.9	14.6	17.1	21.5	26.1	31.0
9	1.8	2.4	3.2	4.3	5.0	6.4	7.7	9.5	11.0	12.2	14.9	17.4	21.9	26.6	31.6
10	1.8	2.5	3.2	4.3	5.1	6.5	7.8	9.7	11.2	12.4	15.2	17.7	22.2	27.0	32.0

Table 4. Iron and Copper Elbow Equivalents

Fitting	Iron Pipe	Copper Tubing
Elbow, 90-deg	1.0	1.0
Elbow, 45-deg	0.7	0.7
Elbow, 90-deg long turn...	0.5	0.5
Elbow, welded, 90-deg....	0.5	0.5
Reduced coupling	0.4	0.4
Open return bend........	1.0	1.0
Angle radiator valve......	2.0	3.0
Radiator or convector	3.0	4.0
Boiler or heater	3.0	4.0
Open gate valve	0.5	0.7
Open globe valve	12.0	17.0

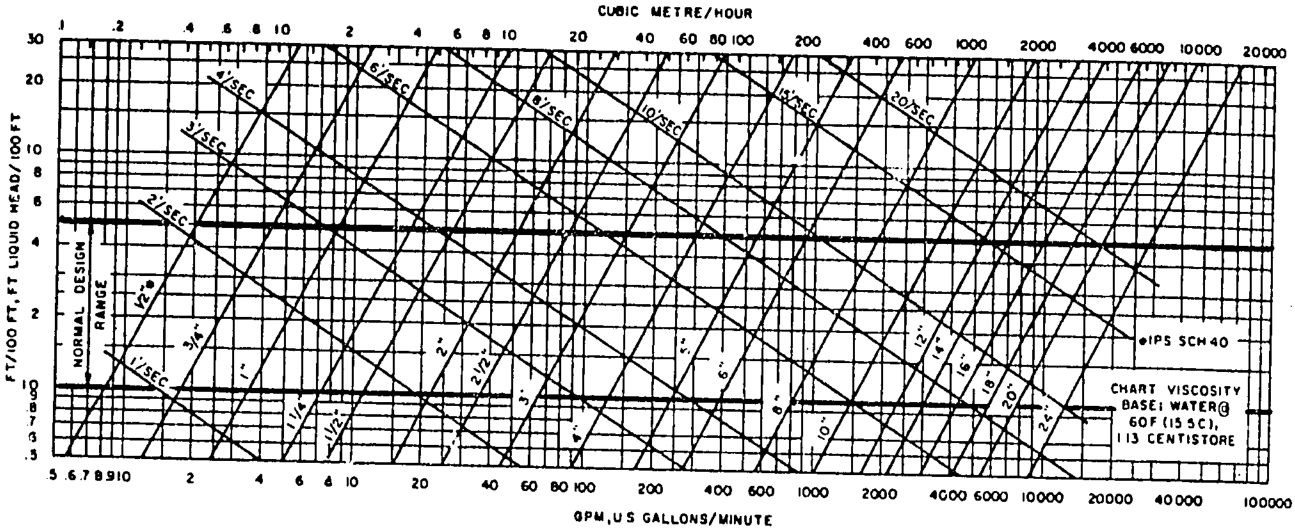


Fig. 16 Friction Loss for Water in Commercial Steel Pipe (Schedule 40)

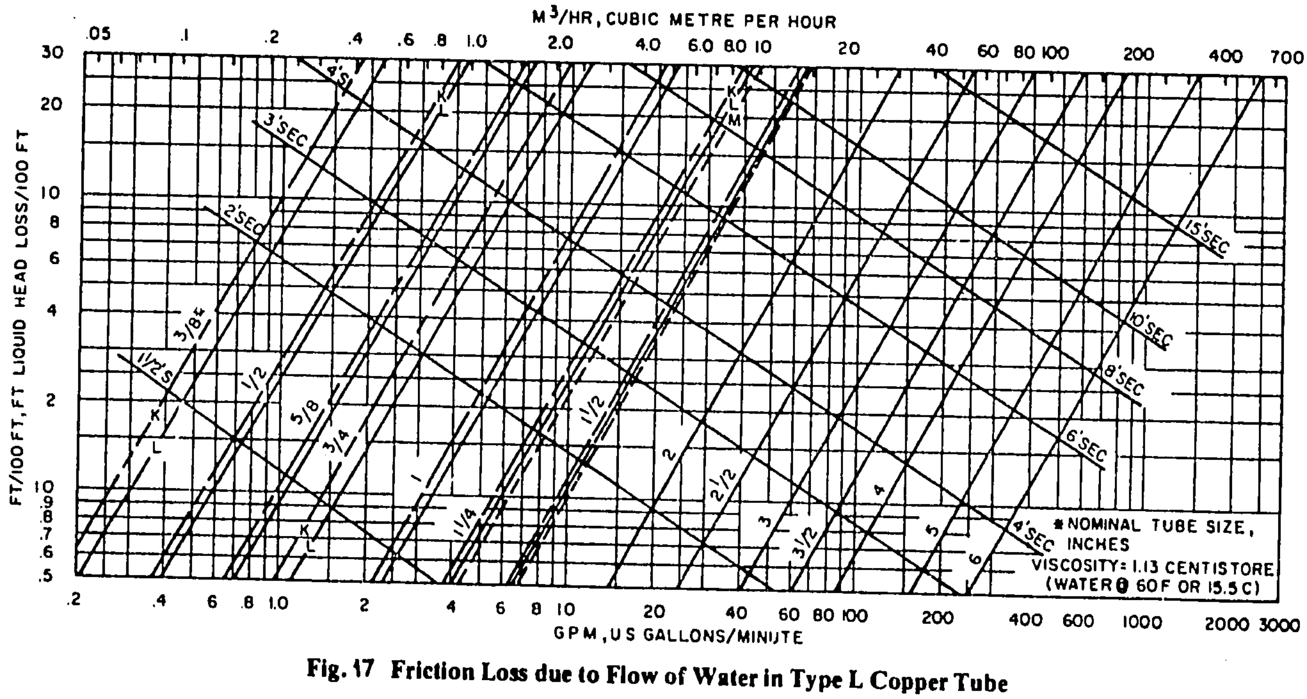


Fig. 17 Friction Loss due to Flow of Water in Type L Copper Tube

Example 3

Determine the pressure drop for a 1-inch open gate valve and 20 ft. of 1-inch type L copper pipe at a flow velocity of 2.5 fps.

Solution

From Table 4, an open gate valve is equivalent to 0.7 elbows. From Table 3, a 1-inch elbow at 2.5 fps fluid velocity is equivalent to 2.6 feet of 1 inch pipe. Therefore, the gate valve is equivalent to 0.7 x 2.6 = 1.8 feet of pipe.

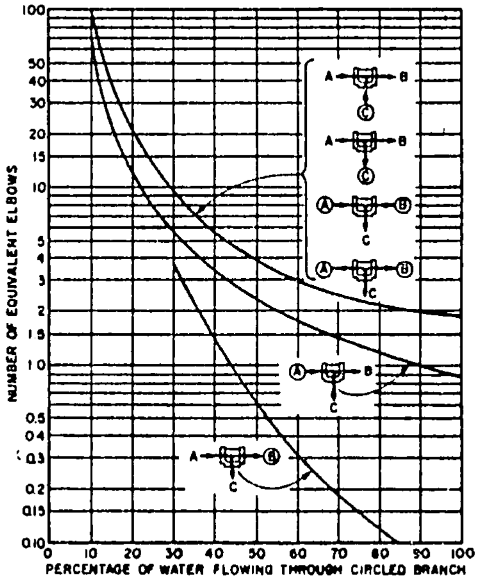
The pressure drop for the system is therefore based on 20 + 1.8 = 21.8 feet of 1-inch type L copper pipe. From Figure 17, the pressure drop at 2.5 fps can be estimated as 3 feet of water per 100 ft. of pipe. For 21.8 feet of pipe the pressure drop would be

$$\frac{3}{100} \times 21.8 = 0.65 \text{ ft } H_2O$$

$$= 0.28 \text{ psi}$$

The following rule of thumb is often used: the equivalent length of pipe for an elbow (in feet of pipe) is approximately twice the nominal pipe diameter in inches. Thus, a 1-inch elbow is equivalent to 2 feet of 1-inch pipe, a 4-inch elbow is equivalent to 8 feet of 4-inch pipe, etc.

Pressure drop through pipe tees varies with flow through the branch. Pressure drops through the functions for tees of equal inlet and outlet sizes are shown in Figure 18.



Notes: 1. The chart is based on straight tees, that is, branches A, B, and C are the same size.

2. Pressure loss in desired circuit is obtained by selecting proper curve according to illustrations, determining the flow at the circled branch, and multiplying the pressure loss for the same size elbow at the flow rate in the circled branch by the equivalent elbows indicated.

3. When the size of an outlet is reduced the equivalent elbows shown in the chart do not apply. Therefore, the maximum loss for any circuit for any flow will not exceed 2 elbow equivalents at the maximum flow (gpm) occurring in any branch of the tee.

4. The top curve of the chart is the average of 4 curves, one for each of the tee circuits illustrated.

Fig. 18 Elbow Equivalents of Tees at Various Flow Conditions

Heat Exchangers

Heat exchangers are often used in solar hot water systems to transfer heat from one fluid circuit to another, to transfer heat from the collector loop to thermal storage, or to transfer heat from storage to the system load. The analysis of heat exchanger performance is therefore an important component of the overall system design.

A convenient method for sizing heat exchangers and estimating their performance uses a parameter called the heat exchanger effectiveness, ϵ. Heat exchanger effectiveness is is the ratio of the actual rate of heat transfer, Q_{HX}, to the theoretical rate, Q_{TH}, that would occur if the heat exchanger area of heat transfer were infinitely large. Consider the simple counterflow heat exchanger shown below. For this configuration

$$Q_{HX} = C_c\,(t_2 - t_1) = C_h\,(T_1 - T_2) \text{ Watts} \tag{8}$$

where C_c = cold fluid heat capacitance rate, W/K

C_h = hot fluid heat capacitance rate, W/K

The fluid heat capacitance rates are simply the product of the fluid heat capacity, in J/kg K, and the mass flow in kg/s.

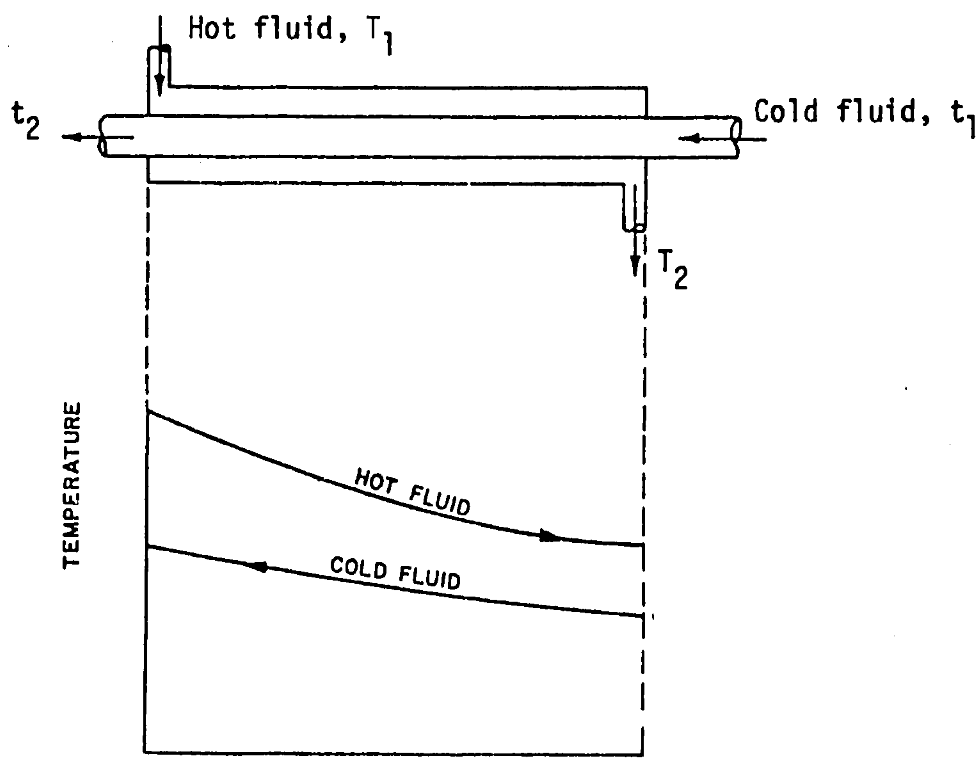


Figure 19. Single-Pass Counterflow Shell-and-Tube Heat Exchanger

The maximum rate of heat transfer that could theoretically occur, Q_{TH}, is given by

$$Q_{TH} = C_{min} (T_1 - t_1) \text{ Watts} \tag{9}$$

where C_{min} is the smaller of C_c and C_h.

since $\varepsilon = \frac{Q_{HX}}{Q_{TH}}$

we have $$Q_{HX} = \varepsilon\, C_{min} (T_1 - t_1) \text{ Watts} \tag{10}$$

This is a very convenient formula since once the heat exchanger effectiveness, ϵ, is determined, the performance of the heat exchanger is only a function of the incoming fluid temperatures, which are usually known with some certainty.

It can be shown that the heat exchanger effectiveness can be found from

$$\varepsilon = \frac{1 - \exp[-NTU(1 - C^*)]}{1 - C^*\exp[-NTU(1-C^*)]} \tag{11}$$

where C^* is the ratio of the fluid heat capacitance rates so that C^* is always less than unity, i.e.

$$C^* = C_{min}/C_{max}$$

and NTU is parameter called the <u>number of transfer units</u>, abbreviated as NTU. This parameter is given by

$$NTU = \frac{UA}{C_{min}} \tag{12}$$

where UA is the heat transfer coefficient and area product for the heat exchanger.

Equation 11 is presented graphically in Figure 20 and is valid for counterflow heat exchangers only. Typical values for the overall heat transfer coefficient, U, for shell-and-tube exchanger made of metal range between 80 and 120 Btu/hr ft^2 °F (450-680 W/m^{2}K) depending on fluid temperatures, flowrates, and the type of fluid used.

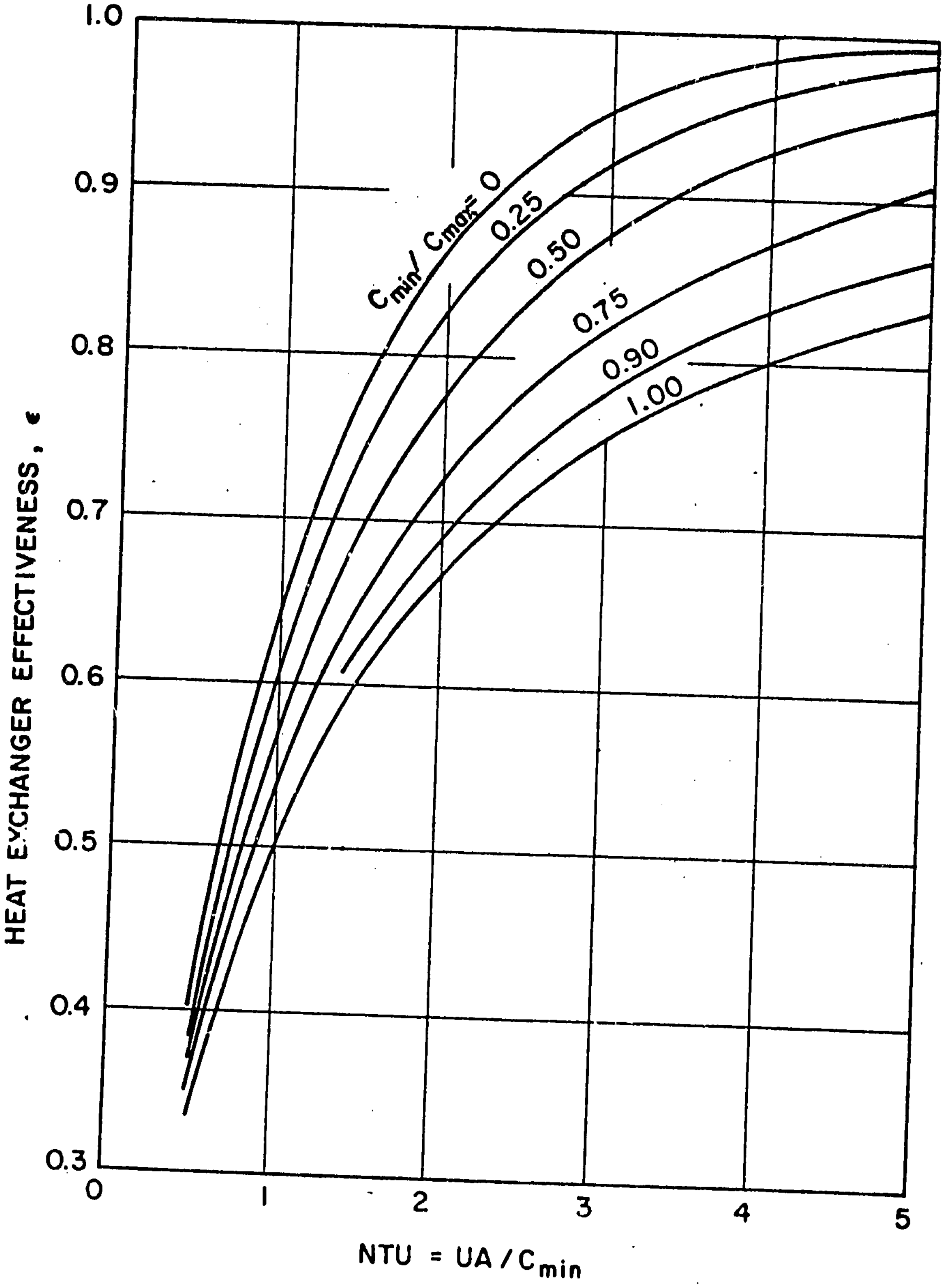


Figure 20. Effectiveness for a Counterflow Single-Pass Shell-and-Tube Heat Exchanger.

If the heat capacitance rates are equal, then $C^* = 1$ and Equation 11 becomes indeterminate. In this case the heat exchanger effectiveness is given by

$$\varepsilon = \frac{NTU}{1 + NTU} \qquad (13)$$

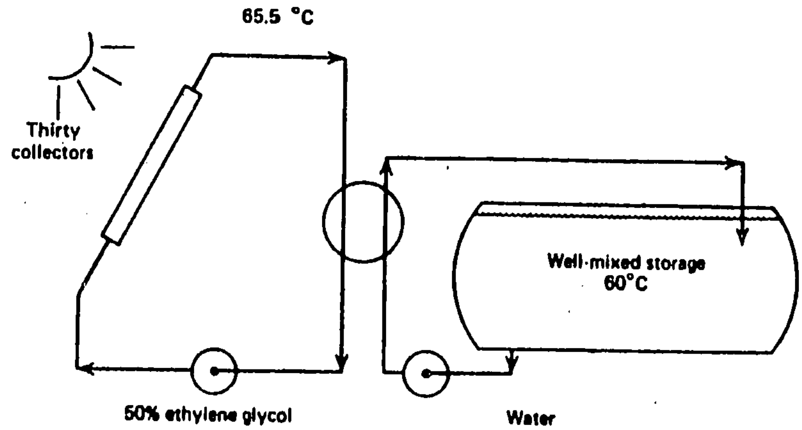
where NTU is determined as before, by Equation 12 When one flow rate is much larger than the other (> 15:1), the equations may be simplified to

$$\varepsilon = 1 - \exp(-NTU) \qquad (14)$$

This equation also applies for situations where one side of the heat exchanger surface may be considered isothermal; for example, a coil inside or wrapped around a tank.

Example 4

An array of collectors of area 90 m^2 transfers heat via a heat exchanger to a a well-mixed storage tank holding water. A 50% ethylene glycol solution (C_p = 3.52 KJ/kg K) circulates through the collector loop at a rate of 0.36 kg/s. Water circulates through the exchanger from storage at a rate of 0.5 kg/s. The exchanger is rated at UA = 5033 W/K. The temperature of the fluid leaving the collectors is 65.5 °C; the temperature in the storage tank is 60°C. What is the temperature of the fluid returning to storage and going to the collectors?



Solution

For the collector loop we have

$$C_h = 0.36 \text{ kg/s} \times 3.52 \text{ kJ/kg K}$$
$$= 1267.2 \text{ W/K}$$

And for the storage loop

$$C_c = 0.5 \text{ kg/s} \times 4.19 \text{ KJ/kg K}$$
$$= 2095.0 \text{ W/K}$$

so
$$C_{min} = 1267.2$$
$$C_{max} = 2095.0$$
$$C^* = 1267.2/2095 = 0.6$$

therefore
$$NTU = \frac{UA}{C_{min}} = \frac{5033}{1267.2} = 3.97$$

From Figure 20, the heat exchanger effectiveness can be estimated as 0.9. The rate of heat transfer in the heat exchanger can therefore be determined from Equation 10 as

$$Q_{HX} = 0.9 \times 1267.2 \times (65.5 - 60)$$
$$= 6272.6 \text{ Watts}$$

From Equation 8 we can now determine the exchanger outlet temperatures as

$$t_2 = t_1 + Q_{HX}/C_c$$
$$= 60 + 6272.6/2095.0 = \underline{63.0^\circ C}$$
$$T_2 = T_1 - Q_{HX}C_h$$
$$= 65.5 - 6272.6/1267.2 = \underline{60.6^\circ C}$$

Heat Exchanger Factor

A useful and convenient analytical treatment of heat exchangers in solar thermal systems has been developed by deWinter [14]. It can be shown that the effect of the exchanger in the collector loop is to reduce the collector heat removal factor, F_R, by a factor, F_X, called the heat exchanger factor.

The useful energy gain from the collector loop with a heat exchanger is then given by:

$$Q_u = F_X \; F_R \, A_c \, [I\tau\alpha - \; U_L \, (T_{in} - T_a)] \qquad (15)$$

The heat exchanger factor is found from

$$F_X = \frac{1}{1 + \dfrac{A_c F_R U_L}{C_{col}} \left(\dfrac{C_{col}}{\epsilon C_{min}} - 1 \right)} \qquad (16)$$

where $\quad C_{col}$ = heat capacitance rate of the collector loop

It is customary to define a modified heat removal factor, $F_R' = F_X F_R$, and then to write Equation 15 as

$$Q_u = F_R' \; A \; [\, I\tau\alpha - U_L \; (T_{in} - T_a)] \quad . \qquad (17)$$

where F_R' accounts for the heat transfer penalty imposed by the presence of the heat exchanger. Since the performance of the collectors is reduced by a factor F_x due to the heat exchanger, the collector area should be increased by a factor equal to the reciprocal of F_x in order for the system to produce as much energy with the exchanger as a similar system without an exchanger.

Example 5

Select a heat exchanger for a liquid system with a collector area of 50 m^2. The collector has the following characteristic parameters:

$$F_R\,(\tau\alpha)_n = 0.7$$
$$F_R U_L = 5.4 \;\; W/m^2K$$

Assume a time of day when the insolation on the plane of the collector is 950 W/m^2, inlet fluid temperature is 55°C, storage water temperature is 55°C, and ambient temperature is 0°C. The collector fluid is a 50 percent mixture of ethylene glycol (specific heat 3.4 kJ/kgK, density 1048 kg/m^3).

Solution

The energy collected is given by

$$Q_u = F_R A_c \; [I\tau\alpha - \; U_L \; (T_{in} - T_a)]$$

$$= 50 \; [0.7 \; \times 950 - 5.4 \; (55 - 0)]$$

$$= 18.4 \; kW$$

To size the heat exchanger, flow rates in both the collector and the storage loops are required. These are not given but the rule-of-thumb for the collector loop is 0.015 L/m^2s, so the total flow should be 50 x 0.015 = 0.75 L/s. The rate of storage water through the exchanger is typically twice the rate through the collectors, or 1.5 L/s.

collector loop: $C = 0.75 \left(\frac{L}{s}\right) \times 1.048 \left(\frac{kg}{L}\right) \times 3400 \left(\frac{J}{kgK}\right)$

$= 2672$ W/K

storage loop: $C = 1.5 \left(\frac{L}{s}\right) \times 1 \left(\frac{kg}{L}\right) \times 4190 \left(\frac{J}{kgK}\right)$

$= 6285$ W/K

so $C_{max} = 6285$ W/K
$C_{min} = 2672$ W/K
$C^* = 2672/6285 = 0.43$

We can find the heat exchanger effectiveness, ε , from Equation 10 rewritten as

$$\epsilon = \frac{Q_{HX}}{C_{min}\ (T_1 - t_1)}$$

We know $t_1 = 54°C$ and can find T_1 from

$$Q_{HX} = C_{col}\ (T_1 - T_2)$$

where C_{col} is the heat capacitance rate of the collector loop. Since $C_{col} = 2672$ W/K and $T_2 = 55°C$

$$T_1 = \frac{18400}{2672} + 55 = 61.9°C$$

thus $\epsilon = \dfrac{18400}{2672\ (61.9-54)} = 0.87$

From Figure 20, with $\epsilon = 0.87$ and $C_{min}/C_{max} = 0.43$ we can estimate NTU as 2.8

Since $NTU = \dfrac{UA}{C_{min}}$

$$UA = 2.8 \times C_{min} = 2.8 \times 2672$$

or $UA = 7482$ W/K

This parameter defines the performance of the required heat exchanger and permits the selection of an appropriate unit from manufacturer's catalogues.

System Control and Configuration

The function of the control system is to control the solar thermal system in such a way that the collection and storage of heat is accomplished as effectively as possible. The basic components of the control system are the temperature sensors, the differential temperature controller, and the output system. The differential controller starts the pump whenever a temperature sensor on the absorber plate of the collector indicates a temperature a few degrees higher than the temperature in the middle or at the bottom of the storage tank. The controller turns the pump off when the temperature differential falls below a preset level. Typical differentials are 15°F (8°C) for turn on, 5°F (3°C) for turn off. It is important that the collector sensor be mounted to a section of the absorber where it is thermally buffered from the temperature drop caused by the flow through the collector when the pump is first switched on. Otherwise, the pump may cycle on-off for a considerable period of time. Temperature sensors may also be used to monitor both freezing conditions and excessive storage temperatures. In each case the control system is designed so that the differential controller makes an appropriate response to the signals it receives from the sensors.

The output system delivers the appropriate control voltages from the differential controller to the pumps, valves, fans, or dampers that control the fluid circulation.

A typical system configuration is shown in Figure 21. This is a drain-down system, which means that in the event of freezing conditions the system is designed to drain away to a sewer. The vacuum breaker above the collectors permits air to enter the system while draining. Horizontal piping is avoided. When the temperature rises sufficiently the solenoid valves isolating the storage tank open, the dump valve closes, and the system is refilled from the cold water supply. The air vent on top of the system allows air to escape as the collectors fill with water. This kind of system is exposed to main line pressures and must be designed accordingly. The biggest disadvantage with this design, however, is that drainage is dependent on the combined action of an electrical dump-valve and a differential controller. If either fail in freezing conditions the system is likely to be damaged.

Figure 22 shows a similar system except that here the system drains back into the storage tank. This configuration is called a drain-back system. Whenever the circulation pump is off the fluid in the collectors drains back into the storage tank. The storage tank is vented to permit air to enter the system. Note that the collector loop is not exposed to main supply pressure which would prevent the system from draining. The collector return line must enter the storage tank above the fluid level in order for the system to drain.

Figure 23 shows a typical closed-loop solar thermal system. Since the collector loop is closed an expansion tank becomes necessary. If freezing is a potential problem the heat transfer fluid must be a water/glycol mixture or some other freeze resistant fluid. Figure 24 shows a similar closed-loop system, this time employing two tanks. This system will return cooler water to the collector thereby improving their efficiency. However, thermal losses from storage will be greater.

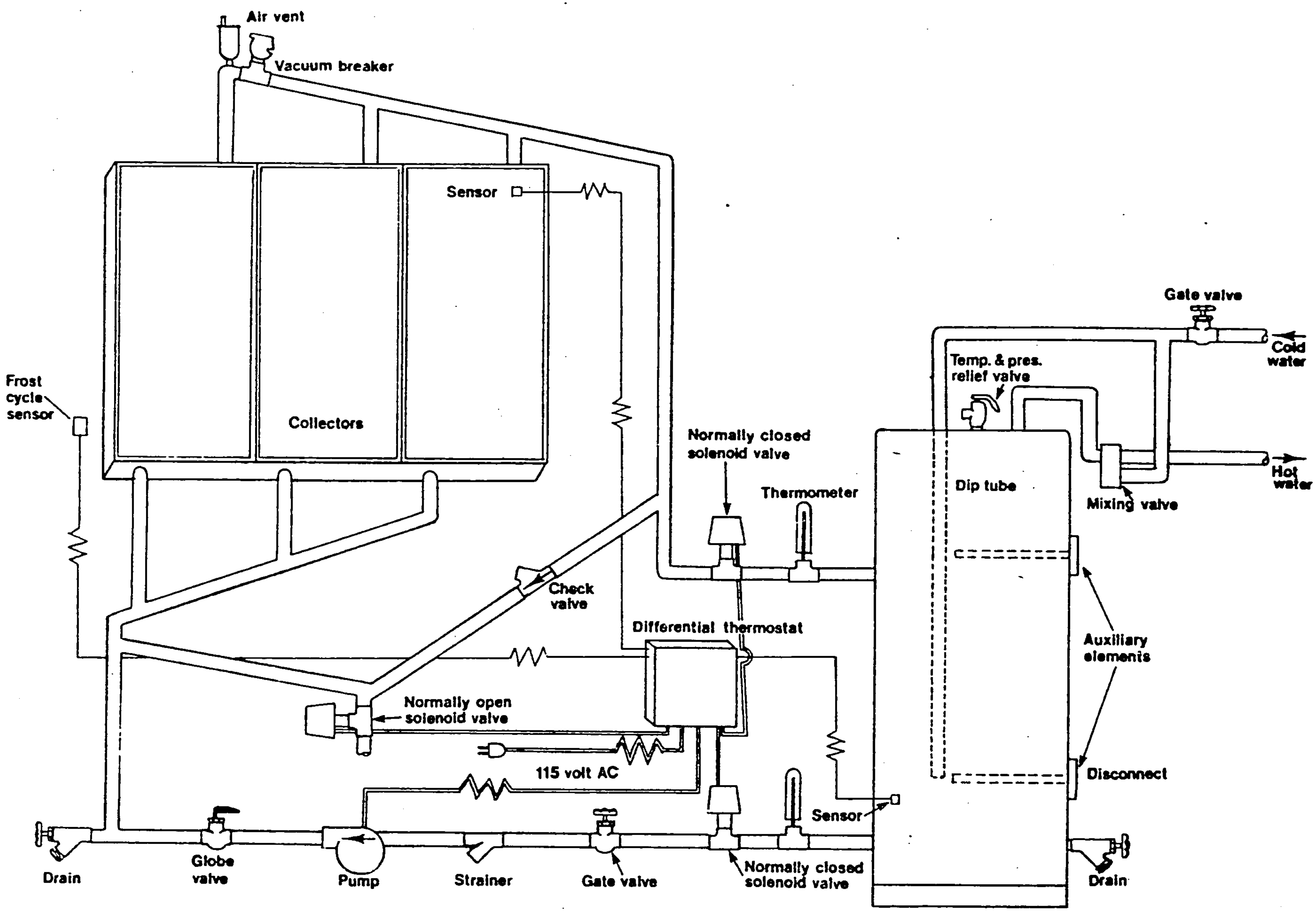


Figure 21. Drain-Down System

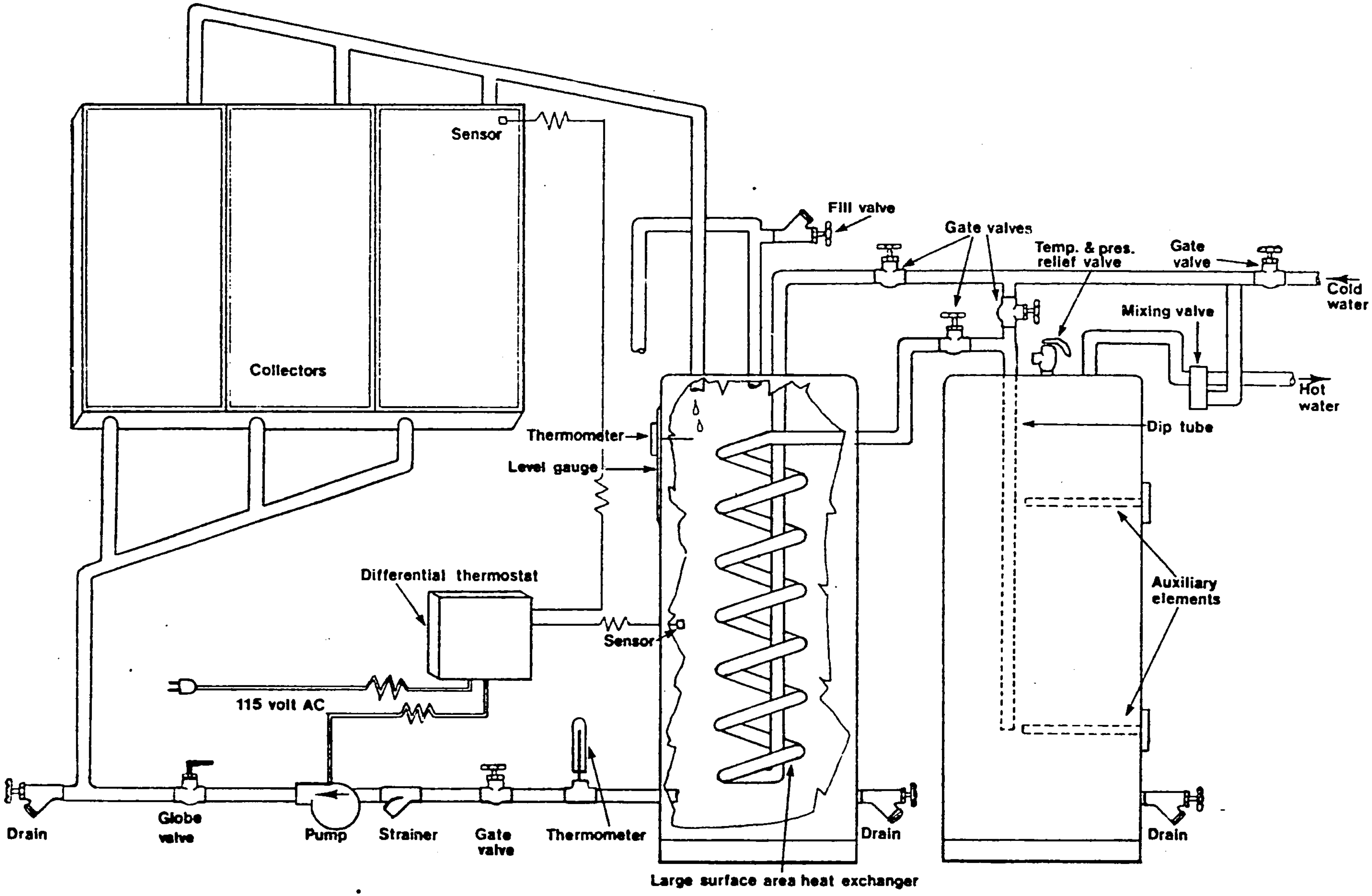


Figure 22. Drain-Back System

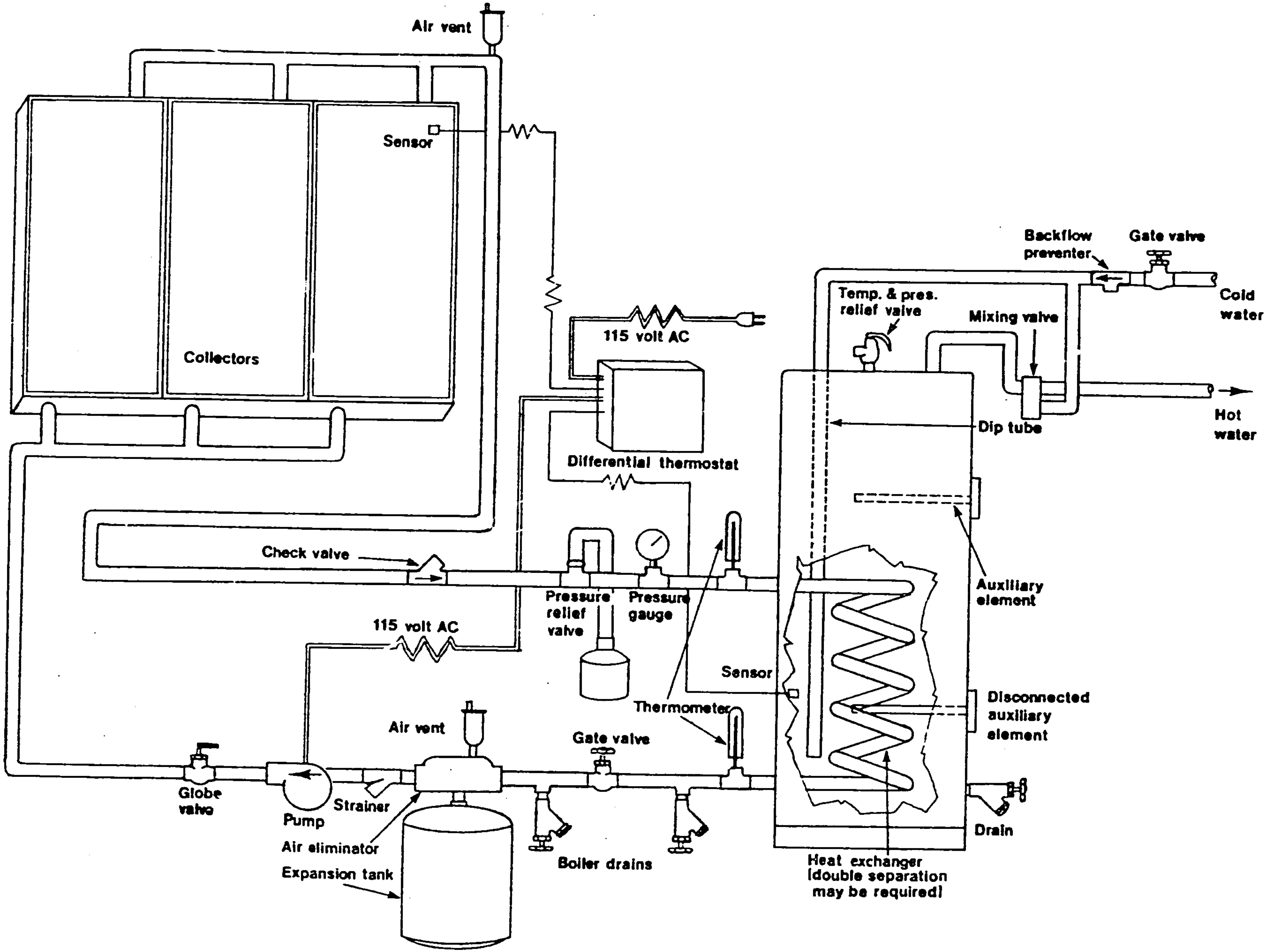


Figure 23. Closed-Loop System -- One Tank

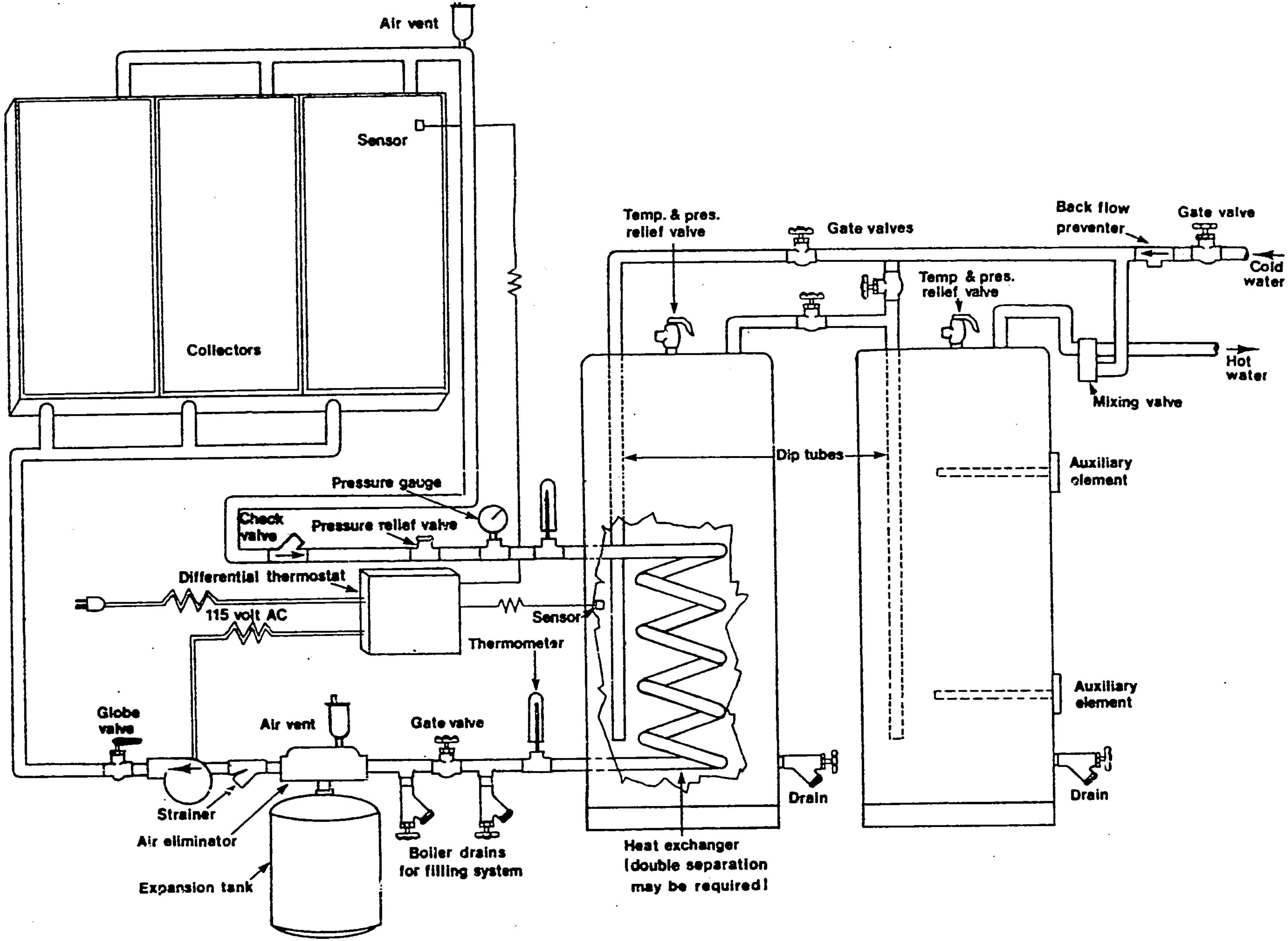


Figure 24. Closed-Loop System -- Two Tanks

By far the simplest solar thermal system is the thermosyphon system. Figure 25 shows a typical system. No control system is required. However, the storage system must be at a higher elevation than the collectors. This is often an impractical arrangement with large solar thermal systems.

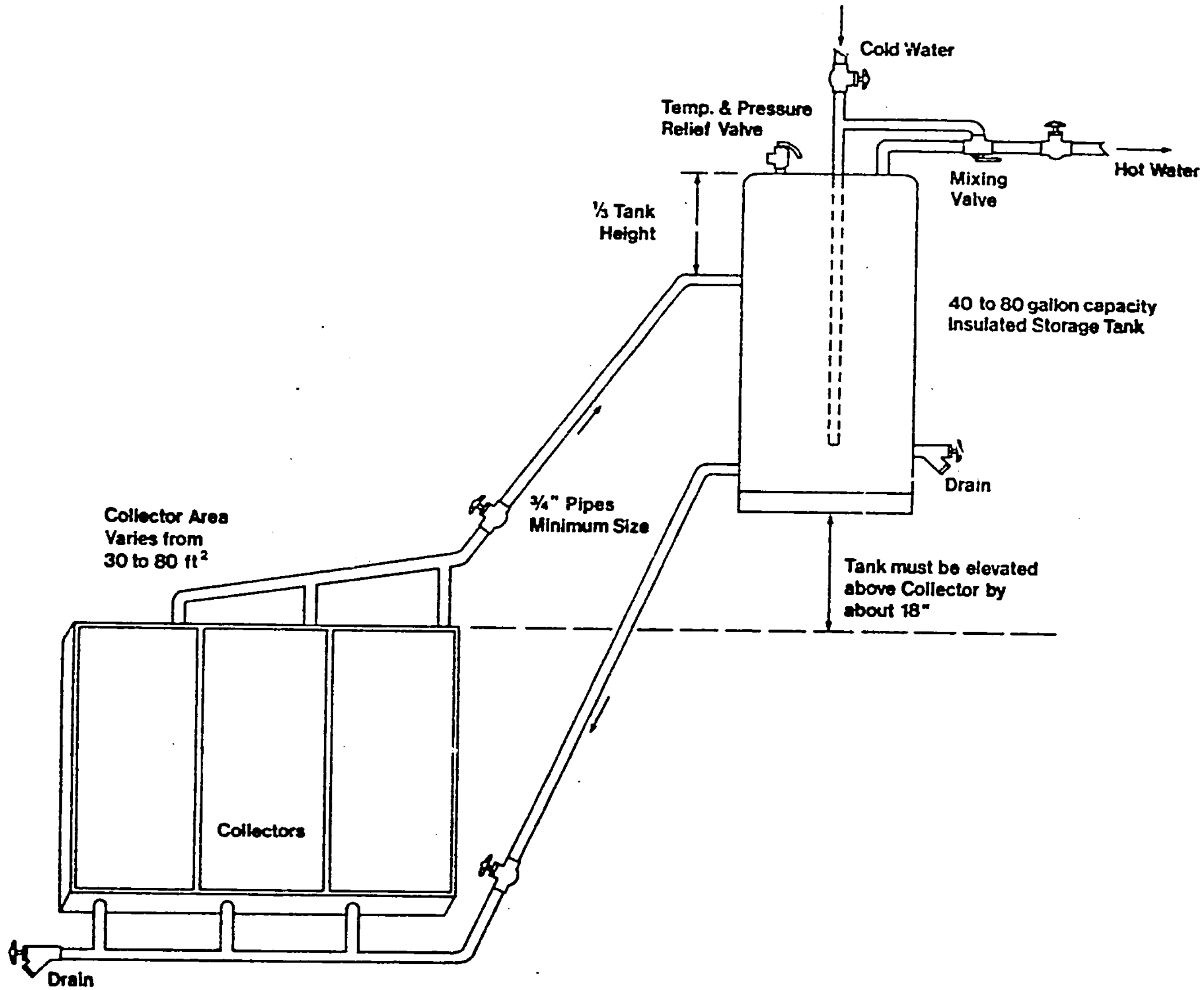


Figure 25. Thermosyphon System

DESIGN PROBLEM

Develop a preliminary design for a solar hot water system for a 100 room hotel. The hotel has a restaurant that serves about 400 meals each day. A local manufacturer can supply flat plate colletors with dimensions 1.25 m by 2.47 m. The effective area of each colletor is 3 m^2. Tests on the collectors indicate they produce a pressure drop of 4 cm water gauge at a flow of 0.06 litres/sec. The system is to be a drain-back system. The collectors are to be located on the roof of the hotel whichis 13 m above ground. There is no basement so storage will be at ground level.

The table below shows typical hot water demands for various types of buildings. Design the system to provide hot water at 60°C.

Hot Water Demands and Use for Various Types of Buildings

Type of Building	*Maximum Hour*	*Maximum Day*	*Average Day*
Men's Dormitories Women's Dormitories	3.8 gal/student 5.0 gal/student	22.0 gal/student 26.5 gal/student	13.1 gal/student 12.3 gal/student
Motels: No. of Units[a] 20 or less 60 100 or More	 6.0 gal/unit 5.0 gal/unit 4.0 gal/unit	 35.0 gal/unit 25.0 gal/unit 15.0 gal/unit	 20.0 gal/unit 14.0 gal/unit 10.0 gal/unit
Nursing Homes	4.5 gal/bed	30.0 gal/bed	18.4 gal/bed
Office Buildings	0.4 gal/person	2.0 gal/person	1.0 gal/person
Food Service Establishments: Type A—Full Meal Restaurants and Cafeterias Type B—Drive-Ins, Grilles, Luncheonettes, Sandwich and Snack Shops	1.5 gal/max meals/hr 0.7 gal/max meals/hr	11.0 gal/max meals/hr 6.0 gal/max meals/hr	2.4 gal/avg meals/day* 0.7 gal/avg meals/day*
Apartment Houses: No. of Apartments 20 or less 50 75 100 130 or more	 12.0 gal/apt. 10.0 gal/apt. 8.5 gal/apt. 7.0 gal/apt. 5.0 gal/apt.	 80.0 gal/apt. 73.0 gal/apt. 66.0 gal/apt. 60.0 gal/apt. 50.0 gal/apt.	 42.0 gal/apt. 40.0 gal/apt. 38.0 gal/apt. 37.0 gal/apt. 35.0 gal/apt.
Elementary Schools	0.6 gal/student	1.5 gal/student	0.6 gal/student*
Junior and Senior High Schools	1.0 gal/student	3.6 gal/student	1.8 gal/student*

* Per day of operation.
[a] Interpolate for intermediate values.

[source: ASHRAE]

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SOLAR THERMAL ELECTRIC SYSTEMS

Solar thermal electric systems include the following:

- Central receiver ("power tower") systems (CRS) which are composed of a field of heliostats (mirrors) which are controlled to reflect incoming direct solar rays to a common absorber (receiver) elevated above the field by a central tower. The energy, in the form of heat, is transferred from the absorber to a working fluid (steam, air, helium, sodium potassium eutectic or salts), which in turn is the source of heat for a thermodynamic cycle (Rankine, Brayton, or combined Rankine/Brayton) to convert the heat into electricity.

- Line concentrators which are fields of distributed (discrete) parabolic concentrating collectors which focus direct insolation upon a line with single axis tracking and an open or cavity receiver or absorber. The heat is transported from the array via the absorber pipeline and is transferred to the working fluid of a Rankine power cycle.

- Point concentrators which are fields of distributed (discrete) paraboloidal concentrating collectors which focus direct sun rays at a point, with dual axis tracking and a cavity receiver (absorber). The heat is transported from the array in one concept via steam, oils, or chemical mixtures to a central Rankine power conversion system. An alternate concept is to use individual power converters (Brayton or Stirling engines) for each collector module, to produce electricity, and then transport electric current to the power conditioning facility, then to the busbar.

Flat plate collector systems could also be used but they are uneconomical when used to produce electricity. There are essentially three thermodynamic cycles that can be used separately or in combination for energy conversion: Rankine, Brayton, and Stirling.

DISTRIBUTED COLLECTOR SYSTEMS (DCS)

1. COOLIDGE

An experimental solar irrigation project sponsored by the U.S. DOE and run by the University of Arizona has been in operation since 1980. It provides electricity to pump water from three 91 m deep wells at Coolidge, Arizona to irrigate cotton crops. The power plant uses 2140 m^2 of parabolic trough concentrating collectors to focus sunlight on receiver tubes within which circulates the primary circuit heat transfer fluid: Caloria HT-43, a synthetic oil stable at high temperatures. The primary fluid vapourizes a low-boiling point secondary fluid (toluene) that drives a Rankine-cycle turbine that generates electricity.

Electrical power is fed into local electric-utility lines, from which power is drawn as needed to pump about 5300 litres per minute from the three wells, each of which requires about 50 KW. Maximum power output is rated as 175 KW of which approximately 25 KW is for power plant pumps and motors. Figure 1 shows schematically the elements of the system. The Acurex collectors raise the temperature of the Caloria HT-43 oil to around 290°C. The oil is then circulated through a 30,000 gallon (114 m^3) thermal storage tank. A disadvantage with this particular thermodynamic cycle is that the pressure in the condenser is below atmospheric, thus raising the possibiity of air leaking into the system. Toluene forms explosive mixtures with air at low concentrations.

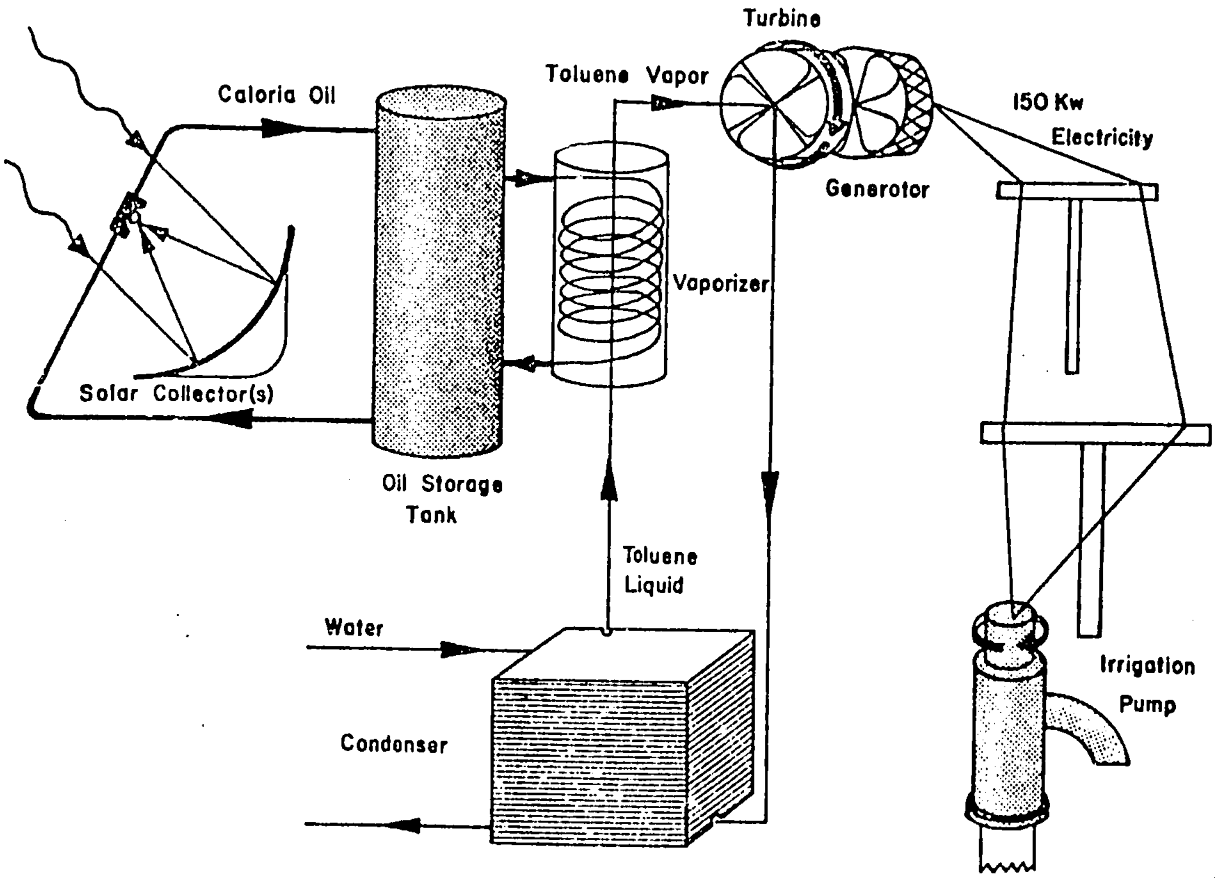


Figure 1. Diagram of Coolidge Pumping System

2. ALMERIA

As another example of a DCS system we look at the Small Solar Power Systems Project (SSPS) initiated by the International Energy Agency (IEA). Supported and funded by eight European countries and the U.S., the SSPS is part of the IEA Research and Development program which is aimed at applying and demonstrating those new or improved energy technologies that offer significant potential for contributing to future energy needs.

The principal objective of the SSPS project is to examine in some detail the feasibility of using solar radiation to generate electric power for possible application either in established grids or in communities whose geographical situation renders conventional electrical supply techniques difficult and costly. Evaluation is to be performed with respect to two dissimilar engineering approaches. A solar farm or DCS using parabolic trough collectors is to be located adjacent to a central receiver system (CRS) using a field of heliostats.

The technical and operational objectives are to compare both technological concepts, based on the same design philosophy and operated under the same environmental conditions. The SSPS-DCS plant, which has a rated output of 500 KWe, utilizes the pilot-system experience of Acurex in building irrigation plants in New Mexico and Arizona, as well as of the German company M.A.N. in operating similar systems in Spain, Mexico, and Australia. The plant has two collector fields of approximately equal size (see Table 1). One field is made up of 10 loops of 60 collectors manufactured by Acurex; the other field consists of 14 loops of 6 collector modules developed by M.A.N. Both of these collector designs are line-focusing parabolic trough types.

The Acurex collector is arranged to track the sun in a single-axis mode, the rotational axis being oriented in the east-west direction. The M.A.N. collector modules employ two-axis tracking for orientation in azimuth and elevation. Application of the two design concepts in the same location offers the opportunity to compare life-cycle costs versus annual energy output under realistic conditions.

The heat transfer and power conversion systems of the DCS have been designed with three heat transfer loops.

1) The first loop takes low temperature oil, Caloria HT-43 at 225°C, from the bottom of a thermal storage tank, circulates it through the collector fields, and returns it at a temperature of 295°C to the top of the storage tank.

2) In a second loop, a boiler takes the hot oil from the storage tank, discharges the thermal energy to the steam loop, and returns the oil to the thermal storage tank.

3. The third loop circulates water through the boiler and then expands the generated steam through a turbine generating electricity. The low-enthalpy steam is condensed and pumped back to the boiler.
Figure 2 shows a simplified diagram of the DCS process flow.

Table 1. SSPS Distributed Collector System—Performance Data

Design point:	day 80, 12:00 (equinox noon) solar insolation	0,92 kW/m²
Collector fields:	ACUREX collector, model 3001 60 groups in 10 loops	2674 m²
	MAN collector, model 3/32, "HELIOMAN" 84 modules in 14 loops	2688 m²
	total aperture area	5362 m²
	concentration ratio	ca. 40
	land-use-factor (ACUREX/MAN)	0,27/0,32
	heat transfer medium	thermal-oil (HT-43)
	collector inlet temperature	225°C
	collector outlet temperature	295°C
Thermal storage:	one-tank-thermocline, storage medium	thermal-oil (HT-43)
	capacity equivalent to	0,8 MWh_e
	hot/cold temperature	295°C/225°C
Steam generator:	HT-43 inlet temperature	295°C
	HT-43 outlet temperature	225°C
	steam outlet temperature	285°C
	steam pressure	25 bar
Power (at design point):	solar insolation	4933 kW
	thermal	2580 kW
	gross electric	577 kW
	net electric	500 kW
Efficiencies (at design point):	thermal/gross electric	22,4%
	thermal/net electric	19,4%
	insolation/net electric	10,1%

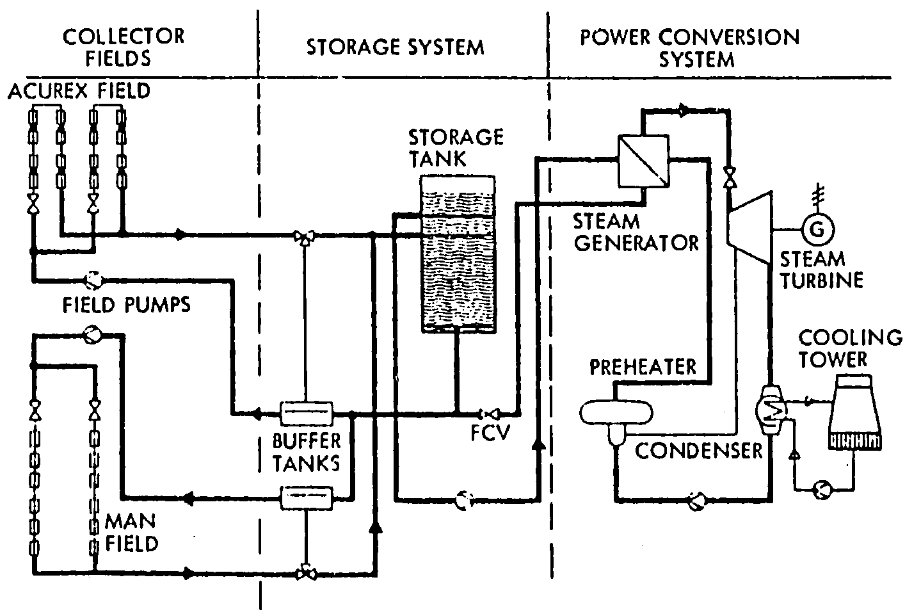


Fig. 2. *Simplified schematic diagram of DCS process flow*

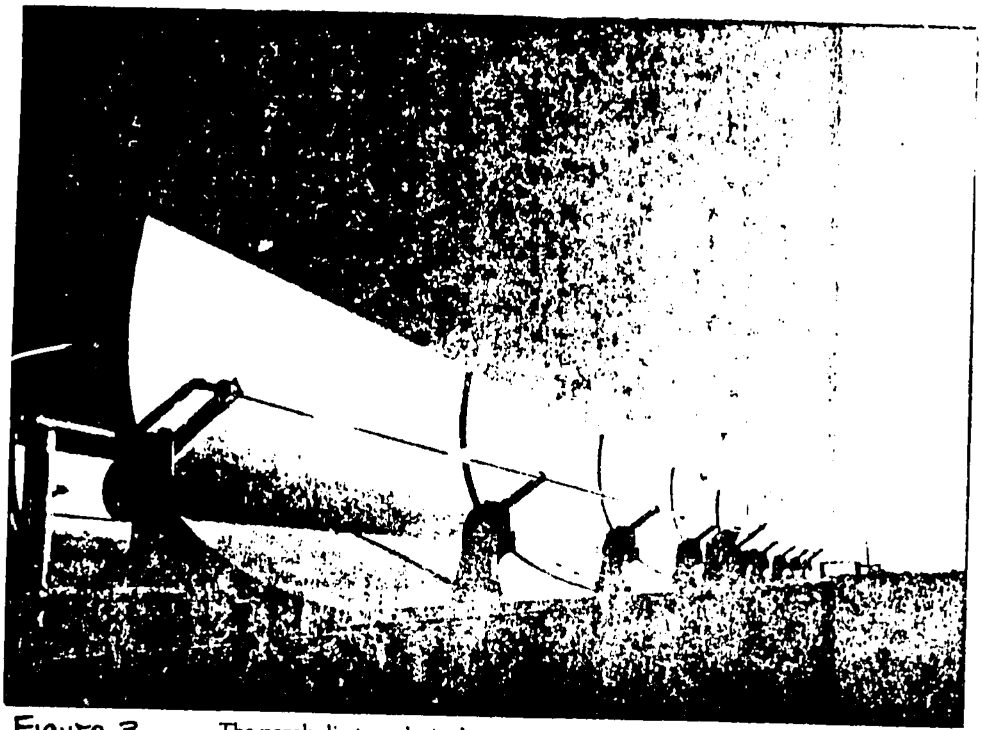


Figure 3. The parabolic-trough single-axis tracking collector by Acurex.

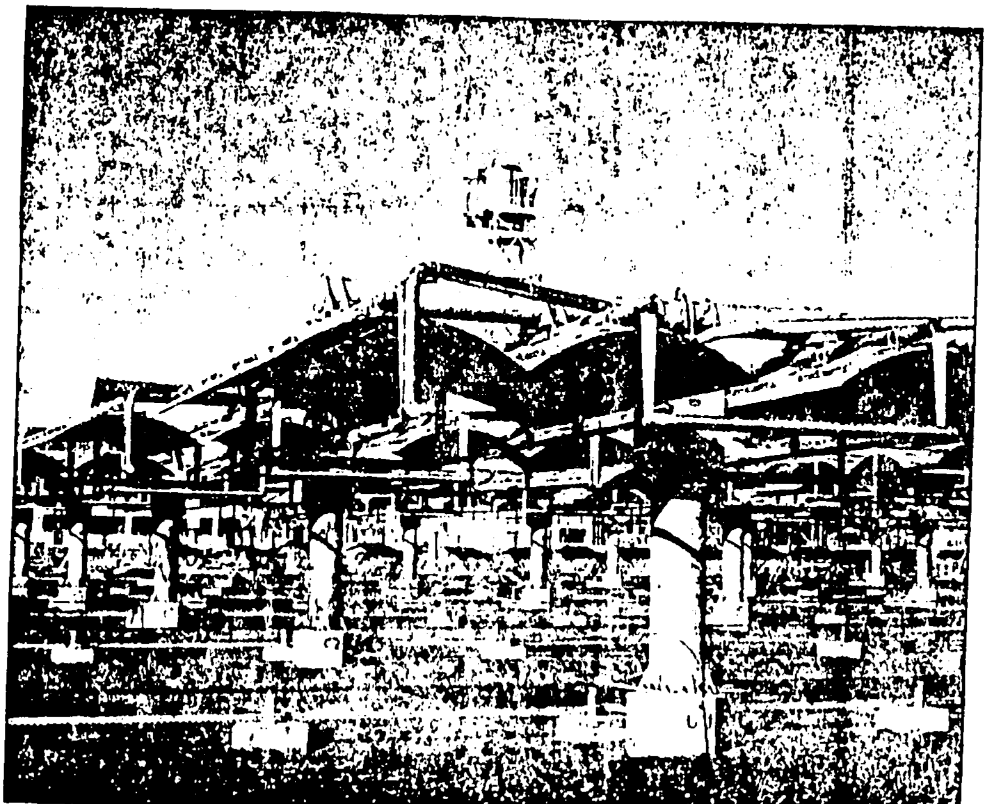


Fig 4. The two-axis tracking concentrator by M.A.N., in stow position with faces down.

CENTRAL RECEIVER SYSTEMS

Eight central receiver system experiments and pilot plants are now in operation or under construction throughout the world, each with the output power of one megawatt or more of thermal energy. Two of these systems are now operating in the United States and France, and six more--located in the United States, France, Italy, Japan, and Spain--are under construction. All told, they represent an investment of at least $250 million.

One of the first relatively large systems, a 1 MWt solar furnace constructed by the French at Odeillo in the Pyreness Mountains, was converted in the late 1970's to generate electricity for demonstration purposes. In this application, the thermal power was removed from the receiver by means of a hot-oil heat-transfer loop to thermal storage, or directly to an oil-to-steam heat exchanger to operate a steam turbine coupled to an electric generator.

As a solar furnace, the Odeillo plant develops temperatures up to 3,000°C without the need for direct flame-firing of test specimens or use of heat-exchanger enclosures. Sixty-three heliostat mirrors, controlled by computers, reflect the sun's radiation onto a parabolic mirror which in turn concentrates the radiation on the fixed-focus area.

The Central Receiver Test Facility (CRTF) installed in 1977 at Sandia National Laboratories in New Mexico is a test bed for components and subsystems for the Barstow, California, pilot electric plant. Its sophisticated tower contains three test bays served by an elevator. The field consists of 222 heliostats which can focus five megawatts of thermal power into a test bay.

The 10MWe Barstow steam plant now under construction will be connected to Southern California Edison utility grid, and is expected to serve the needs of a community of 6,000. Its storage system will be designed to provide 7MWe for four hours.

A two-megawatt electric plant is under construction at Targasonne, near Odeillo. Two towers will allow the testing of one receiver subsystem while another subsystem is being installed or modified. Molten salt will be used as the heat-transfer fluid and also as the thermal-storage material.

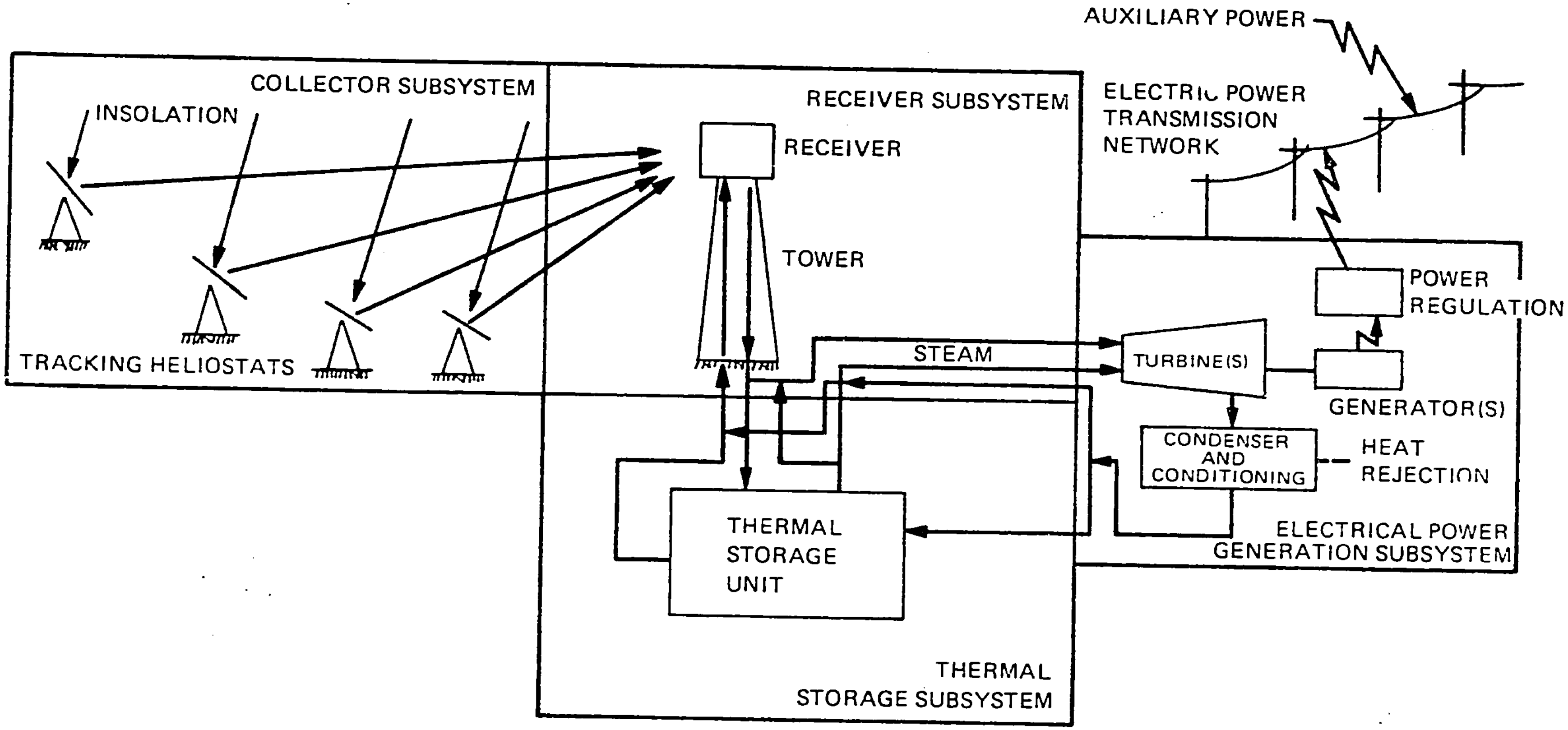
A third Spanish plant, a 1MWe facility to be built at the Almeria site, is receiving assistance from the United States in the use of newly-developed design methodologies and computer programs. All three plants are expected to be operational by 1982.

Under Japan's Project Sunshine program, two pilot plants with different design approaches are under construction at Nio, Kagawa Prefecture, on Shikoku, one of Japan's major southern island. Capable of producing 1,000 kWe each, the two plants are now operational.

Figure 5 shows the basic components of a central receiver system.

FIGURE 5.

CENTRAL RECEIVER SOLAR THERMAL POWER SYSTEM



Source: ERDA, Central Receiver Solar Thermal Power System — Phase I, 10 MWe Pilot Plant, Washington, D. C., 1976.

1. ALMERIA

The SPSS - CRS plant has a rated output of 500 kWe. Solar radiation is concentrated about 450 times by a heliostat field with approximately 4000 m^2 of reflective surface. The Martin Marietta first-generation heliostats track the sun both in asimuth and elevation, with a maximum pointing error of about 2 mrad whenever the wind speed is less than 13 km/h. The field is designed to survive wind speeds of up to 144 km/h, seismic activities of 0.6 m/s^2, and the impact of 20 mm hail at 20 m/s. Additional performance data is indicated below in Table 2.

Table 2. SSPS Central Receiver System—Performance Data

Design point:	day 80, 12:00 (equinox noon) solar insolation	 0,92 kW/m^2
Heliostat field:	total reflective surface area concentration ratio land-use-factor	4000 m^2 450 0,22
Cavity receiver:	heat transfer medium aperture size active heat transfer surface inlet temperature outlet temperature	Sodium 9 m^2 16,9 m^2 270°C 530°C
Thermal storage:	two-tank-system, storage medium capacity equivalent to hot storage temperature cold storage temperature	Sodium 1,0 MWh_e 530°C 275°C
Steam generator:	sodium inlet temperature sodium outlet temperature steam outlet temperature steam pressure	525°C 275°C 510°C 100 bar
Power (at design point):	solar insolation thermal gross electric net electric	3675 kW 2283 kW 600 kW 517 kW
Efficiencies (at design point):	thermal/gross electric thermal/net electric insolation/net electric	26,3% 22,6% 14,1%

Transfer of thermal energy in the sodium cooled system is performed at high temperature (530°C) and low pressure (4 bar). The incoming energy (2.7 MWt at the design point), which produces peak fluxes on the tube bundle of the receiver of 0.63 MW/m^2, is passed through a storage system to the boiler. The third loop generates steam and delivers it to the turbines at 510°C and 100 bar.

The German designed cavity-type receiver has a vertical octagonal aperture of 9.7 m^2. Sodium flows in six horizontal parallel tubes which wind back and forth from the bottom to the top of the cavity. Sodium enters the inlet header at 270°C at the bottom of the panel and leaves the outlet header at 530°C near the top. The receiver is mounted on top of a 43 m high steel tower with a concrete foundation.

A cold sodium vessel and a hot sodium vessel, each having a volume of 70 m^3, provide storage for the CRS. Sodium enters the hot storage vessel from the receiver at 530°C, is pumped to the helical-tube steam generator, then is returned to the cold sodium vessel at 275°C. The power conversion unit is a steam-driven five-piston motor coupled to a three-phase generator. The operating conditions of this unit are indicated below.

Thermal input (steam)	2200 kWt
Inlet pressure	100 - 102 bar
Inlet temperature	500 - 520°C
Back pressure	0.3 bar
Speed	1000 rpm
Motor	845 Hp
Gross output	600 kWe
Net output	562 kWe
Efficiency (gross/thermal)	27.3 %
Efficiency (net/thermal)	25.5 %

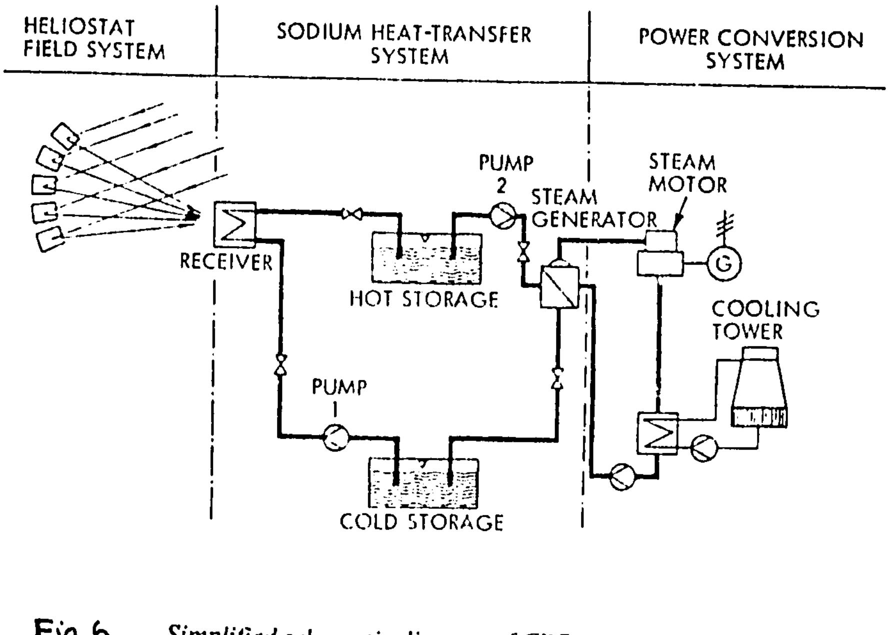
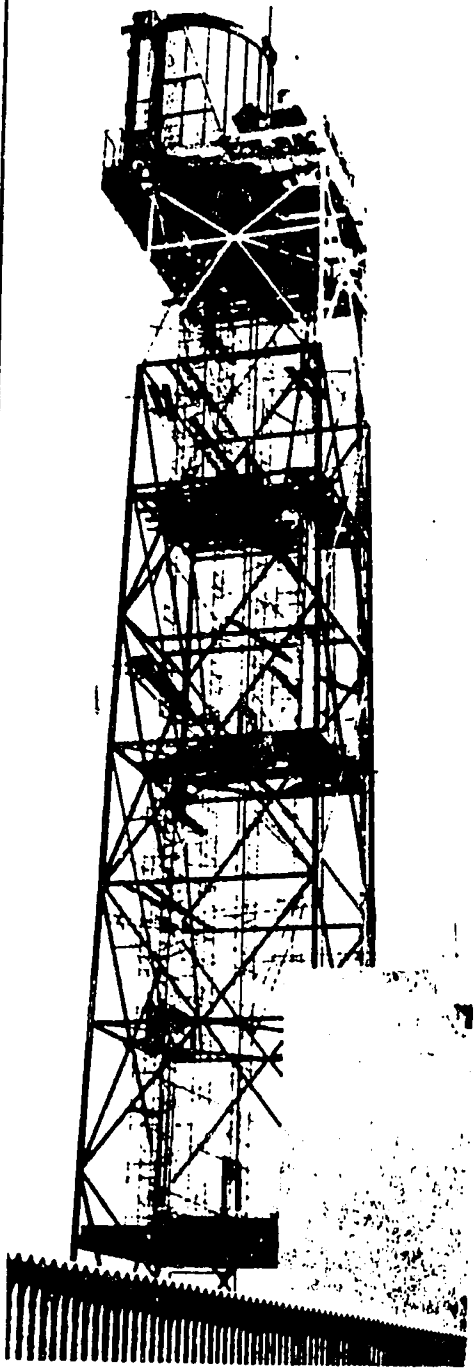
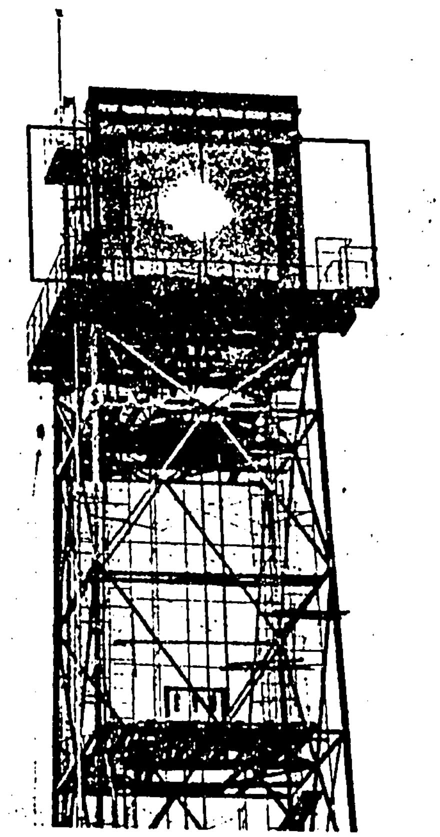


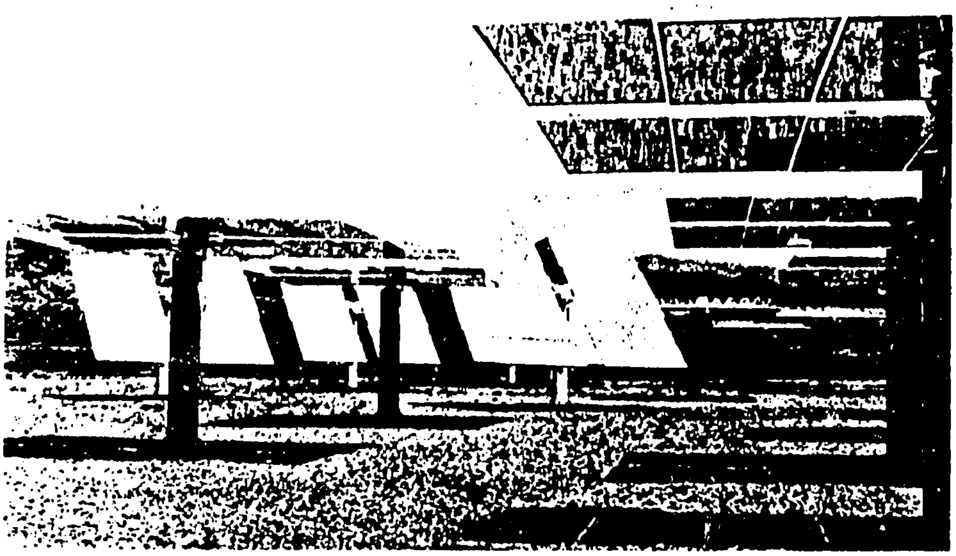
Fig 6. *Simplified schematic diagram of CRS process flow*



The 43-meter-high receiver tower as seen from the ground, with a view of the back of the sodium receiver.



The cavity-type sodium receiver of the CRS is shown mounted on the receiver tower. In this photograph, the receiver doors are closed, showing a single heliostat image.



Closeup of the CRS heliostat field shows the Martin Marietta first-generation-type heliostats, some of which are in the stowed position with the reflecting surface facing downward (foreground).

2. BARSTOW

Construction of a CRS pilot plant capable of generating 10 MWe is in the process of being completed near Barstow, California. This project is the first of its kind in the U.S. and will be a pilot operation for judging the feasibility of central receiver systems.

Seven major systems are involved in total plant operation: the collector, receiver, thermal storage, master control, plant support, beam characterization, and electric power generating systems. (The first six of these make up the solar facility.) The heliostats of the collector system reflect solar energy onto the receiver mounted on a 90.8 m (298 ft) tower. In the receiver, water is boiled and converted to high-pressure steam (516°C and 10.3 MPa; 960°F and 1465 psia), which is then converted to electrical energy by the turbine/generator. Any steam from the receiver in excess of the energy required (35.7 MWt) for the generation of 10 MWe net power to the utility grid is diverted to thermal storage for use when output from the receiver is under that needed for rated electrical power.

When the turbine operates directly on steam from the receiver, the pilot plant's rated output is 10 MWe plus 1.8 MWe parasitic loads (internal plant loads). When operating from the thermal storage system alone (274°C and 2.7 MPa; 525°F and 385 psia), the net electrical output is 7 MWe. Overall efficiency of the system ranges from 13.5% (full insolation day) to 11.1 % (full energy storage operation).

Collector System

The collector field, consisting of 1818 Martin Marietta sun-tracking heliostats, has a total reflecting area of 72,538 m^2 (781,740 ft^2) and is divided into four quadrants. Each heliostat is made of 12 slightly concave mirror panels totaling 40 m^2 (430 ft^2) of mirrored surface that focus the sun's rays on the receiver. The mirror assembly is mounted on a geared drive unit for azimuth and elevation control.

There are a total of 1240 heliostats in the two northern quadrants and 578 heliostats in the two southern quadrants. In the southern quadrants, the heliostats are focused on each of the 6 preheat panels under optimum conditions. In the northern quadrants, the heliostats are focused on each of the 18 boiler panels so that the heat is distributed over the length of the panels.

The collector control subsystem consists of a micro-processor in each heliostat, a heliostat field controller for groups up to 32 heliostats, and a central computer called the heliostat array controller. The annual and diurnal sun position information for pointing each heliostat are stored within this control subsystem. The heliostats can be controlled individually or in groups in either manual or automatic modes. The heliostat array controller is located in the plant control room and is functionally tied into the master control system. The plant operator can control the collector field through either the heliostat array controller or the master control system.

The heliostats are designed to operate in winds up to 36 mph and will be stowed in a mirror-down position in higher winds. Design specifications include survivability in a stowed position in winds up to 90 mph. Several heliostats have satisfactorily passed tests in which wind-induced structural loads were simulated.

Receiver System

The receiver system consists of a single-pass to superheat boiler with external tubing, a tower, pumps, piping, wiring, and controls necessary to provide the required amount of steam to the turbine. Steam demand can be varied from the control room by the operator, or the receiver system can react to a demand from the electric power generating system up to the receiver's rated output.

The receiver is designed to produce 516°C (960°F) steam at 10.3 MPa (1465 psia) at a flow rate of 112,140 lb/h. The receiver has 24 panels (6 preheat and 18 boiler), each approximately 0.9 m (3 ft) wide and 13.7 m (45 ft) long. The panels are arranged in a cylindrical configuration with a total surface area of 330 m^2 (3252 ft^2). Each panel consists of seventy Incoloy 800 tubes through which water is pumped and boiled. The external surface temperature of the receiver tubes at rated output will be approximately 621°C (1150°F). Each receiver tube is 0.69 cm (0.27 in.) inside diameter and 1.27 cm (0.5 in.) outside diameter. These boiler tubes are made with thick walls and special metal in order to withstand the effects of diurnal cycling, wich can cause premature metal fatigue. In contrast to a solar boiler, conventional boilers are kept heated even when steam and/or electrical demand is low. In a solar receiver, the heat source disappears when the sun is obscured or not shining, and the boiler cools. When insolation returns, the boiler is reheated.

Within each panel, all tubes are welded to the adjacent tubes for their full length on the outside surface only. The receiver panel exterior is painted with a special black paint ("Pyromark") to increase thermal energy absorption. The interior surface of the receiver panels is insulated.

The tower, holding the receiver 90.8 m (298 ft) above the desert floor, has a 7.6 m (25 ft) deep footing and a 1500 ton concrete base. The tower is equipped with a temporary crane for installation of the receiver panels. The wide area of the tower beneath the receiver houses air-conditioned rooms where the receiver computer controls and some of the beam characterization system are located.

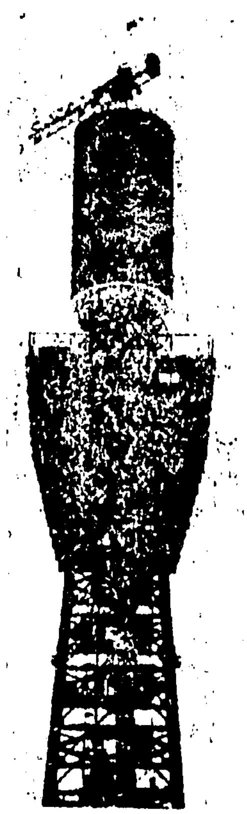
Thermal Storage System

The thermal storage system provides for storage of thermal energy to extend the plant's electrical power generating capability into nighttime or during periods of cloud cover. It also provides steam for keeping selected portions of the plant warm during non-operating hours and for starting up the plant the following day. Sealing steam is required in the turbine casing even when it is not running. Even though the primary source for this turbine sealing steam is thermal storage, a small auxiliary electric boiler is standing by in case the thermal storage system is depleted or not operating.

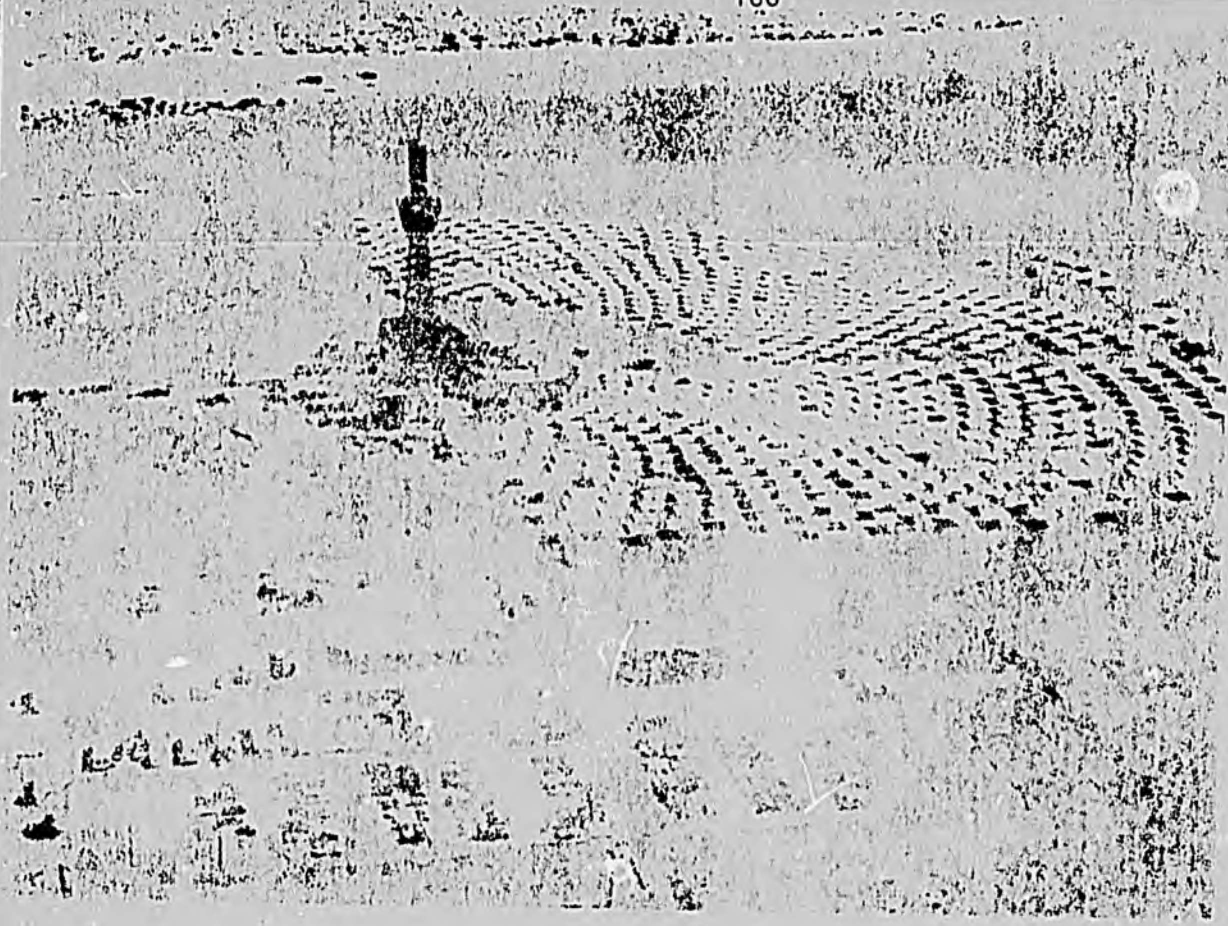
The storage tank is 13.7 m (45 ft) high, 19.8 m (65 ft) in diameter (inner), and built on a special lightweight, insulating concrete for reducing heat loss to the ground. The walls are made of steel and 30.5 cm (1 ft) of insulation and the roof is aluminum plus 61 cm (2 ft) of insulation. The 3581 m^3 (946,000 gal) capacity tank, filled with rock, sand and about 908 m^3 (240,000 gal) of thermal oil (Caloria HT 43), acts as a heat storage vessel or unit.

Desuperheated steam from the receiver is routed through dual heat exchangers in which thermal storage oil is heated. The heated oil is pumped back into the tank and thermal energy is transferred to the rock and sand. When fully charged, the temperature of the thermal storage mixture (oil, rock, and sand) will be approximately 302°C (575°F). When discharging, the heated oil is pumped through another heat exchanger to boil water. Steam at 274°C (525°F) and 2.7 MPa (385 psia) can be delivered to the turbine at a rate of 105,000 lb/h. The rated electrical capacity of the plant operating on thermal storage energy is 28 megawatt-hours (28 MWe-h) net output, i.e., 7 MWe power for 4 hours. After discharging, sufficient thermal energy will be available for heating, sealing steam, and restarting the plant the next day.

As do other plant systems, the thermal storage system has its own controls and also can be controlled both manually and automatically through the master control system.

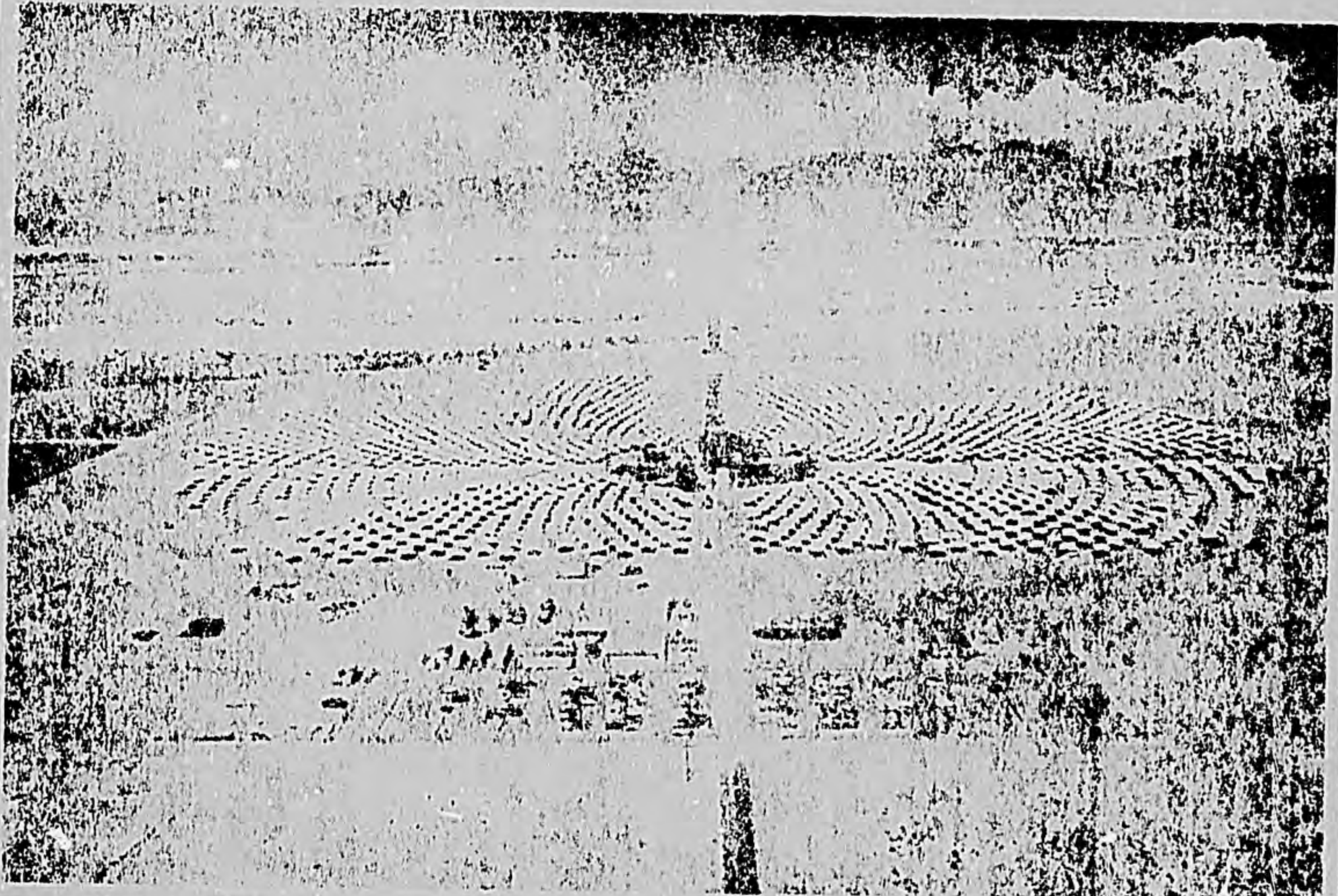


View of the tower with the boiler receiver mounted at its apex. A man is standing beneath and to the left of the receiver behind the railing.



June 1981

September 1981



The capital cost of the Barstow 10 MWe CRS plant is estimated at 141 M$. Annual operating and maintenance costs are estimated as 3.7 M$. Construction took 5 years. These data permit us to make an estimation of the cost of the electricity that will be produced by the plant.

The capital cost of 141 M$ does not include interest charges on the loan. Construction costs are 141/5 = 28.2 M$/yr for 5 years. We assume interest is charged at 15% annually. The total capital cost, including capital charges, is therefore given by:

$$I = \frac{(1 + 0.15)^5 - 1}{0.15} \times 28.2 = 190.14 \text{ M\$}$$

The capital recovery factor (CRF) for this debt, based on 15% interest rate and a 40 yr lifetime, would be:

$$\text{CRF } (0.15, 40) = \frac{0.15}{1 - (1 + 0.15)^{-40}} = 0.15056$$

So the amount paid annually in capital charges is 190.14 x 0.15056 = 28.63 M$/yr.

Operation and maintenance costs are 3.7 M$/yr, so total annual costs may be estimated as 32.33 M$/yr.

How much electricity will the plant produce? This is very difficult to estimate at this stage in the development and demonstration of CRS plants.

Data for California suggest that the direct insolation is about 3200 kWh/m^2 per year. Using this figure and the total heliostat area of 72,538 m^2, the gross insolation is about 232.12 x 10^6 kWh/yr.

The efficiency of the Almeria CRS plant is estimated as about 14% (insolation kWh to net electric kWhe). So an approximate estimate of net electric output for the Barstow 10 MWe plant would be:

$$0.14 \times 232.12 \times 10^6 = 32.5 \times 10^6 \text{ kWhe/yr}$$

Assuming that all routine maintenance is performed at night, so that the daytime plant factor is 100%, the cost of electricity produced by the system is approximately:

$$\frac{32.33 \times 10^6 \text{ \$/yr}}{32.5 \times 10^6 \text{ kWhe/yr}} = 0.995 \text{ \$/kWhe}$$

or very nearly $1 per kWhe.

Wind Power

Wind power technology is one of the very oldest of the renewable energy technologies. The first windmills are mentioned at the beginning of the Moslem era, in the second half of the seventh century. The earliest systems were primitive devices consisting of a vertical rotating shaft turning a millstone. Attached to the shaft were simple wings or paddles probably constructed of woven matting. The device was positioned inside a circular wall with a large opening facing the prevailig wind in such a way that the wind impinged on one side of the windmill blades. Mills of this type (called panemones) were certainly constructed on the plains of Persia which are swept by steady winds.

In Asia and China, in the tenth century, windmills came into use for irrigation and drainage. By the thirteenth century the technology was in evidence across Europe from Portugal to Holland and beyond. At this time the windmill in Europe was used almost exclusively for the grinding and milling of grain (hence its name). Only in Holland, at a later time, was the technology used to pump water - an application in which there remains considerable interest, particularly in the developing countries.

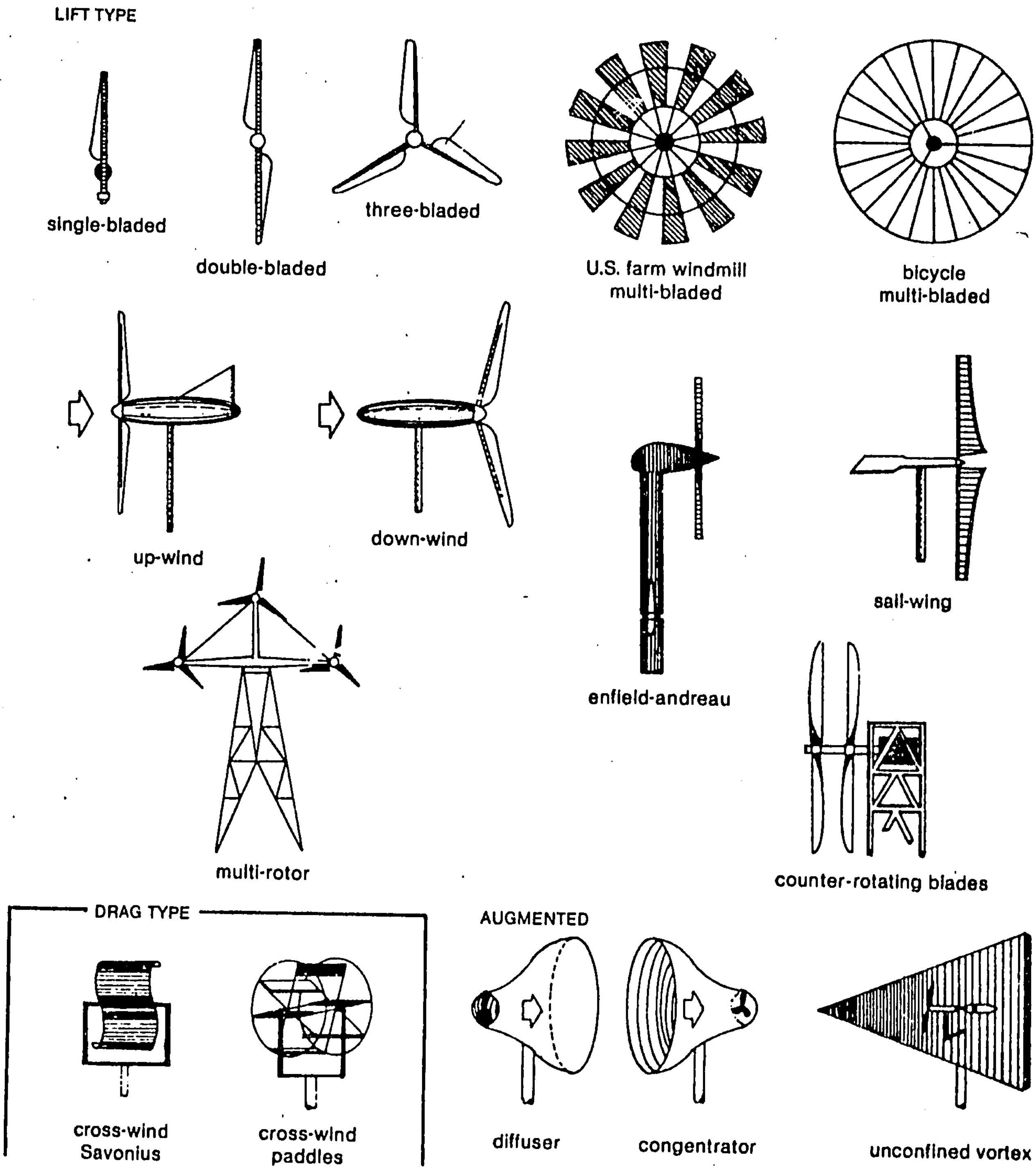
The earliest windmills in the New World were modeled on the European machines. But by the middle of the nineteenth century Daniel Halliday had begun to experiment with the design that developed into the familiar multi-bladed water-pumping machine still to be seen across the American plains. More than six million small multibladed windmills have been built and used in the United States to pump water. It is estimated that over 150,000 are still in operation.

During the 1930's small wind machines that generated electricity came onto the market. Between 1930 and 1960, thousands of wind-powered electric generators were sold and installed in many countries. Production faltered in the 1960's after the Rural Electrification Administration succeeded in supplying most American farms and rural homes with inexpensive electricity from a central power station and transmission system.

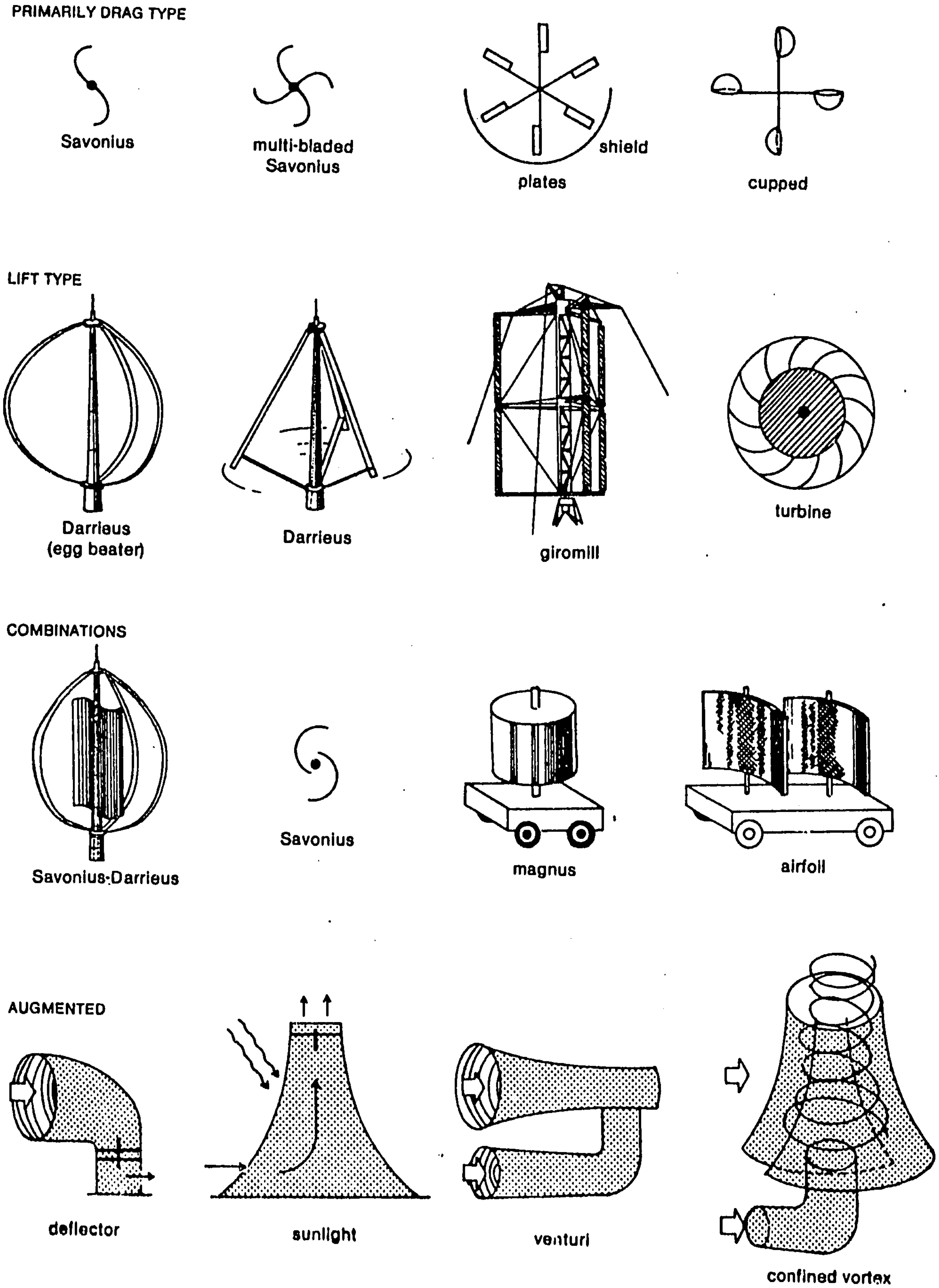
Many of the old wind electric systems are now being renovated and pressed back into service as escalating oil prices spark a renewed interest in this technology.

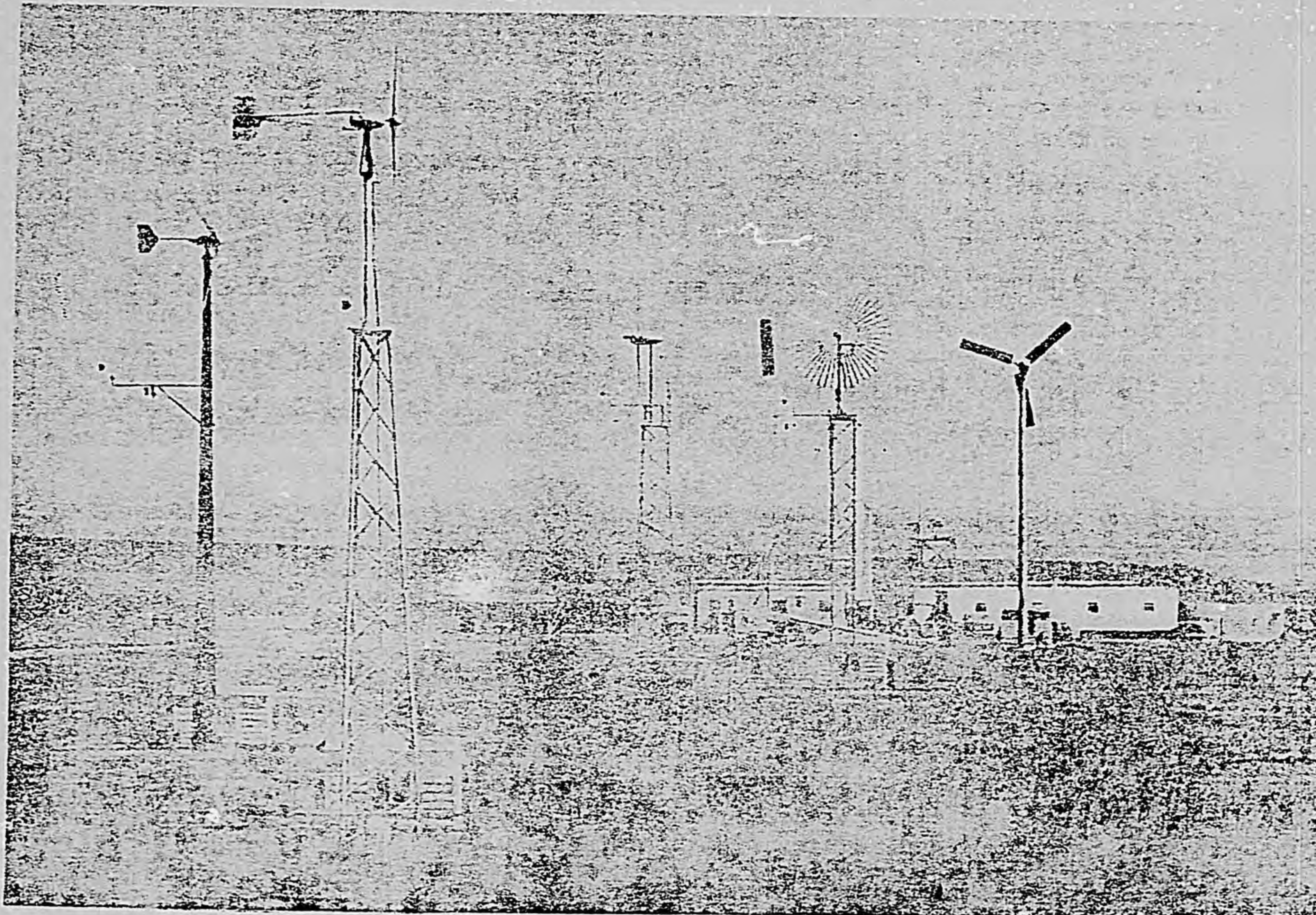
Wind energy conversion systems (WECS) come in a wide variety of shapes, sizes, and configurations. Many of the different types of wind machines are shown diagrammatically overleaf.

HORIZONTAL AXIS WIND MACHINES



VERTICAL AXIS WIND MACHINES





Rocky Flats Wind Energy Test Site

Basic Princples

Wind is a fluid in motion, its energy is kinetic. The power in the wind, P_w, is therefore given by

$$P_w = 1/2\ \dot{m}\ V^2 \qquad \text{Watts} \qquad (1)$$

where
$\dot{m}$ = air mass flow rate, kg/s
V = air speed, m/s

This expression is usually rewritten as

$$P_w = 1/2\ \rho\ AV^3 \qquad \text{Watts} \qquad (2)$$

where
ρ = air density, kg/m^3
A = flow area, m^2

The density of air varies with both temperature and pressure. At 20°C and atmospheric pressure the density of air is 1.2 kg/m^3. Since the air pressure decreases with increasing altitude the temperature corrected density should also be corrected for altitude. The density of dry air at atmospheric pressure is tabulated below.

Density of Dry Air at Atmospheric Pressure

Temperature °C	Density kg/m^3	Temperature °C	Density kg/m^3
0	1.29	20	1.20
5	1.27	25	1.18
10	1.25	30	1.16
15	1.23	35	1.15

The correction factor for variation in altitude, C_A, is shown below. This factor should be multiplied by the density at atmospheric pressure to determine the correct air density.

Altitude Correction Factor, C_A

Altitude, ft	C_A	Altitude, ft	C_A
0	1	5000	0.832
1000	0.964	6000	0.801
2000	0.930	7000	0.772
2500	0.912	8000	0.743
3000	0.896	9000	0.715
4000	0.864	10000	0.688

It can be shown that an ideal wind turbine can extract a maximum of exactly 59.3% of this potential power. In practice, wind energy conversion systems (WECS) can convert at most about 45% of the power in wind. The efficiency of a wind system is often called the Coefficient of Performance.

It is important to note that the available wind power is proportional to the cube of the wind speed. If the wind speed doubles the available power increases by a factor of eight. Small differences in average wind speed between potential sites can therefore make a great deal of difference in terms of power generation.

Wind system performance is often correlated in terms of a dimensionless parameter called the Tip Speed Ratio, TSR. The tip speed ratio is the ratio of the speed at which the blade tip is travelling to the speed of the wind.

$$TSR = \frac{2\pi r N}{V} \tag{3}$$

where r = radius of the swept area, m
N = revolutions per second
V = wind speed, m/s

The efficiency of a wind turbine (the fraction of the power in the wind that is extracted by the turbine) varies with tip speed ratio. The efficiency curve shows a well-defined peak for each type of machine. Figure 1 shows the efficiency of the different types of wind systems as a function of tip speed ratio. The American farm multiblade windmill is an example of a machine which has low efficiency and turns no faster than the wind. One advantage of this design is its high torque at low speeds: essential for pumping water. The Dutch four-arm windmill is a lift-type that shows poor performance because of the crude design of its sail-like rotor. However, it also produces high torque at low wind speeds. The Savonius rotor, which looks like a vertical cylinder with the two halves displaced in opposite directions, is cheap and simple to fabricate but relatively inefficient. Modern high performance wind machines have blades with carefully designed airfoil sections and perform best when they are moving faster than the wind. In general, a low tip speed ratio means greater torque (suitable for pumping water and mechanical work), and a high tip speed ratio means lower torque but higher rotational speed (suitable for generating electrical power). These characteristics are related to the solidity of the rotor. Solidity is the ratio of total blade area to the swept area of the rotor. Figures 2 and 3 show how torque and solidity vary with tip speed ratio.

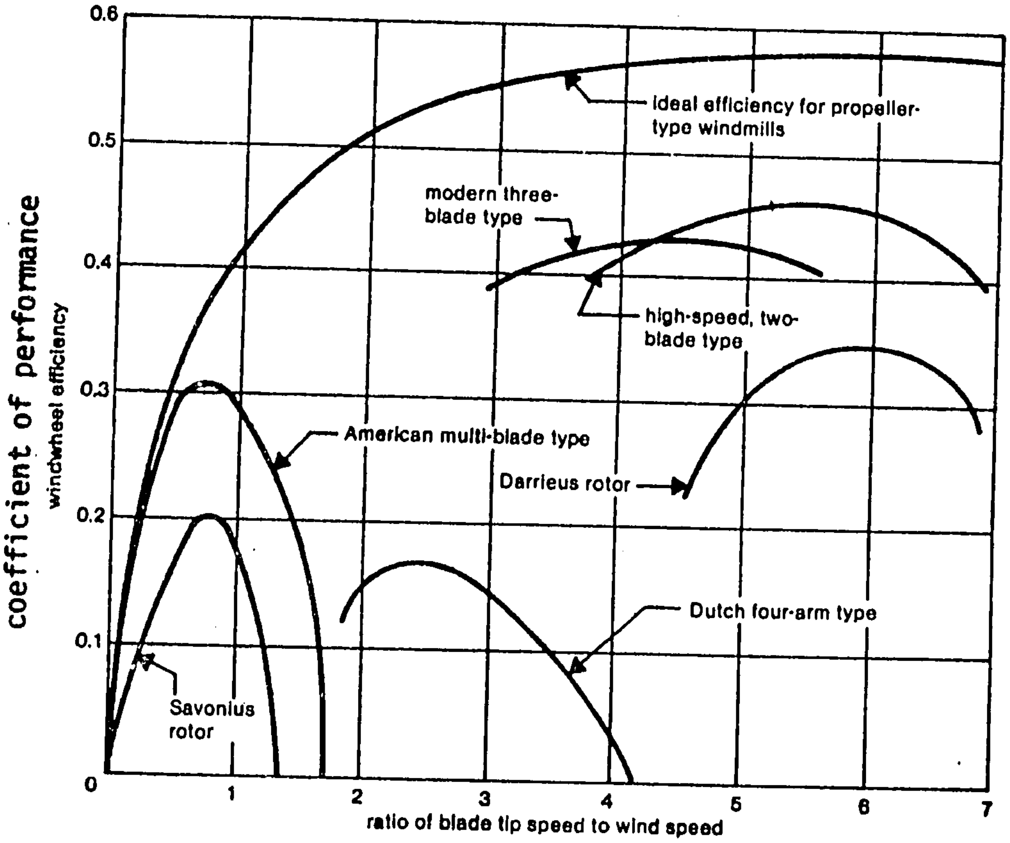


Figure 1 Typical performance of several wind machines.

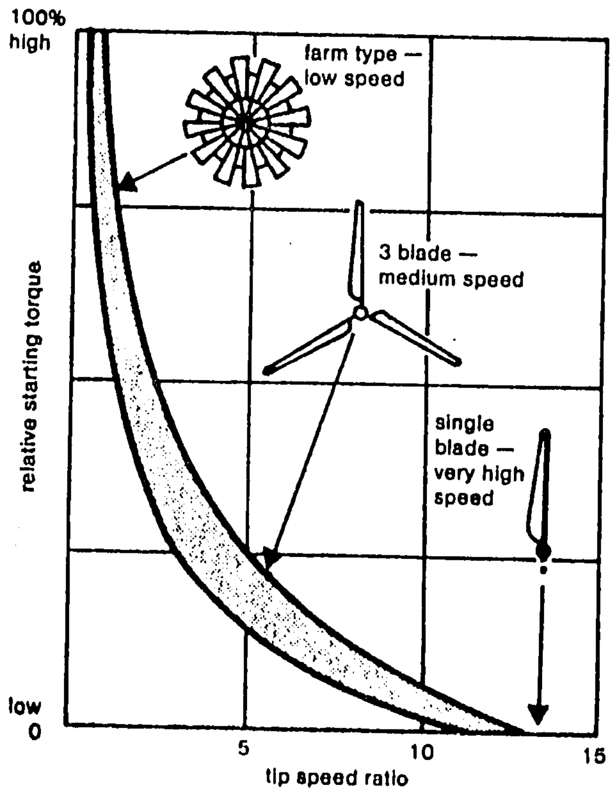


Figure 2 Relative starting torque.

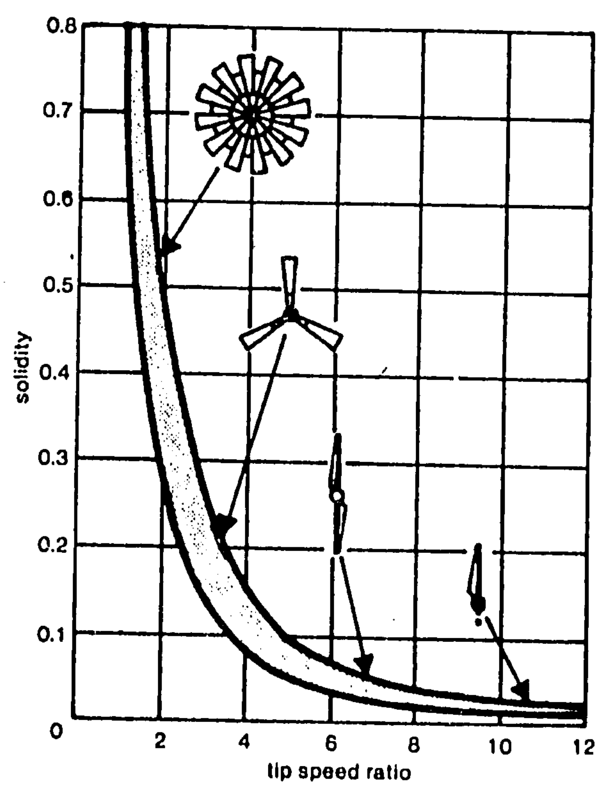


Figure 3 Solidity of several wind machines.

Example 1

A high speed two-blade rotor has a blade length of 3 metres. What is the maximum power the rotor will produce in 10 m/s wind and what is the rotational speed at this power output?

Solution:

From Figure 1, for a two-blade rotor, the maximum efficiency occurs at a tip speed ratio of about 5.5. At this point the efficiency is about 0.47. The power in the wind is

$$P_W = 1/2\ \rho A V^3 \qquad \text{Watts}$$

assuming

$$\rho = 1.2\ kg/m^3$$
$$A = 28.3\ m^2$$
$$V = 10\ m/s$$

therefore

$$P_W = 0.5 \times 1.2 \times 28.3 \times 10^3 = 17000 \qquad \text{Watts}$$

The power produced by the rotor is 0.47 X 17000 = 8 kW.

The optimal tip speed ratio is 5.5

since

$$TSR = \frac{2\pi r N}{V}$$

$$N = \frac{5.5 \times 10}{2\pi \times 3} = 2.92\ rev/s$$

$$= \underline{175\ rpm}$$

Wind Machine Characteristics

All wind turbines exhibit certain fundamental characteristics which are related to wind speed. At low wind speeds the rotor does not turn until the wind reaches a value called the cut-in wind speed. Since the energy in the wind is proportional to the cube of the speed, there is very little energy in the wind at wind speeds below the cut-in speeds. As the wind speed increases the rotor generates power at a level proportional to the cube of the wind speed until, at the rated wind speed, the wind system produces its rated output. At wind speeds above the rated wind speed most wind turbines produce approximately constant power by the employment of some kind of governor, until at high wind speeds, the wind system is designed to cut-out or furl. The wind speed at which the system is designed to shut itself down is called the furling windspeed. These characteristics are illustrated in Figure 4 which shows the power produced by two different wind machines as a function of windspeed. Machine A is rated a 2 kW machine at a wind speed of 25 mph; machine B is rated at 1 kW machine at a wind speed of 15 mph. Both machines have a cut-in speed of about 7-8 mph.

The rotational speed of the rotor at a given wind speed will depend on how the rotor is loaded. At zero load the rotor will attain a high rotational speed but will produce very little power. As the load is increased, the rotational speed drops as the rotor produces power until, at high load levels, the rotor speed is slowed to zero. These curves can be plotted for different windspeeds as shown in Figure 5. For each windspeed the power curve shows a well-defined maximum. The loads of the maxima defines the maximum power output of the wind machine as a function of windspeed, and the optimum rotor loading at a particular wind speed.

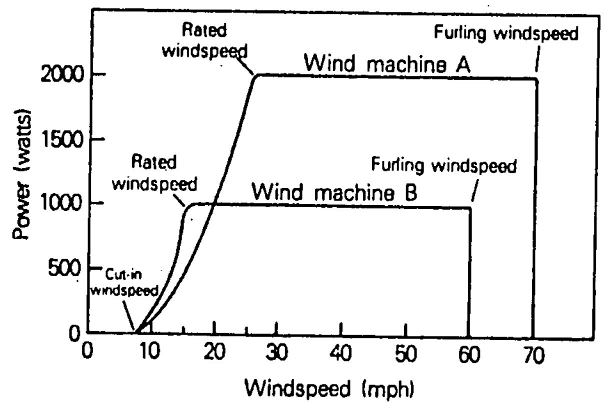


Figure 4. Output power for two typical wind machines. A rotor produces its maximum power at windspeeds between the rated and furling windspeeds.

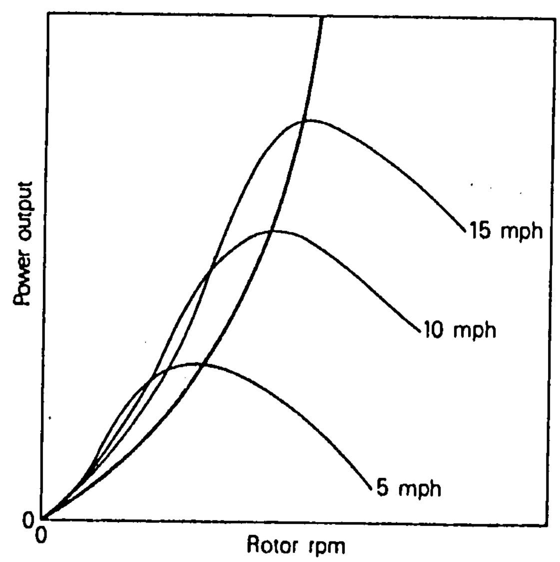


Figure 5. Variation of power output with rpm for a typical rotor. At each windspeed, there is a point of optimum performance (heavy line).

Energy Production

Manufacturers of wind turbines will generally provide promotional material showing power output curves similar to the curves shown in Figure 6. Both curves are for 1 kW systems but at different rated windspeeds. Machine B is rated 1 KW at 20 mph; machine A is rated 1 kW at 32 mph. How much energy will each of these machines produce?

To answer this question we need to know something about the wind regime. Ideally we would have available a wind duration curve, which has the characteristic profile shown in Figure 7. The wind duration curve in this case covers a period of one month only; 720 hours.

From the information shown in Figures 6 and 7 it is possible to determine the energy produced over the 30 day period by both machines. Divide the abscissa of Figure 7 into 20 hour increments and read off the average wind speed for each 20 hour period. From Figure 6 estimate the power produced by each machine at this wind speed in watts. This value multiplied by 20 gives the energy produced (Wh) during the period. Table 1 shows the results of such a calculation.

What can be clearly seen is that machinc B produces more than twice the energy produced by machine A. Even though both machines are nominally 1 kW, the difference in their rated speeds makes a very large difference to the amounts of energy each wind turbine produces. A lower rated speed implies a higher energy yield.

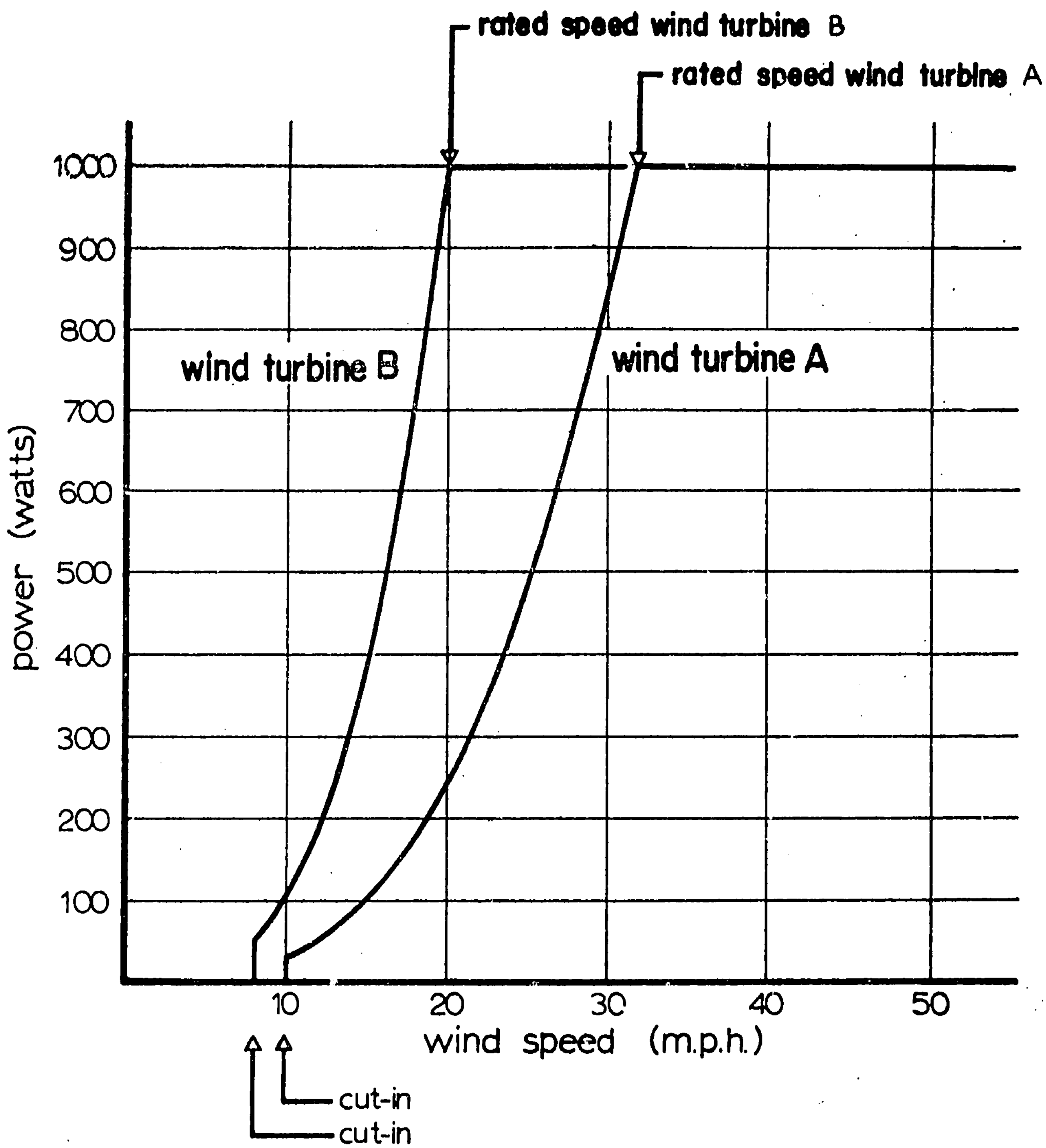


FIGURE 6 Power curves for two example wind turbine generators.

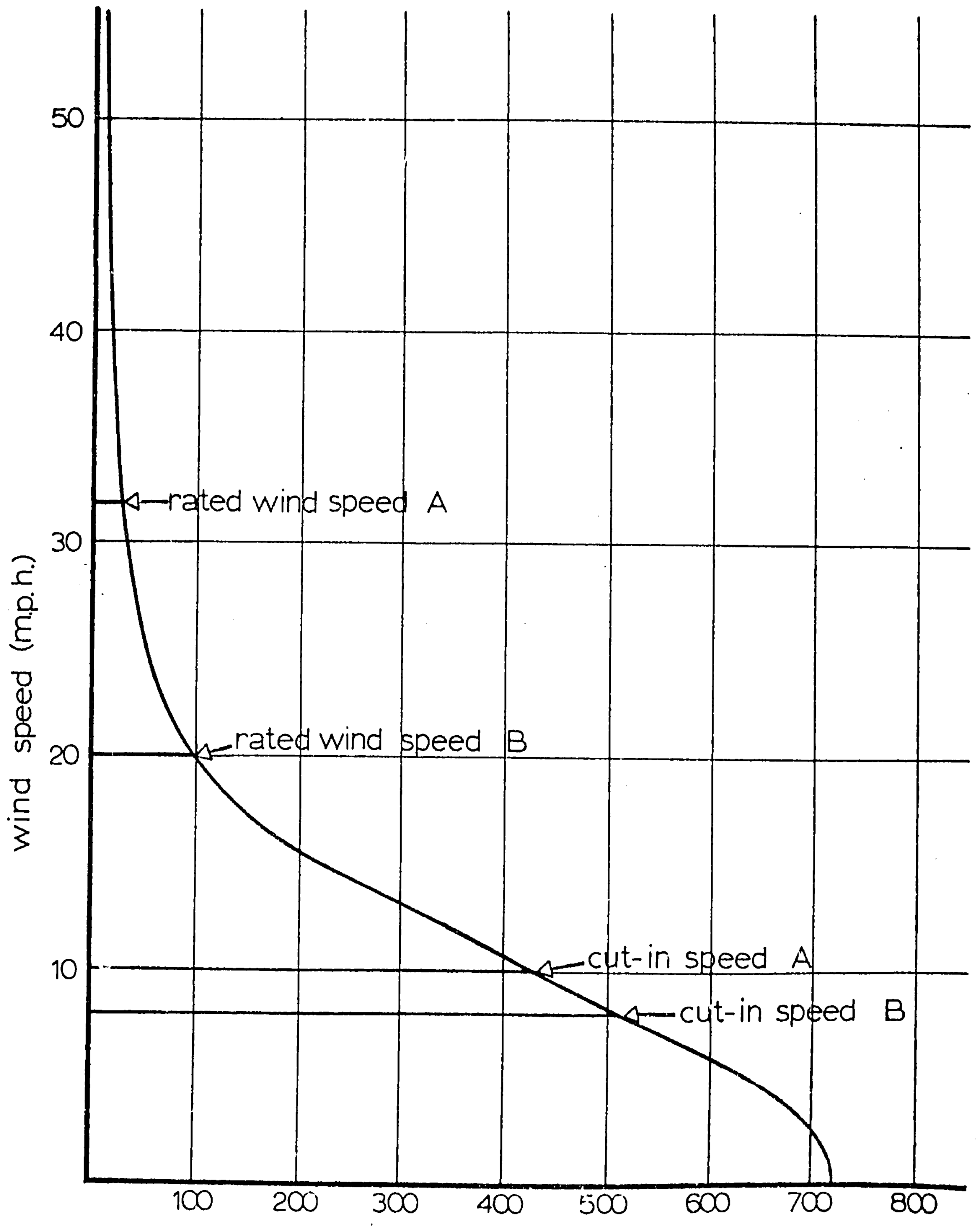


FIGURE 7. Example wind duration curve.

Table 1. Energy extracted per month from the wind distribution shown in Figure 7 by wind turbines A and B.

		WIND TURBINE A		WIND TURBINE B	
No.	*V mph*	*Power watts*	*Watts × 20 Hrs.*	*Power*	*Watts × 20 Hrs.*
1	40	1000	20,000	1000	20,000
2	35	1000	20,000	1000	20,000
3	26	650	13,000	1000	20,000
4	22	320	6,400	1000	20,000
5	20	250	5,000	1000	20,000
6	19	238	4,760	950	19,000
7	17.5	162	3,240	650	13,000
8	17	150	3,000	600	12,000
9	16.5	138	2,760	550	11,000
10	16	125	2,500	500	10,000
11	15.5	113	2,260	450	9,000
12	15	100	2,000	400	8,000
13	14.5	90	1,800	360	7,200
14	14	80	1,600	320	6,400
15	13.5	70	1,400	280	5,600
16	13	60	1,200	240	4,800
17	12.5	50	1,000	200	4,000
18	12	45	900	180	3,600
19	11.5	40	800	160	3,200
20	11	35	700	140	2,800
21	10.5	30	600	120	2,400
22	10	25	500	100	2,000
23	9.5	0	0	95	1,800
24	9	0	0	87	1,740
25	8.5	0		70	1,400
26	8			50	1,000
27	7.5			0	0
28	7				
29	6.5				
30	6				
31	5.5				
32	5				
33	4.5				
34	4				
35	3.5				
36	2				
Total watt hours			95,420 = 95.4 kWh	229,940 = 229.9 kWh	

Example 2

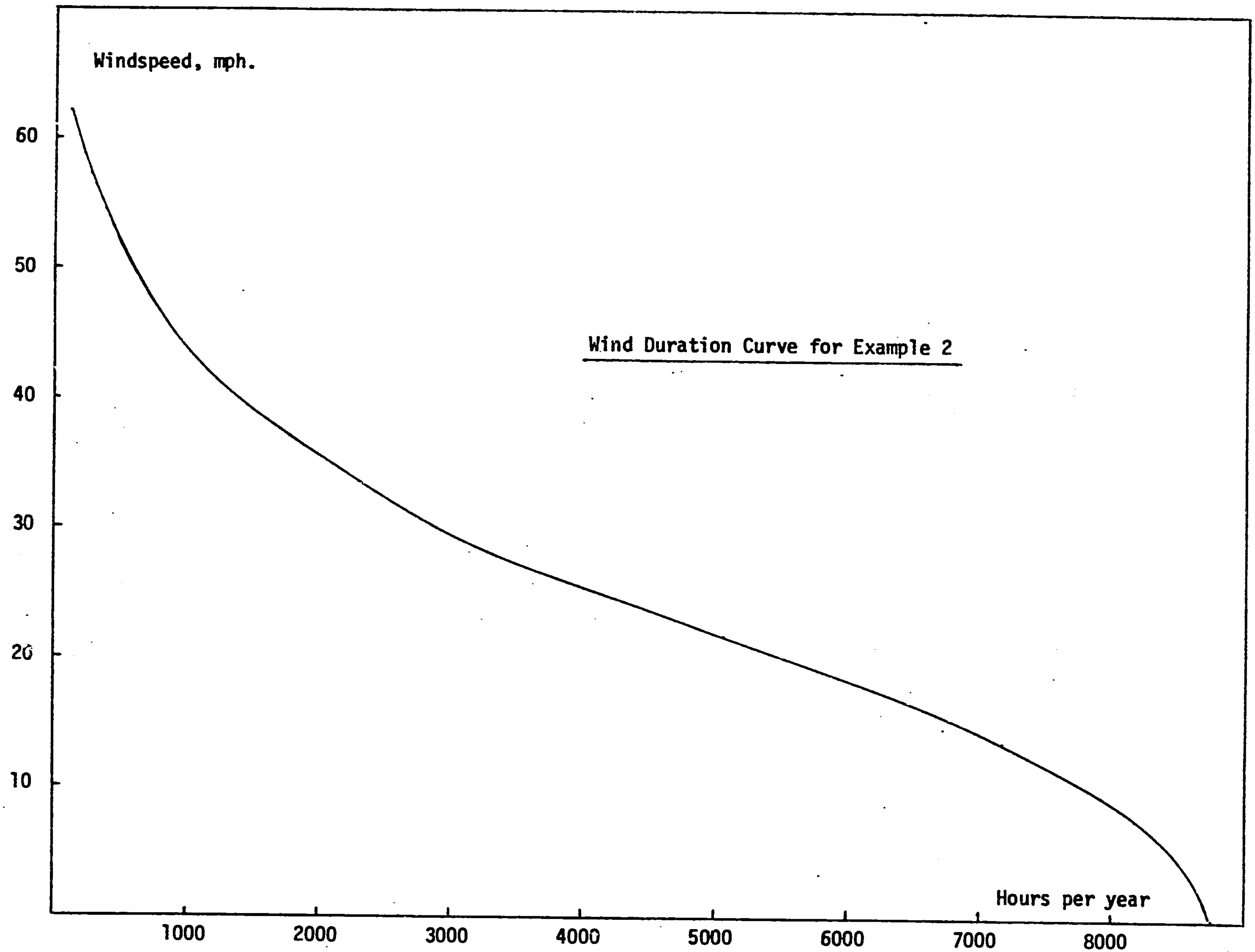
The table below shows annual wind data for a very windy site. Construct the wind duration curve for this location.

SPEED MPH DIR.	1-3	4-6	7-10	11-16	17-21	22-27	28-33	34-40	41-47	48-55	≧56	%	MEAN WIND SPEED
N	.1	.3	1.3	1.1	1.0	.5	.2	.1	.0	.0	.1	4.8	15.8
NNE	.2	.1	.5	.7	.9	.8	.6	.1		.0	.0	3.8	19.8
NE	.2	.1	.6	.8	1.2	.8	.7	.1	.0	.1		4.5	19.4
ENE	.1	.1	.3	.8	1.3	1.5	1.2	1.0	.3	.2	.9	7.7	30.0
E	.2	.4	1.1	1.9	1.5	1.0	.9	1.0	1.0	.3	.2	9.5	23.7
ESE	.1	.1	.5	1.0	1.3	1.3	1.1	.8	.8	.7	.1	7.9	27.4
SE	.1	.3	.6	.7	1.1	1.6	1.3	.9	.4	.3	.1	7.6	25.1
SSE	.0	.2	.3	.2	.3	.4	.4	.7	.5	.6	.3	3.9	32.1
S	.3	.3	.6	.9	1.0	.7	.7	.8	.5	.4	.2	6.4	24.9
SSW	.1	.1	.5	.6	.4	.7	.7	.3	.0	.2	.2	3.7	24.9
SW	.1	.2	.6	1.2	.9	1.1	.9	.5	.3	.2	.2	6.3	23.9
WSW	.0	.2	.8	.9	1.1	1.1	.9	.6	.3	.2	.0	6.1	23.0
W	.2	.3	.7	.9	1.3	1.4	.9	.4	.1	.1		6.3	20.7
WNW	.2	.3	.5	1.1	1.0	1.1	.7	.6	.2	.1		6.0	21.3
NW	.3	.4	1.7	1.9	1.4	1.4	.7	.6	.2	.0		8.7	18.1
NNW	.0	.2	.8	1.7	1.5	1.1	.4	.2	.0	.1		6.0	18.6
VARBL													
CALM												.9	
%	2.3	3.7	11.4	16.4	17.3	16.6	12.2	8.7	4.8	3.7	2.1	100.0	23.0

Solution

The wind duration curve is a cumulative frequency diagram. Starting at the highest windspeed group, determine the time the wind blows at this level. We assume here that the highest windspeed is 65 mph. The calculation is shown below.

range mph	mid-point mph	frequency %	time hrs	cumulative hrs
56-65	60.5	2.1	184	184
48-55	51.5	3.7	324	508
41-47	44	4.8	420	928
34-40	37	8.7	762	1690
28-33	30.5	12.2	1069	2759
22-27	24.5	16.6	1454	4213
17-21	19	17.3	1515	5728
11-16	13.5	16.4	1437	7165
7-10	8.5	11.4	999	8164
4-6	5	3.7	324	8488
1-3	2	2.3	201	8689
Calm	0	0.8	71	8760


Windspeed, mph.
Wind Duration Curve for Example 2
Hours per year
60
50
40
30
20
10
1000
2000
3000
4000
5000
6000
7000
8000

Windspeed Distributions

Winds at many locations exhibit some remarkably similar characteristics. A typical windspeed distribution for a site with 12 mph average winds is shown in Figure 8. Note that the average windspeed is greater than the most frequently occurring windspeed, which in this case is about 9 mph. However, one is primarily interested in the amount of energy in the wind, where this can be calculated from Equation 2. Figure 9, for example, shows windspeed and wind energy distributions for a typical site. At this particular site a windspeed of 8 mph occurs frequently, yet the amount of energy in the wind at this speed is much less than the amount of energy available at windspeeds between 10 and 25 mph. There is very little energy available below 8 mph or above 30 mph at this site.

How does one go about measuring the windspeed distribution at a particular site? There are essentially three options:

1. Measure and record the windspeed at the site for a period of at least one year.

2. Measure and record the windspeed for a shorter period and try to correlate the data with long-term data taken from a nearby weather station or airport.

3. Determine only the annual average windspeed and use a mathematical distribution function to estimate the duration at each windspeed.

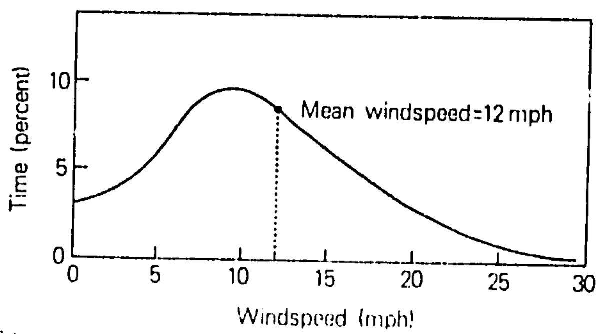


Figure 8. Windspeed distribution for a typical site with 12-mph average winds. This graph indicates the percentage of time, over a period of one year, that the wind blows at any given speed.

4/p

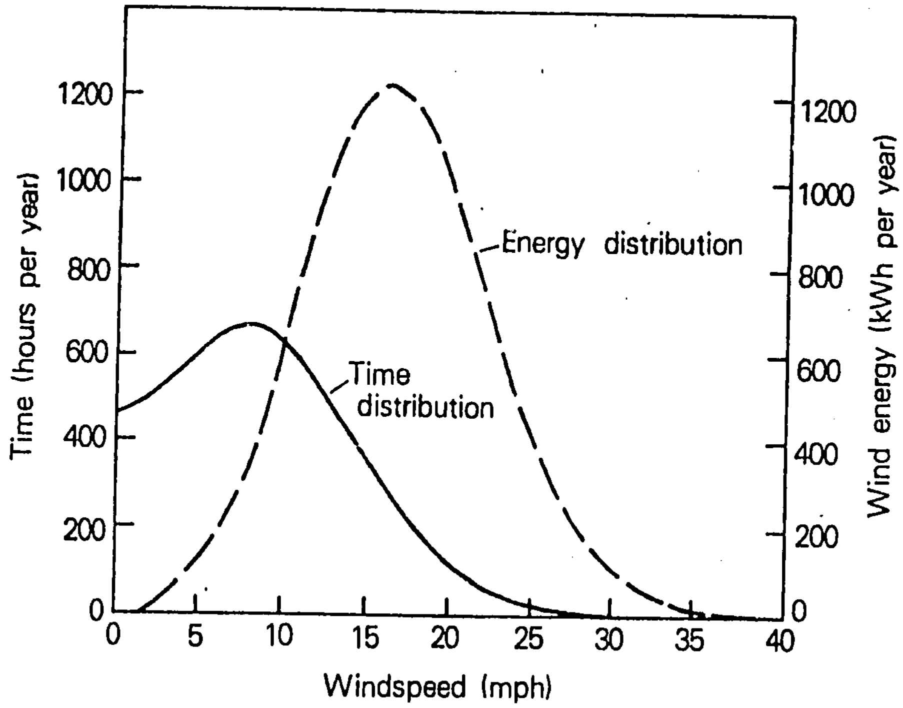


Figure 9. Windspeed and wind energy distributions for a typical wind site.

From an overall planning point of view, the <u>annual</u> windspeed distribution is the most important information to determine. This distribution may be estimated from the annual average windspeed by using the <u>Rayleigh distribution</u>.

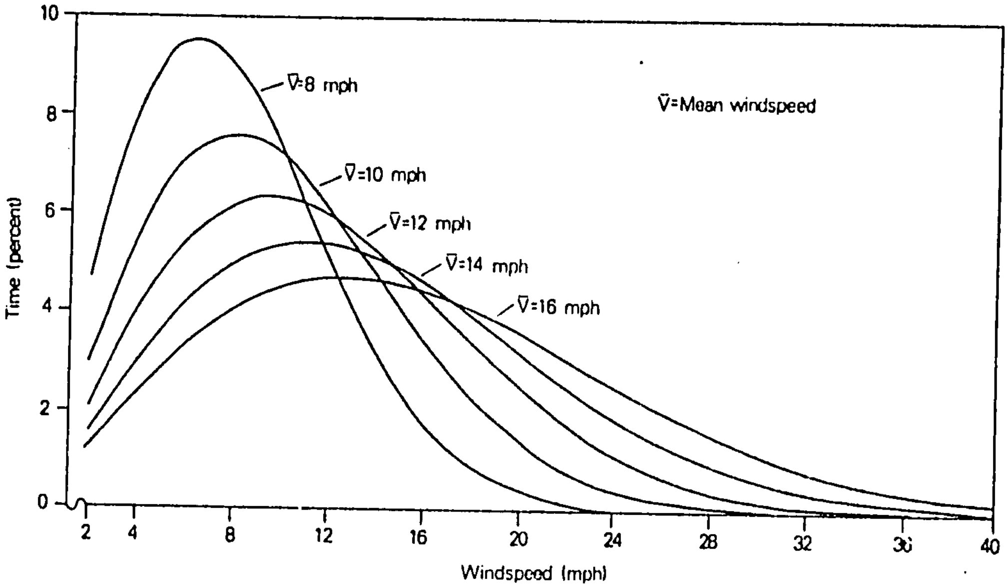
The Rayleigh distribution takes the following form

$$t = 8760 \frac{\pi}{2} \frac{V}{\bar{V}^2} \exp\left[-\pi V^2/4\bar{V}^2\right] \qquad (4)$$

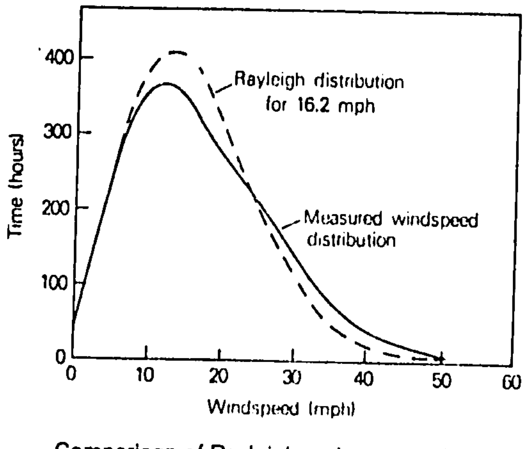
where
- t = time, hours per year
- V = windspeed, mph
- $\bar{V}$ = annual mean windspeed, mph

This equation will predict the total number of hours per year, t, that the wind will blow at windspeed, V, at a site where the mean windspeed is $\bar{V}$. At mean windspeeds below 10 mph the Rayleigh distribution starts to become inaccurate. It should not be used for sites where the mean annual windspeed is below 8 mph. Figure 10 shows predicted windspeed distributions for mean annual windspeeds between 8 and 16 mph. Accuracy should be better than 90% for typical sites on unobstructed terrain. Table 2 shows Rayleigh windspeed distributions in hours per year for a range of annual mean windspeeds [3].

Figure 10.



The Rayleigh distribution versus windspeed for sites with mean windspeeds between 8 and 16 mph.



Comparison of Rayleigh and measured wind-speed distributions for St. Ann's Head, England.

TABLE 2

RAYLEIGH DISTRIBUTION FOR VARIOUS MEAN WINDSPEEDS										
Windspeed mph	Mean Windspeed, mph									
	8	9	10	11	12	13	14	15	16	17
8	784	731	666	601	539	484	435	391	353	320
9	716	697	656	605	553	503	457	415	377	344
10	630	644	627	594	554	512	470	431	395	363
11	536	578	585	570	543	510	476	441	408	377
12	441	504	533	536	523	500	473	443	415	386
13	351	429	474	494	494	483	464	441	416	391
14	272	356	413	446	459	458	448	432	412	391
15	204	288	353	396	420	429	427	418	404	387
16	149	227	295	345	378	396	403	400	392	380
17	105	175	242	296	336	361	375	379	377	369
18	73	132	194	250	294	325	345	355	358	355
19	49	97	153	207	253	289	314	330	337	339
20	32	70	119	170	216	254	283	303	315	321
21	20	50	90	136	181	220	252	275	291	302
22	12	34	68	108	150	189	222	248	268	281
23	7	23	50	84	123	160	194	222	244	260
24	4	15	36	65	99	134	168	197	220	239
25	3	10	25	49	79	111	143	173	198	218
26	1	6	18	37	62	91	122	150	176	197

RAYLEIGH DISTRIBUTION FOR VARIOUS MEAN WINDSPEEDS										
Windspeed mph	Mean Windspeed, mph									
	8	9	10	11	12	13	14	15	16	17
27	.8	4	12	27	48	74	102	130	155	177
28	.4	2	8	20	37	60	85	111	136	158
29	.2	1	5	14	28	47	70	94	118	140
30	.1	.8	4	10	21	37	57	79	102	124
31	0	.5	2	7	16	29	46	66	87	108
32	0	.3	1	5	11	22	37	55	74	94
33	0	.1	.9	3	8	17	29	45	63	81
34	0	0	.5	2	6	13	23	37	53	70
35	0	0	.3	1	4	10	18	30	44	60
36	0	0	.2	.9	3	7	14	24	36	51
37	0	0	.1	.6	2	5	11	19	30	43
38	0	0	0	.4	1	4	8	15	24	36
39	0	0	0	.2	.9	3	6	12	20	30
40	0	0	0	.1	.6	2	5	9	16	25
41	0	0	0	0	.4	1	3	7	13	20
42	0	0	0	0	.3	.9	3	5	10	17
43	0	0	0	0	.2	.6	2	4	8	13
44	0	0	0	0	.1	.4	1	3	6	11

Windspeed distribution curves can be used to determine energy distribution curves. The energy in the wind is the power available at a particular windspeed, calculated using Equation 2, multiplied by the time the wind blows at that speed, as either measured or predicted by a Rayleigh distribution.

Since $$P_W = 1/2\ \rho A V^3 \qquad \text{Watts}$$

and $$t = 4380\ \pi \frac{V}{\bar{V}^2} \exp\left(\frac{-\pi V^2}{4 \bar{V}^2}\right) \qquad \text{hrs/yr}$$

then $$E(t) = \frac{P_W t}{1000} \qquad \text{kWh/yr}$$

The graphs below show windspeed distribution curves and energy distribution curves for two sites with average annual windspeeds of 10 mph and 14 mph.

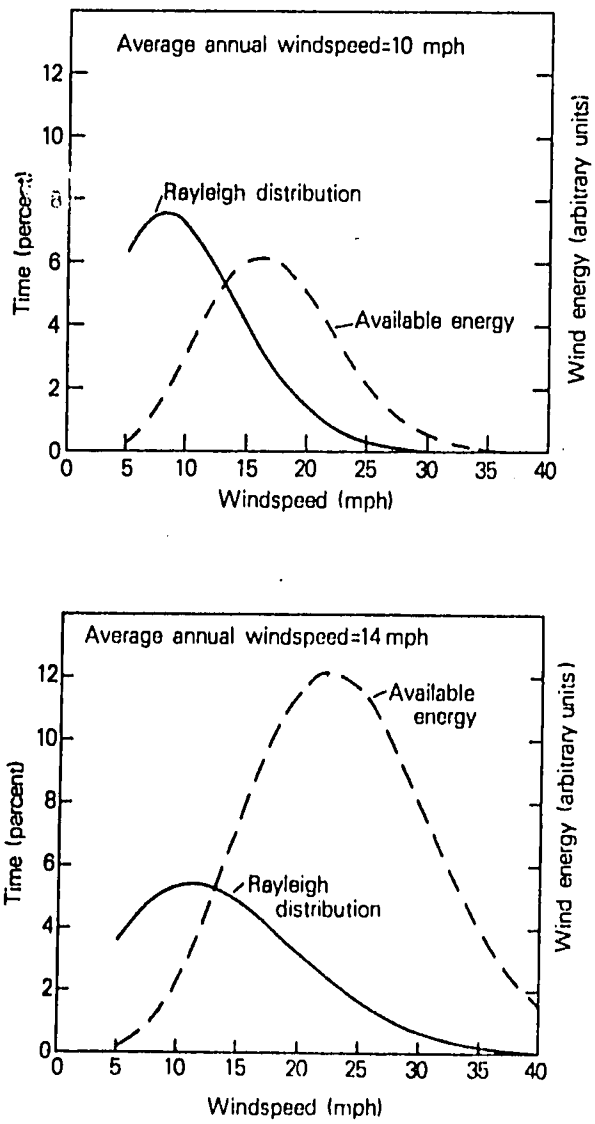


Figure 11 Rayleigh windspeed distributions for sites with 10 mph and 14 mph average winds. The available wind energy is far greater at the windier site.

It is possible to calculate the windspeed at which the peak of each curve will occur. It can be shown analytically that for a Rayleigh distribution the most frequent windspeed, V_f, occurs at

$$V_f = 0.8\,\overline{V}$$

where $\overline{V}$ is the mean annual windspeed, and that the maximum energy available (the peak of the available energy curve) occurs at a windspeed, $V(E_{max})$ given by

$$V\,(E_{max}) = 1.6\,\overline{V}$$

The latter relationship is especially important since it permits a quick calculation of the windspeed at which the maximum energy is available, knowing only one parameter, the mean annual windspeed. The peak-energy windspeed should be about the same as the windspeed at which the system produces its rated power. In this way the wind energy conversion system is matched to the wind regime.

The Rayleigh distribution gives rise to a wind duration curve similar to the one shown in Figure 7. The curve can be constructed using the equation below

$$t_D = 8760 \exp\,[-\pi V^2/4\overline{V}^2)\ \text{hours} \qquad (5)$$

where t_D is the time the wind blows at a velocity greater than V for a distribution with a mean annual windspeed $\overline{V}$.

Lift and Drag

Wind machines use a combination of lift and drag forces to convert the kinetic energy in the wind into a torque applied at the axis of the rotor. The drag force on an airfoil occurs in a direction parallel to the relative wind. The lift force acts in a direction perpendicular to the relative wind. The forces are shown in the diagram below.

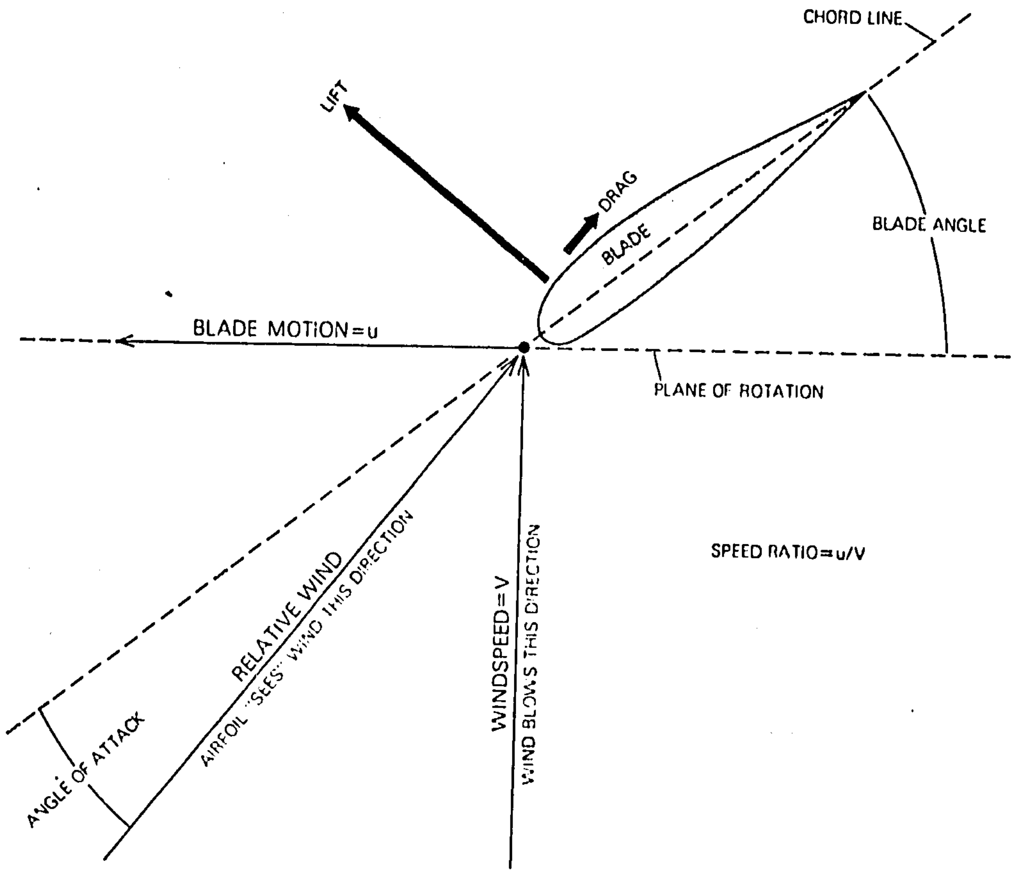


Figure 12. Vector diagram of the airflow at a single rotor blade. The lift force pulls the blade along its rotary path.

The principal attribute of the airfoil is the ability to produce a high lift while incurring only a small drag. Airfoils characteristically have a blunt nose and a finely tapering tail. They can be symmetrical or cambered as shown below.

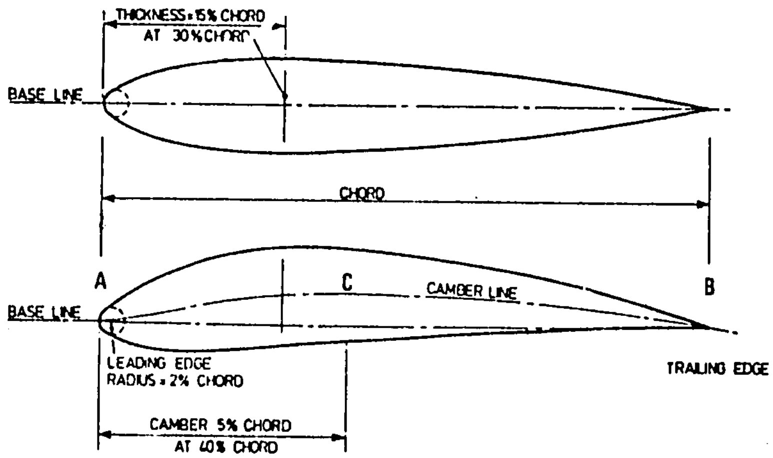


Fig. 13 Geometry of symmetrical and cambered aerofoil

The airfoils above illustrate the characteristic features of (1) the nose radius, (2) the position and magnitude of the point of maximum thickness, and (3) the angle of the trailing edge. These parameters are usually expressed in terms of the chord length. The upper airfoil is symmetrical; it will produce no lift until the airflow makes an angle with the chord line as depicted in Figure 12. This angle is called the angle of attack. Symmetrical airfoils, used on boats for rudder and keels, are essential for lift-dominated vertical axis wind turbines like the Darrieus rotor. If the airfoil is to develop lift at zero angle of attack, then it must be cambered as shown by the lower airfoil in the figure above.

The airfoil properties of interest to the wind system designer are, of course, the lift and the drag, and sometimes the turning moment which the flow exerts on the airfoil. Typical curves of lift and drag, plotted against the angle of attack for a low-speed airfoil are shown in Figure 14, together with the ratio of the forces, called the lift to drag ratio or L/D ratio.

As the angle of attack is increased, the lift increases at a faster rate than the drag does until the blade stalls: the angle of attack at which the lift falls dramatically and drag rapidly increases.

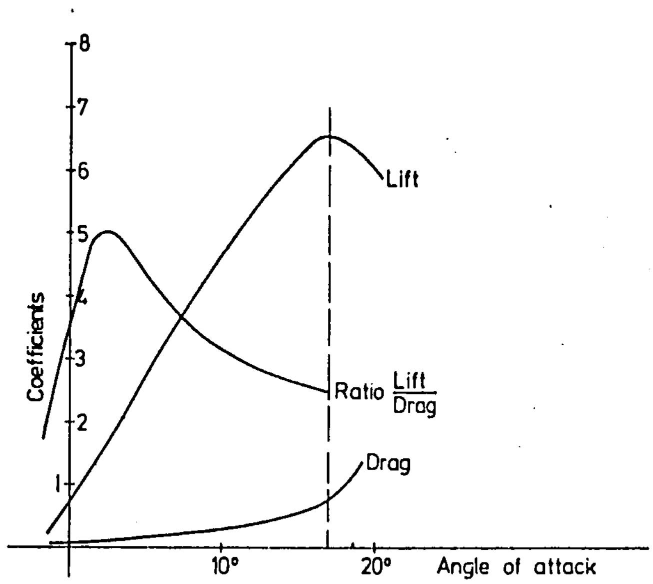


Fig. 14 Properties of an aerofoil

The forces on the blade due to lift and drag may be calculated from

Lift = $1/2\ C_L \rho S V^2$ Newtons

Drag = $1/2\ C_D \rho S V^2$ Newtons

where C_L and C_D are the dimensionless coefficients of lift and drag respectively.

also ρ = air density, kg/m^3
s = blade surface area, m^2
V = airspeed, m/s

Generally, the higher the blade L/D ratio, the faster the rotor will spin, and the greater will be the coefficient of performance. But high L/D ratios arise only from sophisticated blade design which generally means an expensive rotor. We find, therefore, in windmill blade design the usual trade-offs between complexity, sophistication, performance, and cost. The table below shows how these trade-offs are usually resolved for different applications for typical horizontal axis wind systems.

Wind Machine	Design TSR	Blades	Blade Type	Blade L/D
Water pumper	1	6-20	Flat Plate	10
	1	6-20	Curved Plate	20-40
	1	4-10	Sail wing	10-25
Small wind-electric	3-4	4-6	Simple airfoil	10-50
	4-6	2-4	Twisted airfoil	20-100
	3-5	3-6	Sail wing	20-35
Large wind-electric	5-15	1-3	Twisted airfoil	20-100

Rotor Configurations

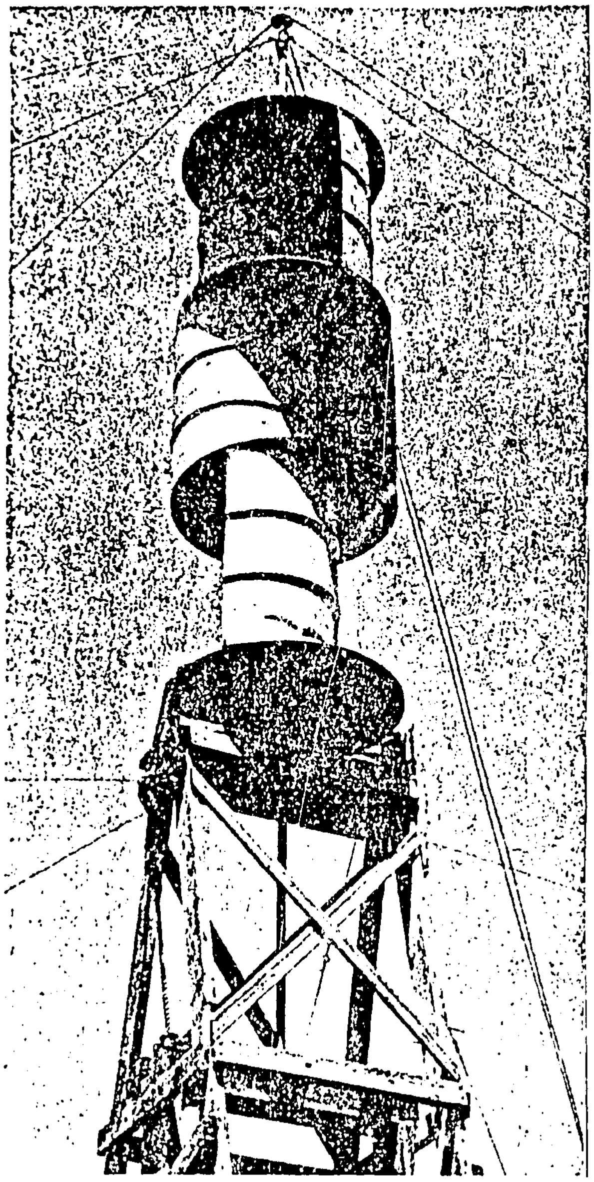
There are basically four common types of wind machine: the Savonius rotor, the Darrieus rotor, both vertical axis configurations; multibladed low-speed rotors like the U.S. farm windmills, and high-speed propellor-type rotors, both horizontal-axis configurations. These rotors have different aerodynamic and power characteristics and a particular rotor configuration can be selected that is well-suited for the mechanical task at hand. Savonius rotors and the multibladed hign-solidity rotors are low-speed rotors that have a high starting torque and are appropriate for mechanical work such as pumping water or milling and grinding grain. The vertical-axis Darrieus and the propellor-type rotors spin much faster and have little or no starting torque. Their high rotational speeds make these rotor types appropriate for driving electric generators.

Savonius Rotor

The Savonius rotor is an extremely simple and robust wind energy conversion system. Sometimes called the S-rotor because of its dinstinctive shape, it looks rather like an oil drum that has been cut in half along its length and the halves separated sideways. More often than not, this is exactly how it is built. Figures 15 and 16 show a couple of typical configurations.

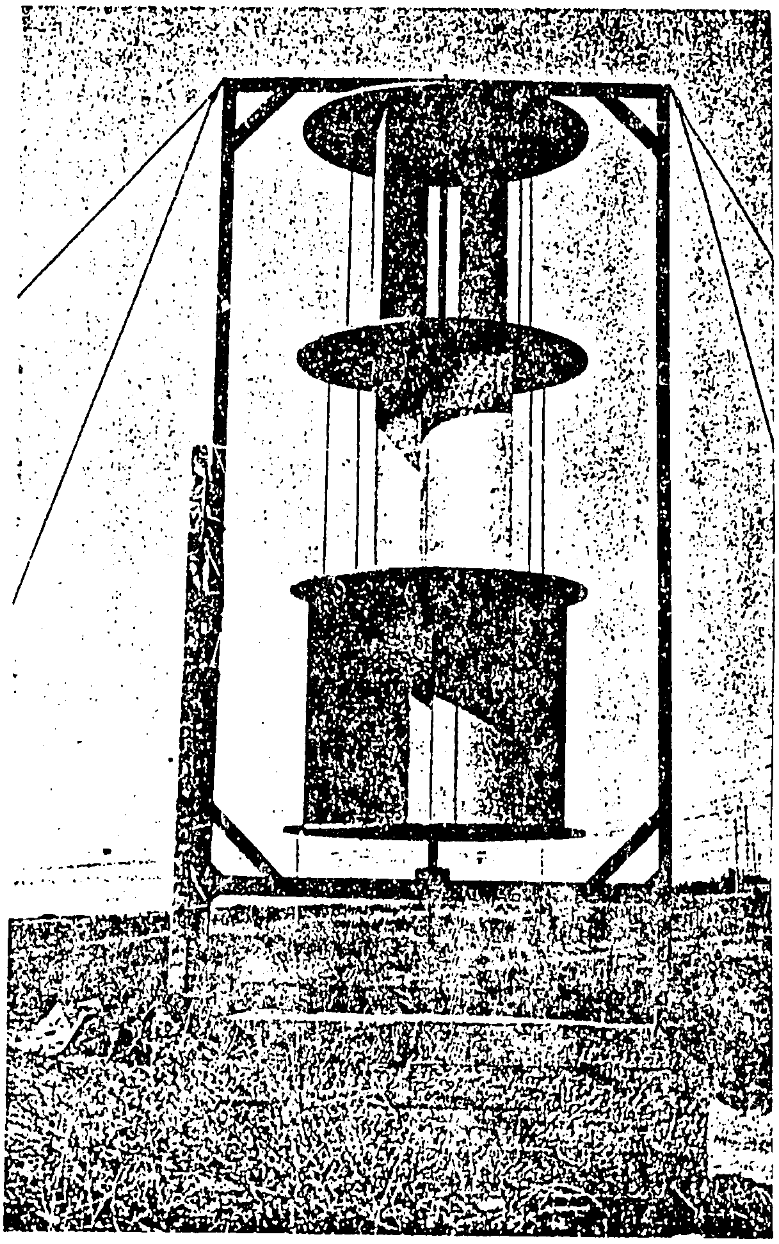
The advantages of the Savonius rotor include its simplicity and ease of construction, and its high starting torque which permits it to start up under load. However, it suffers from rather low efficiency (a coefficient of performance of about 10-20 per cent), and difficulties with overspeed control.

Savonius rotors come in all shapes and sizes. Figure 17 shows some of the more common design configurations. The more simple two-vane design seems to work as well as the multivane types. The aspect ratio, the ratio of vane height to rotor diameter, has an effect on the torque produced. Higher aspect ratio rotors (taller and slimmer) will generally run at higher rotational speeds and lower torque than those systems with a low aspect ratio.



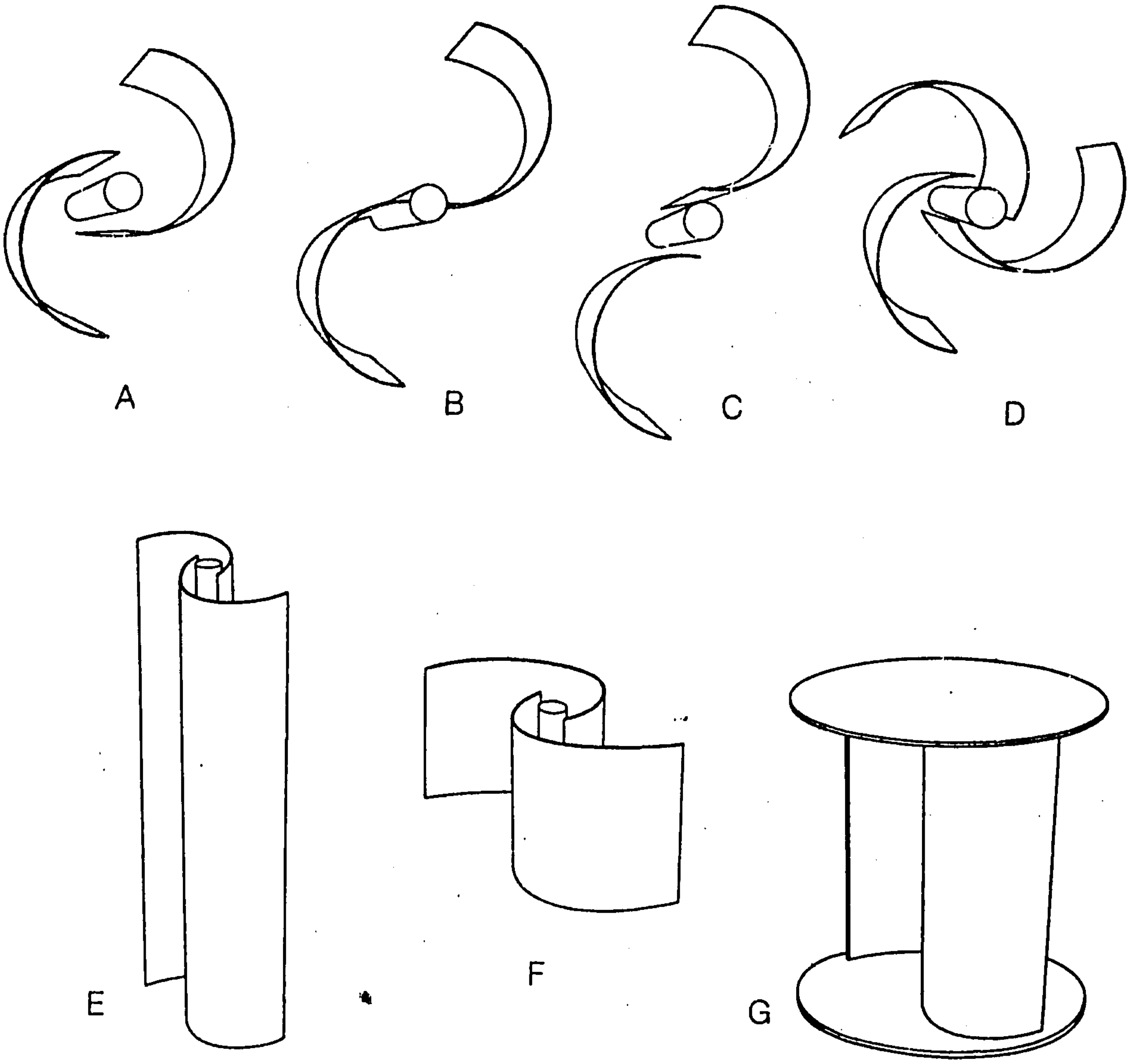
A low-technology Savonius rotor. Easily fabricated from surplus oil drums, this drag-type machine offers only limited power.

Figure 15.



A three-tiered Savonius rotor designed to generate electricity.

Figure 16.



Savonius rotor design options include the intervane gap, number of vanes, aspect ratio, and tip plates. Option E has a much higher aspect ratio than F, and the tip plates in option G improve the rotor performance at low rpm.

Figure 17.

Multibladed Horizontal Axis Rotors

The multibladed horizontal axis rotor is by far the most common wind energy conversion system. In the United States the old farm windmills can still be seen in operation doing what they do best: pumping water. The desirable features of the multiblade rotor are:

- High starting torque
- Simple design and construction
- Simple control requirements
- Durability

Among its disadvantages are:

- Poor compatability with high rpm loads
- High rotor drag load on the tower

The basic components of the American farm windmill are shown in Figure 19. Multibladed rotors are high-solidity low-speed rotors. Their optimal tip speed ratio is about 1 at which point their efficiency may be as high as 30 per cent, but 15-20 per cent is more realistic.

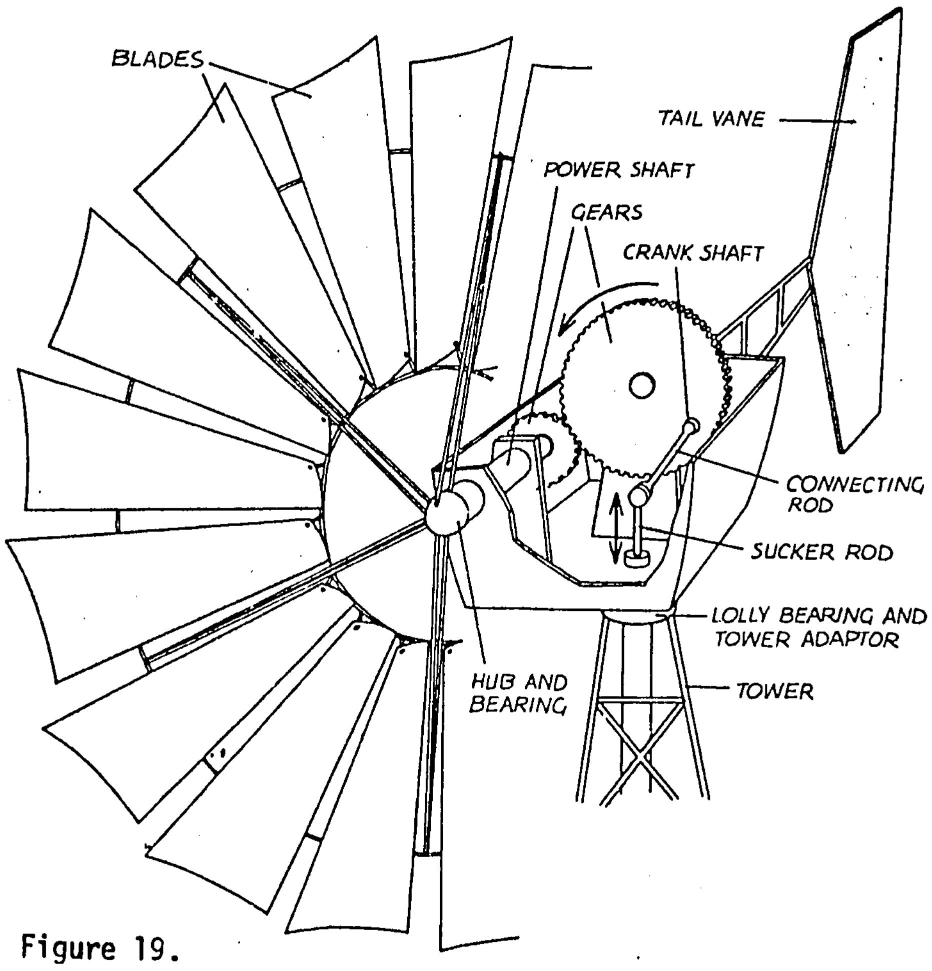


Figure 19.
Components of an American Farm windmill. Gears and crankshaft convert rotary power into the up-down motion of the sucker rod.

Many kinds of multivane wind turbines suitable for pumping water have now come into use in the rural areas of many developing countries. Figure 20 shows a design offered by VITA which uses a recycled automobile axle as a transmission system. The design has flat blades.

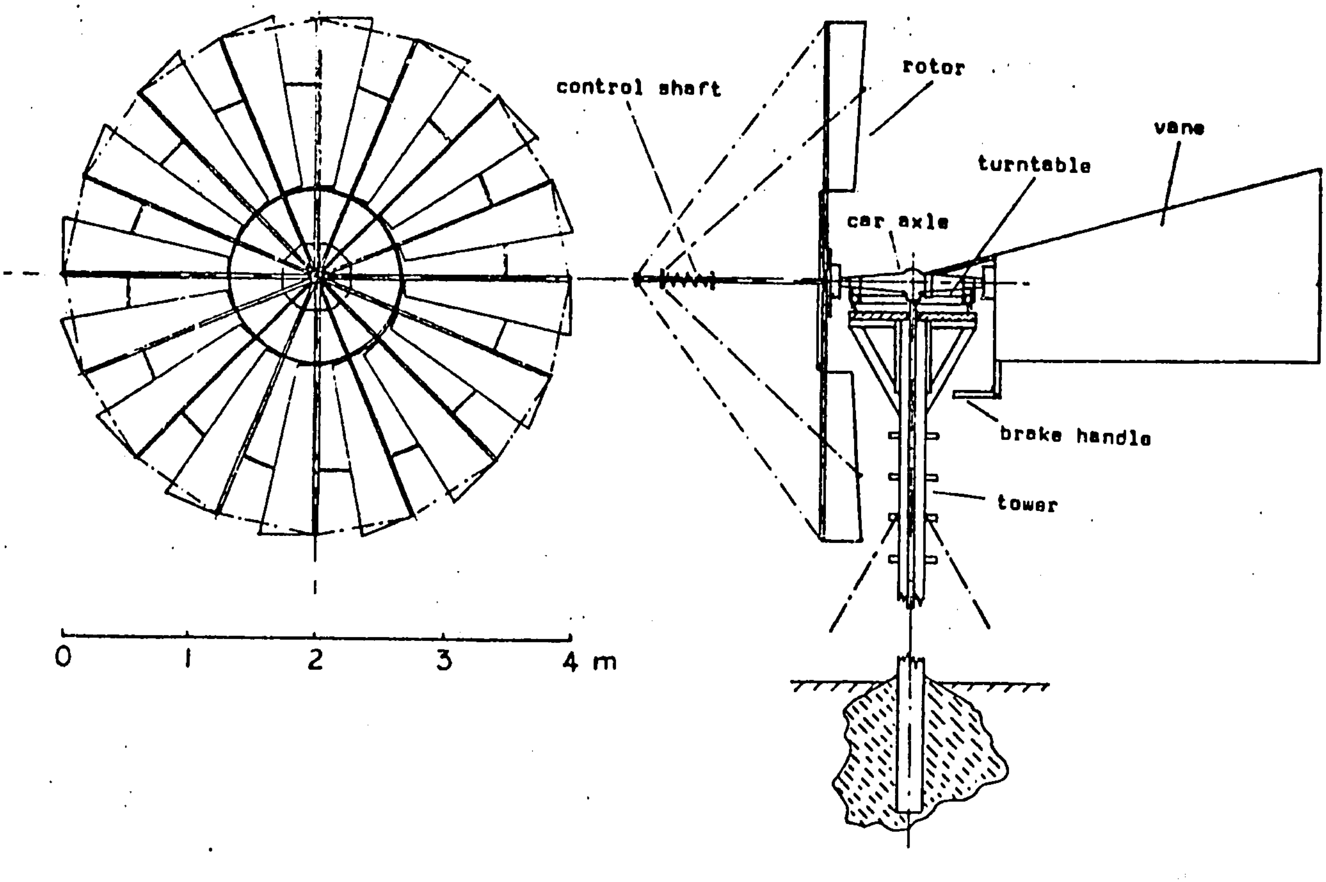
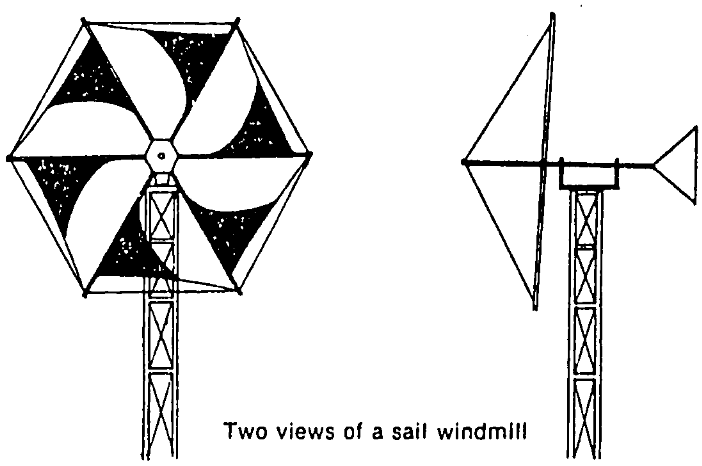


Figure 20. Windmill design suggested by VITA

Perhaps the most simple type of horizontal axis windmills are the sail windmills. This type of windwheel has 4-12 radial arms to which are attached triangular sails. These machines are common on the islands of the Aegean and in other parts of the Mediterranean. The mountain plateau of Lasithi in Crete boasts so many sail windmills that it has come to be known as the "Valley of 10,000 Windmills".

A feature of these windwheels is the forward extension of the axial shaft to provide an attachment for wire stays bracing the radial arms. Stays also extend from tip to tip of the arms. The stays brace the windwheel and generally stiffen the structure. One side of each sail is attached along the rotor arm while the opposed corner is attached at a point along the circumferential bracing in the manner shown below

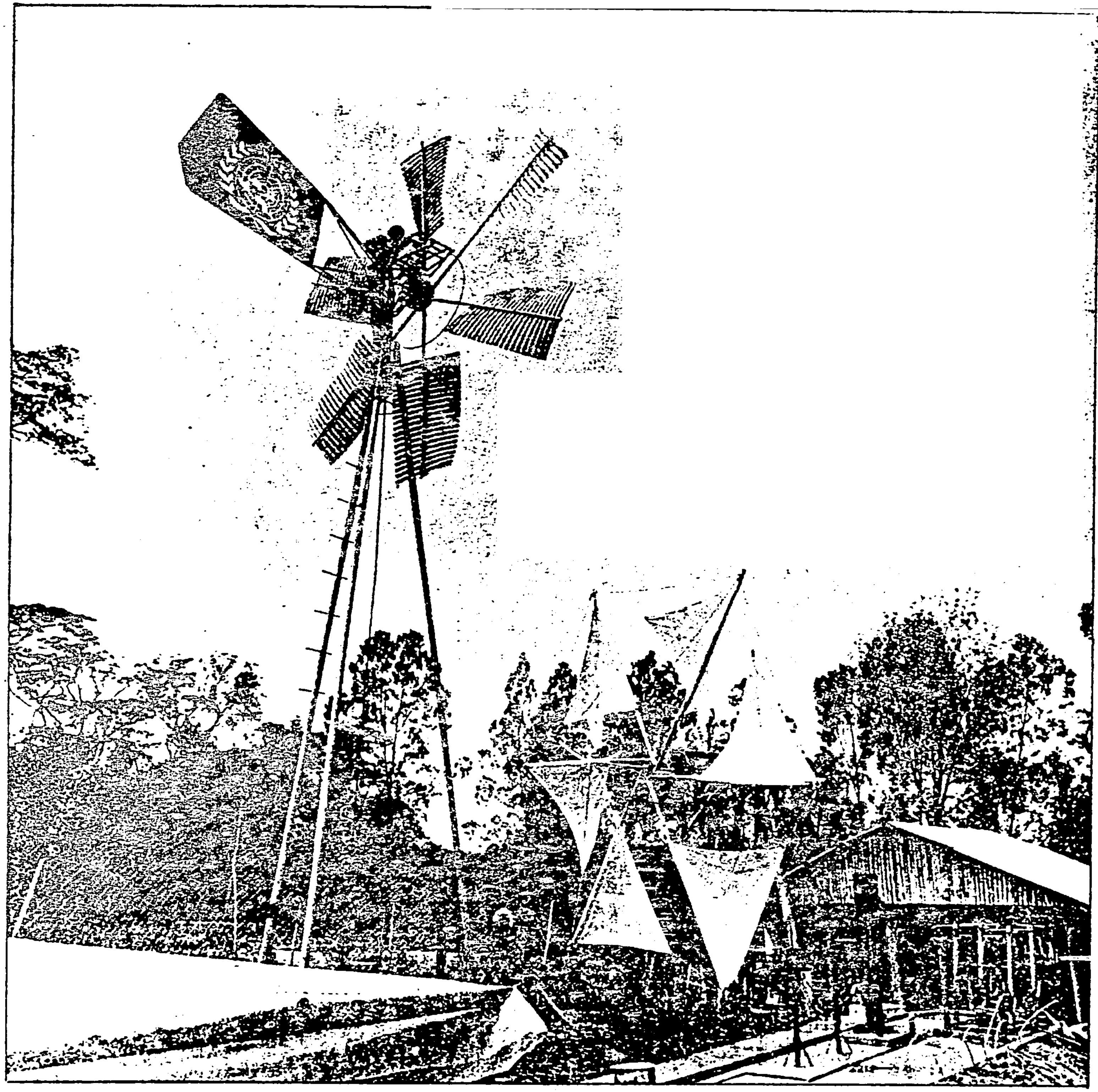


Two views of a sail windmill

In the traditional Cretan design, the circumferential bracing is commonly made of chain, and the setting of the sails is effected by engaging a hook, attached to the corner of the sail, with an appropriate link in the chain. When the wind is strong, and reefing is required, the sails are wound around the poles so as to reduce their area.

The sail mill has many advantages. It has great strength, the aerodynamic surface is self-forming and flexible, and it shows a high degree of self-regulation. It is also a simple and inexpensive machine.

It is interesting to note that the traditional Cretan windwheel design has been successfully used in Africa. The American Presbyterian Mission at Omo Station in Ethiopia has introduced a water pumping version of the sail wind turbine to the local people with great success. The windwheel is used to pump water from the River Omo and irrigate small plots of land along the banks of the river. The project is interesting because the people involved compared a modified Cretan windwheel with a Savonious rotor with respect to their ability to pump water. The sail windmills were found to be superior. The first wind systems used were imported U.S. multibladed farm windmills, (Dempsters), but these proved too expensive a proposition for the local Geleb farmers. The locally fabricated sail wind rotors were produced for about one third the cost of the Dempster water pumpers and were built almost entirely of locally available materials.



The Arusa windmill and a sail cloth windmill at the UNICEF village technology unit display in Nairobi, Kenya.

Darrieus Rotor

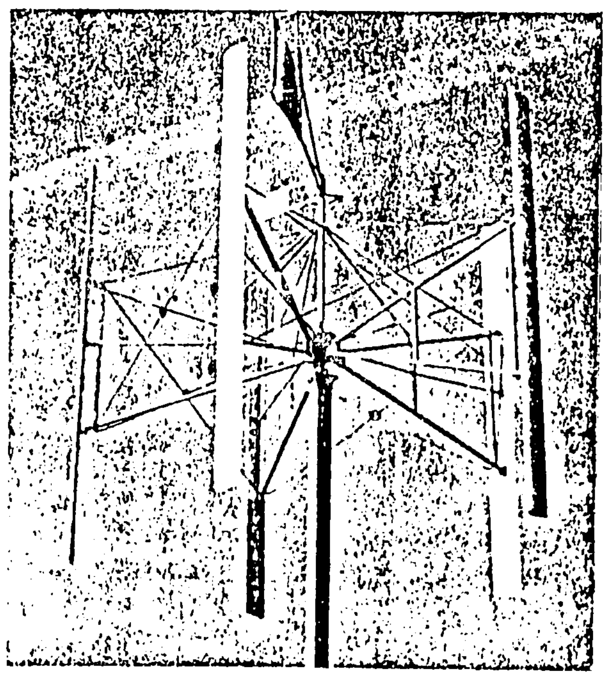
In 1925, G. J. Darrieus applied for a U.S. patent for a wind energy conversion system designed to generate electrical power. Figure 21 shows the basic system configuration. Each blade is a symmetrical airfoil and is curved in the shape that a perfectly flexible cable of uniform density and cross-section would assume if rotated about a vertical axis. The blade shape is called a troposkien. The advantage of this unusual shape is that rotation does not cause the blade to bend and thus the stresses will be only tension.

The swept area of the Darrieus rotor is approximately 2.67 X radius X half-height. Generally, the height is about equal to the diameter.

The Darrieus rotor exhibits some rather unusual aerodynamic characteristics. At low rotational speeds the airfoil is stalled over an appreciable portion of a revolution. The rotor therefore produces almost no torque at low rotational speeds. The Darrieus rotor must therefore be provided with a starting system. This can be an electric motor that disengages when the rotor gains speed or, more simply, one or more small Savonius rotors mounted on the main rotor shaft. Since the Savonius develops maximum torque at start-up it is a useful complement to the more sophisticated and efficient Darrieus rotor.

One version of the Darrieus rotor uses straight blades held parallel to the vertical axis of rotation. Such a machine is sometimes called a cyclogiro or a giromill. The straight bladed Darrieus has the advantage that the blades can be easily hinged. By changing the blade pitch the low-speed stall region can be reduced.

The performance of a Darrieus rotor is very sensitive to tip speed ratio. At low rotational speeds the blade is stalled, at high speeds torque falls off rapidly. The optimal tip speed ratio is about 6 at which point the rotor efficiency is about 35 per cent.



A straight-bladed Darrieus rotor. The pitch angle of the blades is changed automatically.

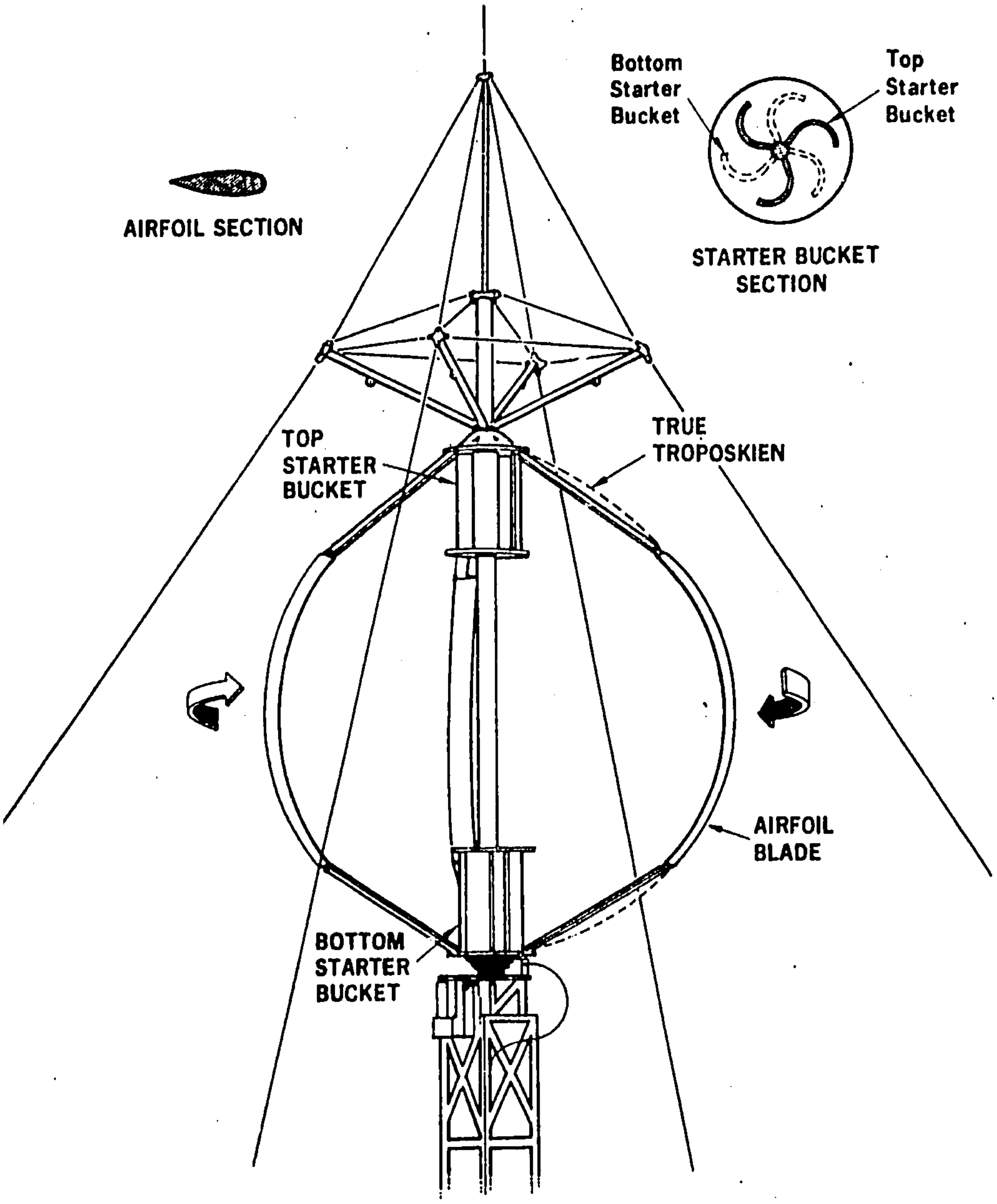


Figure 21. Darrieus vertical axis wind turbine

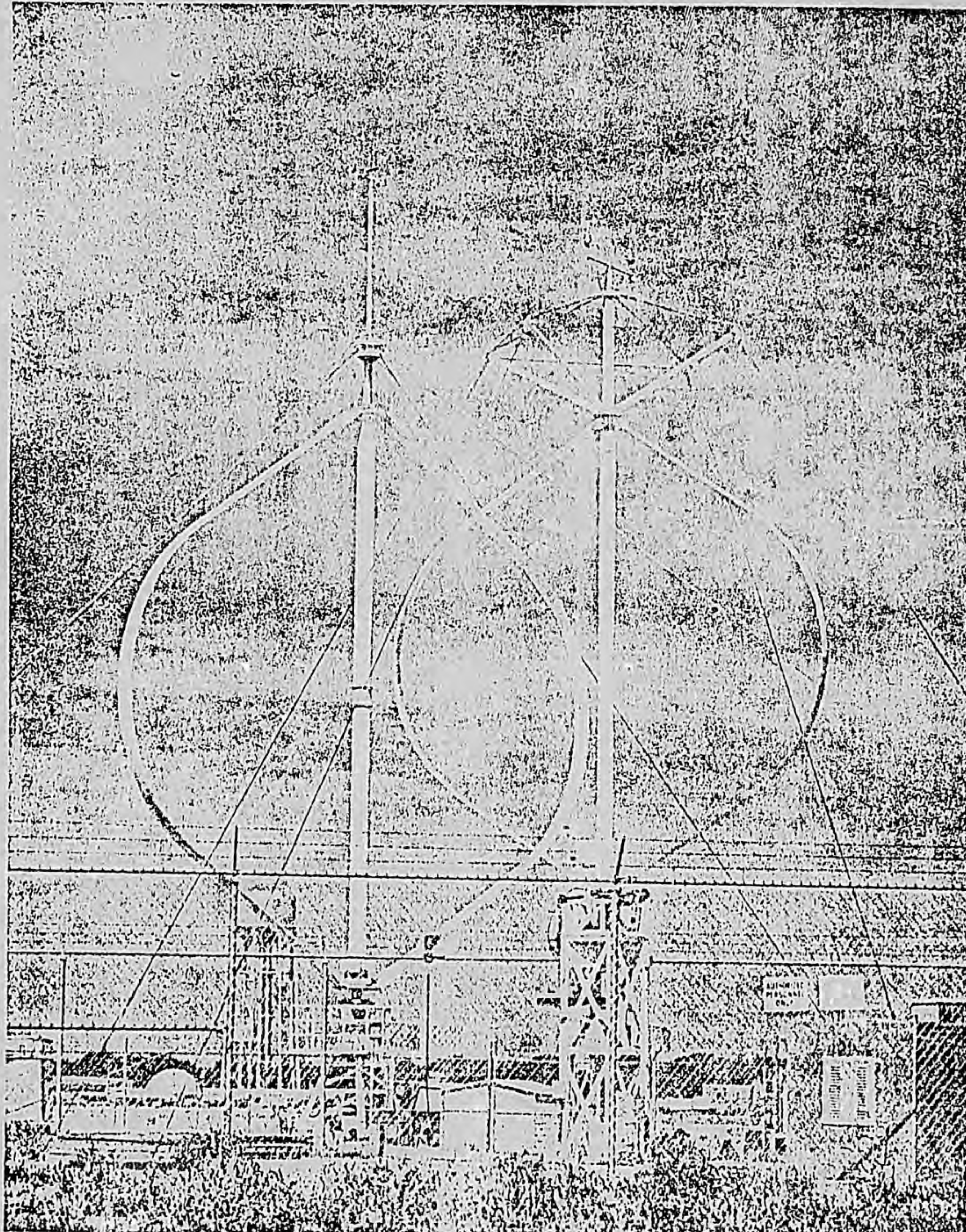
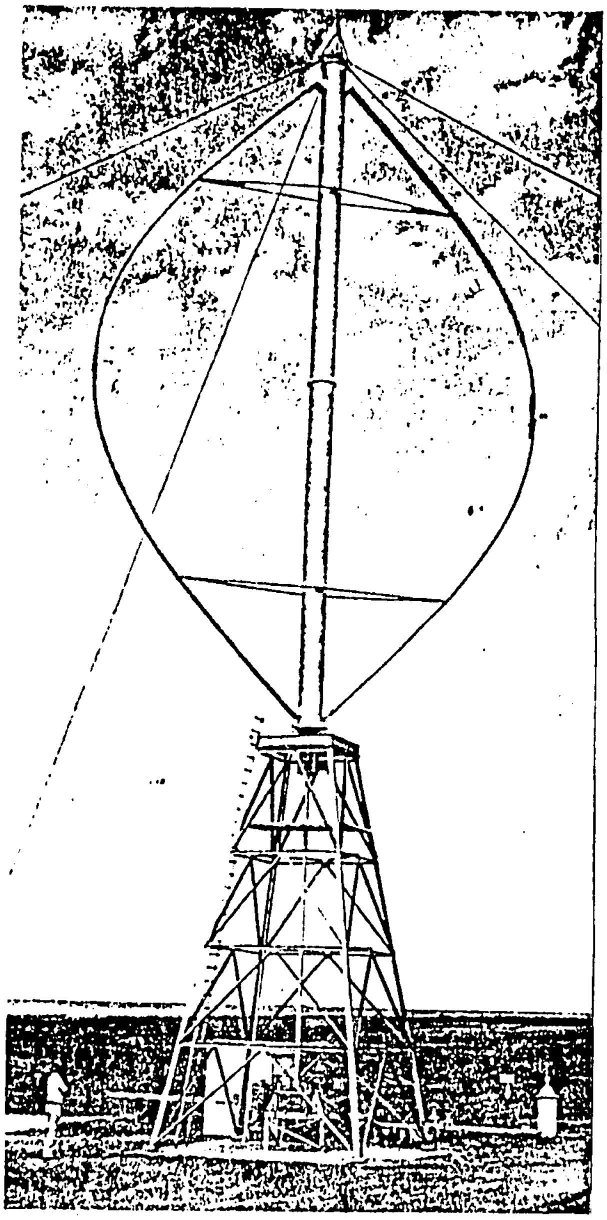
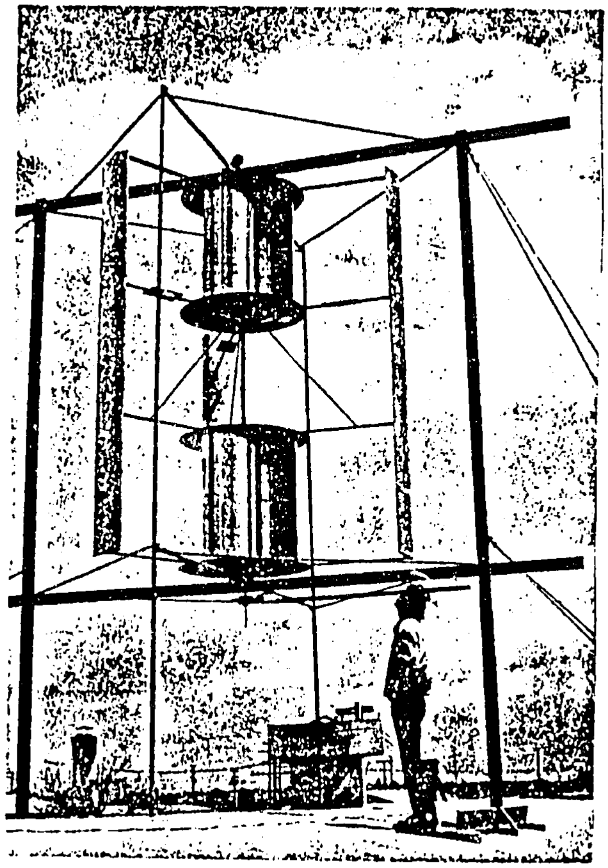


Figure 22. 5 m and 17 m Darrieus Turbines



The Darrieus rotor used for irrigation in Bushland, Texas.



Small Savonius rotors along the axis help accelerate this straight-bladed Darrieus rotor through the stall region.

High Speed Rotors

High speed propeller-type rotors represent the most sophisticated form of the wind energy conversion technologies. The aerodynamic characteristics of the blade are fundamentally important. Rotor efficiencies of up to 45 per cent are possible and 40 per cent is common. High rotational speeds produce the high tip speed ratios shown in Figure 1. High rotational speeds are desirable for electrical power generation since they reduce the need for gearing systems and their associated losses. The high efficiency is attained, however, at some cost. The blades are aerodynamically complex and must be precisely fabricated. Long slender blades may suffer from vibration problems; starting torque is low.

Water Pumping

There are two principal tasks for which wind machines are designed: pumping water and generating electricity. Each task requires a different type of wind machine. The water-pumping system must develop a high torque at start-up. Low speed multibladed rotors operating at tip speed ratios of about 1 are used. Figure 23 shows a typical wind driven water pumping system of the kind still common in the U.S.

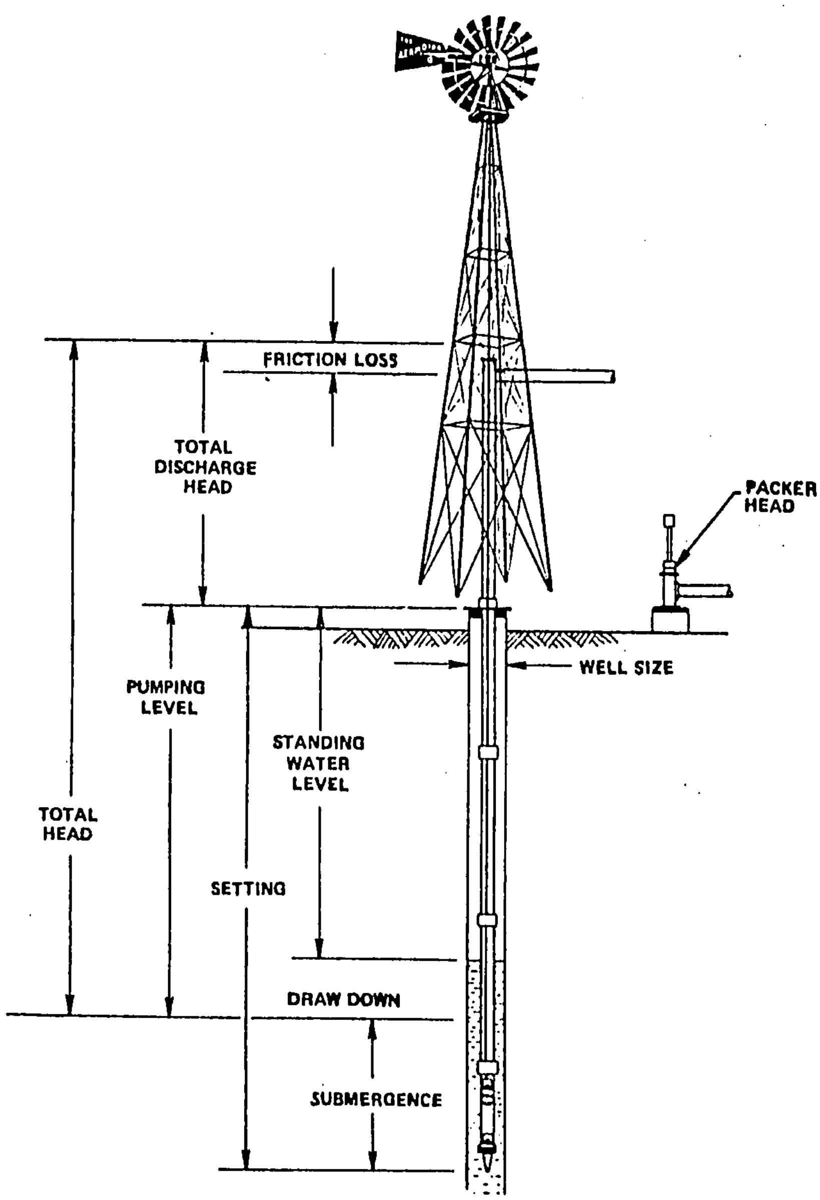


Figure 23. Typical water pumping wind system

Irrigation methods fall into two categories: surface irrigation and well irrigation. In surface irrigation, water is led from rivers, lakes, tanks, etc. to the land to be irrigated by means of gravity flow or low-lift irrigation pumps. Well irrigation utilizes ground water resources by tapping underground aquifers through the construction of shallow open wells or deep tube wells.

Designing a water-pumping wind system is relatively straightforward. The first calculation is to determine how much water needs to be supplied, and how much power is required to pump the water from its point of supply to its point of use. Table 3 shows water requirements for rural communities and should be applicable to developing countries; Table 4 shows water requirements for farm animals based on U.S. experience.

TABLE 3.

Approximate water requirement for various purposes

Use	*Daily requirement*
Domestic	
minimum for survival	5 l/person
water carried home from distant communal supply	10 l/person
water carried home from nearby communal supply	30 l/person
one tap in each house	50 l/person
multiple tap connections	200 l/person
Livestock	
cattle	35 l/head
horses, mules and donkeys	20 l/head
sheep and goats	5 l/head
poultry	25 l/100
pigs	15 l/head
Irrigation	
including conveyance and field application losses	5 to 10 mm or 50 to 100m^3/ha

The power, P, required to pump water is given by

$$P = \dot{m}gH \qquad \text{Watts}$$

where $\dot{m}$ = mass flow, kg/s
g = acceleration due to gravity, 9.81 m/s^2
H = total head, metres

In this expression, the total head must also include friction losses in the piping system.

TABLE 4. Water Requirements for Farm Animals

Effect of External Temperature on Water Consumption

Water Consumption of Hogs
(Pounds per Hog per Hour)

Temperature (°F.)	75-125 lb. hogs	275-380 lb. hogs	Pregnant Sows
50	0.2	0.5	0.95
60	0.25	0.5	0.85
70	0.30	0.65	0.80
80	0.30	0.85	0.95
90	0.35	0.85	0.90
100	0.60	0.85	0.80

Water Consumption of Pigs
(Pounds of Water per Day)

Conditions	
Body Weight—30 lbs.	5-10
Body Weight—60-80 lbs	7
Body Weight—75-125 lbs	16
Body Weight—200-380 lbs	12-30
Pregnant Sows	30-38
Lactating Sows	40-50

Water Consumption of Dairy Cows
(Gallons per Day per Cow)

Temperature	Lactating Jerseys	Lactating Holsteins	Dry Holsteins
50	11.4	18.7	10.4
50-70	12.8	21.7	11.5
75-85	14.7	21.2	12.3
90-100	20.1	19.9	10.7

Water Consumption of Chickens
(Gallons per 100 Birds per Day)

Conditions	
1-3 weeks of age	0.4-2.0
3-6 weeks of age	1.4-3.0
6-10 weeks of age	3.0-4.0
9-13 weeks of age	4.0-5.0
Pullets	3.0-4.0
Nonlaying hens	5.0
Laying Hens (moderate temperatures)	5.0-7.5
Laying Hens (temperature 90°F)	9.0

Water Consumption of Hens
(Milliter per Bird per Day)

Temperature	White Leghorn	Rhode Island Red
70	286	294
80	272	321
90	350	408
100	392	371
70	222	216
70	246	286

Water Consumption of Growing Turkeys
(Gallons per 100 Birds per Week)

Conditions	
1-3 weeks of age	8- 18
4-7 weeks of age	26- 59
9-13 weeks of age	62-100
15-19 weeks of age	117-118
21-26 weeks of age	95-105

Water Consumption of Sheep
(Pounds of Water per Day)

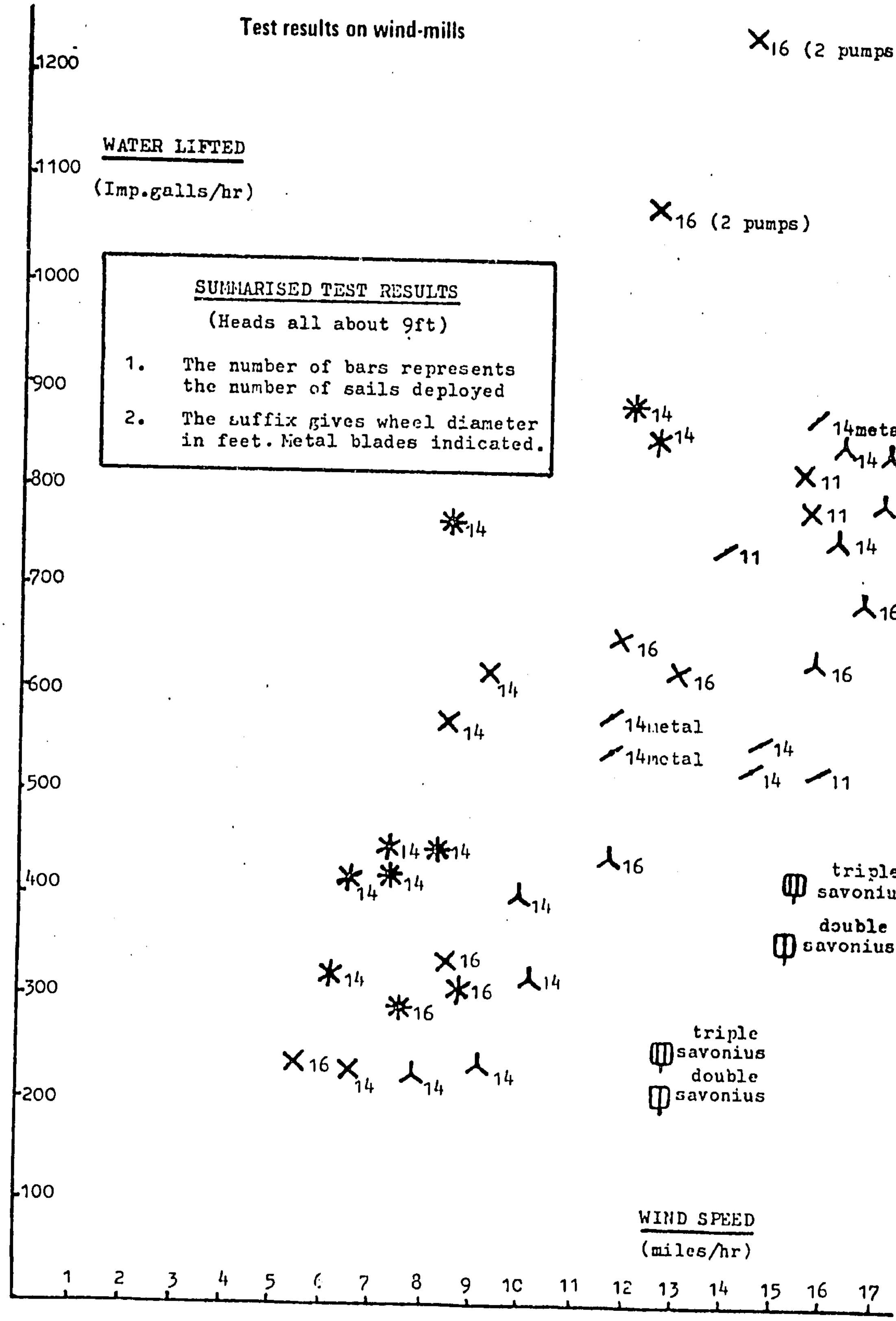
On range or dry pasture	5-13
On range (salty feeds)	17
On rations of hay and grain or hay, roots and grains	0.3-6
On good pasture	Little (if any)

Water Consumption of Cattle

Class of Cattle	Conditions	Pounds per Day
Holstein calves (liquid milk or dried milk and water supplied)	4 weeks of age	10-12
	8 weeks of age	13
	12 weeks of age	18-20
	16 weeks of age	25-28
	20 weeks of age	32-36
	26 weeks of age	33-48
Dairy Heifers	Pregnant	60-70
Steers	Maintenance ration	35
	Fattening ration	70
Range Cattle		35-70
Jersey Cows	Milk Production 5-30 lbs/day	60-102
Holstein Cows	Milk Production 20-50 lbs/day	65-182
	Milk Production 80 lbs/day	190
	Dry	90

SOURCE: Water, *Yearbook of Agriculture*, U.S. Department of Agriculture *1955*.

Some interesting results from the field are presented by Fraenkel [11] in the chart shown below. The 16 ft. diameter four bladed sailwing rotors driving twin pumps clearly perform much better than the other systems. The vertical axis Savonius rotors are notably inefficient.



Example 3

Suppose we want to irrigate 4 hectares of land, supplying water at a rate of 100 mm of water per month throughout the year. A water source is available that can provide water at a maximum rate of 30 gpm. The water must be lifted a height of about 6 metres. The monthly average windpower is shown below in Watts/square metre.

J	F	M	A	M	J	J	A	S	O	N	D	Ave
148	106	155	141	123	115	75	74	86	120	132	127	114

What size wind system is required

Solution

The amount of water to be supplied each month is given by 4 X 10,000 m^2 X 100 mm = 4000 m^3/month.

Assume a 30 day month, the average rate at which water is to be pumped, $\dot{m}$, is

$$\dot{m} = \frac{4000 \times 1000}{30 \times 24 \times 3600} = 1.543 \text{ kg/s}$$

This water must be pumped against a head of about 6.6 meters (adding 10% for head losses); the power delivered to the fluid is therefore

$$P = 1.543 \times 9.81 \times 6.6 = 99.9 \quad \text{Watts}$$

We assume that a pump is available with an efficiency of about 60%, and that the coefficient of performance of a simple water pumping rotor is about 10%. Then the wind power required to pump the water is given by:

$$\frac{99.9}{0.6 \times 0.1} = 1665 \quad \text{Watts}$$

The site wind data is given in Watts/m^2. The rotor area is therefore given by dividing the power required, 1665 Watts, by the average monthly windpower in Watts per square meter. Based on the average monthly figure of 114 W/m^2 the required rotor area is

$$A = \frac{1665}{114} = 14.6 \text{ m}^2$$

rotor diameter = 4.3 metres

Based on the minimum monthly figure of 74 W/m^2 the required rotor area is

$$A = \frac{1665}{74} = 22.5 \text{ m}^2$$

rotor diameter = 5.4 metres

It now becomes a matter of economic analysis and judgement whether to select the larger or smaller system. It will depend on the kind of crop under irrigation, whether auxillary power sources are available, the influence of seasonal rains, and the relative cost of the different size wind systems.

Electric Power Generation

Wind electric systems are generally low-solidity designs that operate at high tip speed ratios. Some of the early wind turbines used direct-drive generators where the generator armature turned at the same speed as the rotor. However, low speed generators, although robust and durable, are heavy and expensive. Modern wind electric systems usually have a gear system designed to gear up the rotor speed to a higher level. This permits the use of a smaller, lighter, less costly generator, but this saving is offset by the cost and maintenance requirements of the transmission system.

Generators installed in wind electric systems can produce either direct current (DC) or alternating current (AC). Alternating current is generated in an AC generator or alternator. The frequency of the generated current is governed by the rotational speed of the generator. To produce a constant frequency output the wind turbine must therefore spin at a constant speed even when the wind velocity is changing. This is accomplished by automatically altering the pitch of the blades; however, this is an expensive mechanism for small wind-electric systems.

Generators used to produce AC power at the same frequency as the utility supply are called <u>synchronous generators</u>. This type of system increases the complexity of the blade control mechanism and thus the cost of the wind machine. On very large wind turbines, synchronous generators are a practical concept. Generators that produce a constant frequency output under variable speed conditions are under development. These generators are called <u>field modulated generators</u>.

Generation of direct current, in the past, usually involved generation of AC inside the generator, then conversion to direct current by means of brushes and a commutator. The method commonly used now is to rectify the AC output of an alternator to direct current. This technique eliminates the need for brushes and a commutator and takes advantage of the superior low-speed characteristics of alternators. The three basic generator configurations are shown schematically in the figure below.

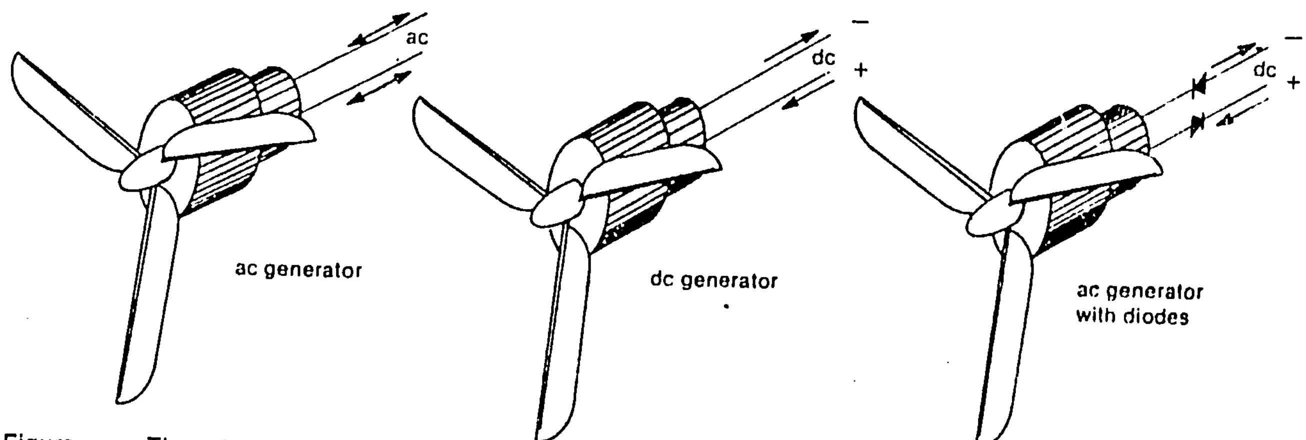


Figure Three types of generators.

Output Regulation

Generally, three methods are used for regulating or controlling the electric output of the generator:

1. Voltage regulators are used on field wound units to control the strength of the field, which in turn controls the output voltage.

2. Voltage controllers may be used on permanent magnet units to adjust voltage levels according to the output of the generator and the needs of the system.

3. No regulation at all. The output of the permanent magnet generator is used as is, while that of the wound field is fed back to the field either directly, or through a resistor to give a variable-strength field according to the strength of the generator output.

Figure is a schematic wiring diagram for a simple DC system with a back up generator. The upper load monitor senses situations when the wind system generates more power than the batteries and loads A and B can accomodate, and responds by switching in load C. This load could be a resistance heater heating water, another battery bank, or any load that can take the excess power. The other load monitor is coupled to an automatic starting system for the auxiliary generator. When this monitor senses a low-voltage condition, which could occur during periods of light winds and heavy energy demand, the monitor starts up the generator to supply the load and to charge the batteries.

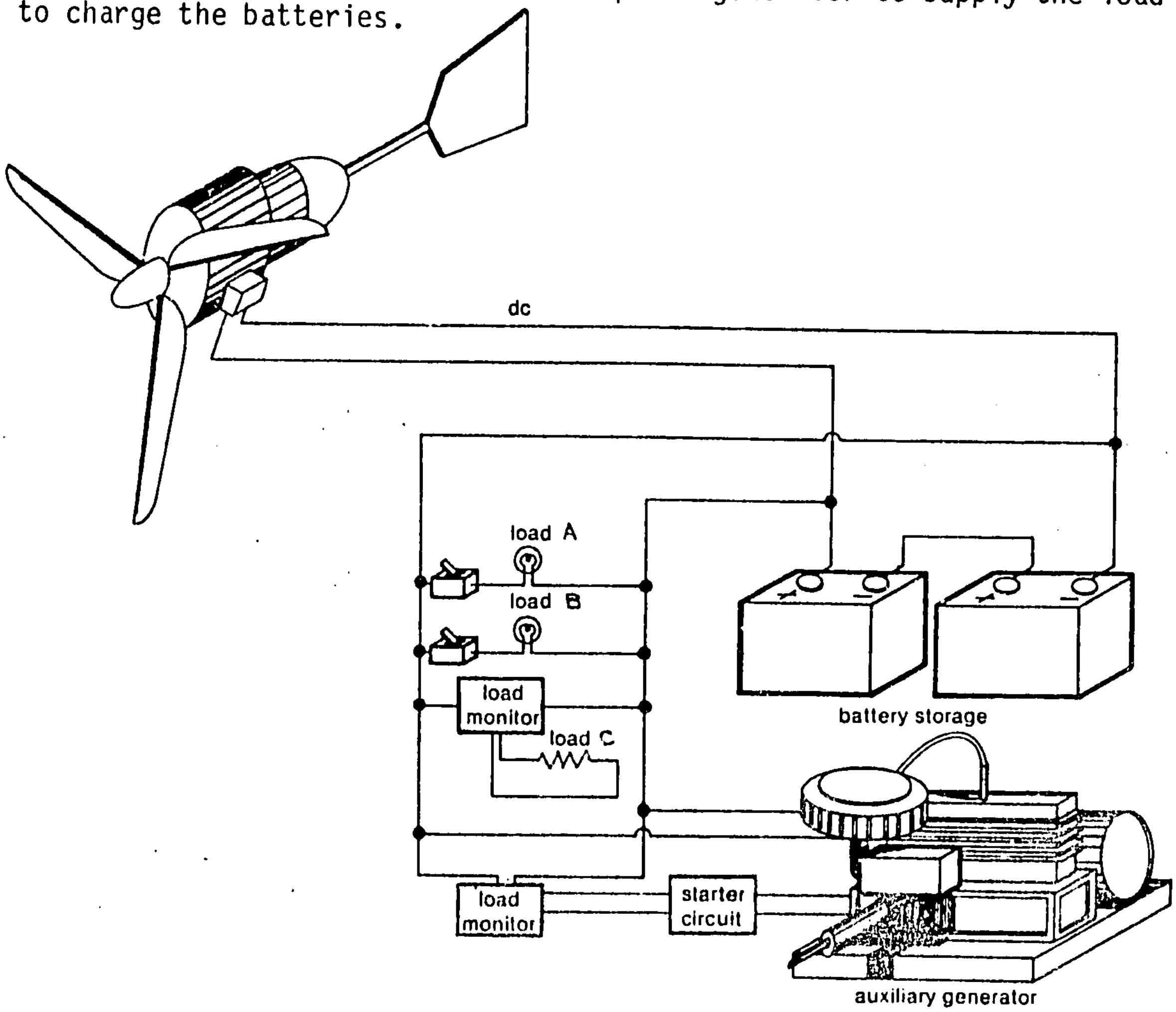


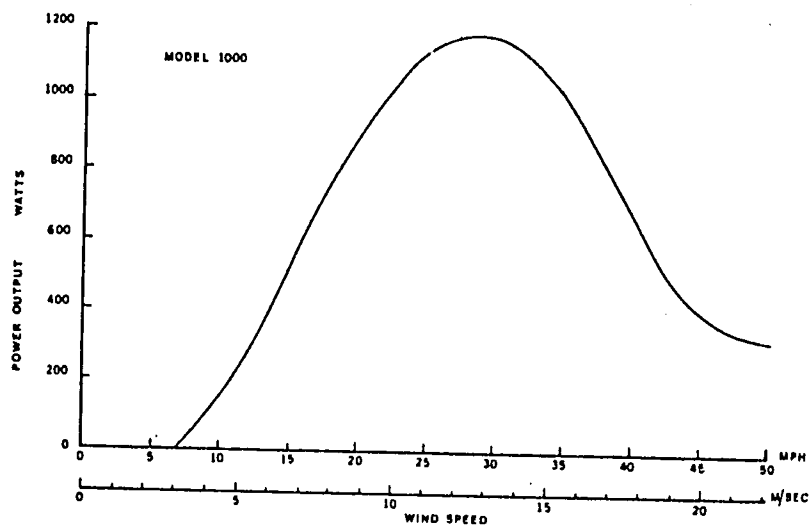
Figure Complete wind-electrical system with backup generator.

Problems

1. A wind electric machine is rated 2 kW at a windspeed of 10 m/s. The cut-in windspeed is 3 m/s and the furling or cut-out windspeed is 20 m/s. The mean annual windspeed at the site is 6 m/s. Assume that the output increases linearly between windspeeds of 3 m/s and 10 m/s and then remains constant until the machine cuts out. If the windspeed follows a Rayleigh distribution estimate the annual energy production in kWh/yr.

2. If the machine in problem 1 costs $3,600 calculate the cost of energy produced if the system lifetime is 15 years. Assume the machine is financed with a loan charged at 10% per annum. The system includes an inverter and batteries costing an additional $3,000. Annual operation and maintenance charges are estimated at $200 per year.

3. The figure below shows the output from the Sencenbaugh Model 1000 wind generator as a function of windspeed. The system is rated 1 kW at 9.8 m/s. How much energy does this system produce annually if site mean annual windspeed is 8 m/s ?



4. The data below pertain to the VITA designed multibladed windmill shown on page Determine the coefficient of performance and the tip speed ratio at the operating point indicated.

Windspeed	4 m/s
Rotor speed	21 rpm
Rotor torque	8.8 m-kg_f

The rotor diameter is 4 metres.

5. Show analytically that for a Rayleigh distribution the most frequently occurring windspeed, V_f, occurs at 80% of the mean annual windspeed, $\bar{V}$.

6. Show analytically that for annual windspeeds following a Rayleigh distribution, the energy available in the wind peaks at a windspeed equal to 1.6 times the mean annual windspeed.

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BIOGAS

Biogas is produced by the anaerobic digestion of biomass material. Animal wastes, when fermented by methane-forming bacteria in the absence of air, will produce over a period of a month approximately 30 - 60 litres of gas per kilogram of dung. The gas, which is predominantly methane, can be used for heating, lighting, cooking, and for operating gasoline or diesel engines.

There is now considerable interest in this simple technology. There are reportedly 7 million biogas units in China (including the world's largest biogas plant which generates 90 kW of electrical power), 90,000 units in India, 30,000 in Korea, 9,000 in Taiwan, over a thousand in Nepal, and lesser numbers in Japan, Philippines, Vietnam, Indonesia, Thailand, Pakistan, Bangladesh, and Sri Lanka, as well as throughout Africa and Central and South America.

The Digestion Process

In anaerobic digestion, organic waste is mixed with large populations of microorganisms under conditions in which air is excluded. Under these conditions, bacteria grow which are capable of converting the organic waste to carbon dioxide (CO_2) and methane (CH_4). The anaerobic conversion to methane yields relative little energy to the microorganisms themselves. Thus, their rate of growth is low and only a small portion of the degradable waste is converted to new bacteria, most is converted to methane. Since this gas is insoluble it escapes from the digester fluid where it can be collected and used as fuel. As much as 80 - 90% of the degradable organic portion of a waste can be stabilized in this manner, even in highly loaded systems.

Anaerobic treatment of complex organic materials is normally considered to be a two-stage process, as indicated in Figure 1.

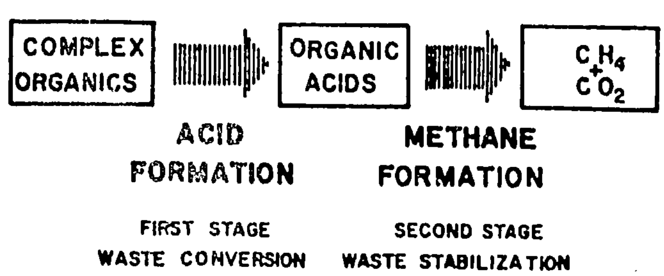


Figure 1. The two stages of anaerobic methane digestion.

In the first stage, there is no methane production. Instead the complex organics are changed in form by a group of bacteria commonly called "acid-forming bacteria". Complex materials such as fats, proteins, and carbohydrates are converted to more simple organic materials — principally fatty acids. Acid-forming bacteria bring about these initial transformations to obtain small amounts of energy for growth and reproduction. This first phase is required to transform the organic matter to a form suitable for the second stage of the process. This is the stage that produces the methane.

During the second stage the organic acids are converted by a special group of bacteria into carbon dioxide and methane. The methane-forming bacteria are strictly anaerobic and even small amounts of oxygen are harmful to them. There are several types of these bacteria, and each type is characterized by its ability to convert a relatively limited number of organic compounds into methane. Consequently, for complete digestion of the complex organic materials, several different types are required. The most important variety which utilizes acetic and proprionic acid, grows quite slowly and hence must be retained in the digester for four days or longer; its slow rate of growth (and low rate of acid utilization) usually represents one of the rate-limiting steps around which the anaerobic process must be designed.

The methane-forming bacteria have proved to be very difficult to isolate and study, and relatively little is known of their basic biochemistry. Figure 2 indicates schematically the general biochemical anaerobic digestion process.

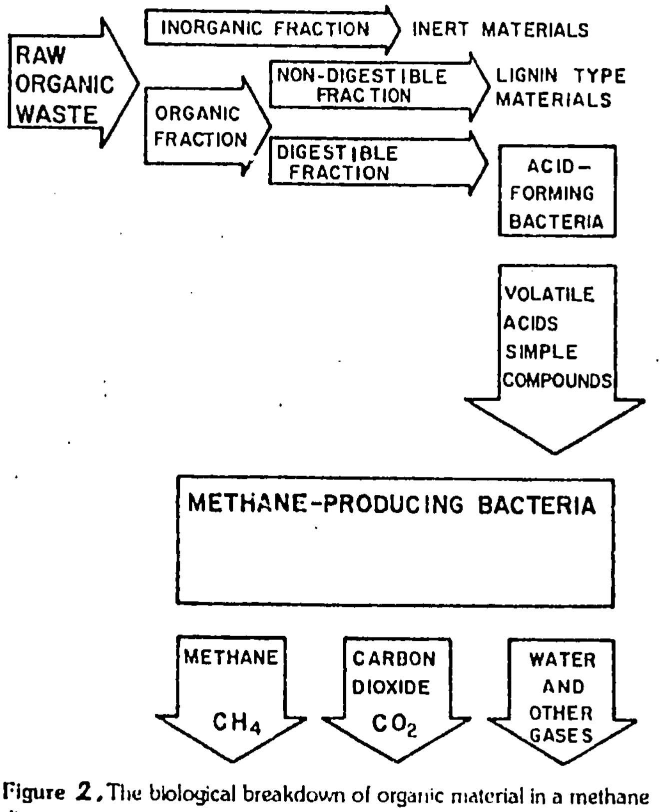


Figure 2. The biological breakdown of organic material in a methane digester.

The amount of gas produced and its composition will depend on the characteristics of the feed material and the conditions under which the digester operates.

Animal manures, when slurried with water, are excellent feed materials and under optimal conditions produce good quality gas. Tables 1 and 2 indicate the estimated gas production from the dung of cattle, pigs and poultry. These figures (Table 1) imply biogas generation rates as follows:

Dairy cattle	30	litres gas/kg dung
Beef cattle	42	litres gas/kg dung
Swine	53	litres gas/kg dung
Poultry	116	litres gas/kg dung

Carbon - Nitrogen Ratio

The ratio of carbon to nitrogen (C:N) in the digester feed critically affects the operation of the digester and the composition of the gas. Gas production can be increased by supplementing substrates that have a high carbon content with substrates containing nitrogen, and vice versa. If the C:N ratio is too high, the process is limited by the availability of nitrogen; if the C:N ratio is too low, ammonia may be produced in quantities large enough to be toxic to the bacterial population. For optimum production of methane the C:N ratio should be about 30:1. Tables 3 and 4 show C:N ratios for many common animal and agricultural wastes. Further qualitative information on the influence of the C:N ratio on digester performance is shown in Table 5.

It should be noted that in order to evaluate the feasibility of using a particular biomass material for biogas production, both the C:N ratio and the biodegradability need to be known. The wide range of values reported in the literature is an indication that a degree of caution is advisable in designing digesters utilizing waste materials for which no direct experimental or operating data are available.

pH Level

The bacterial population in anaerobic digesters is sensitive to pH levels. The optimal pH range lies between 7.0 and 7.2 but gas production will proceed satisfactorily between 6.6 and 7.6. When the pH falls below 6.6 there is an inhibitory effect on gas production. Acid conditions below about 6 will suppress the methanogenic bacteria and shut down gas production. Under normal operating conditions, however, the biochemical reactions tend to automatically maintain the pH level in the proper range.

During the start up of the digester acidic conditions may sometimes occur since the acid-forming bacteria at first multiply much more rapidly than the methanogenic bacteria. To alleviate this problem artificial means to raise the pH to about 7 may be required. Bicarbonate of soda is reportedly an effective anti-acid agent. It should be mixed with the feed slurry: about 10 grams of bicarbonate to 30 litres of slurry.

Table 1 Estimated manure and bio-gas production from animal wastes (1)

	Dairy Cattle	Beef Cattle	Swine	Poultry
Manure production (lb/day/1000 lb live weight)	85	58	50	59
Volatile solids (lb dry solids/day/1000 lb live weight)	8.7	5.9	5.9	12.8
Digestion efficiency of the manure solids (%)	35	50	55	65
Bio-gas production				
(ft^3/lb VS added)	4.7	6.7	7.3	8.6
(ft^3/1000 lb live weight/day)	40.8	39.5	43.1	110.9

(lb x 0.454 = kg: ft^3/lb x 0.062 = m^3/kg)

Table 2. *Gas yield of some common fermentation materials.* (17)

Material	*Amount of gas produced per tonne of dried material in cubic metres*	*Percentage content of methane*
General stable manure from livestock	260-280	50-60
Pig manure	561	
Horse manure	200-300	
Rice husks	615	
Fresh grass	630	70
Flax stalks or hemp	359	59
Straw	342	59
Leaves from trees	210-294	58
Potato plant leaves and vine etc.	260-280	
Sunflower leaves and stalks	300	58
Sludge	640	50
Waste water from wine or spirit making factories	300-600	58

Table 3. Nitrogen Content and C/N Ratio[a] (2)

Material	Total Nitrogen (% dry weight)	C/N Ratio
Animal wastes		
Urine	16.0	0.8
Blood	12.0	3.5
Bone meal	—	3.5
Animal tankage	—	4.1[b]
Dry fish scraps	—	5.1[b]
Manure		
Human feces	6.0	6.0–10.0
Human urine	18.0	—
Chicken	6.3	15.0
Sheep	3.8	
Pig	3.8	
Horse	2.3	25.0[b]
Cow	1.7	18.0[b]
Steer	1.35	25.3
Sludge		
Milorganite	—	5.4[b]
Activated sludge	5.0	6.0
Fresh sewage	—	11.0[b]
Plant meals		
Soybean	—	5.0
Cottonseed	—	5.0[b]
Peanut hull	—	36.0[b]
Plant wastes		
Green garbage	3.0	18.0
Hay, young grass	4.0	12.0
Hay, alfalfa	2.8	17.0[b]
Hay, blue grass	2.5	19.0
Seaweed	1.9	19.0
Nonleguminous vegetables	2.5–4.0	11.0–19.0
Red clover	1.8	27.0
Straw, oat	1.1	48.0
Straw, wheat	0.5	150.0
Sawdust	0.1	200.0–500.0
White fir wood	0.06	767.0
Other wastes		
Newspaper	0.05	812.0
Refuse	0.74	45.0

Notes: a. From "Anaerobic Digestion of Solid Wastes" (Klein) and *Methane Digesters for Fuel Gas and Fertilizer* (Merrill and Fry).

b. Nitrogen is the percentage of total dry weight while carbon is calculated from either the total carbon percentage of dry weight or the percentage of dry weight of nonlignin carbon.

Table 4. Approximate values for the carbon/nitrogen ratios of some of the common materials used for biogas pits. (17)

Material	Carbon as a percentage of total weight %	Nitrogen as a percentage of total weight %	Carbon/nitrogen ratio %
Dry straw	46	0.53	87:1
Dry rice stalks	42	0.63	67:1
Maize stalks	40	0.75	53:1
Fallen leaves	41	1.00	41:1
Soya bean stalks	41	1.30	32:1
Wild grass: i.e. weeds etc. (in China often narrow, thin leaved)	14	0.54	27:1
Peanut vine stalks	11	0.59	19:1
Fresh sheep manure	16	0.55	29.1
Fresh cow/ox manure	7.3	0.29	25:1
Fresh horse manure	10	0.42	24:1
Fresh pig manure	7.3	0.60	13.1
Fresh human manure	2.5	0.85	2.9:1

Table 5 C/N Ratio and Composition of Bio-gas[a] (2)

Material	Gas: Methane	CO_2	Hydrogen	Nitrogen
C/N low (high nitrogen)	little	much	little	much
Blood				
Urine				
C/N high (low nitrogen)	little	much	much	little
Sawdust				
Straw				
Sugar and starch				
potatoes				
corn				
sugar beets				
C/N balanced (near 30:1)	much	some	little	little
Manures				
Garbage				

Notes: a Adapted from *Methane Digesters for Fuel Gas and Fertilizer* (Merrill and Fry).

Temperature Effects

The temperature of an aerobic digester strongly effects its performance. The optimum temperature for mesophilic anaerobic digestion is about 35°C. Gas production and digestion will proceed at lower temperatures but the rate of digestion is reduced. An example of the relationship between residence time, temperature, and gas production that was obtained in one study is shown in Figure 4. As a rough approximation, for every ± 5°C temperature change (mean daily ambient) from 25°C, the daily gas production will vary by about 20%. This applies to a temperature range between 10°C and 35°C. Below 10°C gas production drops off rapidly. As long as the digester volume is not too small, there is sufficient thermal capacity to smooth out diurnal variations in ambient temperature. The slurry temperature will be approximately equal to the mean 24 hour ambient temperature. Small digesters (less than 1 cubic metre) can be expected to show stronger temperature variations throughout the day.

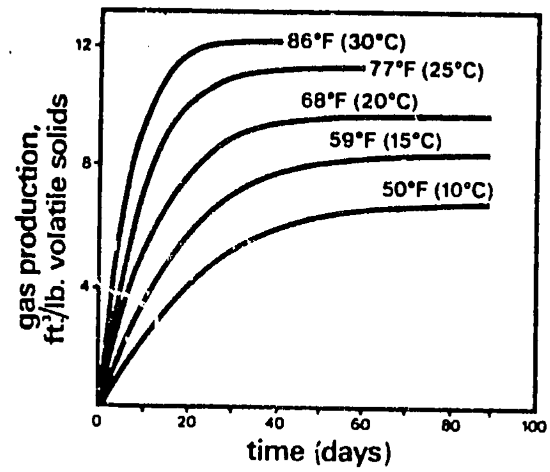


Figure 3. Bio-Gas Production as Related to the Temperature of the Digester and the Time of Digestion (1)

It may be difficult, especially in colder climates and in the winter, to maintain mesophilic temperatures in the anaerobic digestor. Supplemental heat can be used to maintain the necessary microbial activity. Note that this in an ideal application of a simple, cheap, flat-plate solar collector. Where it is found necessary to heat the digestor, it should be well insulated.

In summary, factors that cause poor digestor performance or even complete failure include:

1. Sudden change in temperature (either due to climatic changes or failure of the heating system if one is used).
2. Sudden change in the rate of loading.
3. Sudden change in the nature of raw materials.
4. Presence of toxic materials
5. Extreme drop in pH (acidic conditions)
6. Slow bacterial growth during start-up.

SLUDGE UTILIZATION

The residue from a methane generation process will contain lignin, lipids, material protected from bacterial degradation, synthesized microbial cells, metabolic degradation products such as volatile acids and other soluble compounds, inert material in the original waste, and water. The residue will be a liquid with a solids concentration of 4-8 percent. Anaerobically digested sludges can be stored and spread on land with less risk of creating conditions for odor and insect breeding problems than exists with similar handling procedures for untreated or partially treated organic waste materials.

Methane generation conserves the nutrient elements needed for crop production. The only materials removed from the system, other than in the sludge, are the generated gases. Practically all of the nitrogen present in the waste entering a digester is conserved. If the sludge is properly stored, and when applied to soils is immediately incorporated to reduce the loss of nitrogen by volatilization, most of the nitrogen present in plant residues can be available for use by the growing plants. To minimize ammonia nitrogen losses, the digested sludge should be stored in lagoons or tanks which present a minimum of surface area for ammonia evaporation. Other chemical elements contained in the added waste will be conserved in the digested sludge.

The end result of applying digested sludge on soils is the same as that resulting from the application of any other kind of organic matter. The humus materials can improve soil physical properties such as aeration, moisture holding capacity, increase cation exchange capacity, and improve water infiltration capacity. The sludge can serve as a source of nutrients for crops grown on the soil. When human, animal, and agricultural wastes are used for methane generation, there is little likelihood that any items in the sludge will cause adverse conditions to the crops or to animals fed the crops grown on land where digested sludge is used as a fertilizer.

The application of the digested sludge to the crop land should be done in an environmentally sound manner, generally at rates consistent with the need of the crops being grown. Runoff that can contaminate surface waters and loadings that result in ground water contamination must be avoided.

An important aspect to be considered with methane generation is that the total volume of sludge that must be handled for final disposal is equal to or greater than the initial amount of dry wastes that are to be digested because of the liquid added to obtain a solids concentration that can be mixed. Although considerable solids decomposition occurs in a digester, approximately 50%, little reduction of the total volume to be handled results.

Gas Utilization

Pure methane is a colorless and odorless gas which generally constitutes between 60 and 70% of the gas produced by anaerobic digestion. The other constituents are primarily carbon dioxide and smaller quantities of other gases such as hydrogen sulphide and hydrogen. Biogas burns with a blue flame and has a heat value of about 600 - 700 Btu/ft^3 or 22 - 26 MJ/m^3.

Many options exist for utilizing the digester gas. It can be used directly in gas-burning appliances for heating, cooking, lighting, and refrigeration, or it can be used as a fuel in internal combustion engines. It may be necessary to remove the hydrogen sulphide from the biogas. This may be accomplished by passing the biogas through a box filled with iron filings (18). If a gas with a higher heating value is required, the carbon dioxide may be removed by bubbling the biogas through limewater. Tables 6 and 7 indicate the amounts of biogas required for various applications and for utilization in internal combustion engines.

Gasoline engines will run on biogas producing about 80% of their rated power. Diesel engines can be converted to dual-fuel engines, again producing about 80% of their rated output. Biogas substitutes for about 80 - 90% of the diesel fuel: anywhere from 2 - 4 cubic metres of biogas will substitute for a litre of fuel [23, 24]. Biogas consumption falls in the range of about 0.35 - 0.5 cubic metres per HP-hour [23, 24]. For electrical power generation, biogas consumption is generally between 0.6 to 1.1 cubic metres per kWh electric.

Table 6. Quantities of bio-gas required for a specific application (1)

Use	Specification	Quantity of Gas Required ft^3/hr	m^3/hr
Cooking	2" burner	11.5	0.33
	4" burner	16.5	0.47
	6" burner	22.5	0.64
Gas lighting	per mantle	2.5-3.0	0.07-0.08
	2 mantle lamp	5	0.14
	3 mantle lamp	6	0.17
Gasoline or diesel engine[(a)]	Converted to bio-gas	16-18 per hp	0.45-0.51 per hp
Refrigerator	per ft^3 capacity	1	0.028
Incubator	per ft^3 capacity	0.45-0.6	0.013-0.017
Gasoline	1 liter	47-66[(b)]	1.3 - 1.9
Diesel fuel	1 liter	53-73[(b)]	1.50-2.07[(b)]
Boiling water	1 liter	2.2[(c)]	0.62[(c)]

[(a)] Based on 25 percent efficiency

[(b)] Absolute volume of bio-gas needed to provide energy equivalent of 1 liter of fuel

[(c)] Absolute volume of bio-gas needed to heat 1 liter of water to boiling

BIO-GAS CONSUMPTION RATE PER HOUR IN CUBIC FEET AND AIR REQUIRED FOR OPERATION OF DIFFERENT TYPES OF ENGINES (4)

NAME OF THE ENGINE	FUEL SYSTEM	CYLINDER AND CYCLE DETAILS OF ENGINE	BRAKE HORSE-POWER	HORIZ. OR VERT.	SPEED IN RPM	DIA. OF THE GAS SUPPLY LINE	GAS PRESSURE IN WATER COLUMN INCHES REQUIRED FOR
Stuart engine coupled with 250 watts, 110 volts. D.C. generator made in England	petrol engine	Single-cylinder 4-cycle	1 bhp	Vertical	High Speed 1,800	3/8"	2" to 4"
Onan engine coupled with 1 K.V.A. 110 volts. A.C. generator made in U.S.A.	"	"	3 bhp	Vertical	High Speed 1,500	"	"
Kubota engine made in Japan.	powerine engine	single-cylinder 2 stroke	5 bhp	Horizontal	Low 900	1/2"	1" to 3"
Kubota engine made in Japan.	Kerosene or oil engine	single-cylinder 4-cycle	10 bhp		Low 600 to 700	3/4"	1" to 3"

SYSTEM OF OF COOLING	GAS CONSUMPTION (CU. FT.) PER HR. NORMAL LOAD	FULL LOAD	AIR IN CU. FT. REQUIRED FOR NORMAL LOAD	FULL LOAD	WORKING EFFICIENCY OF THE ENGINE LIQUID FUEL	BIO-GAS
Air-cooled	16.5	18.5	95	108	250 watts	225 watts
"	48.0	58.0	264	318	1,000 watts	850 watts
Water-cooled	82.0	95.0	–	–	5 hp	4.03 hp
"	155	175	–	–	10 hp	8.2 hp

TABLE 7.

INDIAN GOBAR GAS PLANTS

A typical Indian gobar (cow-dung) gas plant is shown below.

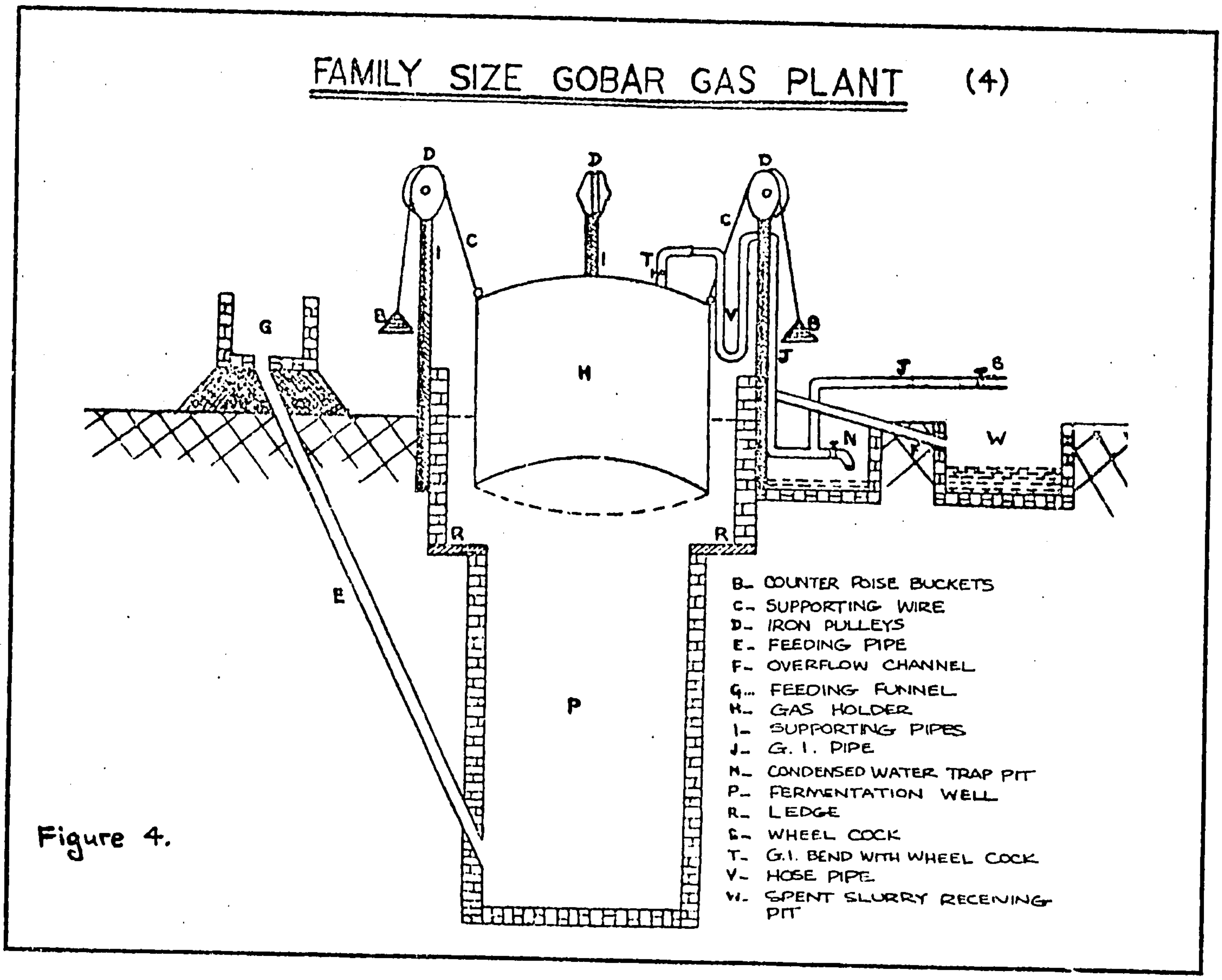


Figure 4.

The cow-dung (gobar) slurry enters the digestor through the loading tank, G, and spent slurry overflows into pit W. The flow is by gravity: the base of tank G being a little above the overflow pipe leading to pit W. The biogas is collected under the mild steel or galvanized iron drum, H, and taken off, via a flexible pipe, V, through a water trap and a flame trap. The pressure of the gas, in the system shown in Figure 4, may be controlled by adjusting the counter poise weights B.

The design of the gas - holder system has recently been improved as shown in Figure 5. This design removes the need for a flexible gas pipe— generally a source of leaks. The structure of the metal drum is shown in greater detail in Figure 6. Another innovation is the two-chamber system shown in Figure 7. This produces rather better digestion and therefore improved gas production.

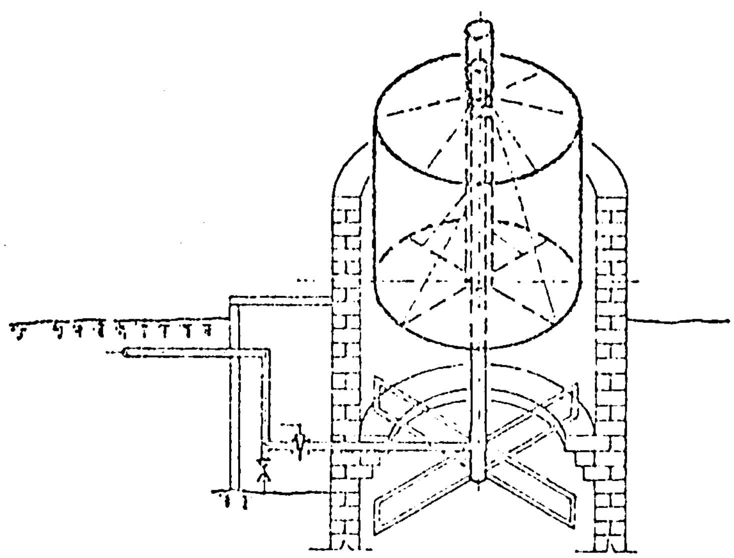


Figure 5.

THE GAS DRUM (9)

A gas drum is generally made of mild steel sheeting or galvanized-iron sheeting of any thickness from SWG 16 to SWG 30 (1.63 - 0.32 mm). India favours the heavier gauge further strengthened with angle iron or iron rods; Taiwan often uses gauge 30 fixed to a rectangular wooden framework for rigidity. (their fermenting tanks too are often rectangular). There is no point in making the drum too heavy and then having to counterbalance it. Except in plants where the whole day's gas has to be stored for use at one time, it is customary to make the drum one-third the depth of the pit, and its diameter 10 cm less than that of the pit. A 2½" G.I. pipe almost double the height of the drum passes through it and is welded to the top centre; its lower end is held firmly by thin iron tie-rods. The top of this pipe (called the "slide pipe" is closed). A few holes are drilled in the pipe, inside the drum and just below the top. Note that the 2½" G.I. pipe is suitable for small family-size plants; larger installations of course require surdier pipes of bigger diameter.

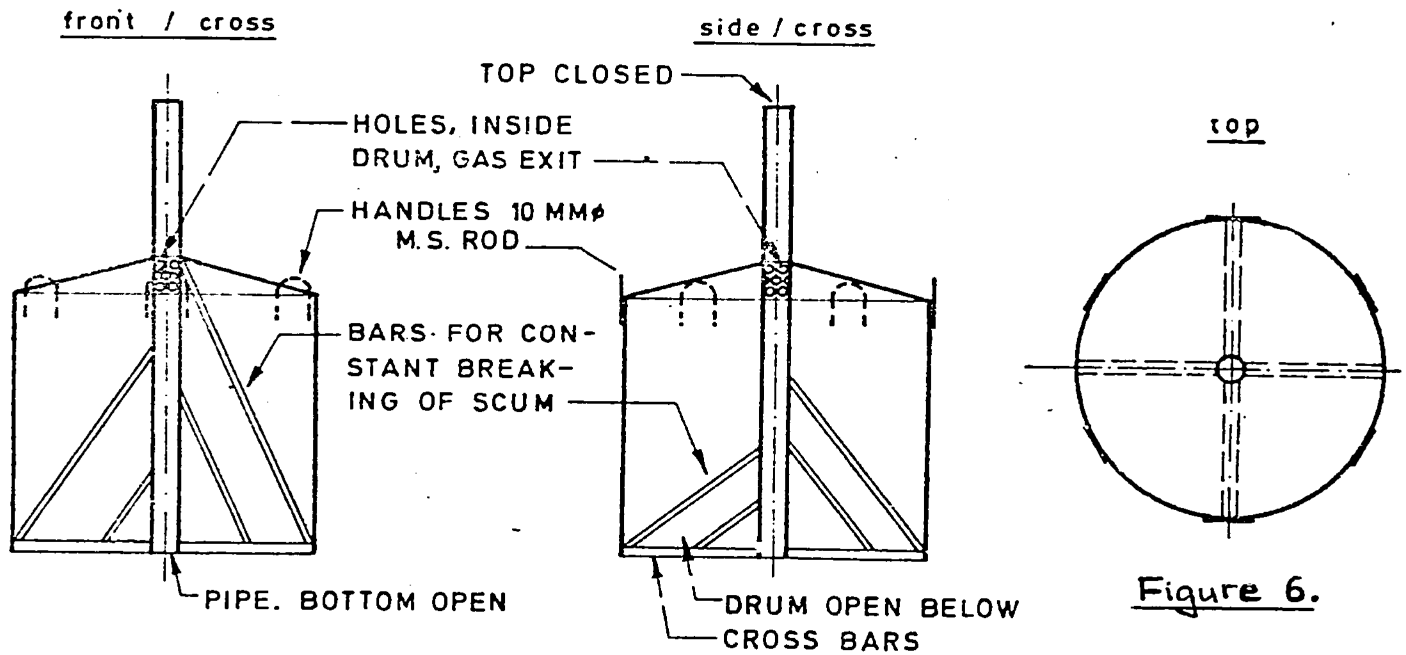
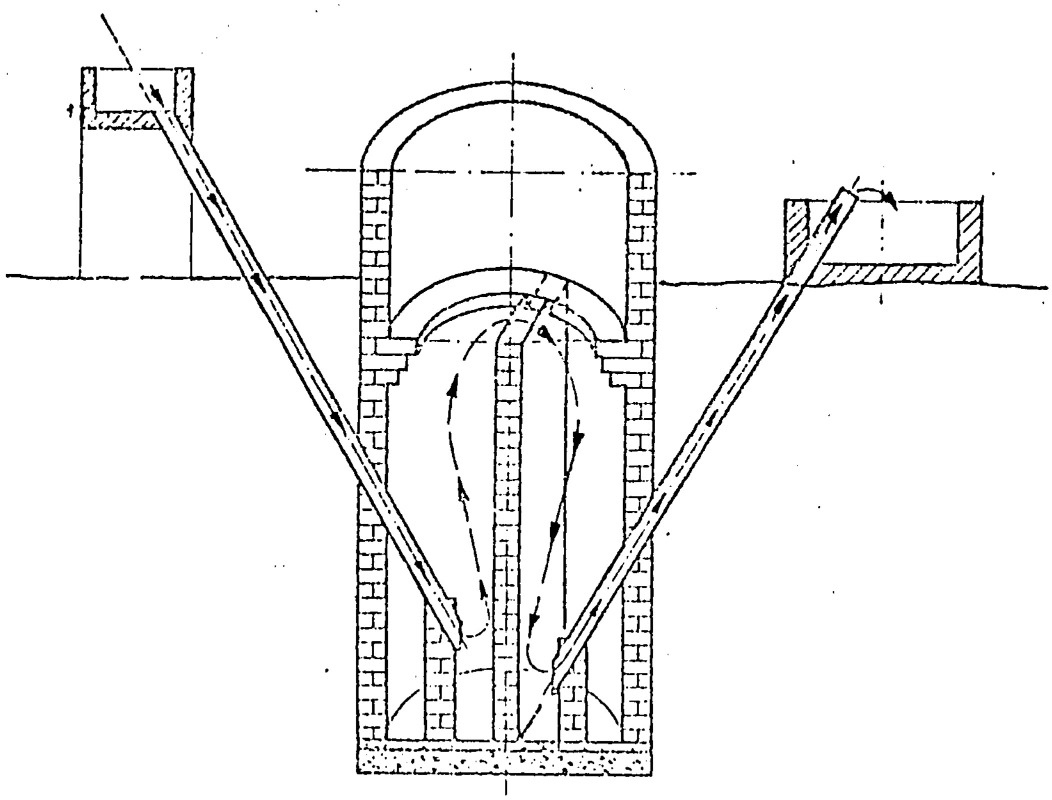


Figure 6.

This pipe slips onto a 2" G.I. pipe held perfectly vertical in the dead center of the pit by two strong cross-bars, or imbedded in a ferro-concrete beam. As the drum fills and empties it rides up and down on this center guide pipe, the top of which should stand at least half of the height of the drum above overflow level. Make sure the drum is perfectly airtight. Attach handles to the outside on top for lifting. Weld for metal rods from the tie-rods to the slide pipe sloping sideways. Stagger them. With a slight rotation of the drum, or with its mere rise and fall, these break up a troublesome scum that forms on the slurry and tends to harden and prevent the passage of gas. Protect the outside and inside of the drum with a coat of paint.



TWO CHAMBER CONSTRUCTION

Figure 7.

Gobar digestors range from about 2 - 12 cubic metres in size - this is the volume of the pit. These units would produce about 1 - 6 m of gas per day - an amount of gas approximately equal to one half the volume of the pit. Table 8 gives some indication of the size of these small scale systems and the number of people served by each.

Size number	1	2	3	4
Pit diameter, m	1.2	1.5	1.75	2.0
Pit depth, m	3.3	3.4	3.8	3.82
Volume, m^3	3.73	6.0	9.0	12.0
Feed (slurry), l/day	90	150	225	300
Gas production, m^3/day	1.5-2	2.5-3	3.5-4.5	4.5-6
Cattle required	2-3	4-5	6-7	8-9
Persons served	3-4	5-7	8-10	11-14

Table 8
Gobar Gas Plant Capacities

These figure are approximate and assume:
1. A 40 day retention time.
2. Digester temperatures of 30°C or more.
3. Per capita gas use of 425 litres daily.
4. Feed slurry of 1 part water, 1 part cow dung.
5. Animals produce about 18 litres of dung per day.

ECONOMICS

Estimates of the cost of Indian gobar gas plants vary widely. The table below is from the KVIC booklet (ref. 3) and applies to construction in Nepal:

Capacity		1978 Cost	
m^3/day	ft^3/day	RS	$(8Rs = 1$)
2	70	2800	350
4	140	4035	500
6	210	5010	626
8	280	6000	750
10	350	7320	915
20	700	13800	1725
45	1237	22080	2760
60	2120	31200	3900
140	5000	67500	8440

Ram Bux Singh gives these figures:

Capacity		Installed Cost (1975 $)	
m^3/day	ft^3/day	In India	In U.S.
3	100	140	400
7	250	350	900
14	500	600	1800
34	1200	1500	4000
57	2000	2250	5500

According to one recent article (6) the 'typical' Nepalese gobar plant (which would be about 2 m^3/day capacity) costs about $400. A larger plant, constructed in southeast Nepal and designed to produce 14 m^3/day of biogas, cost $3000 in 1978.

An interesting communal system has been projected for the village of Pura in India (11). A biogas system generating 37 m^3/d of gas is to be employed. All village cooking can be accomplished with 26 m^3/d, leaving 11 m^3/d for water pumping and electric lighting. In addition, 3 kg/d of nitrogen will be obtained from spent slurry. The total capital costs for the system is estimated at approximately $7500 (1978), compared to the 'typical' capital costs of $10,000 for village electrification. The $7500 cost includes the cost of the biogas plant, piping, the biogas fired generator, electrical distribution, a water pump, storage tank, building costs, a cement mill to use rice-husk ash, and a calcium carbonate extraction plant.

Capacities and Costs of Biogas Plants (16)

Size of Plant[a] m^3	Size of Plant[a] ft^3	Estimated Cost as of Feb. 1975 Rupees	Approx. No. Animals Required	Grant (25% of estimated cost)	Loan (75% of estimated cost)
2	70	2,332	2-3	583	1,749
3	105	3,016	3-4	754	2,262
4	140	3,360	4-6	840	2,520
6	210	4,175	6-10	1,044	3,132
8	280	5,000	12-15	1,250	3,750
10	350	6,100	16-20	1,525	4,575
15	525	8,500	25-30	2,125	6,375
20	700	11,500	35-40	2,875	8,625
25	875	12,800	40-45	3,400	9,400
35	1,237	18,400	45-55	4,600	13,800
45	1,590	20,740	60-70	5,185	15,555
60	2,120	26,000	85-100	6,500	19,500
85	3,004	38,800	110-140	9,700	29,100
140	4,948	58,000	400-450	14,500	43,500

[a] Volume of gas produced daily.

Cost/Income Analysis of a 6-m^3/day Biogas Plant[a] (16)

Basis:	Cost of Gas Holder	Rs 1,670
	Pipeline and Appliances	450
	Civil Construction	2,056
	Total Cost of Plant	Rs 4,176
	Subsidy (25%)	1,044
	Cost Basis	Rs 3,132

	Previous Use of Cattle Dung	
Annual Working Costs	Farmyard Manure	Fuel
Interest on capital @ 15%[b]	Rs 281.88	RS 281.88
Gas holder (10-yr life) 10%	125.25	125.25
Pipeline and appliances (30-yr life) 3.3%	11.14	11.14
Civil work (40-yr life) 2.5%	38.55	38.55
Painting gas holder	100.00	100.00
Maintenance	100.00	100.00
Cost of dung as manure (14.8 tons)	592.00	-
Cost of dung as fuel (in terms of kerosene equivalent @ Rs 1.01/liter)	-	711.00
Total Costs	Rs 1,248.82	Rs 1,367.82
Annual Income		
Manure (22.4 tons @ Rs 50/ton)	1,112.00	1,112.00
Gobar gas (2,190 m^3/yr @ Rs 1.01/ liter kerosene equivalent)	1,371.30	1,371.30
Total Income	Rs 2,483.30	Rs 2,483.30

	Income	Costs	Net Income
Dung formerly used as manure	2,483.30	1,248.82	1,234.48
Dung formerly used as fuel	2,483.30	1,367.82	1,115.48

Notes:

(a)Adapted from Fernandez.

(b)Five equal payments of interest on declining balance.

(c)Note that the loan repayment of Rs 3,132 ÷ 5 = Rs 626.40 will have to be paid annually. For the first 5 years, therefore, the net income for the two bases are Rs 608.08 and Rs 489.08, respectively. However, after repayment of interest installments, the figures increase to Rs 1,516.36 and Rs 1,397.36, respectively.

Material drawn from: Fernandez, A., ed. 1976. Gobar gas plant–why and how. *Seva Vani* (May-June):15-21. Further information available from: The Director, Gobar Gas Scheme, Khadi and Village Industries Commission, Irla Road, Vile Parle (W), Bombay 400056, India.

Biogas Cheaper than Kerosene or Electricity [13]

Costs of Biogas (1976 dollars)

Item	Private Owner+	Community Scale*
Capital Cost	200	21,111
Annual Cost	31	5,589
Fertilizer Credit	23.7	1,111
Net Energy Output Per Annum	8×10^9 J	10^{12} J
Net Annual Cost	7.3	4,478
EnergyCost Per kWh(t)	0.003	0.0016
Per Bbl-equivalant	$5.29	$2.60

Three authors' estimates of biogas costs for small and large biogas plants. It is apparent that biogas at $0.01-0.02/kWh(t) is competitive on an enthalpic basis (for heating, cooking and probably pumping) with currently subsidized Indian electric power at $0.01-0.06/kWh(e) and kerosene at $0.013/kWh(t). Marginal electrical supply is estimated to cost much more, from $0.046-0.085/kWh(e).

+ From Parikh and Parikh. a plant with 1.8 m^3/d output. The annual cost is computed with a 12% cost of money, 15 year depreciation period, and $5.00 per year paint cost. The fertilizer credit assumes 52.6 kgs of nitrogen fertlizer are produced and sold annually at $0.45/kg.

* From Makhijani, for a plant with 150 m^3 average daily output and a 400 m^3 total capacity.

Note that Makhijani's scheme includes compression, land, distribution equipment and farmer extension services for the irrigation fuel biogas unit.

Sources: R Bhatia, "Economic Appraisal of Bio-Gas Units in India Framework for Social Benefit Cost Analysis", Economic and Political Weekly, August 1977. A Makhijani, Energy Policy for Rural India", Economic and Political Weekly, August 1977. K.S. and J.K. Parikh, "Mobilization and Impacts of Biogas Technologies", Energy, volume 2, no 4 December 1977.

SOCIO-ECONOMIC IMPACTS

Problems associated with individual gobar units relate primarily to the investment and operating costs for the economically least well off village members. As a low cost energy option to be used on a wide scale, biogas plants are still too expensive for most rural families. Some 70% of the Indian population earn less than $100 per year, or one-third to a quarter of the investment costs required for a gobar gas plant. In addition only about 10% of India's rural households own the 3 - 4 cattle needed to provide the manure to run the biogas plant. "Costwise this scheme (gobar gas plants) cannot be extended to a significant majority of the rural population. At best, it can only cater to the cattle-owning upper-income stratum of the rural population which accounts for not more than 8 to 10 per cent of the total rural population. In fact, it was found in a recent survey in Gujerat, one of the most successful states in the gobar gas scheme, that the average landholding size of the gobar gas plant owners was as high as 26 acres". (7)

According to Subramanian (12), Indian banks insist on ownership of 5 - 6 animals and 2 hectares of cultivated land before extending credit. Many observers feel that the solution to this problem is the construction of village scale plants. However poor villagers do not necessarily perceive the new technology as beneficial. For example, Bhatia (8), studied a community biogas plant in a village in Uttar Pradesh, designed to serve the landless as well as the landed. The only incentive to collect dung was to receive equivalent amounts of slurry. Street lighting was the free payoff for the village, while each home received cooking gas and gas for two lamps of 100 candle power. Apparently the villagers felt they were only replacing one form of cooking fuel with another, one which required longer walking by the women. The result from better fertilizers was not quickly apparent to them. The cost of the plant seemed therefore only an added expense. Bhatia argues that soft coke is a cheaper solution for cooking since few villagers actually used the gas for lighting.

A potential problem with community-based or village-scale plants concerns ownership and control. Who owns the digester? Who controls the amounts of gas and fertilizer available? How does one ensure cooperation from "rich" families who could set up their own plants and also involve poor families who spend hours collecting fuel? Some observers feel that "the installation of biogas units by rich farmers would result in an increasing tendency to have stable-bound cattle in order to augment the collection of dung and urine. This would deprive the poorer sections in rural society (landless labourers and marginal and small farmers) of a cheap source of fuel for their energy needs." (14)

A biogas plant is also likely to raise the value of dung. It is then possible that poor landless people will find it increasingly difficult to get access to this fuel. "If a biogas plant is established, then cow dung would not be used directly but would be transformed into another type of fuel and fertilizer. One could anticipate that many of the types of tensions associated with commercialization of the village economic system and with the adoption of new farm technologies would recur. As in these previous cases, the new development provides novel opportunities for gain for some and threatens to deprive others of existing income rights. In turn, these economic changes will interact with caste and status relationships and with the village political system. At present, dung is a non-marketed commodity subject to the rules and rights that govern its sharing. As with crops, labour, land, and water, dung will become a marketed and priced item subject to distribution through the market system. It is reasonable to suppose that the village cattleowners, who are likely in the main to be the wealthier farmers, will attempt to assert latent property rights to their beasts' dung. Their power and status are likely to enable them to get their way. The losers will be the poorer families whose women and children collect dung for use as a cooking fuel. They will have to seek alternative sources of fuel. Some may even lose small incomes from the sale of dried dung." (15)

The problems that may arise from the introduction of biogas plants are not entirely economic. For example, an 18-stall women's community latrine/gas digester system was installed in 1977 in an urban suburb of Kathmandu. The system was designed to generate gas while mitigating a serious sanitation problem. For both cultural and technical reasons, the gas from this digester was less valuable than anticipated. The Hindu people of the village considered food cooked on any sewage byproduct to be ritually polluted, and could therefore use it only for cooking animal food or for lighting. The gas that was produced had too high a concentration of carbon dioxide to burn continuously.

As a local sanitation facility the system operated well for over a year, mostly through the efforts of a janitor whose monthly wage was paid out of the development project fund. When his salary was discontinued, the city government was unable to supply funds. So community residents arranged with the ward representative to levy a small tax (about four cents per month for each patron) to pay the janitor directly. This procedure, however, worked only for two months, and when the janitor left, water supply problems and lack of maintenance made the latrine unusable (6).

Another community biogas project, in the village of Dobare in southeast Nepal, faced a different set of problems. The $3000, 14 m^3/d digester was designed to serve the cooking and lighting needs of five households. Most of the labour for plant construction was donated by the five beneficiary families, who also agreed to supply it with their daily wastes.

One year after the plant's completion only two of the mantle lamps were in working order. Corrosive elements in the gas (presumably hydrogen sulphide) and inadvertent improper usage made mantle burnout common. Two of the five households were using neither gas lamps nor stoves, and had withdrawn from the cooperative arangement. Waste collection and gas distribution systems had not worked smoothly. In addition, many Nepalese people feel that the burner does not emit a sufficient flame, though it is in fact hot enough for most cooking. Evidently it is the smoke that is missed. Firewood smoke cures and protects the thatch of their homes and its structural members (6).

CHINFSE BIOGAS SYSTEMS

Only a limited amount of information is available on the Chinese biogas plants. Although the Chinese have been experimenting with biogas since the 1950's it was only in the last ten years and mainly in Szechuan that there was a movement to develop and extend the practice on any scale. The biogas systems built in Szechuan are of two kinds. The first type, by far the most common, is based on a pit size of 8 - 10 cubic metres, built and used by an individual family. The second type is built by a 'production team' and has a capacity of about 100 cubic metres. The gas produced from the latter unit is used to power agricultural machinery, machine tools, to pump water and to generate electricity (17). The diagrams below show some of the more common designs.

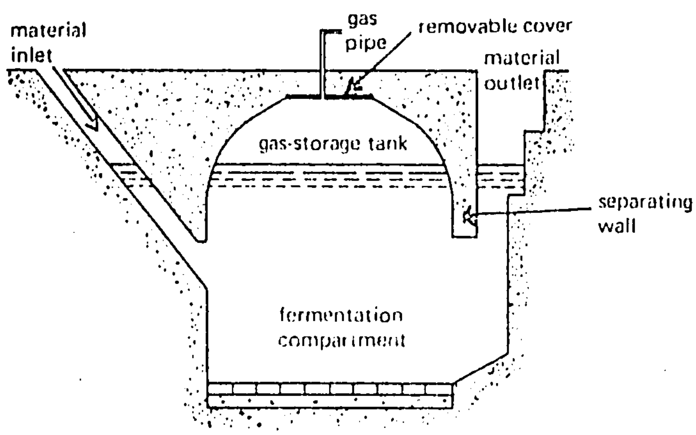


Diagram of a circular biogas pit.

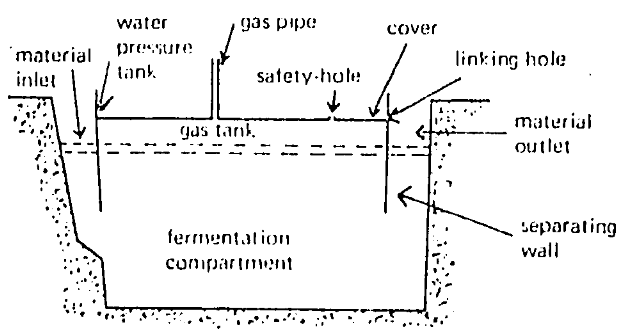
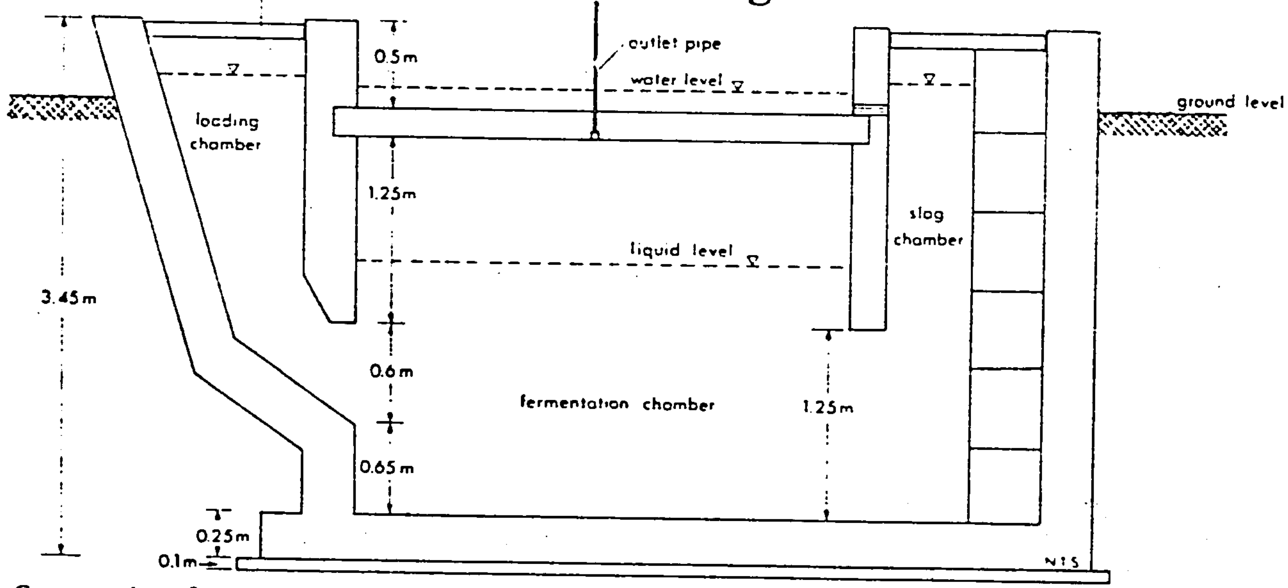


Diagram of a rectangular biogas pit.

Sichuanese Digester



Cross-section of a typical Sichuanese digester.

The principal difference between the Indian and Chinese designs is that the Chinese design does <u>not</u> incorporate a floating gas-holder. This makes the Chinese units <u>very</u> cheap since almost the whole system can be built by hand with locally available materials.

"The capital cost of a digester is low - roughly 1 Yuan per cubic metre or per person when using home-made concrete, and 5 - 6 Yuan or £1.50 per cubic metre when using commercial cement. Thus, for a pit for a family of seven, the cost of materials ranges from £2 - £3. It is, however, impossible to translate these figures to other currencies. In assessing the cost of building a biogas pit, it is the labour time which is significant. Usually, 35 working days are required - and these can extend well beyond 10 hours each - to build a 7 cubic metre pit for a family of seven." (17)

$4m^3$ DRUMLESS BIOGAS DIGESTER
(Erected at Cazanove Garden)

Base diameter 3m, dome base dia. 2.6m, dome height 60cm, cylindrical portion 170cm high. Total volume 12 m^3 and gas volume $4.0m^3$. (Using 80kgs of cow dung per day.)

CONSTRUCTION DETAILS, MATERIALS & COST

INTRODUCTION:

Biogas on farms is an autonomous means of obtaining power, and fuel from the animal and farm wastes and fertilizer as a byproduct. On an integrated farm, it is a distinct and effective possibility. A design has been tried with the following objectives:

1) To study the feasibility of replacing compost pit with a drumless biogas plant.

2) To study the feasibility of constructing a drumless biogas plant with village level expertise and suiting local conditions.

3) To explore the possibility of using slurry for growing animal food, fish, etc.

METHOD OF CONSTRUCTION

A. FOUNDATION:

1) Foundation is dug to a depth of 2m of 3.6m diameter. The bottom is rammed to make it approximately level. Over this brickjelly is laid.

2) Brickjelly: Use 2cm to 2.5cm brick bats. Mix with stone lime in proportion 2 brick bats 1 lime. Dry mix them first then add sufficient water to make it workable. Place this initially, 23cm to 24cm thick which when consolidated properly by ramming will settle to 20cm.

3) Cement concrete: This should be 1:2:4 proportion (1 cement, 2 sand, 4 pebbles). This should also be thoroughly dry mixed and water added approximately 25 to 27 litres per bag of cement. Place this mixture first in a layer of 3cm thick. Over this place the grid (6mm bars at 15cm centres both ways). Over this place the remaining 7cm concrete. Use 1% soap water for mixing instead of plain water (1kg of soap in 100 litres of water. The soap is easily dissolved in boiling water).

B. DIGESTER SUPERSTRUCTURE

4) Brickwork: It should be started after one day of laying the cement concrete. The base circle should be drawn first (3m inside diameter and 3.5m outside diameter). The brick layer should follow these circles. Usual english bond and 1:1:8 combination mortar with soap water should be used for the brickwork here. As the wall is sloping inside (for reducing the gas pressure on the dome and increasing earth pressure on the walls), which reduces the base diameter by 40cm at the

base of the dome, each layer of bricks should project inside by 8 mm. It would be better to make a template as shown which should be used for checking the slope of the brickwork. Over the openings for the inlet and outlet, 10cm deep R. C. lintel with three 6 mm bars should be provided. The inlet and the outlet chamber brickwork up to ground level should now be done and left till the last, i.e., till all inside work is over. This will provide easy access to the inside. At the base of the dome the brickwork is extended in the form of a cornice as shown in the drawing, this is for the rigidity of the dome at the base.

5) Construction of the dome: First build a small temporary pillar, 30 x 30 x 70cm high at the centre of the base. This should be removed after the dome construction. A nail should be driven exactly at the centre with a small projection where a nylon or cotton string can be attached. The string should be about 2 metres long and a mark should be prominently made at a distance of 160cm from the nail by tying, say a coloured thread. This is the radius of the dome. Next get 7 to 8 thin bamboo strips. The nail should be partly driven, i.e., it will project nearly 1" from the bamboo. This is for the support of freshly laid bricks on the dome. Now the construction may be started. The inner face of the cornice, which has been provided at the base of the dome, is now plastered at the proper angle in line with the centre, in 1:4 cement mortar, with soap water. Two 6mm bars should be laid all round in a circle at the outer edge of the base and embedded in this mortar. The brickwork for the dome should also be carried out in 1:4 cement mortar, but here add 200 gms of washing soda per bag of cement for quicker setting. Ordinary bricks should be laid in this mortar and during laying each brick should be first checked with the mark of the string for its correct position and then should be supported by the bamboo strip, the projecting nail supporting the brick.

After laying 5 to 6 bricks in this manner it will be found that the cement has set sufficiently for the second brick to carry its own weight. The bamboo support may be removed and may be used for there seventh brick. The support for the first brick should not be removed until the ring is complete. If there is some difficulty experienced regarding the setting then some more time should be allowed for the cement to set before removing the support. Obviously, it would be much better if the bricks are cut to shape and size beforehand to follow the curvature of the dome. (Shown in the drawing). But it may be rather laborious to do it, so one can use two or three ordinary bricks and then cut one brick to shape and size for the layers near the base of the dome. It will make the curvature slightly undulating and which can be corrected by plastering. Thus each layer of brick will be concentric circle following the dome curvature. The dome construction can be carried out like this without preparing any form-work beforehand. It takes nearly 2 days to complete the dome of this size with one mason and two helpers. Of course, as the radius of the opening of the dome becomes smaller and smaller, the bricks will require shaping more frequently. The nail on the bamboo should support the freshly laid brick so the bamboo may have to be placed on bricks to obtain the correct height. The 2 1/2" G.I. pipe for stirrer should be fixed at the centre of the dome. 2cm thick plaster in 1.4 cement mortar with Accoproof should be applied on outside of the dome. Then one layer of chicken wire mesh should be placed over the whole area and another layer of 2cm thick, 1:4 cement mortar should be applied over it. Over this, one layer of acchakal bricks should be laid, not in layers but in a criss-cross way, in combination mortar (1:1:8) and plastered over it with the same combination mortar, using soap water instead of plain water. It is understood that scaffolding should be provided wherever it is necessary. No weight should come on the dome until it is set and complete.

C. FINISHING & SEALING

6) Plastering: Before starting inside plastering of the digester, all joints should be raked with a thick nail to a depth of at least 1cm all over, including the dome. Then 1:4 cement mortar with Accoproof should be used for plastering the inside including the walls and the dome and a neat cement finish should be given all over. The plastering should be cured for at least 3 days with frequent spraying of water all over. It should then be allowed to dry for 3 days and then hot bitumen in liquid form should be applied to the dome and the walls. This will make it fully gas and water-tight. Now complete the inlet and the outlet chambers in combination mortar. Inside plastering should be finished in neat cement. The outside of the digester need not be plastered and should be filled with excavated earth and the dome also should be covered with 15cm of earth. Now loading may be started if it is ten days after completion of the dome. Otherwise allow at least 10 days after completion of dome.

Note I: The digester is designed to take gas pressure of 100cm water head. Although the pressure will be released from the digester if the slurry level goes down, it would be better to provide additional safety arrangement.

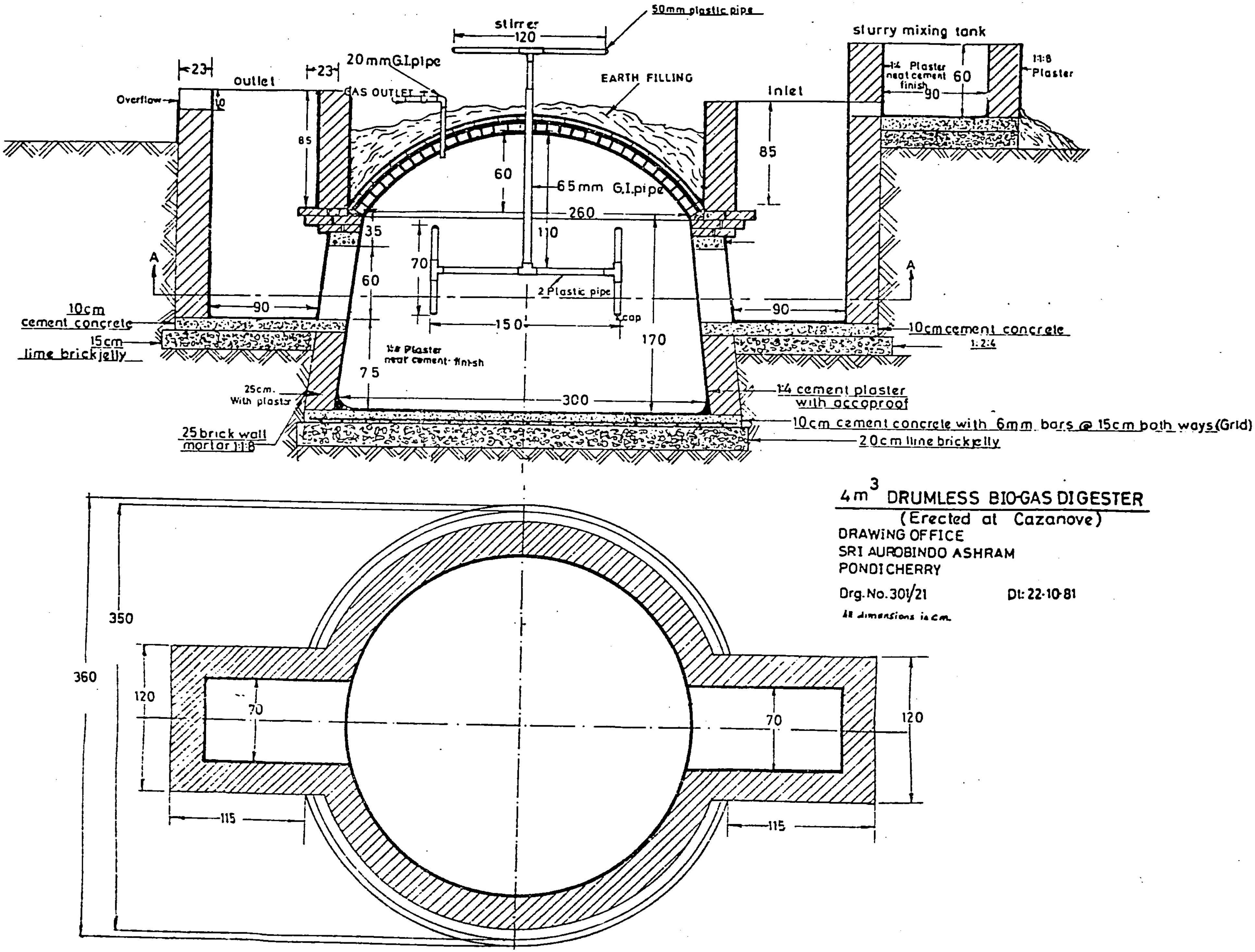
For this purpose, there should be one water manometer provided to check the gas pressure inside the digester and it should be so designed that it also acts as safety valve.

The design shown in the drawing may be adopted and connected at any convenient point on the gas line, where it could be observed easily.

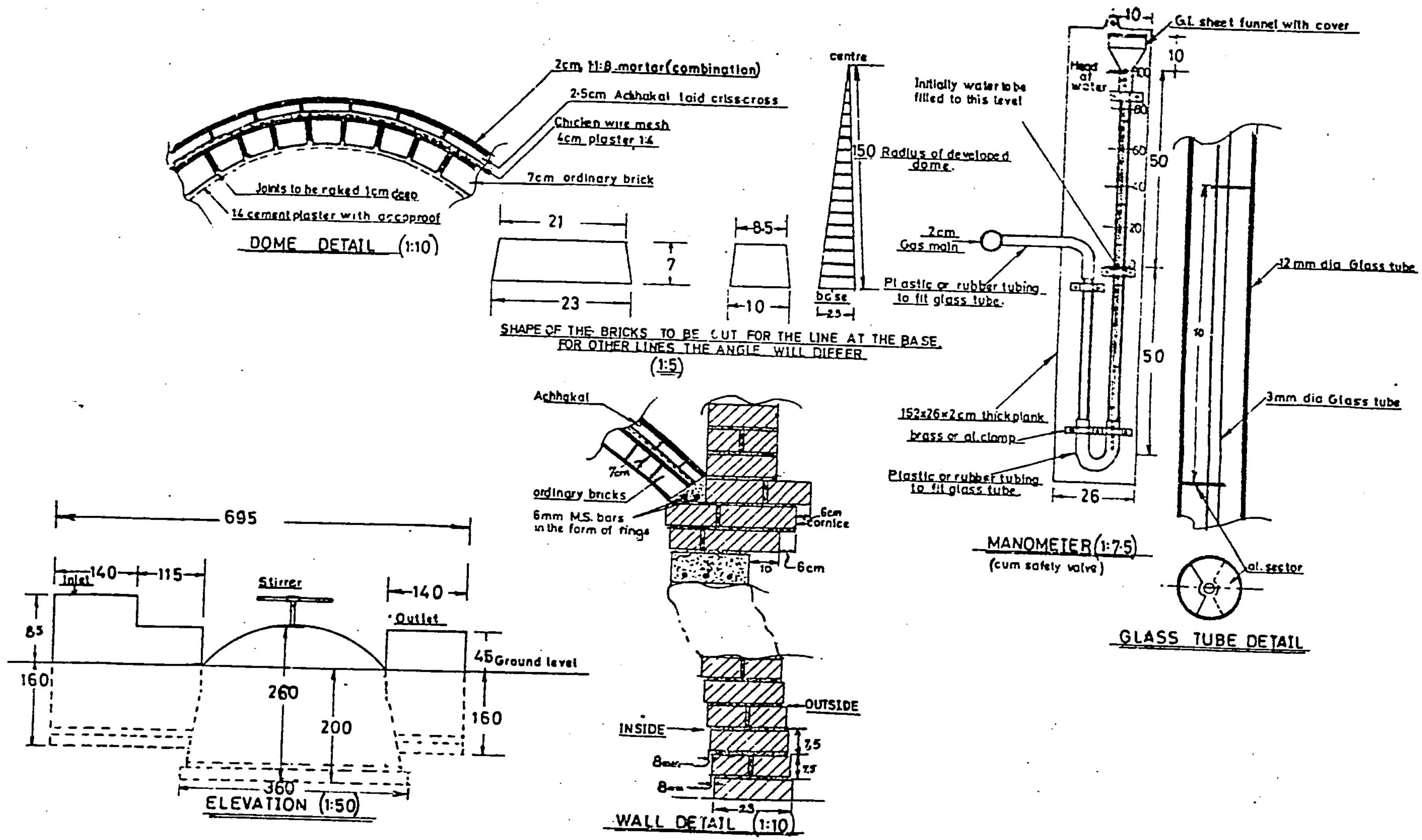
Note II: Gas is likely to leak slowly and continuously if there is the slightest crack or porosity in the space where gas is collected. This, obviously is most undesirable as the output of the digester will be reduced and even it may become nil. All masonry and plastering work should therefore be done properly under good supervision so that there is no chance of having any porous area or cracks.

Drawing Office
Sri Aurobindo Ashram, Pondicherry

Date: 28.10.81



stirrer
120
50mm plastic pipe
slurry mixing tank
1:4 Plaster neat cement finish
60
90
1:1:8 Plaster
23
23
20mm G.I.pipe
outlet
GAS OUTLET
Overflow
10
85
EARTH FILLING
inlet
85
60
65mm G.I.pipe
260
35
110
70
60
2 Plastic pipe
cap
A
A
90
90
10cm cement concrete
15cm lime brickjelly
150
10cm cement concrete 1:2:4
170
1:4 Plaster neat cement finish
75
25cm. With plaster
300
1:4 cement plaster with accoproof
25 brick wall mortar 1:1:8
10cm cement concrete with 6mm bars @ 15cm both ways (Grid)
20cm lime brickjelly
4 m3 DRUMLESS BIO-GAS DIGESTER
(Erected at Cazanove)
DRAWING OFFICE
SRI AUROBINDO ASHRAM
PONDICHERRY
Drg. No. 301/21
Dt: 22-10-81
All dimensions in cm.
350
360
120
70
70
120
115
115


2cm 1:1:8 mortar (combination)
2·5cm Achhakal laid criss-cross
Chicken wire mesh
4cm plaster 1:4
7cm ordinary brick
Joints to be raked 1cm deep
1:4 cement plaster with accoproof
DOME DETAIL (1:10)
centre
150 Radius of developed dome
base
SHAPE OF THE BRICKS TO BE CUT FOR THE LINE AT THE BASE. FOR OTHER LINES THE ANGLE WILL DIFFER (1:5)
G.I. sheet funnel with cover
Initially water to be filled to this level
Head of water
2cm Gas main
Plastic or rubber tubing to fit glass tube
152x26x2cm thick plank
brass or al. clamp
Plastic or rubber tubing to fit glass tube
MANOMETER (1:7·5)
(cum safety valve)
12 mm dia Glass tube
3mm dia Glass tube
al. sector
GLASS TUBE DETAIL
695
Inlet
Stirrer
Outlet
Ground level
ELEVATION (1:50)
Achhakal
7cm ordinary bricks
6mm M.S. bars in the form of rings
6cm cornice
6cm
OUTSIDE
INSIDE
WALL DETAIL (1:10)

$4m^3$ Drumless Biogas Digester Cost as on July 1981

A. MATERIALS

1.	Bricks - -	- 4300 nos	= Rs.	623.50
2.	Achhakal - -	- 400 nos	= Rs.	32.00
3.	Cement - -	- 32 bags	= Rs.	900.50
4.	Sand -($8.5m^3$)	- 1.5 lorry	= Rs.	120.00
5.	Lime -($2.3m^3$)	- 3 carts	= Rs.	288.00
6.	Pebbles -($1.5m^3$)	- 2 carts	= Rs.	50.00
7.	Accoproof -	- 12 pkts	= Rs.	84.00
8.	6mm M.S. bar -	- 120 m	= Rs.	145.00
9.	Binding wire -	- 1 kg	= Rs.	7.50
10.	Chicken wire mesh	- 10m x 1m	= Rs.	50.00
11.	Washing soap -	- 10 kgs	= Rs.	30.00
12.	Bitumen - -	- 3 kgs	= Rs.	13.50
13.	"Araldite" -	- 1 pkt	= Rs.	16.00
14.	G.I. pipe - size 2 1/2"	- 1.5m	= Rs.	50.00
15.	Plastic pipe 2" dia.	- 6m with fittings	= Rs.	235.00
16.	Sundries - -	- - -	= Rs.	10.00
			Rs.	2655.00

B. LABOUR

1.	Excavation -	- $20m^3$	= Rs.	100.00
2.	Mason - ..	- 25 man days	= Rs.	375.00
3.	Helper - -	- 80 man days	= Rs.	400.00
			Rs.	875.00

A. Materials - - - -	= Rs.	2655.00
B. Labour - - - -	= Rs.	875.00
	Rs.	3530.00
Manometer cost - - - -	= Rs.	100.00
Total	Rs.	3630.00

- - - - - - - - - - - - -

Drawing Office,
Tata Energy Research Institute
Field Research Institute
Sri Aurobindo Ashram,
Pondicherry 605002

Date: 29.10.81

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IMPROVED STOVES AND SOLAR COOKERS

There is another 'energy crisis' in many of the developing countries. This is a crisis that arises, not from the insecurity and expense of securing an adequate supply of oil, but from the decreasing availability, and rising costs, of the traditional fuels -- particularly of fuel wood. Shortages of traditional fuels are not new. But in the past, urban dwellers and industrial users have typically been able to shift to the commercial, more efficient fuel sources. Today, while the costs of traditional fuels are high in many areas, those of petroleum and other commercial sources are often higher still. One response has been to increase consumption of traditional fuels, further endangering this renewable, but always insecure, resource base.

Wood is the preferred traditional fuel in developing countries. In most cases people only burn crop or animal residues when wood is unavailable or expensive. Because fuel wood cannot usually be economically transported over long distances, large demands for wood fuels by urban dwellers and industrial users (including charcoal making) can quickly place stress on forests in local regions. In drought-prone regions such as the African Sahel, even relatively small and scattered populations can outpace the ability of fragile ecosystems to renew themselves sufficiently rapidly to sustain fuelwood production. Cities located in these areas are particularly prone to creating desert-like conditions. While land-clearing for agricultural use and overgrazing are probably more important causes of deforestation and desertification, fuel wood collection has been an exacerbating factor in North Africa, the Indian subcontinent, and the Andean and Caribbean regions of Latin America.

Although globally, 97 million hectares or 2 percent of forests were added between 1965 and 1975, tropical forests are under much greater stress, being lost at a rate of perhaps 16 million hectares annually. The preliminary results of a study of fuelwood supply and needs indicate that about 100 million people in developing countries live in areas where there is already an acute shortage of fuelwood. About another 1 billion are able to meet their minimum firewood requirements only by cutting in excess of the sustainable supply. According to this report, with current trends of population growth, of fuelwood demands, and depletion of tree resources, over 2 billion rural people in developing countries will need, within two decades, to be provided with large supplies of alternative fuels [1].

Growing populations increase the demand for food and farmland and push agriculture into previously forested regions. Between 1900 and 1965 half the forested area in developing countries was cleared for agriculture. Between 1950 and 1970 the rural population in India alone expanded by 136 million and cultivated land increased by 22 million hectares. During the same period, forest reserves in India declined by 3.4 million hectares.

In Africa and parts of Latin America, the deforestation problem is complicated by a transition from shifting cultivation to sedentary agriculture. Slash-and-burn farming, which was appropriate as long as the people-to-land ratio was low, now contributes to progressive deforestation as rotations are shortened and forest regeneration is prevented. Recent droughts in Africa have accelerated this process, especially in the Sahel.

In the Chittagong Hill tracts of southeastern Bangladesh, about 40% of the total forested area of the country is being rapidly destroyed by illicit timbering and by shifting cultivators who have reduced the traditional 10-year slash-and-burn rotation to 3 years [2]. In the hills of Nepal, forests are being depleted for fuel and cattle feed at an annual rate of more than 3 percent. This could lead to total depletion within 30 years. Between 1963 and 1971 in Thailand the total land area covered by healthy forests was apparently reduced from 53% to 39% [4]. In Sri Lanka, the forest cover has been reduced from 44% in 1956 to about 20% of the total land area by a combination of clearing for settled agriculture, slash-and-burn cultivation with progressively shorter rotation times, and exploitation of trees for construction and household and industrial fuels-chiefly in manuufacturing tea, ceramics, bricks, and tobacco [5]. Sri Lanka's major development program the Mahaweli River development scheme will reduce the area of forest still further to about 17% of total land area. In India, potentially usable forests are said to cover around 66 million hectares, mainly in mountainous regions, and in four states that have less than 20 percent of the population. The forests are being cut down faster than they can grow, partly to make room for new farmlands and partly for use as fuel. In consequence, the upland areas are subjected to destructive erosion, which in turn results in rapid silting of irrigation and power reservoirs and serious flooding of downstream areas [6]. In India, people clearly recognize the growing crisis as fuel for cooking becomes increasingly scarce. Carpenters making wheels for bullock carts near Bardole in Gujerat complain that the rising costs of wood are jeopardizing their livelihood. Sweepers in the streets of New Delhi collect and sell leaves swept up outside offices [7].

The problems discussed above strike the poor particularly hard. It is the poor who are the largest users of traditional fuels and the poor who have the least access to their means of production. For them, the only possible responses to difficulties in obtaining fuels are to reduce their energy consumption from their already very low levels, or to devote more time and labour to collection. When traditional fuels become scarce they may start to be bought and sold in markets and to take a growing proportion of family incomes. Buying wood fuels in some cities in the wood-short Sahel has been estimated to require as much as a third of the average labourer's income [8]. Distances of 100 to 500 kilometres and more have been reported for the transport of both fuelwood and charcoal to cities in a number of developing countries. In parts of upland Nepal and central Tanzania, 200 to 300 person-days of work per year are required to gather a family's fuelwood. In other cases landowners may exact more services from the landless in order to permit them to gather wood or crop residues on their property. The added burden and drudgery of this labour usually fall on women and children, the traditional gatherers of fuel.

Another response to rising fuel prices has been to reduce energy consumption and to lower nutritional standards. In some areas of West Africa, the number of cooked meals per day has been reduced. In some hill regions of Nepal and Haiti, fuel shortages even appear to have influenced the choice of crops produced in favour of those requiring less cooking. In Guatemala, Indians in the high plateau have stopped cooking beans as often because they require longer cooking. The food substituted is lower in protein and has resulted in visibly lower nutritional levels. In Upper

Volta women used to cook millet twice a day, then once a day. Now many women cook only every other day or so. In between they feed their families on flour mixed with water. In the Senegal there has been a switch from millet to rice because the latter does not take as long to cook. But the rice now used is inferior and nutrition levels have suffered accordingly [7].

Where shortages of traditional fuels exist, the poor are forced to exert even greater pressure on the sustainability of renewable energy resources. People may cut fruit trees and other economically valuable species for wood, reducing both food and income. Seedlings, and tree roots may be destroyed for fuelwood; leaves and grasses may be raked from hillsides leaving only bare earth. Crop and dung residues may be stripped from fields, reducing agricultural yields and creating further pressures for expansion of agricultural land. In a vicious cycle soil fertility and agricultural productivity are reduced, and fuel supplies become ever more difficult to obtain.

There are three main approaches to ameliorating the problems of deforestation and shortages of fuel wood. These are:

(1) Resource augmentation: managing silvicultural resources in a more efficient manner; for example, by introducing fast-growing tree species.

(2) Efficiency improvements: introducing more efficient end-use devices; for example, improved stoves, and more efficient energy conversion technologies, for example, improved methods for making charcoal.

(3) Resource substitution: switching from biomass to another energy source. Direct solar energy is a source that could potentially substitute for biomass used for cooking. Biogas produced from the anaerobic digestion of crop and animal wastes is also a potential substitute for fuelwood.

In this Chapter we look at improved cook stoves: principally the Lorena stove, and the relatively new technology of solar cooking.

Traditional methods of cooking in developing countries, which have been used for centuries with little modification, involve burning wood in an open fire, sometimes enclosed by a horseshoe-shaped alcove made of mud or bricks to act as a windshield. In some cases, three stones are placed around the fire to serve as supports for the cooking vessels. This technique is commonly known as the three stone cooking method. Open-fire cooking is very inefficient because only 5 to 10% of the potential energy in the wood fuel is utilized in the cooking process.

Traditional cooking over an open fire also has a number of undesirable characteristics, for example:

(1) control -- intensity or rate of burn of open fires is difficult to control which may restrict the preparation of some kinds of food or limit the choice of cooking methods;

(2) unhealthy conditions -- family members, especially the cook are exposed to the constant emission of smoke and soot into the kitchen area and, in many cases, throughout the house;

(3) hazards -- open fires subject family members and especially unattended children to possible burns and scalds from sparks from burning logs or from unstable cooking pots and expose combustible structures to possible fire damage;

(4) unsanitary food preparation -- build-up of soot in the kitchen work area and roaming animals, such as dogs, may bring filth in contact with food cooked close to the ground over an open fire;

(5) lack of cooking space -- often only one food item may be prepared at a time and a supply of hot water cannot be kept heated while preparing food items;

(6) fatigue -- open fires often require constant tending or fanning by the cook who is exposed to the heat emitted by the fire;

(7) indirect costs -- family members expend considerable time, labor, or money to obtain the large amounts of firewood needed for open-fire cooking because of the poor efficiency of wood use.

The main advantage of open-fire cooking is that the cooking device is essentially free. In addition, the smoke from open fires often creates a welcome deterrent to insects and predatory animals, which could devour food stores and/or dwelling structures and harm family members. In fact, in some areas wood is burned, in part, specifically to provide protective smoke. However, open-fire cooking methods leave considerable room for improvements to make the chores of food preparation and firewood collection and use easier, more efficient, and less time consuming.

THE LORENA STOVE

Since its conception in 1976, the Lorena stove has been introduced principally among the indigenous people of western Guatemala. The stove was developed at the Estacion Experimental Choqui-ICADA, a small appropriate technology centre near Quezaltenango in highland Guatemala, to meet the cooking needs of rural people using readily available construction materials. Formed out of a large monolithic block of clay and sand (hence the name: lodo plus arena) the stove is designed to contain the heat of the fire and channel if through a system of internal flues thus reducing the amount of firewood required for cooking and eliminating smoke build-up in the kitchen. The stove can be built in a few days using only a shovel, machete, and kitchen spoon as construction tools. In Guatemala, a cottage industry has developed around the new stoves. Trained stovemakers have been able to make a living by building stoves in people's homes. An estimated 2000 stoves have been built in the last 2 or 3 years [9]. The Lorena design has been replicated or adapted for use in West Africa, Java, Nepal, and throughout Latin America.

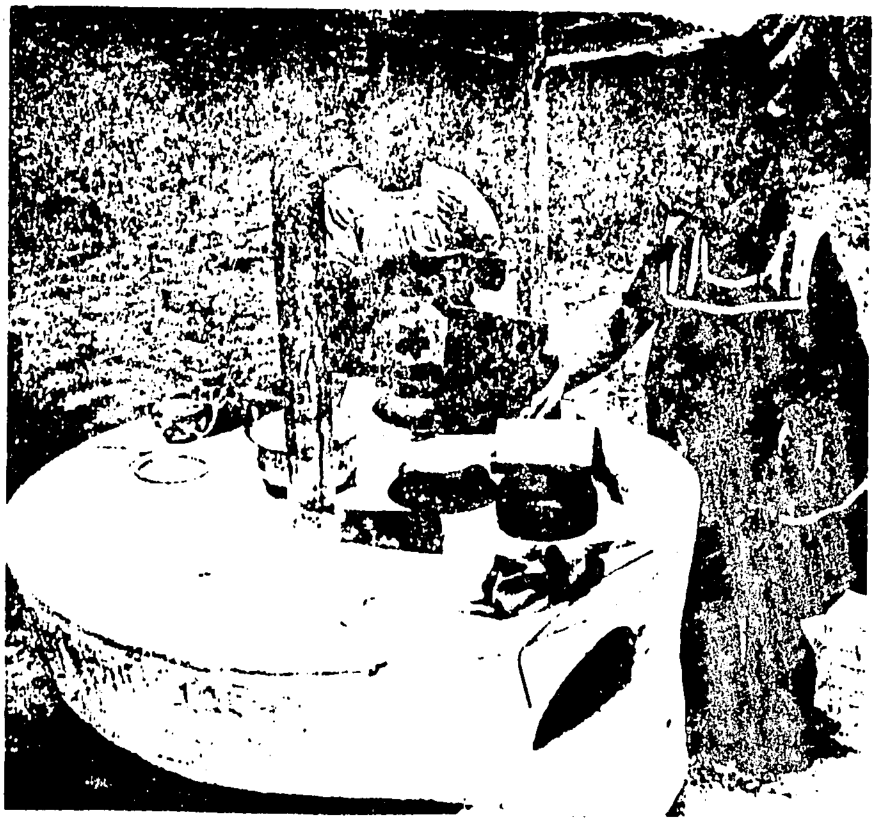


Figure 1. Lorena Stove in Guatemala [10]

The function of an efficient cookstove is to transfer the maximum amount of heat from the combustion process to the cooking pots. Lorena stoves, if designed and used properly, can be very efficient:

1) The thick sand-clay material keeps the firebox walls very hot, which improves the combustion of firebox gases.

2) The tight fit of the pots in the potholes reduces air leaks, helps prevent heat loss, and increases control over burning rate.

3) The firebox is designed for control of combustion processes and is shaped to direct maximum heat to the pot.

4) A chimney and set of dampers can be used to control the air flow through the stove, for more complete burning of fuel, and to hold residual heat in the stove body.

5) Potholes are designed to allow the pots to sit low into the stove to capture as much heat as possible, and to avoid losing heat from the sides of the pots.

6) In areas where more than one pot is used, multiple cooking potholes can be incorporated to allow simultaneous cooking with heat from one fire; heat not used by the first pot can be used by the second pot, and so on.

7) Baffles underneath each pothole create turbulence in the flow of hot gases, so that heat is transferred more effectively to the pots; offset tunnels also contribute to this effect.

8) The flexibility of lorena as a material allows stoves to be designed for maximum efficiency in many differing situations. For example:

    (a) In areas where long, slow cooking is done, the body of a large Lorena stove stores much of the heat, which can then be used for extending cooking time of slow-cooking foods or for baking, after the fire has died out. This heat will also gently heat the outer surface of the stove.

    (b) In areas where cooking is done quickly, small Lorena stoves heat up to efficient operating temperature in a short time because they have less firebox wall and tunnel surface area.

9) Water containers, if used, absorb some of the heat that would otherwise escape up the chimney, reducing the need for additional fuel to heat water. [10] However, field testing of Lorena and traditional stoves in Indonesian villages [11] has shown that Lorena stoves are not always more efficient than the traditional designs. The results of these tests indicated that the reasons for poor performance were:

1) Incorrectly placed chimneys causing inadequate draught;

2) Flues blocked by ash or pieces of ceramic pots. These pieces had originally been used to raise the pot above the hole to allow more heat to reach the cooking pot. This indicates that either the draft was insufficient or the combustion chamber and flues were the wrong shape or dimensions;

3) Incorrectly placed flues, resulting in poor heat transfer.

4) Damaged dampers or damper slots -- the users were unable to control the flow of air and hot gases into and out of the stove;

5) Incorrectly shaped pot holes. The hole over the combustion chamber was too small, thus reducing the area available for heat transfer. Some of the pots did not fit properly into the second, third and fourth pot holes. Since only a limited contact with the hot gases could occur, heat transfer to these pots was markedly reduced;

6) Incorrectly made combustion chamber. Often the combustion chamber was either too small or too large. This led to poor combustion (indicated by low CO_2 levels and high excess air) with a consequent increase in creosote and soot formation.

Another conclusion from this study was that smaller mass Lorena stoves showed consistently better performance than large mass stoves, which were only more efficient when used constantly throughout the day.

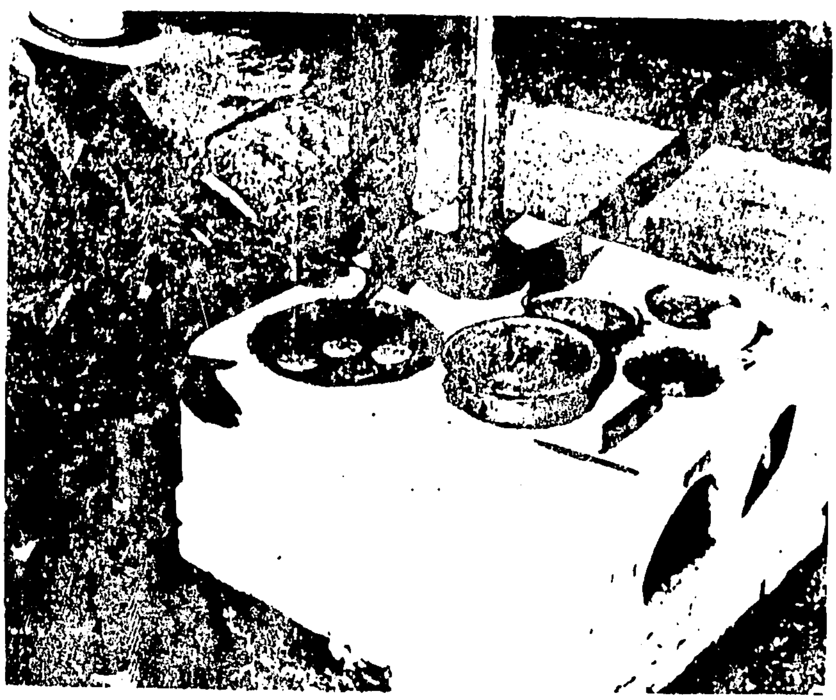
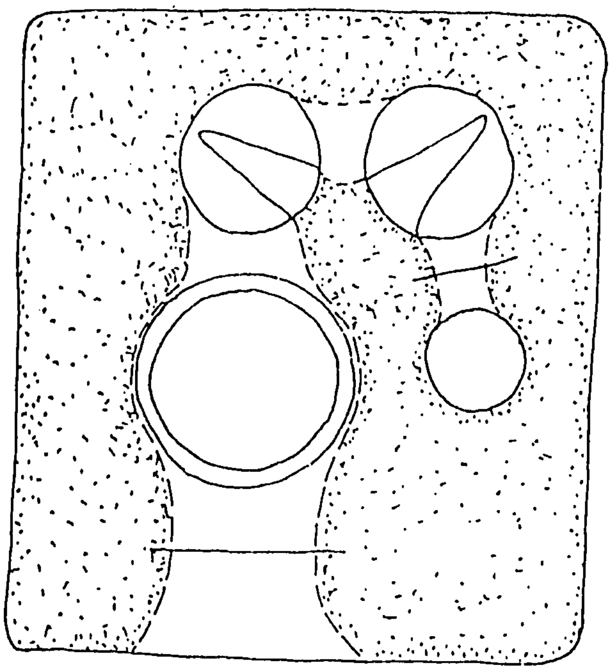


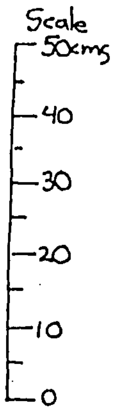
Figure 2. Lorena Stove in Guatemala [10]

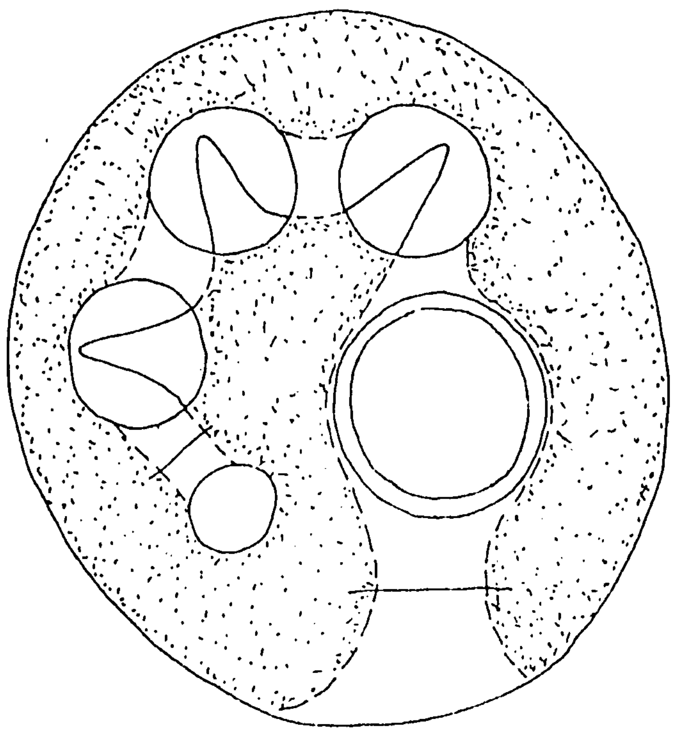
STOVE CONFIGURATIONS

It should be noted that there is no single design of Lorena stove. The stove should be designed to suit local cooking practices and traditions. Some examples are shown below. These sketches are from reference [10].



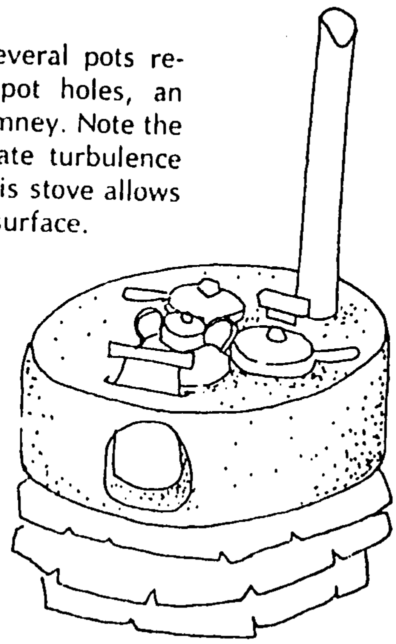
Guatemala





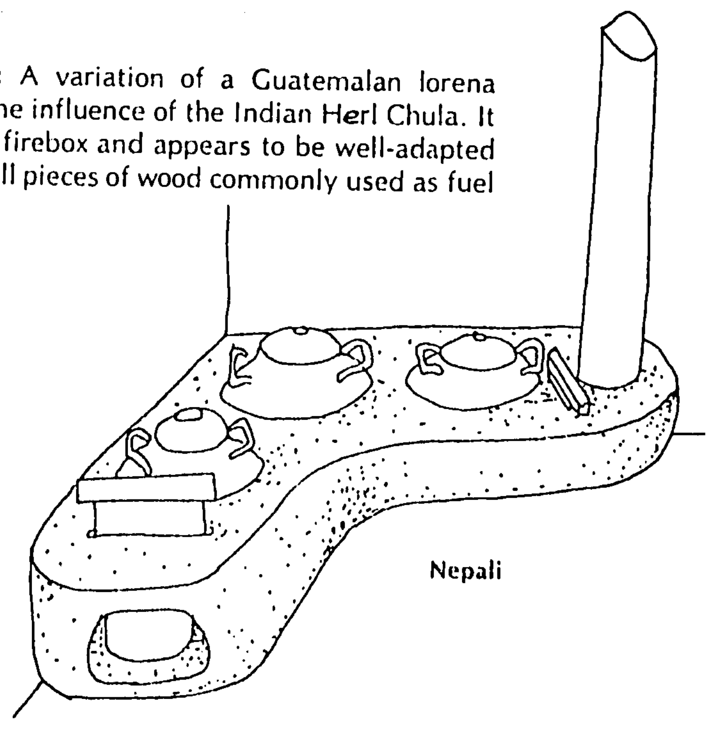
Guatemala

Guatemala: Slow cooking in several pots requires a stove with 2 or more pot holes, an effective damper system, and a chimney. Note the sharp bends in the tunnel to create turbulence under the pots. The large size of this stove allows food preparation on the stove top surface.

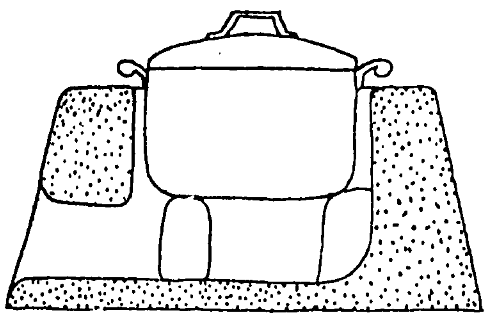


Guatemala

Nepali: A variation of a Guatemalan lorena showing the influence of the Indian Herl Chula. It has a tiny firebox and appears to be well-adapted to the small pieces of wood commonly used as fuel in Nepal.

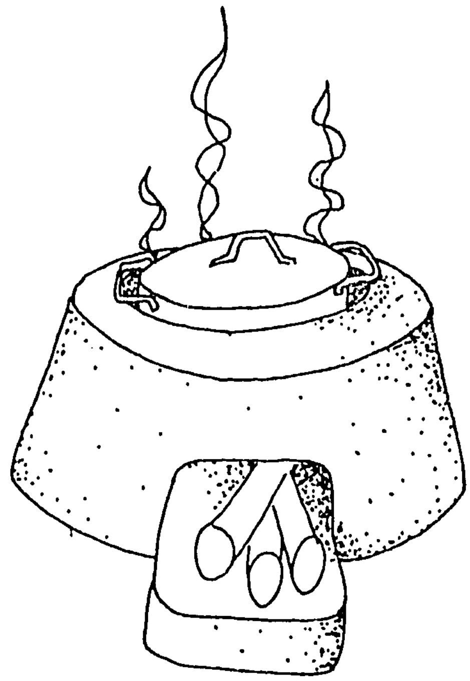


Nepali

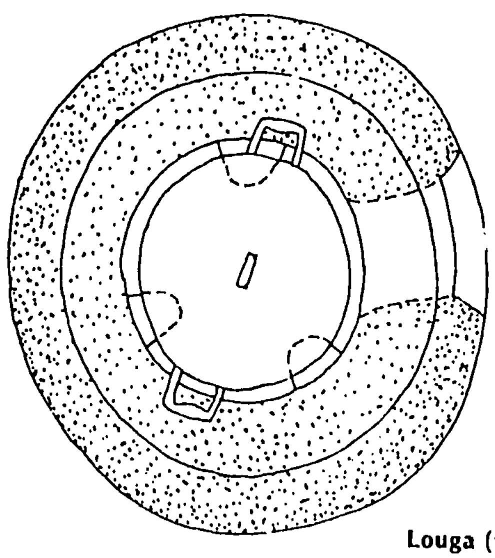


Louga (cross section)

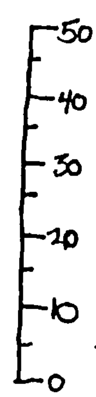
Louga (Senegal): The single pothole of this outdoor stove also acts as a chimney. Smoke escapes through a narrow gap all around the pot, heating its sides. The small firebox entrance prevents too much air from reaching the fire. The stove is built using the pot itself as a form. The lorena material is packed around it, then carved out a little more to make the smoke passage. The pot rests on supports, which may be rocks or small cans.



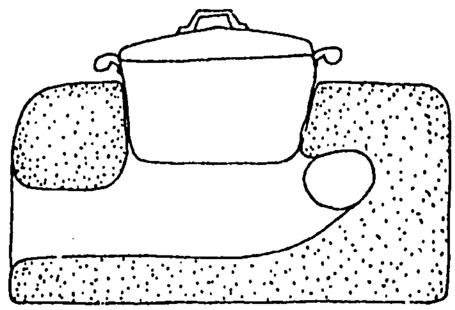
Louga



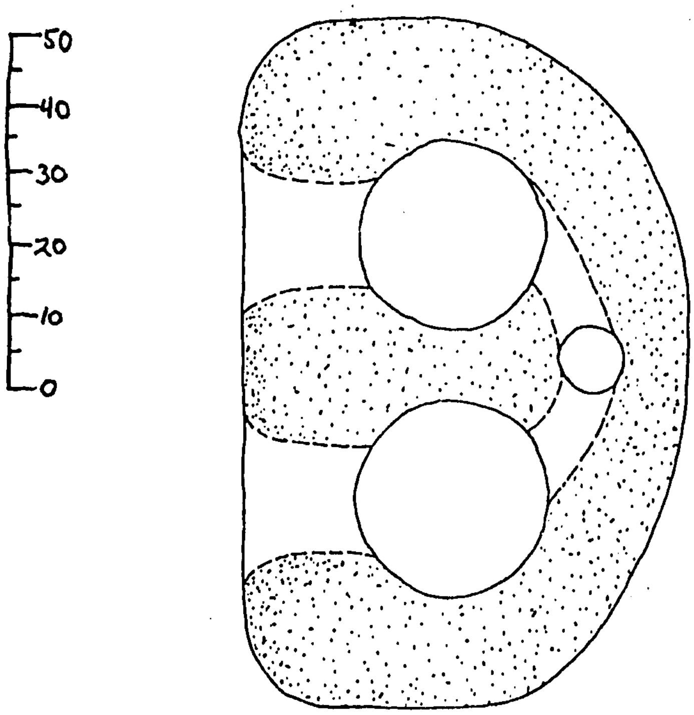
Louga (top view)



Twin Fireboxes (Senegal): This simple stove without dampers allows cooking over one or two fires. It might be a good design to introduce in an area without traditional stoves. After people have become accustomed to cooking on a simple stove, a more sophisticated design with dampers might be introduced.



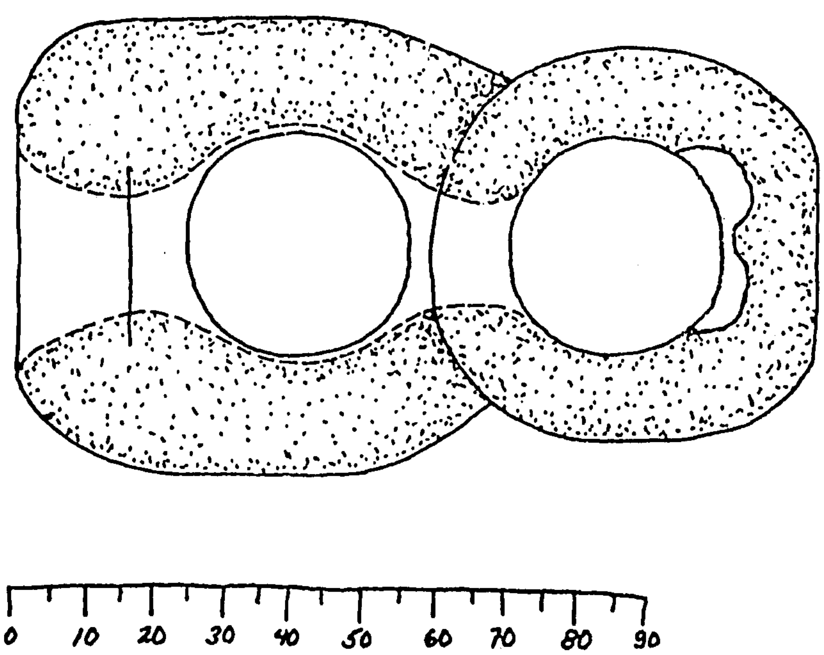
Twin Fireboxes (cross section)

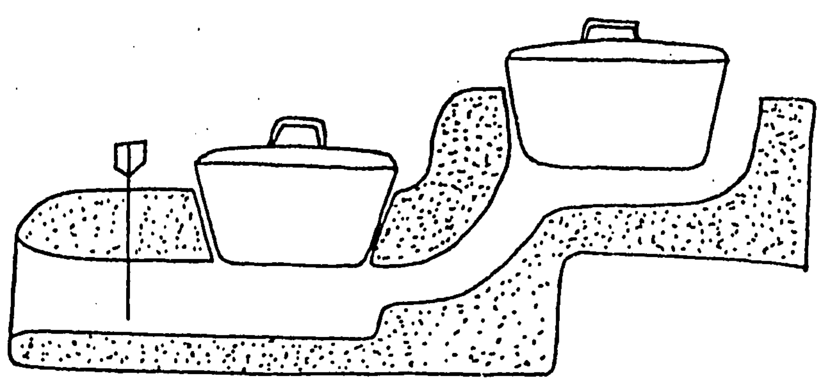


Twin Fireboxes (top view)

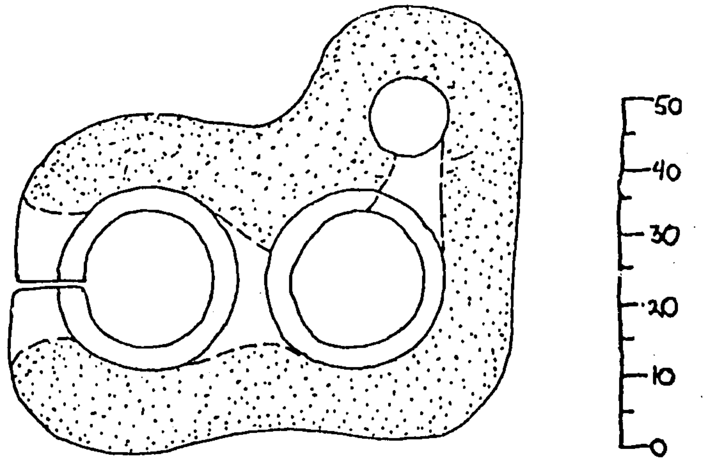
Upper Volta Guitar Stove: Another lorena adaptation for cooking out of doors in an area with scant rainfall. Smoke escapes around the elevated second pot, which acts as the chimney.

Upper Volta Guitar (top view)

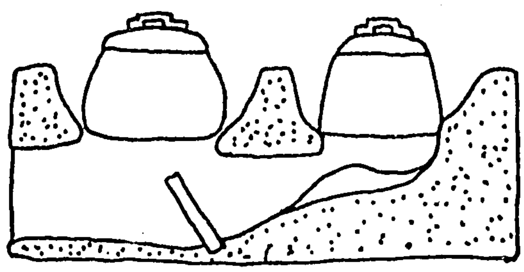




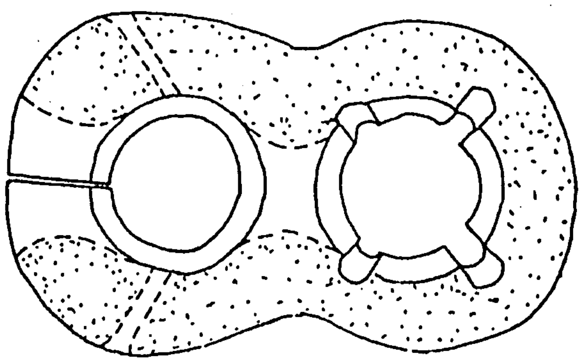
Upper Volta Guitar (cross section)



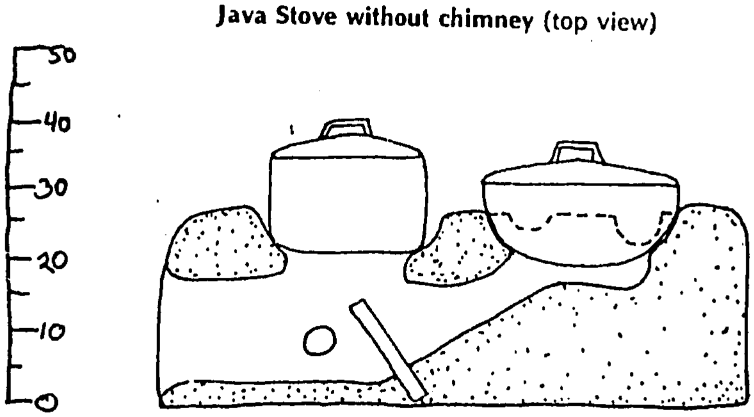
Java Stove with chimney (top view)



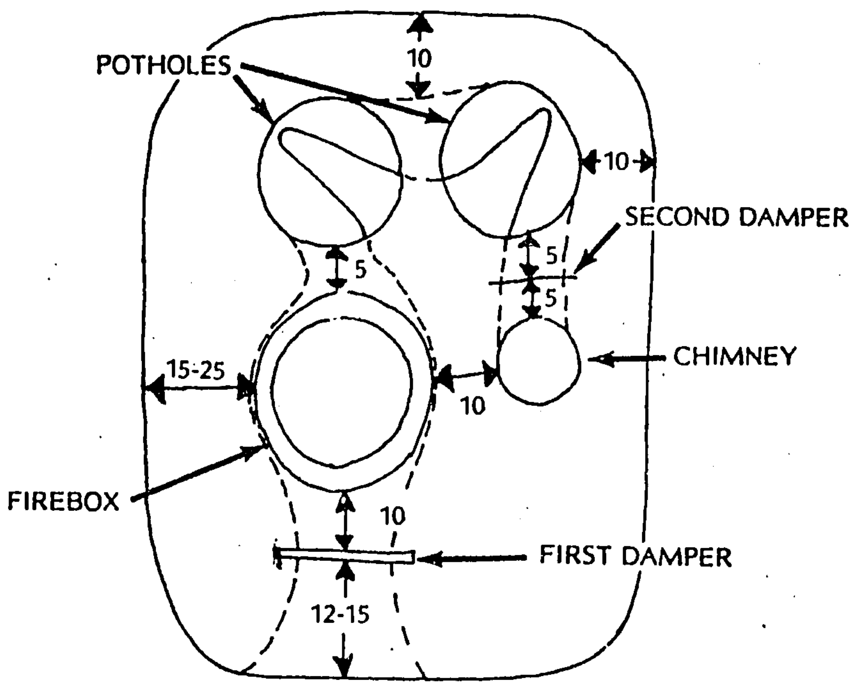
Java Stove with chimney (cross section)



Java Stove without chimney (top view)



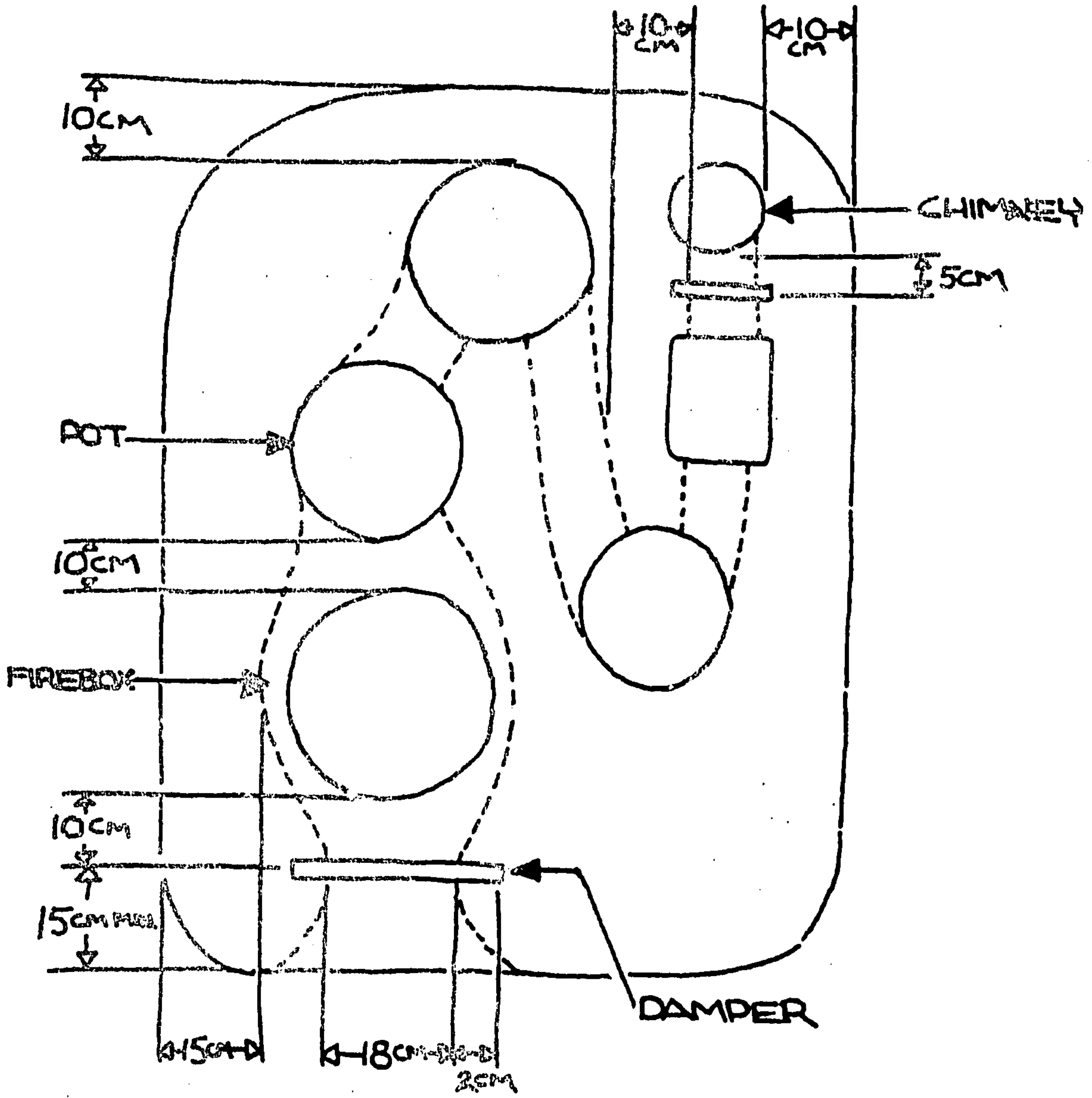
Java Stove without chimney (cross section)

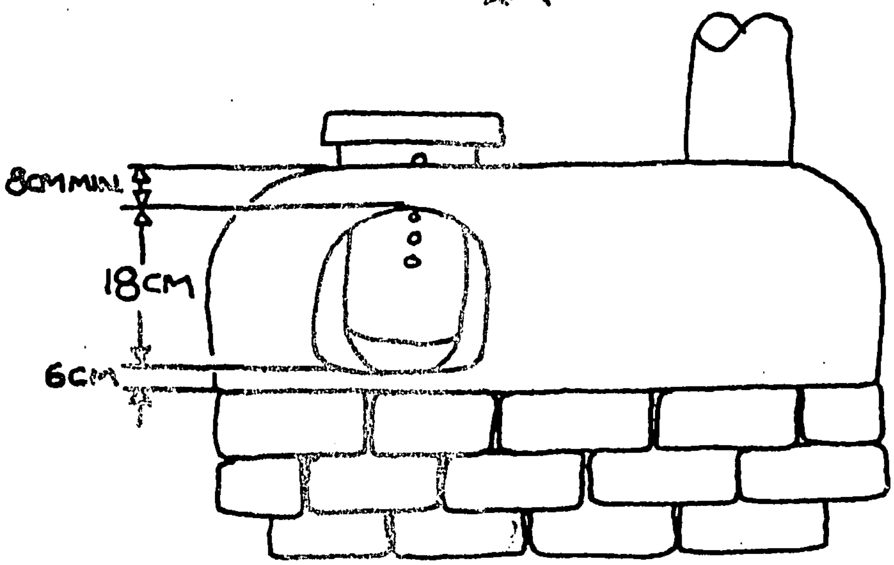


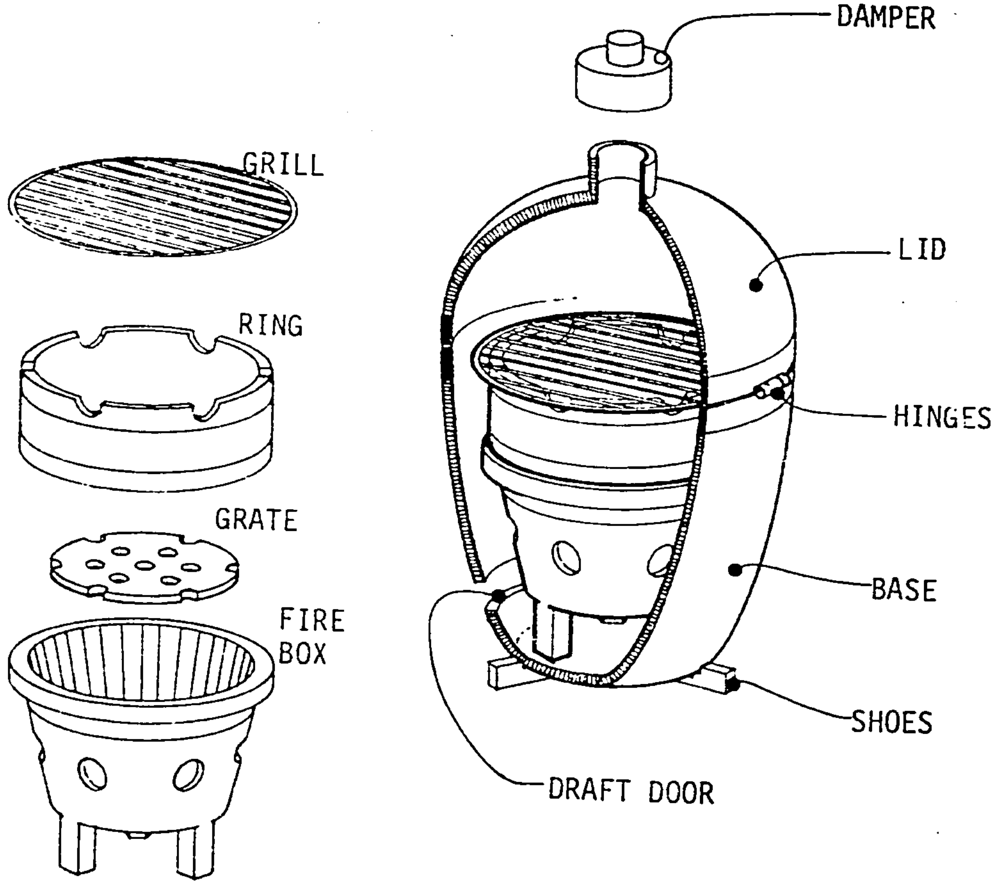
Recommended Distance Between:

Mouth of firebox and first damper	15 cm is best, 12 cm minimum
First damper and pothole over firebox ...	10 cm
Firebox and outside edge of stove	15 to 25 cm
Two potholes	5 cm minimum
Tunnel and outside edge of stove	10 cm minimum
Pothole and outside edge of stove	10 cm minimum
Chimney and firebox....................	10 cm minimum
Second damper and pothole.............	5 cm
Second damper and chimney	5 cm
Top of firebox entrance and top of stove...	10 cm minimum
Bottom and top of firebox entrance.......	10 to 15 cm, depending on whether dampers are used and height of chimney.
Bottom of firebox entrance and base	5 cm minimum, unless base is not flammable.

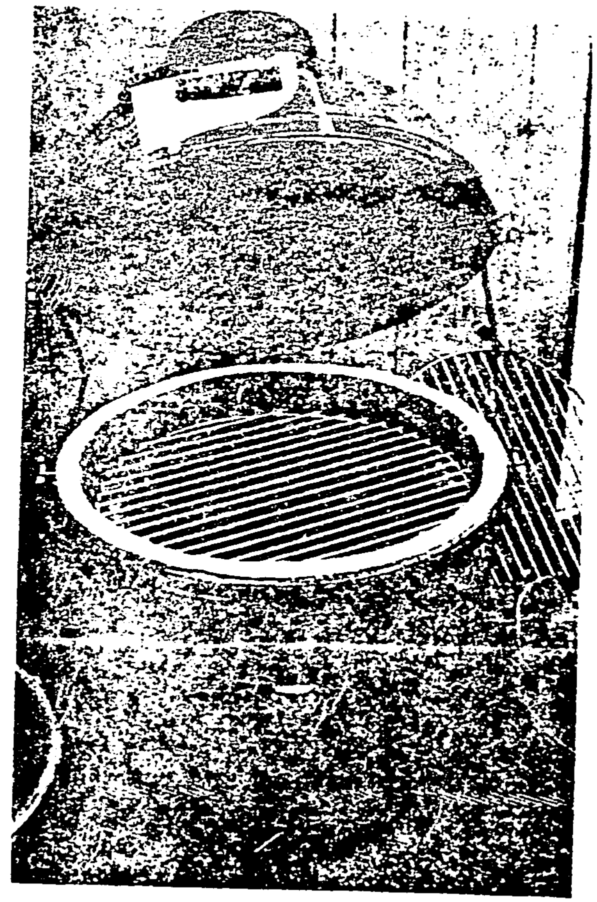
Figure 3.


10 cm
10 cm
10 cm
5cm
CHIMNEY
POT
10cm
FIREBOX
10cm
15cm MIN.
DAMPER
15cm
18cm
2cm


8cm MIN.
18cm
6cm



Sketch Showing Major Components of
Japanese Kamado Cooker.



Photograph of a
Japanese Kamado
Cooker.

Figure 4. BASIC COMPONENTS OF JAPANESE KAMADO COOKER

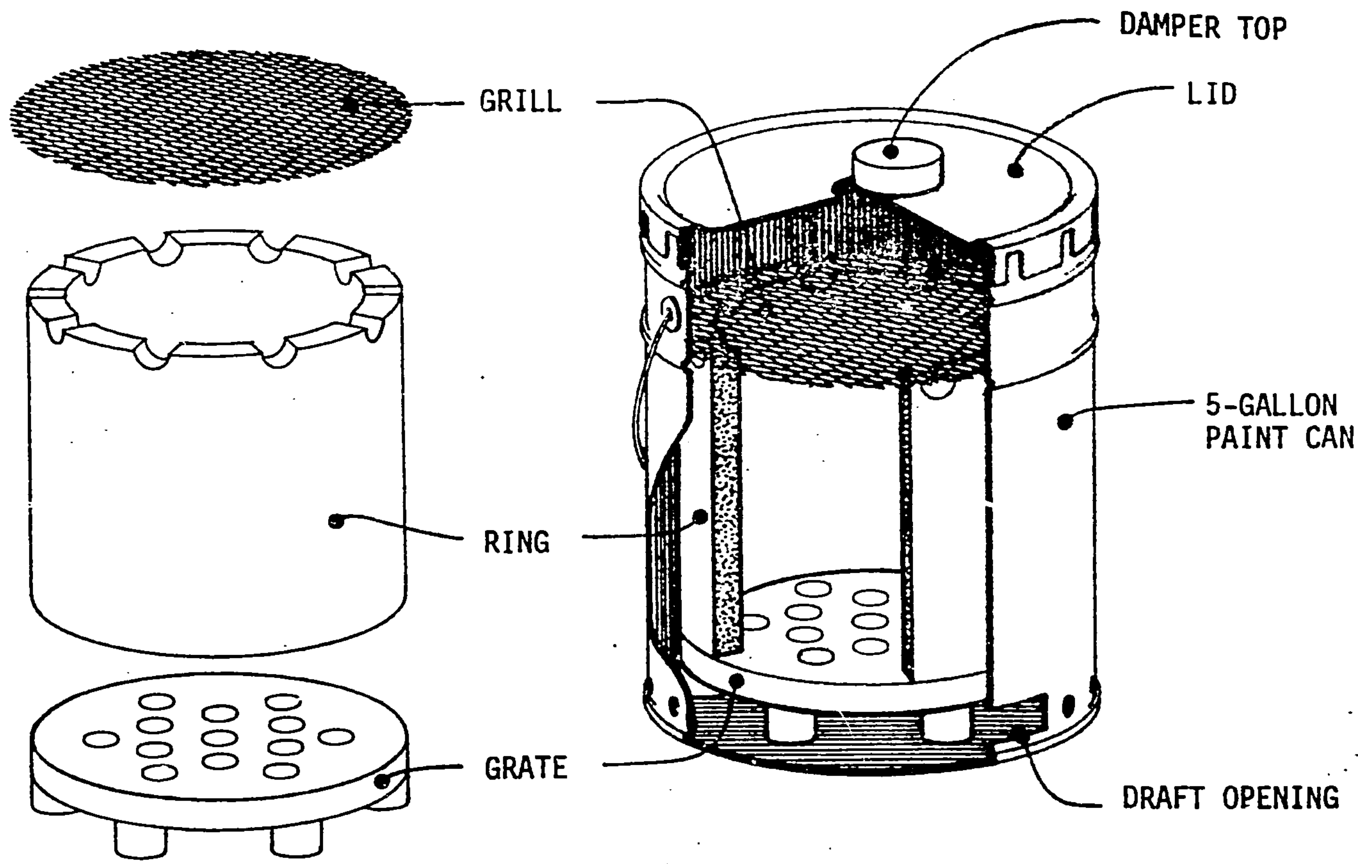


Figure 5. Schematic of Wood Burning Cook Stove Constructed from a 5-Gallon Paint Can.

Technology Extension [16]

Drastic forecasts of potential total deforestation in some regions before the turn of the century have at last promoted both high- and low-level government organizations, as well as private organizations, to consider the active dissemination of improved cookstove designs as an important tool for relieving some of the ecological stresses resulting from deforestation. Included in this group of institutions are large and influential organizations such as the World Bank and Appropriate Technology International (ATI) who are involved either directly or indirectly in financing efforts to expand the use of more efficient cookstoves. The World Bank includes introduction of more efficient wood burning stoves as a possible component of financed forestry projects for developing countries and ATI is currently funding efforts by the Centro Meso Americano de Estudios Sobre Technologia Apropiada (CEMAT) in Guatemala and Save the Children Federation (SCF) in Honduras to encourage the widespread use of Lorena stoves in these Central American countries. This kind of high-level support and institutional commitment on an international and national level to more efficient use of firewood for cooking has been largely missing until recently, which could be a major reason why there has been only moderate success in the dissemination of more efficient cookstoves since the improved designs were first developed in the 1950's.

The main factor required for the success of cookstove dissemination efforts is to ensure that the improved cookstove not only has advantages from the viewpoint of the promoters but also has benefits that can be perceived by rural villagers as being worth the time and investment required to change from a traditional form of cooking which, while inefficient, may be simple to understand and culturally acceptable. The new designs must be capable of demonstrating a substantial improvement in performance, which is readily translated into noticeable improvements in village life, because the introduction of improved cookstoves will inherently mean making basic changes in cooking practices which are often constrained by traditional, religious, and economic factors. For example, in parts of Africa, where women collect the firewood as well as cook, men may have little desire to spend time or money to help obtain a more efficient stove which might cut firewood collection time by one-half. In some regions, women have control or responsibility for the household budget covering firewood and cooking utensils. In this case, women must spend money to acquire a better stove. All of these considerations illustrate the unique problems that will have to be dealt with in each local region to successfully extend the use of more efficient stoves.

The success of a cookstove dissemination program may be enhanced by the involvement of an influential and knowledgeable person or group in the local village that understands the needs of the local people, including skills, materials limitations, and cultural requirements, as well as the basic principles behind the workings of a more efficient cookstove. Careful consideration of this type of information may produce a modified cookstove design which is attractive to the potential users while still performing with a better efficiency and having the necessary durability. For example, one adaptation which might be useful to aid the acceptance of improved cookstoves in areas where villagers value the smoke from traditional cooking devices would be to develop an efficient stove without a chimney. The importance of the adaptability of improved cookstoves to local conditions and/or limitations in materials supply without completely compromising design principles essential to achieving a better stove performance should not be underemphasized. Questions surrounding the adaptability issue should provide a fertile area for future cookstove research efforts.

Perhaps an appropriate vehicle for cookstove extension efforts is an organized program operating through, or in connection with, persons already involved in other development projects in the region, particularly fuel wood plantation or reforestation projects. However, once an improved cookstove design has been selected and presented to villagers, emphasis must be directed toward assisting villagers in thoroughly understanding how to build the stove and how the stove operates so that the technology will find a permanent place in village life. The extension effort in Guatemala where the input of the local reforestation personnel stimulated Lorena stove dissemination is a good example of incorporating these considerations into the extension program.

Although active cookstove dissemination efforts are only in the early stages of development, a number of elements appear to be necessary to achieve the goal of widespread use of improved cookstoves in developing countries. Some of these elements are:

(1) communication channels should be established between villagers and extension workers to identify any required adaptations for cookstove designs to be compatible with local conditions;

(2) involvement of village community leaders and any development institution representatives stationed in the village should be encouraged in the promotion efforts;

(3) financing opportunities for villagers who could not otherwise pay to have a stove built should be made available as part of the extension effort;

(4) whenever possible, extension efforts should be tied in with other community-based development activities occurring in the village;

(5) communication links between extension personnel and technical support groups should be established to ensure that adaptations of an improved design to local conditions do not result in cookstove safety, durability, or performance problems;

(6) feedback from users of improved cookstoves should be used to make improvements in the stove design or dissemination technique and to record the impacts of cookstove dissemination on village life;

(7) commitments from high-level agencies in governments to efforts to ease firewood shortages, through combined efforts of forestry extension workers and cookstove promoters at the village level should be sought.

SOLAR COOKERS

Solar cookers may be classified as:

a) Concentrating parabolic, and spherical dish or trough collectors where the heat generated at the focus of the collector directly heats either a vessel containing the food or the food itself.

b) Ovens or food warmers which are insulated boxes with transparent covers in which solar energy is collected directly or from a reflective surface.

Direct Concentrating Cookers

Perhaps the most familiar type of solar cooker is the parabolic dish collector which heats food either directly or in a bowl or pan placed at the focus of the collector. Some typical systems are shown overleaf. These cookers permit frying, broiling, and boiling to be done. These concentrators may be either rigid or collapsible. The concentrating cookers utilize direct solar radiation only, and therefore are sensitive to short term variations in solar intensity due to cloud movement and haze. Refocussing is required at 15 - 30 minute intervals depending on the size and shape of the pot and on the area of the beam at the focal point.

i. Fixed Soil-Cement Spheroidal Reflector

A fixed, spheroidal reflector may be made by forming a spheroidal depression in the ground. This depression is lined first with a mixture of soil and cement to stabilize the shape and then with aluminized plastic. The symmetrical depression is formed by swinging a pendulum, or blade, fastened to a wire which is secured at a fixed point above the ground on a tripod. After the rough shape is hollowed out, a shallow layer of an aggregate and cement mixture is placed in the depression, smoothed with the blade and wetted. When this has set, and after some further smoothing of the surface, a reflective lining or aluminized Mylar pressure-sensitive tape is applied.

The advantages of this reflector are its simplicity and ease of construction in the field. The orientation can be fixed during construction so as to make the reflector usable during the desired time of day. The cost of material for the laboratory models, exclusive of the reflective lining was reported to be less than a dollar.

The disadvantages of this cooker are its lack of mobility, which might result in physical damage in certain weather conditions, and the limited period of time during which it can be used (estimated at 4 hours a day).

ii. Lightweight Molded Aggregate Reflector

The most successful reflector of this type is a dish 1.07 meter (42 inches) in diameter with a focal length of 0.46 meters (18 inches). It is formed over a mold and is made of a vermiculate aggregate reinforced with wire with a supporting rim of thin-wall tubing. The surface is lined with aluminized plastic and plastic tape. The total weight is about 23 kilograms

(50 pounds). The reflector mount is a wooden post in the ground, arranged to pivot about its axis. The top edge of the reflector leans against the post and its lower edge is supported at variable distances from the post on an arm secured to the post.

The advantages of this type of cooker are the simplicity of the mounting device and its component parts and the possibility of using locally available materials and labor for its construction. Major disadvantages are the weight of the reflector, which makes it cumbersome to handle, and the fact that the reflective lining is difficult to apply.

Another inexpensive method of fabricating this type of reflector has been suggested. A convex, wire-reinforced, plaster paraboloid die is cast from a centrifugally formed concave parabolic mold. The surface of the die is coated with wax, then a layer of wood pulp, paper mache or laminated newspaper is spread over it. After a few layers are built up, they are pressed to squeeze out excess liquid. The process is repeated until the compressed laminate is 3 to 6 millimeters (1/8 to 1/4 inch) thick. Then the rim of a wooden "wagon wheel", reinforcing framework is glued in place. Woven basketry is suggested as an alternative method of reinforcement. After removal from the die, strips of aluminum foil are pasted on the surface. Finally, a stiff hoop is clamped to the front surface. The process would require about 6 to 8 hours of labor. The cost of materials in this reflector is estimated to be about $10 in the United States.

iii. Spun Liquid Plastic Reflector

A liquid in a revolving horizontal pan takes the shape of a paraboloid, and a liquid resin suitably catalyzed to harden after this form is taken will retain this shape. With the development of the epoxy resins, satisfactory material has become available. EPON 828, catalyzed with 5 percent piperidine and cured at 80° and 90° C has been found to give satisfactory results.

The focal length of the finished reflector is controlled by the speed of rotation of the pan. The formula for the focal length may be simplified to $f = 38.4/\text{rpm}$, where f is the focal length in feet. Additional resin may be poured onto the first surface after it has set. In general, superior surfaces are obtained on the second or third pour. Reinforcing material such as fiberglass can be placed on the surface before the second surface is poured.

The quality of the reflectors produced to date is better than anything that can presently be made without the use of grinding and polishing techniques. This is, however, not a significant factor in cooker considerations. The estimated weight of such a 0.91 meter (36 inch) diameter reflector is about 14 kilograms (30 pounds) for a 12 millimeter (1/2 inch) thickness. The cost of the epoxy resin for this size reflector is estimated at $25 - $35. Although this technique could perhaps be used for making a master mold, it does not appear that this offers a practical solution for low cost solar cookers.

iv. Fresnel-Type Reflector

The use of the principle of the Fresnel reflector in a solar cooker is thought to be unique among the various known solar cooker designs. The Fresnel reflector concentrates light and heat by means of several simple curved reflecting surfaces. This is in contrast to the doubly-curved reflecting surfaces of the typical parabolic dish reflector. This reflector was developed and tested by VITA and is constructed of 3 millimeter (1/8 inch) Masonite to which aluminized Mylar is cemented. By cutting a series of rings out of the Masonite, removing sectors and rejoining, a series of nesting collars are formed that focus light. These collars are supported in a simple wooden frame. The design is simple enough to permit construction with the tools, skills and materials (except for the aluminized Mylar) that are locally available throughout most of the world. Figure 10 shows the basic structure of the reflector and the method of construction.

The most serious drawback of this cooker is that which plagues other designs as well: the deterioration of the aluminum reflecting surface of the cooker due to weathering. Even bulk aluminum loses its high reflectivity on exposure to the elements. On the other hand, this design allows convenient replacement of the reflective material whenever dictated by a loss in perforamnce.

v. Drape Formed Polyester Parabola

Only one example was found of a solar concentrating cooker which was fabricated by village labor using essentially locally available material and which also found considerable use in the village itself. The basic unit consists of a parabolic concentrator developed at the University of Wisconsin. These collectors were introduced into three villages in Mexico during 1959 - 1961. Construction of the parabola was accomplished in several stages. First a male mold was formed from a concrete mound with the parabolic contour developed by rotating a wooden template of the proper profile on a vertical axis projecting through the center of the mound. After the concrete mold cured it was coated with a parting agent and covered with newspaper. Alternate layers of polyester resin and fabric were drape formed over the mold until a shell thickness of about 1/16 inch was obtained. The shell was stiffened by incorporating a locally forged metal band into the rim of the plastic shell during the shell forming operation. After the shell had been left to dry (cure) for several hours it was lifted from its mold and the inside surface lined with reflective material. In this case small squares of glass mirrors were cemented to the shell. The basic construction of the cooker was the same as that presently used in Niamey, Niger. In the Mexican village situation the horizontal axis cooking support was formed from iron strips by the village blacksmith. The "U" frame support was fabricated from wood by local carpenters. With the exception of the polyester resin which was obtained in Mexico City, all other materials were provided locally. The process of cooker building took between 8 and 32 hours of labor and the cost of materials was about $15.

Advanced Solar Cookers

In order to overcome the problems of cooking in the direct sunshine and of cooking only when the sun is shining some advanced solar cooking concepts have been developed. The first involves the use of a heat transfer system to permit cooking to be done in a shelter. The second involves the use of some type of energy storage system which permits the cooking to be done in the evening or at other times when the sun is not shining.

Heat Transfer Systems

Various types of heat transfer systems have been proposed for bringing the heat generated in a solar collector into a sheltered area where the heat can be used for cooking. As previously discussed, heat transfer systems form the basis for most solar thermal systems used to operate mechanical devices. For example, in the SOFRETES water pumping system the hot water from the flat plate collector is transferred to the evaporator where the collected heat is used to vaporize an organic liquid. In a similar manner, flat plate collectors have been used to heat water to produce steam which in turn heats a remote hot plate. However, the maximum temperature obtainable with a flat plate collector is of the order of 140° C and at this temperature the efficiency of the system is very low. One example of such a system is illustrated in Figure 6. This photograph was taken at the solar energy laboratory of the University of Khartoum, Sudan. The heat transfer pipe leads from the top of the sloping collector down to a hot plate located to the left of the collector support structure. The collector consists of a series of longitudinal pipes which run the length of the collector and are connected in parallel by headers at the top and bottom. Water is permitted to fill about three quarters of the length of the tubes and is sealed off by means of a valve. The system is then allowed to reach thermal equilibrium which is about 140° C at 3-1/2 atmospheres absolute pressure. By opening a valve leading to the hot plate the steam is allowed to condense on the hotplate releasing the heat of vaporization to the plate. The problems associated with this collector are (a) the relatively low hot plate temperature which would be suitable only for light stewing or water heating chores (b) the low efficiency of the system and (c) steam under pressure presents potential safety hazards in the village environment. A similar system was developed at Brace Research Institute. In this case the heat from the steam passes directly into a cabinet heater or oven.

In order to increase the temperature of the cooking unit, concentrating solar collectors, usually parabolic or cylindrical trough, have been proposed to heat oil which is circulated through a pipe or tube type receiver (heat exchanger) located at the focal line of the trough. The heated oil is then used to transfer the collected heat to a hot plate or stove located in the dwelling or other sheltered area. Oil is proposed instead of water in these heat transfer systems because it will remain a liquid at atmospheric pressure at the temperature provided by the trough collectors (175 - 200° C) Because line-focus, linear trough collectors are used in this concept some. type of one-axis tracking is required in order to keep the concentrated energy focused on the heat transfer pipe. This may be provided through a weighted pulley arrangement in which the downward movement of a driving weight is caused to coincide with the speed of flowing sand in an hour-glass (Figure 12). Through this pulley action the rotation of the trough follows the east-west motion of the sun at 15 degrees per hour. Figure 13 shows a concept which uses an "automatic" tracking system. A bimetal "heliotrope," in the axis of the trough, reacts to direct heat from the sun to rotate the trough until a "sun-shade," attached to the trough, moves between the "heliotrope" and the sun, casting its shadow on the "heliotrope."

Figure 11 is an artist's rendering of the solar cooking heat transfer/ storage system proposed by Farber. This system differs from the previous two in that two separate heat transfer loops are used. One loop receives energy from the collector and transfers it to a thermal storage medium (phase change material). The second loop receives heat from the storage medium and transfers it to the cooking area. This system is more complex than the others shown and involves pumps and valves which would provide the "automatic" type of operation required in a technical society where solar energy was being used to replace or augment energy sources such as electricity or gas. However, such systems are likely to be too complex and expensive for the rural people. Unfortunately, this cannot be determined at this time since these systems are only in the design stage and prototype systems have yet to be built.

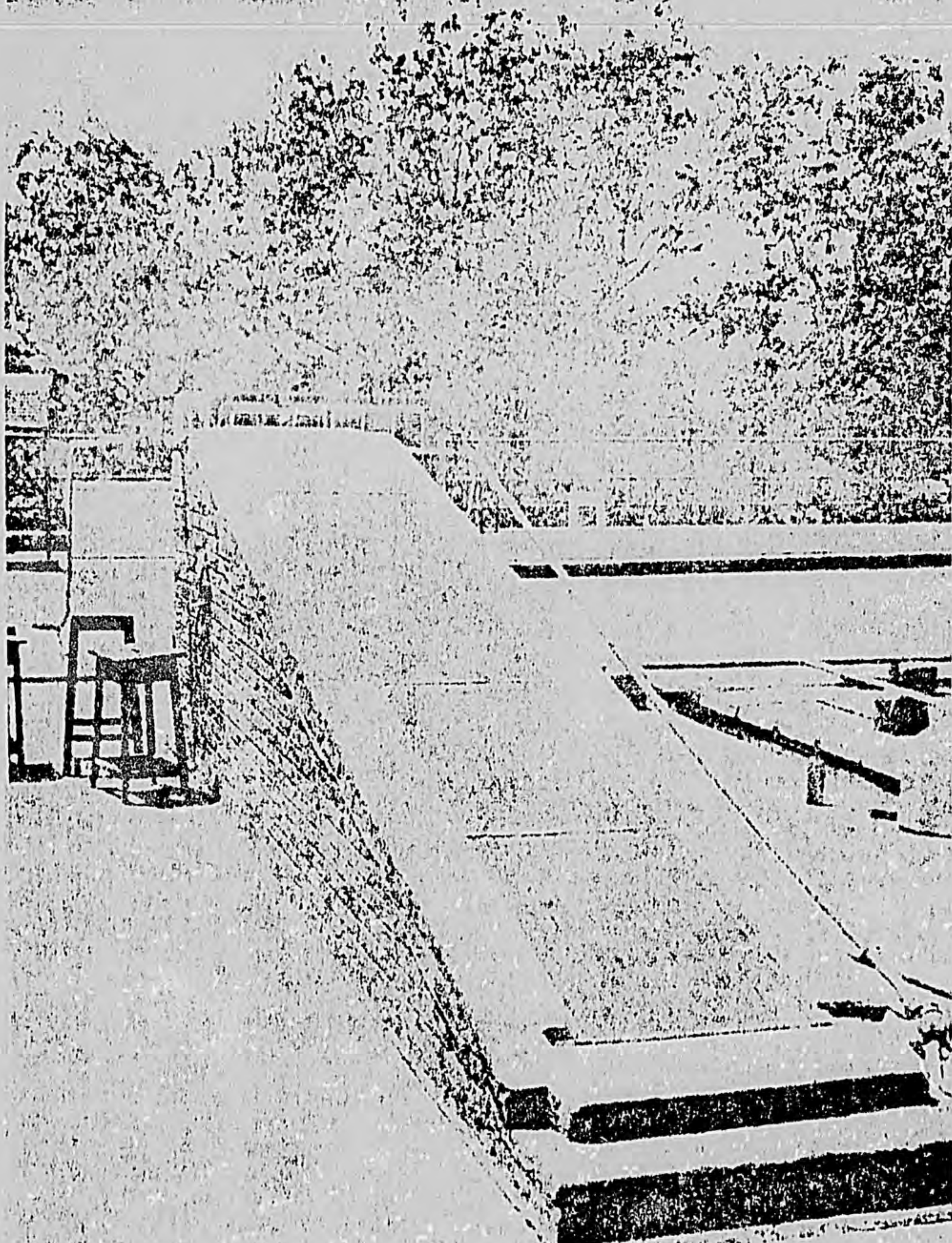


Figure 6. Photograph of a Solar Steam Hot Plate System at the Solar Energy Laboratory of the University of Khartoum, Sudan.

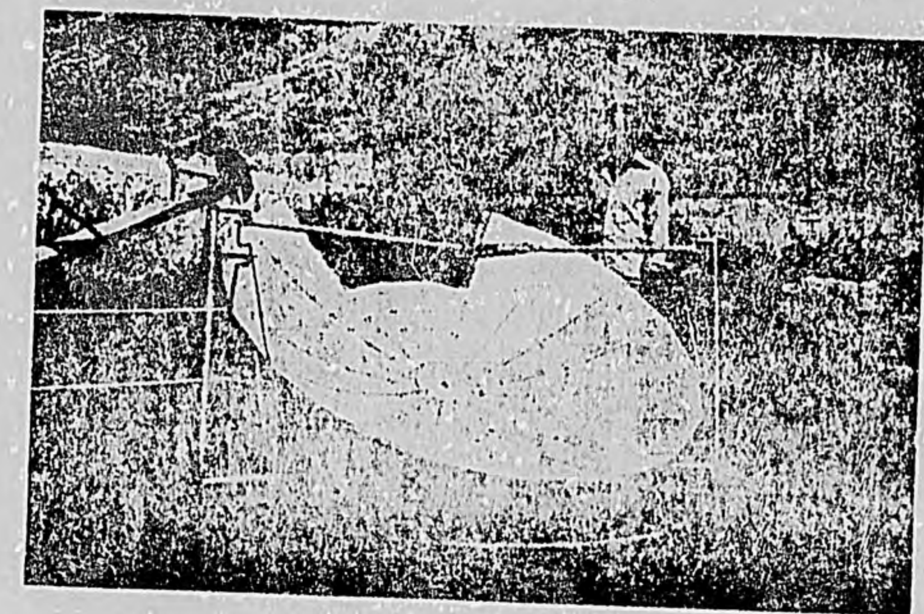


Figure 7. Photograph of Parabolic Dish Solar Cooker at the Solar Energy Laboratory in Bamako, Mali.

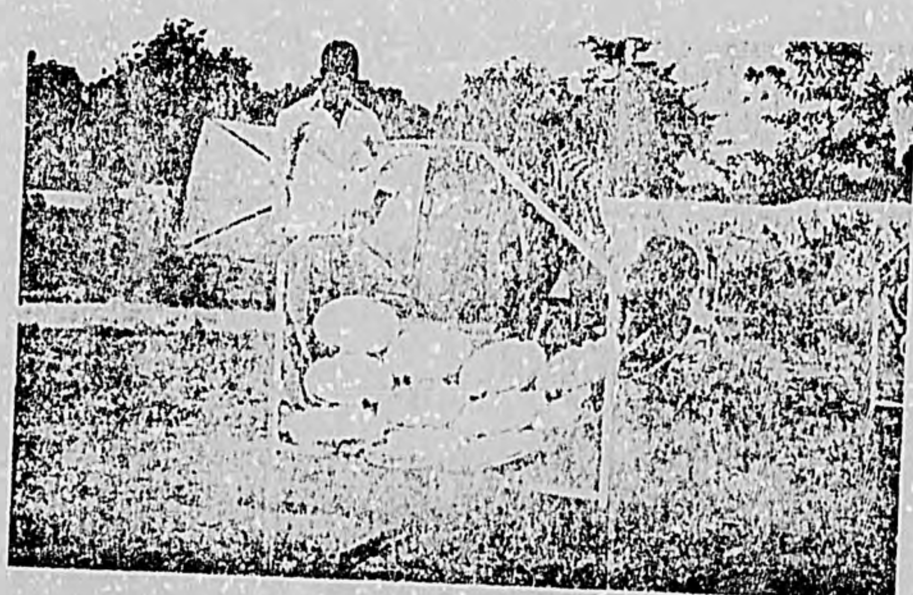


Figure 8. Photograph of Multi-Mirror Solar Cooker at the Solar Energy Laboratory in Bamako, Mali.

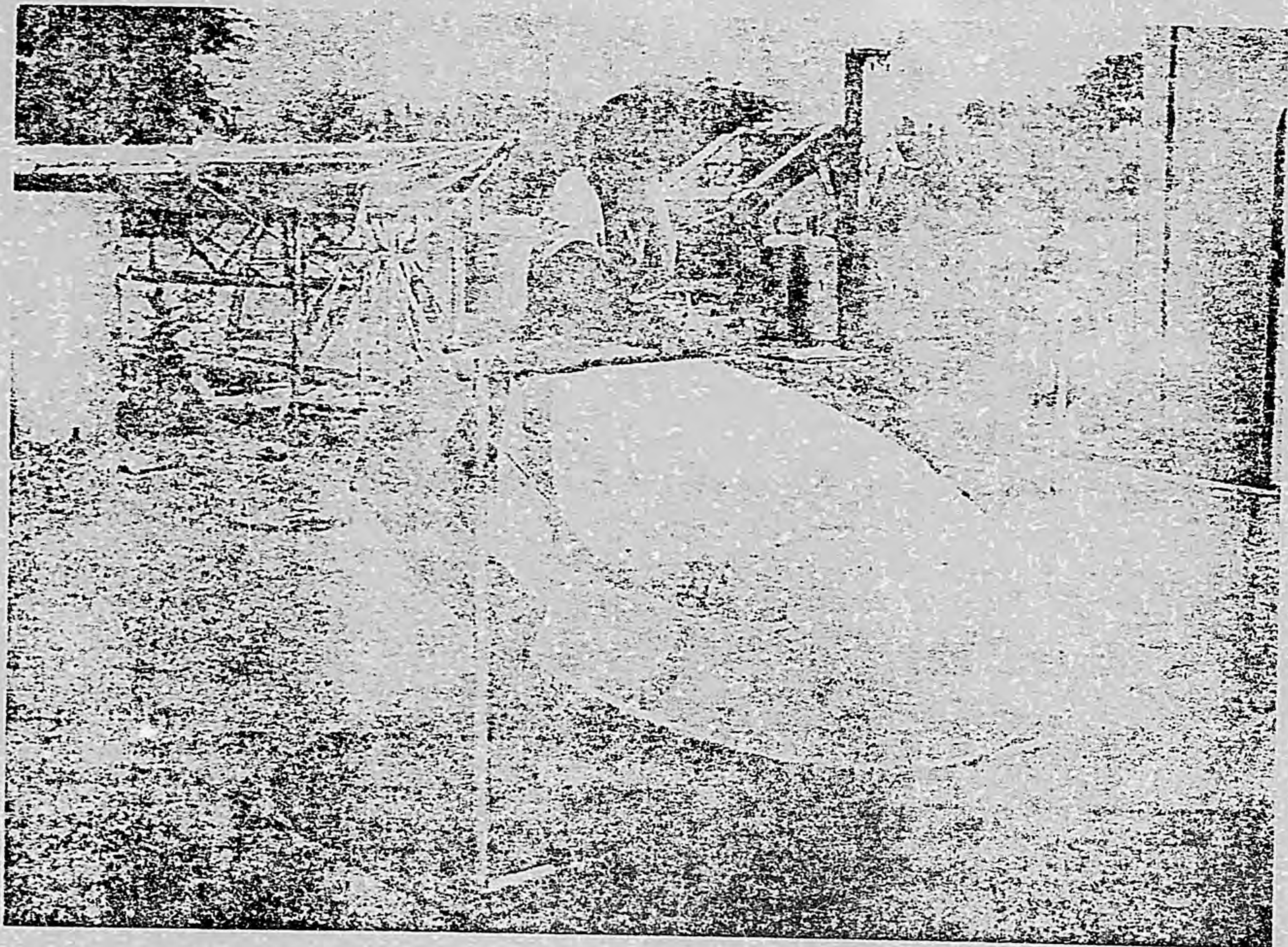


Figure 9. Photograph of a Parabolic Dish Solar Cooker at the Laboratory of l'Office de l'Energie Solaire, ONERSOL, in Niamey, Niger.

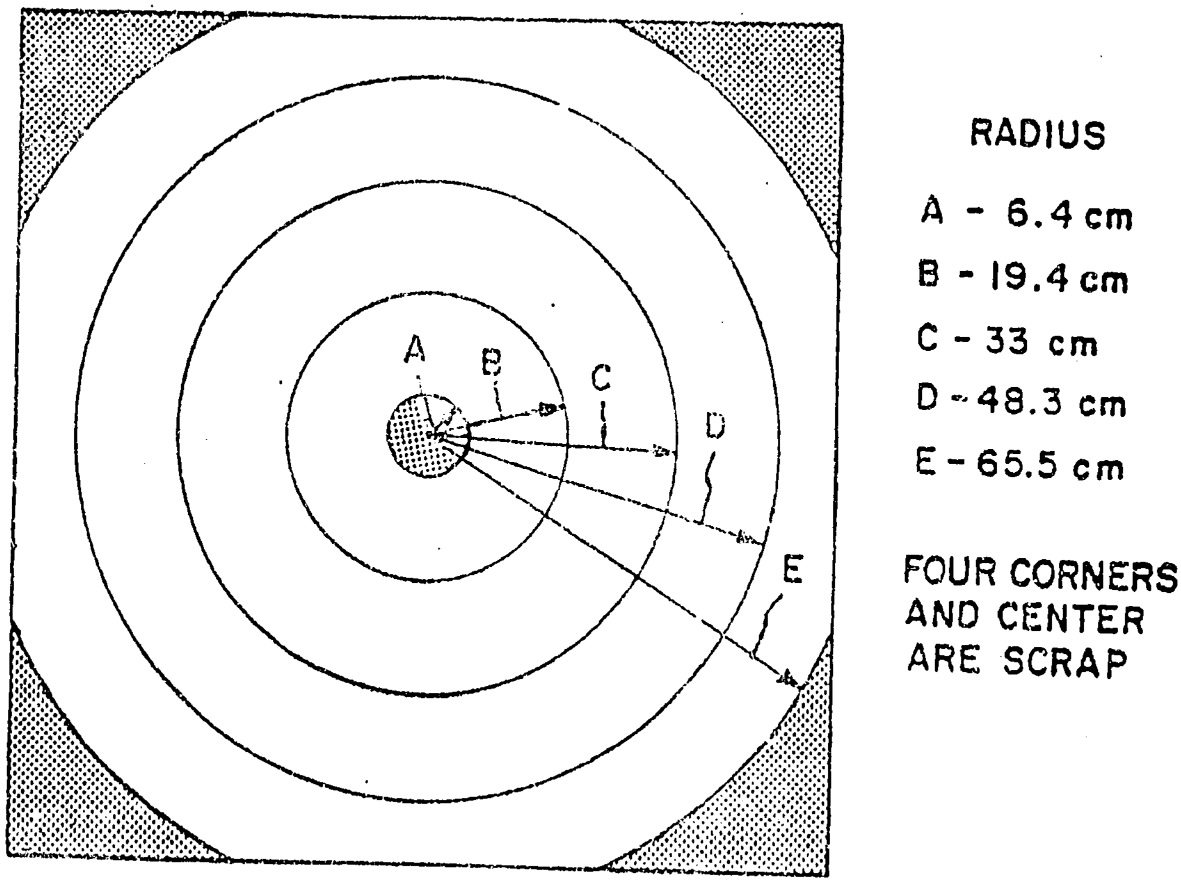


Figure 10(a)

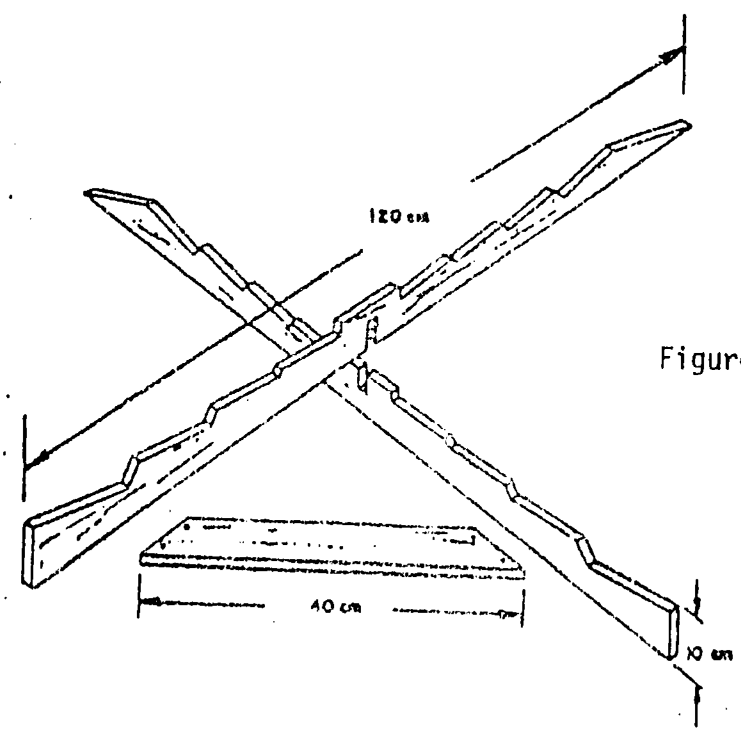


Figure 10(b)

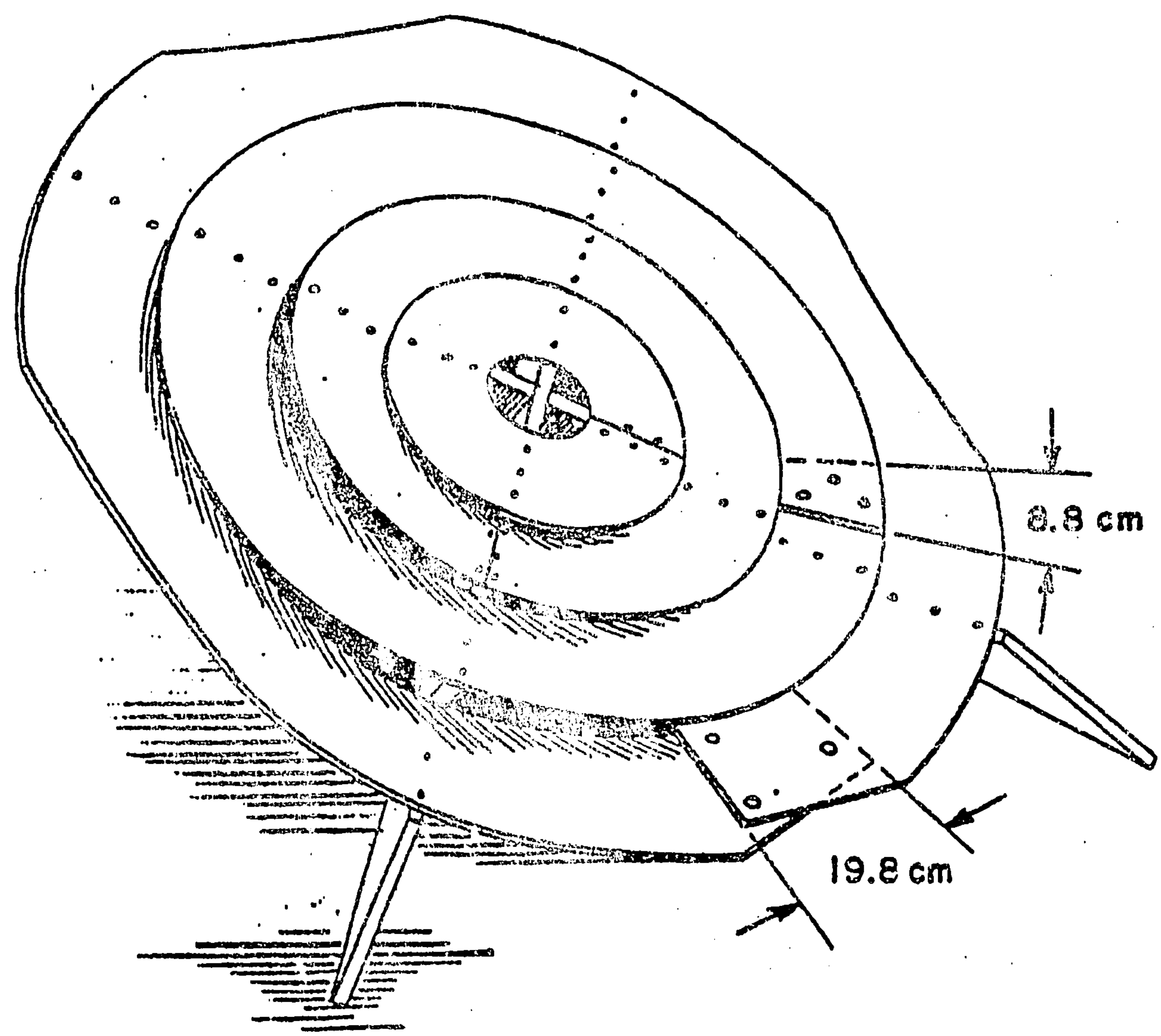


Figure 10(c)

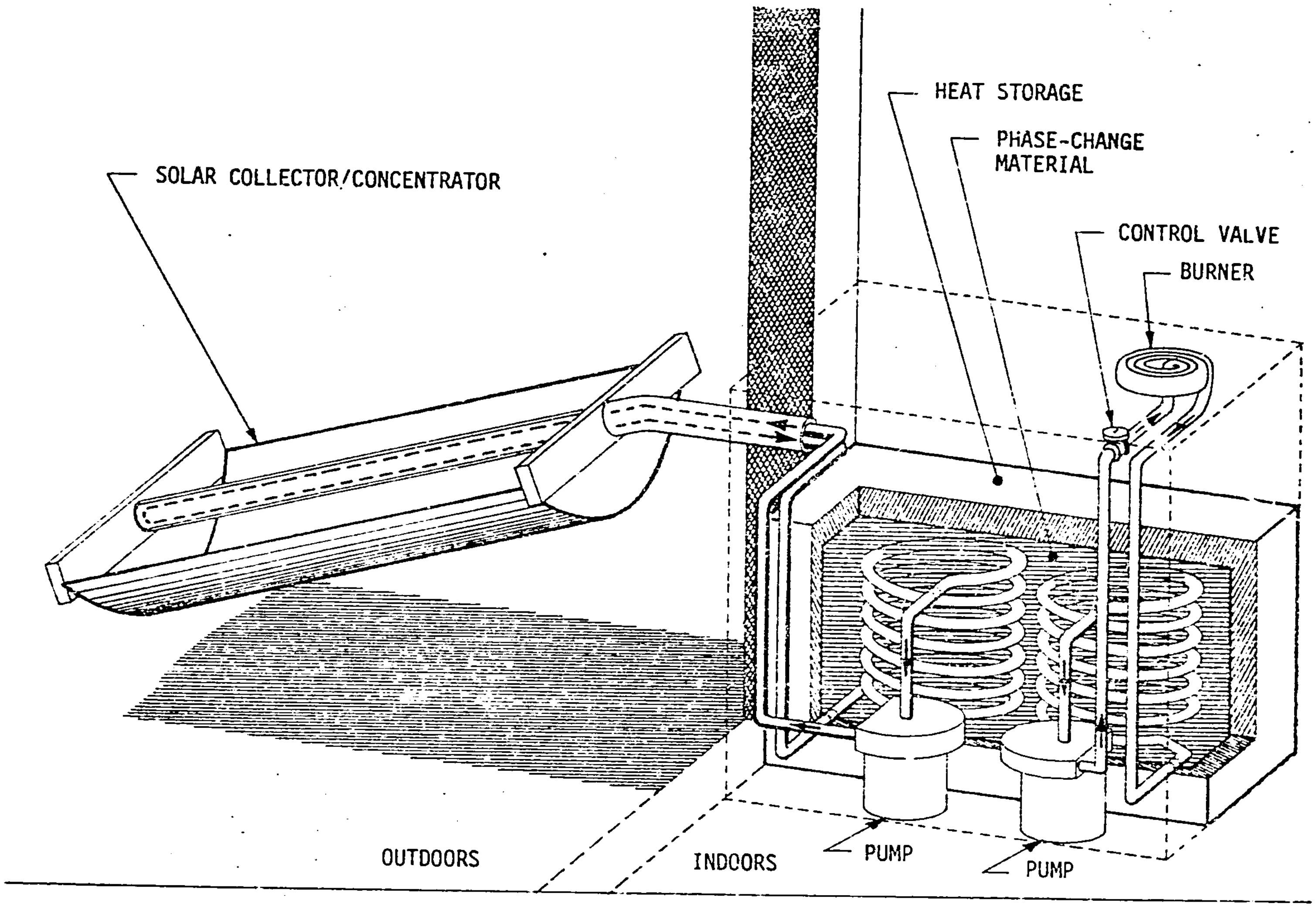


Figure 11. Artist Concept of the Solar Cooking Heat Transfer/Storage System Proposed by Farber.

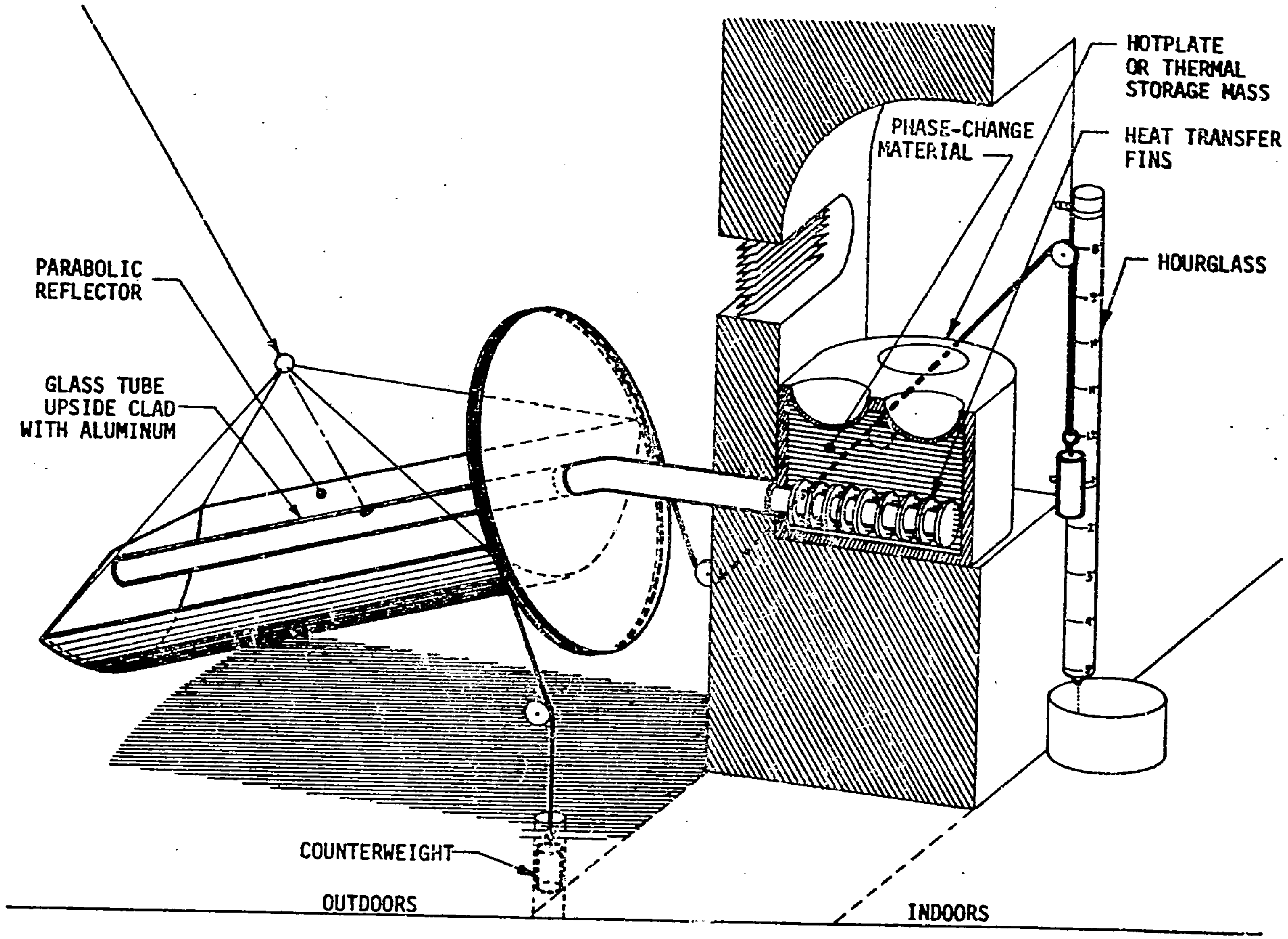


Figure 12. Artist Concept of a Heat Transfer Solar Cooking System Proposed by Stam.

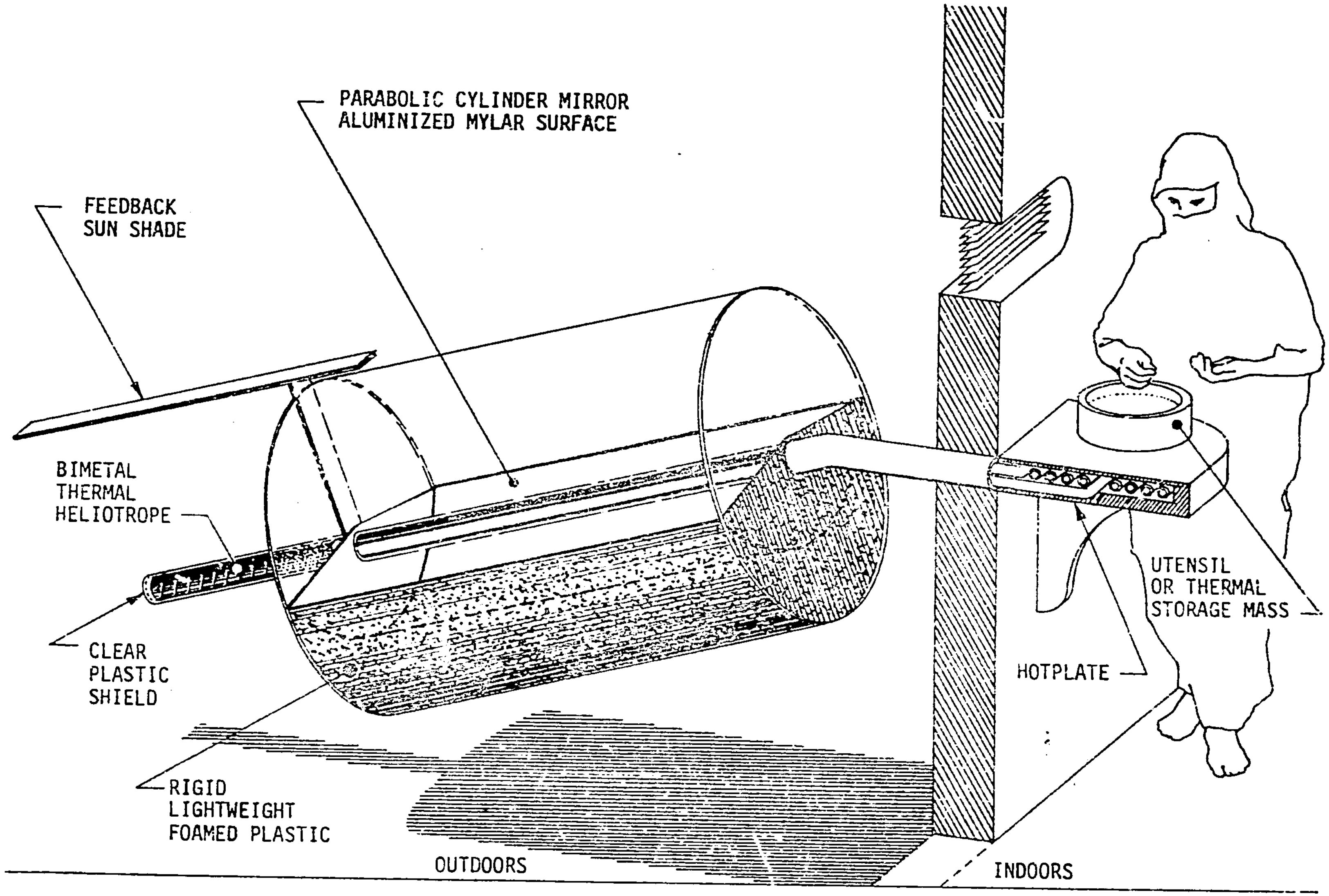


Figure 13. Artist Concept of an "Automatic" Tracking Solar Cooking System Proposed by Swet.

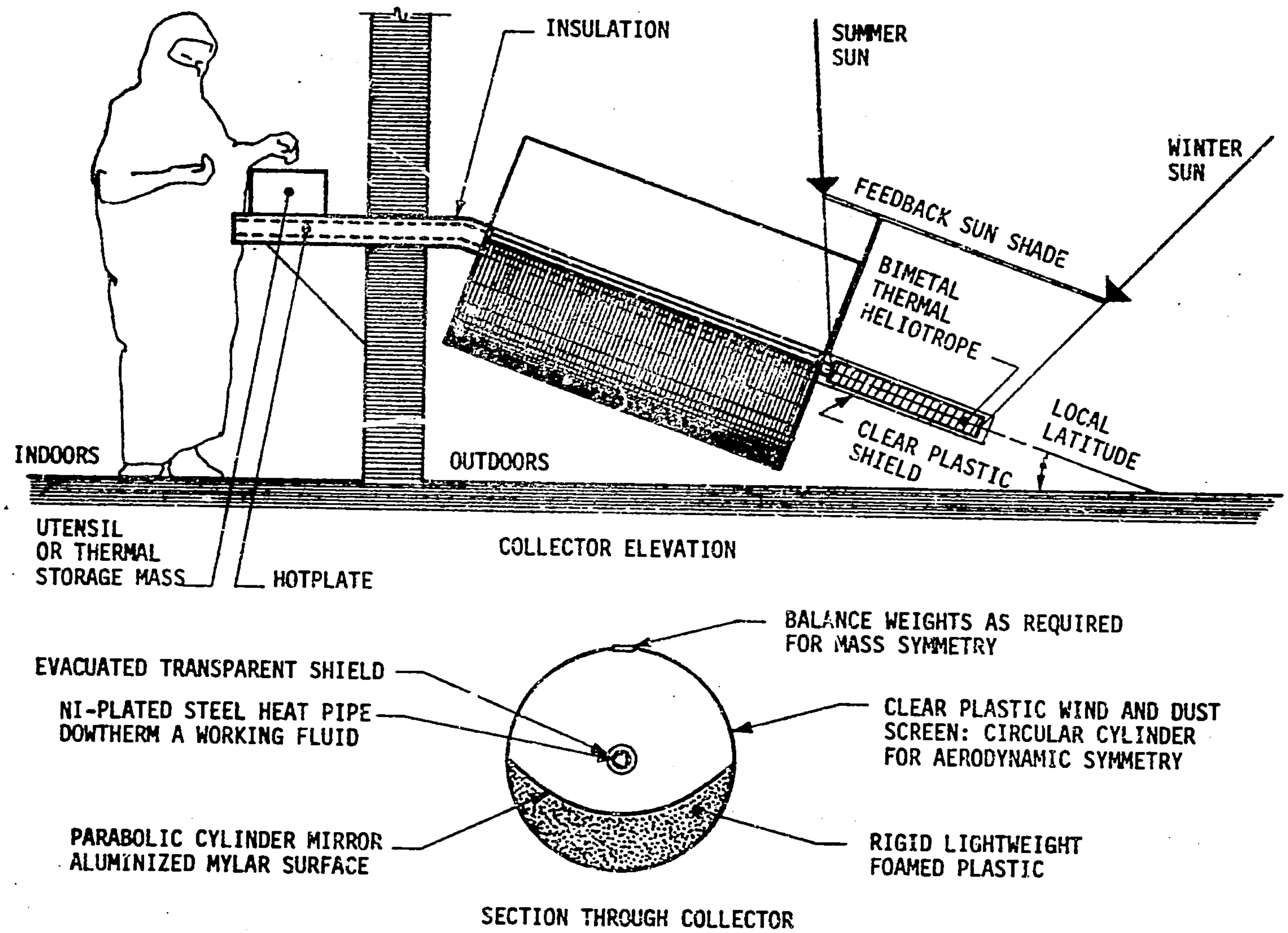


Figure 14. Artist Concept of Heat Transfer Solar Cooking System Using Automatic Tracking Proposed by Swet.

Ovens and Food Warmers

Solar ovens or food warmers typically use some concentration of solar radiation in an insulated chamber with a transparent cover, preferably glass, used to trap thermal infra-red radiation. Although the temperatures attained are lower than those achieved with solar concentration, solar ovens and food warmers have several advantages:

- They can be used for baking.
- They require less focusing and orientation.
- Cooking pots are protected from the wind, and also from insects.
- Several pots or pans can be used at the same time.
- They trap diffuse radiation as well as direct, therefore permitting cooking in partly cloudy and hazy conditions.
- Food can be kept warm after sunset.

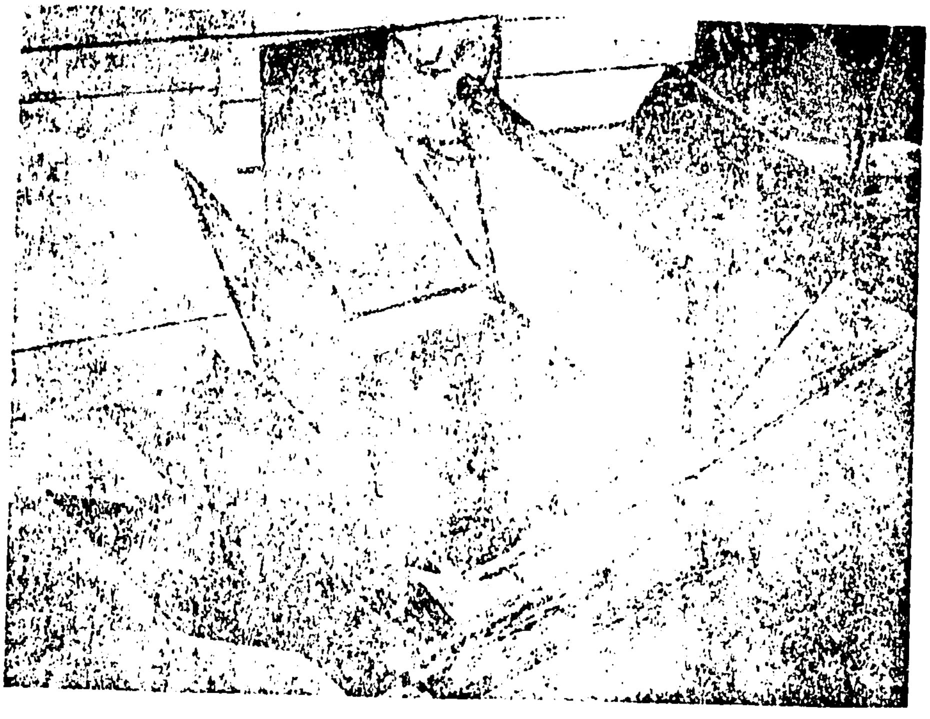


Figure 15. Photograph of a Solar Oven Designed by Telkes.

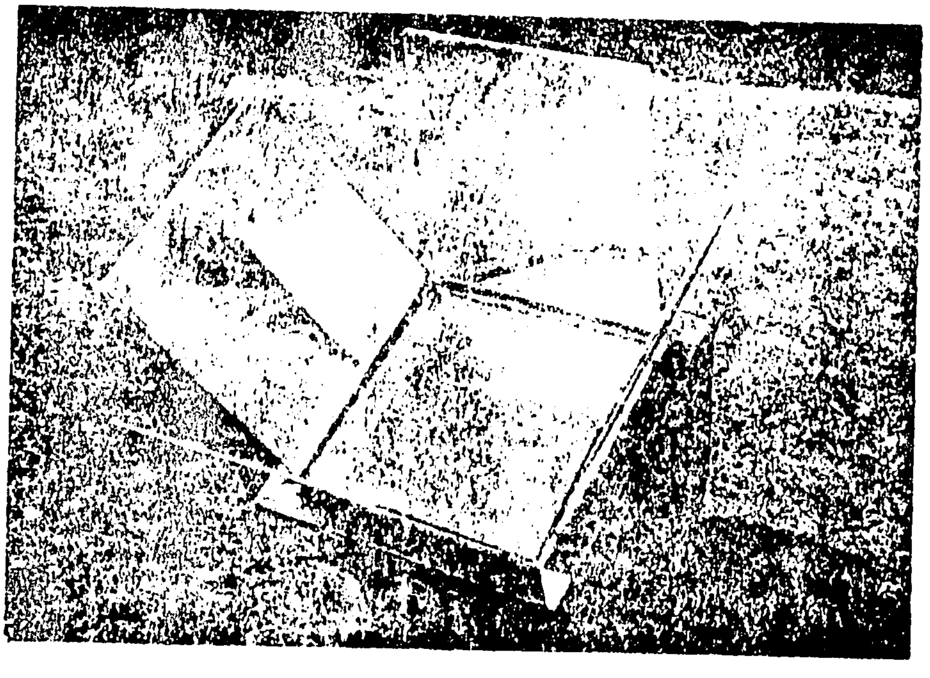


Figure 16. Photograph of a Solar Stove Designed by Telkes.

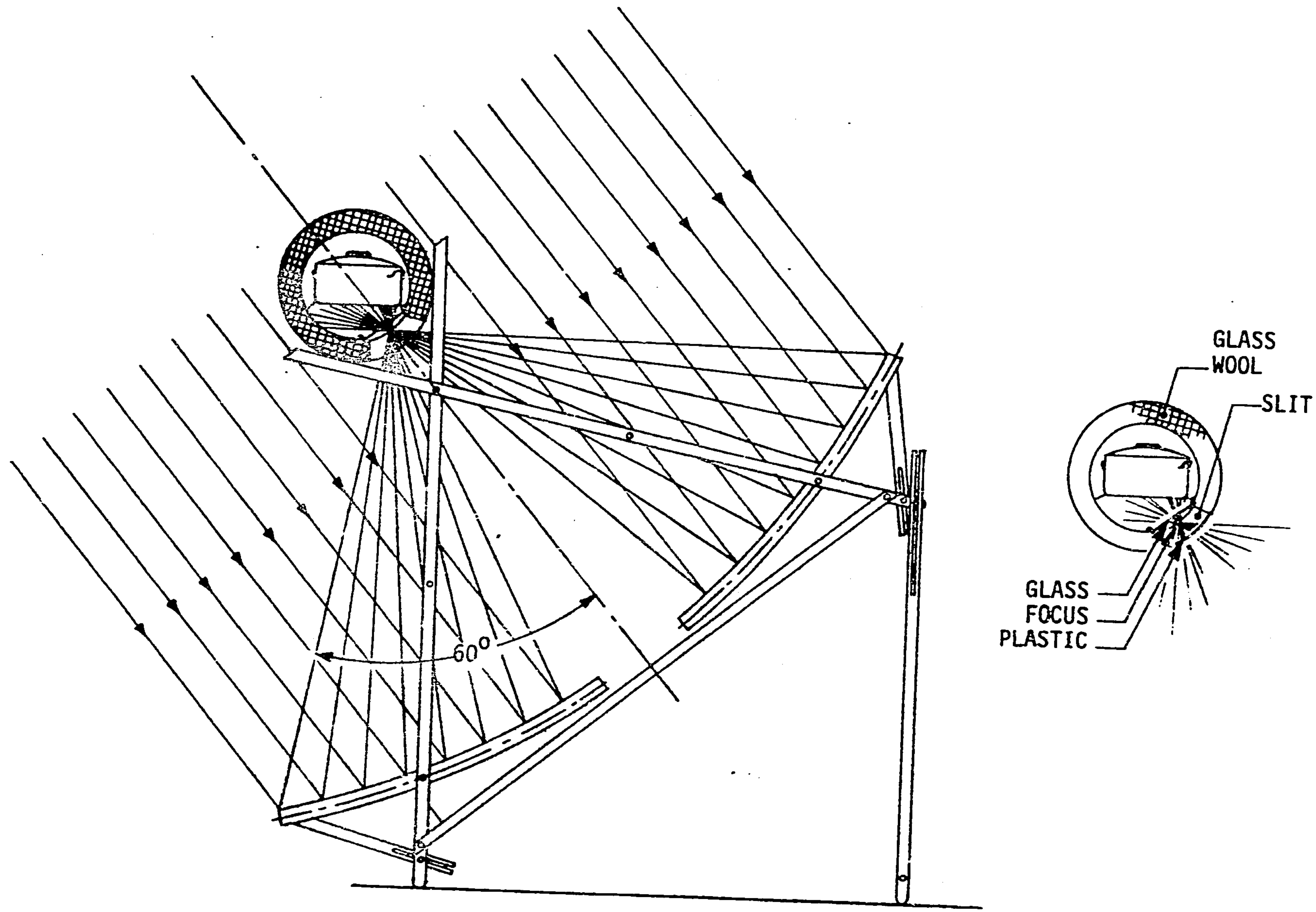


Figure 17. Schematic of a Solar Oven Designed by Prata.

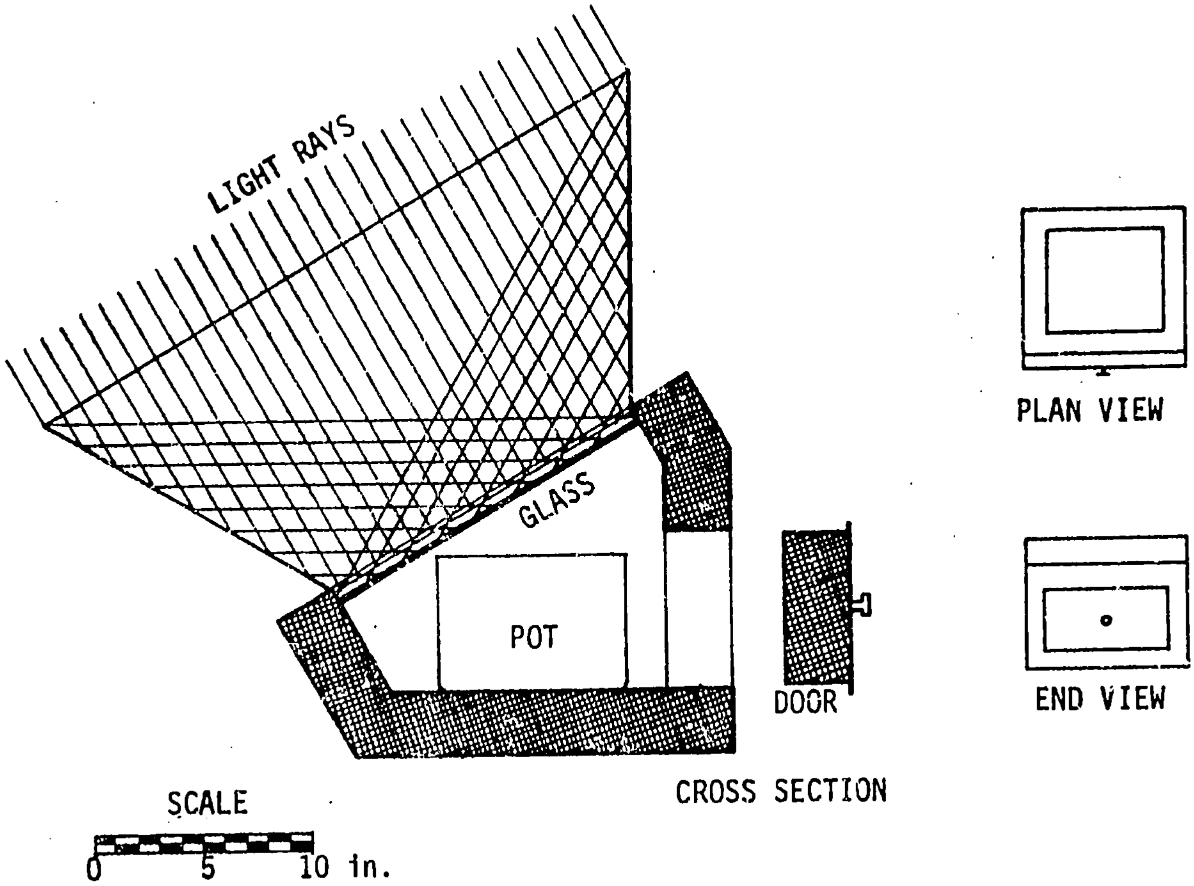


Figure 18. Schematic of a Solar Oven Designed by Telkes.

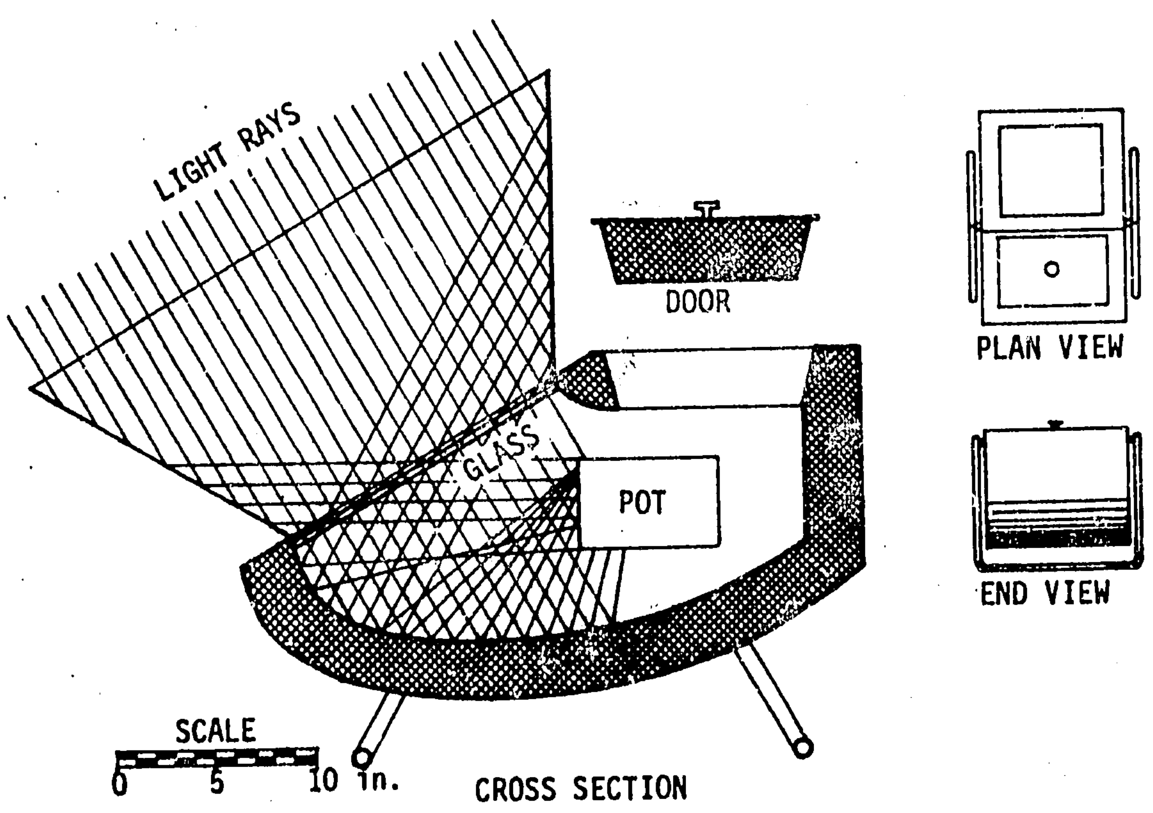


Figure 19. Schematic of a Solar Stove Designed by Telkes.

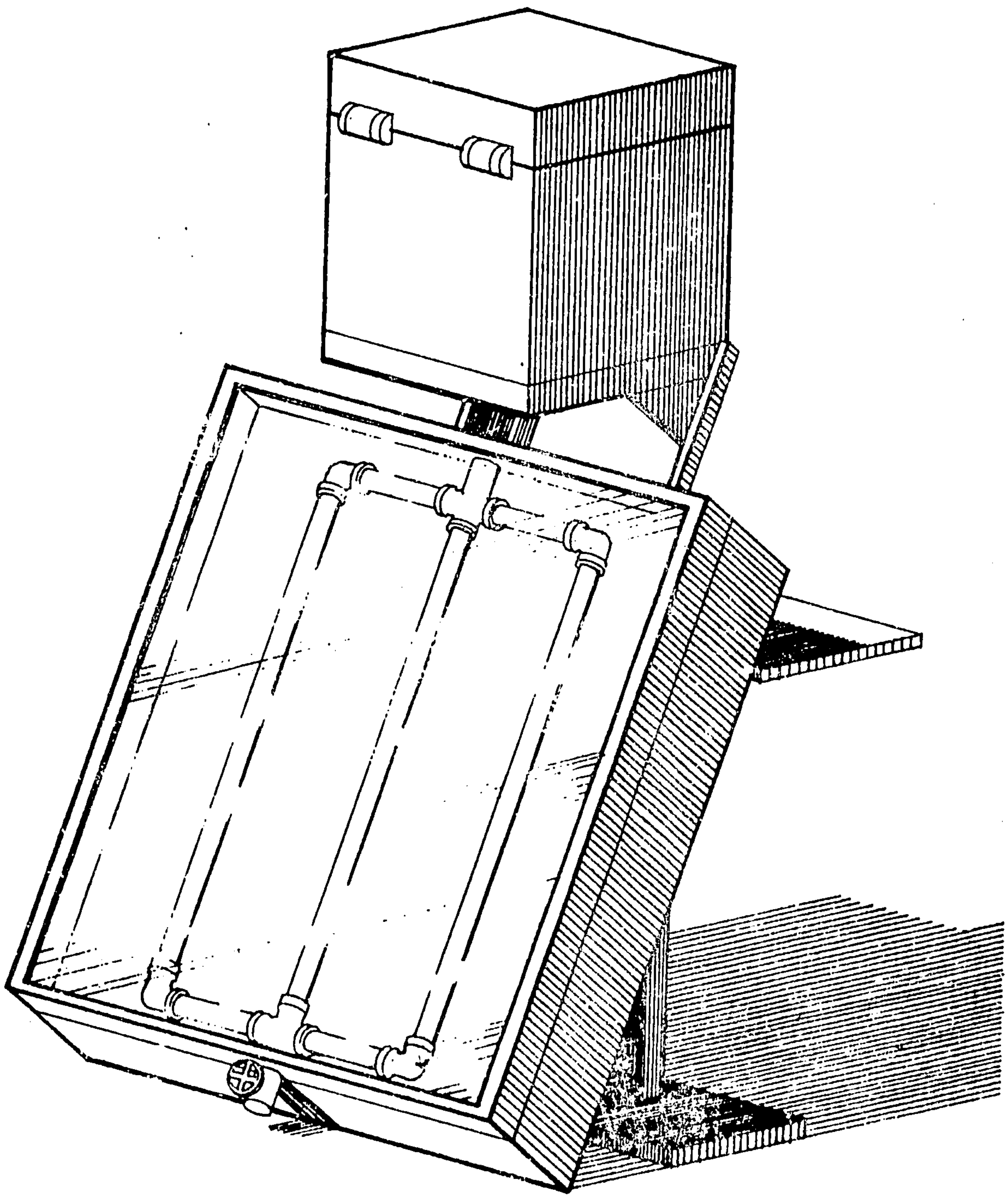


Figure 20. Perspective View of Solar Steam Cooker Developed By Brace Research Institute.

Table 1. SURVEY OF SOLAR COOKERS [13]

	Stam (Proposed)	Duffie Lof, Back	Lof, Fester	Abou-Hussein	Telkes Andrassy	Prata	Ghai	Tabor	Farber, Ingley (Proposed)	Swet (Proposed)
Cooker Type	Focusing	Focusing	Focusing	Oven	Oven	Combination	Focusing	Focusing	Hot Plate	Hot Plate
Reflector Type	Parabolic Cylinder	Paraboloid	Parabolic-Umbrella	Internal Flat Plates	External Flat Plates	Parabolic Cylinder	Paraboloid	Paraboloid	Parabolic Cylinder	Parabolic Cylinder
Reflector Material	Aluminum Foil on Plastic	Aluminized "Mylar" Plastic Film on Polystyrene Shell	Aluminized "Mylar" Laminated to Fabric, on Umbrella Frame	Polished Aluminum Sheets	Anodized Aluminum Sheets (Coated Aluminum Foil)	Nickel Plated Brass Sheet (Nickel Plated Aluminum)	Anodized Aluminum Sheets	Copper Backed, Silvered-glass Plane Mirrors	Polished Aluminum Metal Sheets	Aluminized "Mylar" on Lightweight Foamed Plastic
Reflector Dimension	1.6m x 0.98m	1.22m dia	1.17m dia	(4) 1288 cm^2	(4) 43cm sq	(2) 0.5m x 0.8m	0.11m dia	(12) 29.3cm dia	n.r.	1.22m x 0.91m
Effective Solar Collection Area	n.r.	$1.07m^2$	$1.02m^2$	$0.36m^2$	$0.56m^2$	$0.74m^2$	$0.67m^2$	$0.79m^2$	n.r.	n.r.
Reflector Focal Length	25 cm	46cm	46cm	—	—	~1.05m	45.7cm	76cm	n.r.	23cm
Oven Window	—	—	—	$0.36m^2$ Double Glass	$0.19m^2$ Double Glass	$0.06m^2$ Single Glass	—	—	—	—
Cooking Area	(2) 30cm dia*	~20cm dia	~23cm sq	~20cm sq*	~25cm sq*	~20cm x 50cm*	68cm sq	~29cm dia	n.r.	n.r.
Effective Solar Intensification	n.r.	~34	~20	~3*	~3*	~12*	n.r.	~28	n.r.	n.r.

*Computed or Estimated by Lof.

n.r. - Not Reported.

~ - Approximately.

Table 1. (Continued)

SURVEY OF SOLAR COOKERS

	Stam (Proposed)	Duffie Lof, Beck	Lof, Fester	Abou-Hussein	Telkes Andrassy	Prata	Ghai	Tabor	Farber, Ingley (Proposed)	Swet (Proposed)
Minimum Time Req'd to Heat one Liter of Water 20° to 100° C	n.r.	15 min @ 15 min (minimum)	22 min	n.r.	~30 min @ 46 min (minimum)	26 min*	n.r.	12 min	n.r.	n.r.
Effective Cooking Power kW	n.r.	0.4-0.5 @ 0.28 avg @ 0.38 max	0.25-0.4	n.r.	0.15-0.2* @ 0.10 avg @ 0.12 max	0.15-0.25*	0.13	0.5	0.27-0.6	0.6
Food-Cooking Performance	n.r.	Good	Good	n.r.	Good	Good	Good	Good	Good	n.r.
Approximate Weight, kg	n.r.	10	3	n.r.	~20*	12-18	n.r.	20	n.r.	n.r.
Thermal Storage Considered	Yes	No	No	Yes	Yes	No	No	No	Yes	Yes
Type of Storage Media	Mag. Chloride, Mag. Palmitate	—	—	Sulfate Na,K,Ca	Anhydrous Alkaline Sulfate	—	—	—	Cotton-seed Oil	$LiNO_3$
Temp of Storage Media	Heat of Fusion	—	—	191°-239° C	300°-400°F	—	—	—	800° F	500° F
Total Cooking Capacity	n.r.	~4 kg*	~2 kg*	n.r.	~2 kg*	~4 kg* (2 vessels)	~2 kg*	n.r.	n.r.	n.r.

@ - F.A.O. Tests.

*Computed or Estimated by Lof.

n.r. - Not Reported.

~ - Approximately.

Table 1. (Concluded)

SURVEY OF SOLAR COOKERS

	Stam (Proposed)	Duffie Lof, Beck	Lof, Foster	Abou-Hussein	Telkes Andrassy	Prata	Ghai	Tabor	Farber, Ingley (Proposed)	Swet (Proposed)
Portability	None	Good*	Excellent*	Good*	Good*	Fair*	Fair*	Fair	None	None
Need for Positioning During Cooking	None*	Frequent (15-30 min)	Frequent (15-30 min)	Occasional (30-60 min)	Occasional (30-60 min)	Moderate (25 min)	Frequent (15-30 min)	None	None	None
Suitability For Baking & Roasting	Fair*	Poor*	Poor*	Good*	Good*	Good*	Poor*	Poor	Good	Good
Suitability For Stewing & Frying	Good*	Good*	Good*	Fair*	Fair*	Fair*	Fair*	Good	Good	Good
Durability	n.r.	Good*	Fair*	Very Good*	Very Good*	Good*	Good*	Good	Good	Good
Use of Native Materials	Poor	Fair*	Fair*	Fair*	Fair*, #Good*	Fair*	Good*	Good	Good	Good
Full-Scale Cookers Constructed & Tested	No	Hundreds	See Below	Yes	Yes	Yes	Yes	Yes	No	No
Field Testing	No	Extensive	See Below	No	Some	No	Some	Yes	No	No
Commercial Scale	No	No	Hundreds	No	No	No	No	No	No	No
Approximate Cost or Price	n.r.	$16 (Factory)	$30 (Factory)	n.r.	n.r.	~$35 (Factory)	$14-18	$7-8	n.r.	$100

*Computed or Estimated by Lof.

n.r. - Not Reported.

~ - Approximately.

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ENERGY USE IN AGRICULTURE

According to the World Food Council [8], despite significant increases in food production and improved distribution in some countries in the last few years, the food situation remains precarious for millions of people in many developing countries. Global production of cereals has fallen since 1978/79 and, according to estimates by the Food and Agricultural Organization (FAO), world stocks of grain as a proportion of consumption are at their lowest level in five years. While some developing countries have achieved notable gains in production, others, including some of the least developed and most seriously affected countries, have actually experienced a decrease in per capita food production. In 1974, the World Food Conference set a goal of eradicating hunger within a decade [9]. That goal is no longer considered feasible. The International Development Strategy now projects the achievement of the goal by the end of this century.

In many parts of the world, agriculture depends primarily on energy from animals, people, and of course, the sun. If conditions are favourable traditional agricultural practices can produce very high yields. Modern agriculture, however, is much more energy intensive. It is primarily because of its large energy subsidy that agricultural yields have increased so rapidly in the U.S. and Europe during this century. In 1900 in the U.S., a single farmer could feed about five people. By 1940 he could feed ten, and in 1960 he was feeding 25. Today, food for almost 50 people can be provided by just a single farmer--with a little help from fungicides, herbicides, rodenticides, insecticides, antibiotics, vaccines, fertilizers, irrigation systems, and a large assortment of fuel-burning mechanical equipment.

In the developing countries crop yields are generally lower than in North America and Europe; agriculture is labour-intensive and inputs of commercial energy are usually small. However, the contribution from traditional fuels and from animal power may be significant.

According to the World Bank, there is no readily identifiable yield-increasing technology other than the improved seed-water-fertilizer approach that has been dubbed the Green Revolution. Increased production is unlikely to come from the cultivation of larger areas of land. According to the Global 2000 report [7], arable land per capita is expected to actually decrease over the next twenty years as population rises from 4.5 billion at the present time to about 6.5 billion by the year 2000.

The introduction of high-yielding varieties (HYV's) of wheat and especially rice is an essential part of agricultural policy in areas, such as most of Asia, where land is scarce. In the next two decades it is expected that close to 75% of all increases in the output of basic staples will have to come from yield increases. However, the performance of these high-yielding varieties of wheat and rice is dependent on adequate supplies of fertilizers, pesticides and water. The HYV's demand a large energy subsidy in order to achieve their spectacular yields. The Green Revolution means energy-intensive agriculture.

In these notes we want to examine some basic issues: Can agricultural productivity in the developing countries be significantly improved by a shift to HYV's of wheat and rice, supported by more energy-intensive agricultural practices? And, if so, how much energy will be required and where will it come from?

To examine these questions we will compare and contrast the structure of the agricultural systems in the U.S. with those of certain developing countries.

THE U.S. FOOD SYSTEM

The United States food system is highly productive. Yields of rice are nearly five times larger in the U.S. than those achieved by traditional methods in the Philippines; yields of corn (maize) are also five times larger than achieved by traditional methods in Mexico [1]. It is generally accepted that these gains and improvements are a result of the development of fast growing, high yielding strains of plants and animals, whose productivity, in turn, depends on energy-intensive agriculture.

For example, Figure 1 shows how farm output in the U.S. has varied over the last fifth years when plotted against energy inputs to the food system. The curve suggests that food output cannot be indefinitely raised by increasing the level of energy inputs. Productivity per unit of energy consumption is subject to diminishing returns.

The other notable characteristic of the U.S. and other industrial countries' food systems is an apparently matching decline in the labour intensity of food production. Figure 2 plots man-hours of farm work against energy use for U.S. conditions. This curve suggests that energy has been substituted for labour in agricultural practice. But again one can conclude from this graph that reductions in farm labour resulting from increasing energy inputs cannot continue indefinitely. Figure 2 is deceptive, however. While it is true that the labour force on the farm itself has fallen, the labour employed by the food processing and distribution industries has markedly increased. In effect, labour has shifted from food production to food processing, distribution, and marketing.

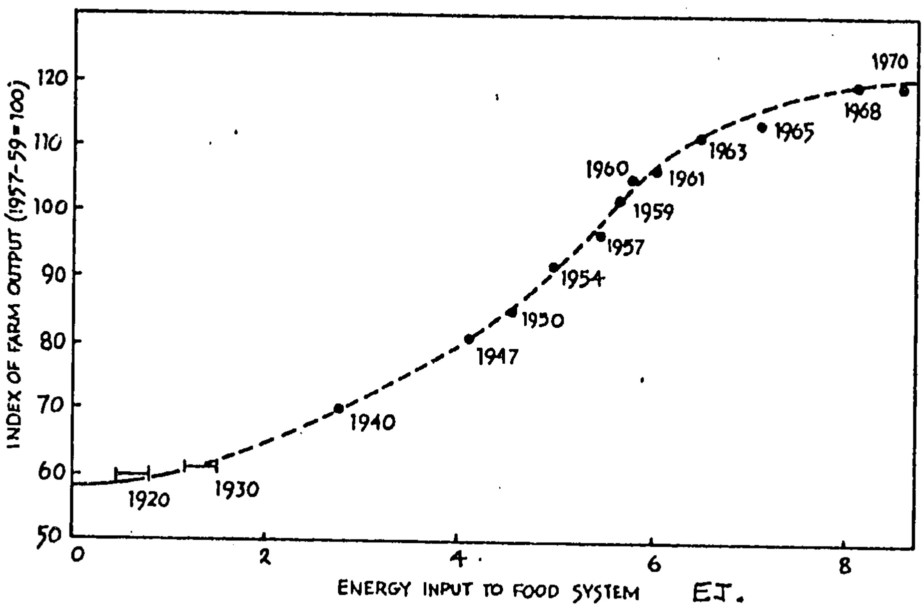


Figure 1 Farm output as a function of energy input to the U.S. food system, 1920-1970.

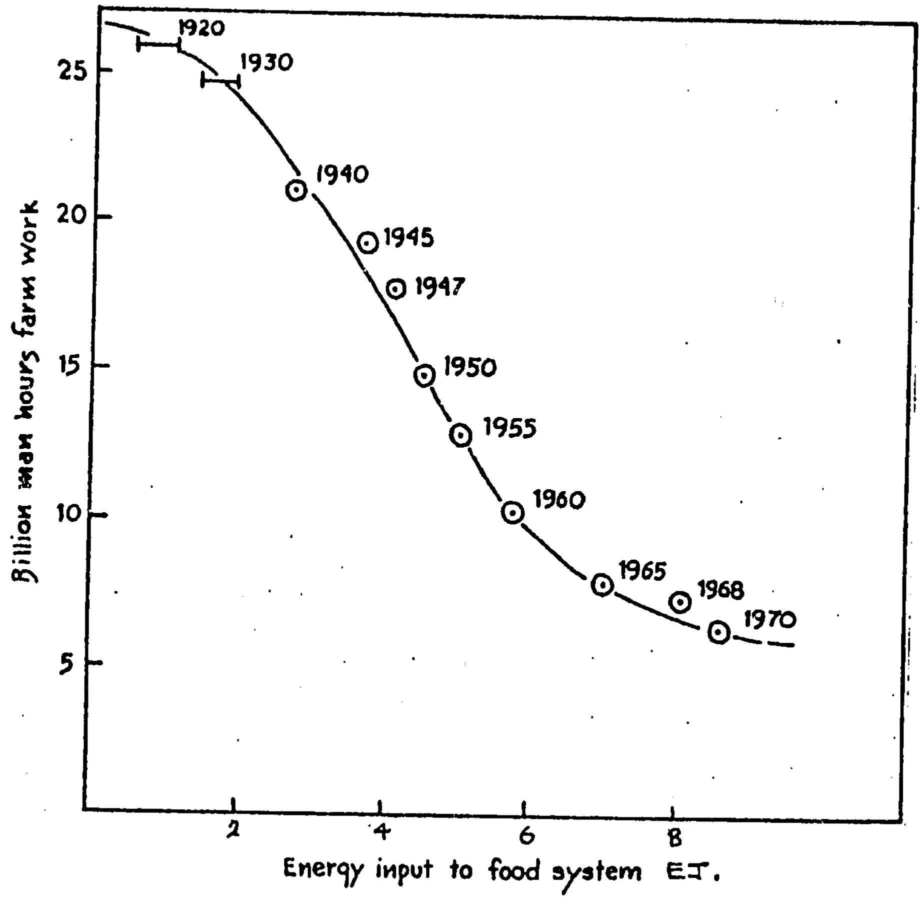


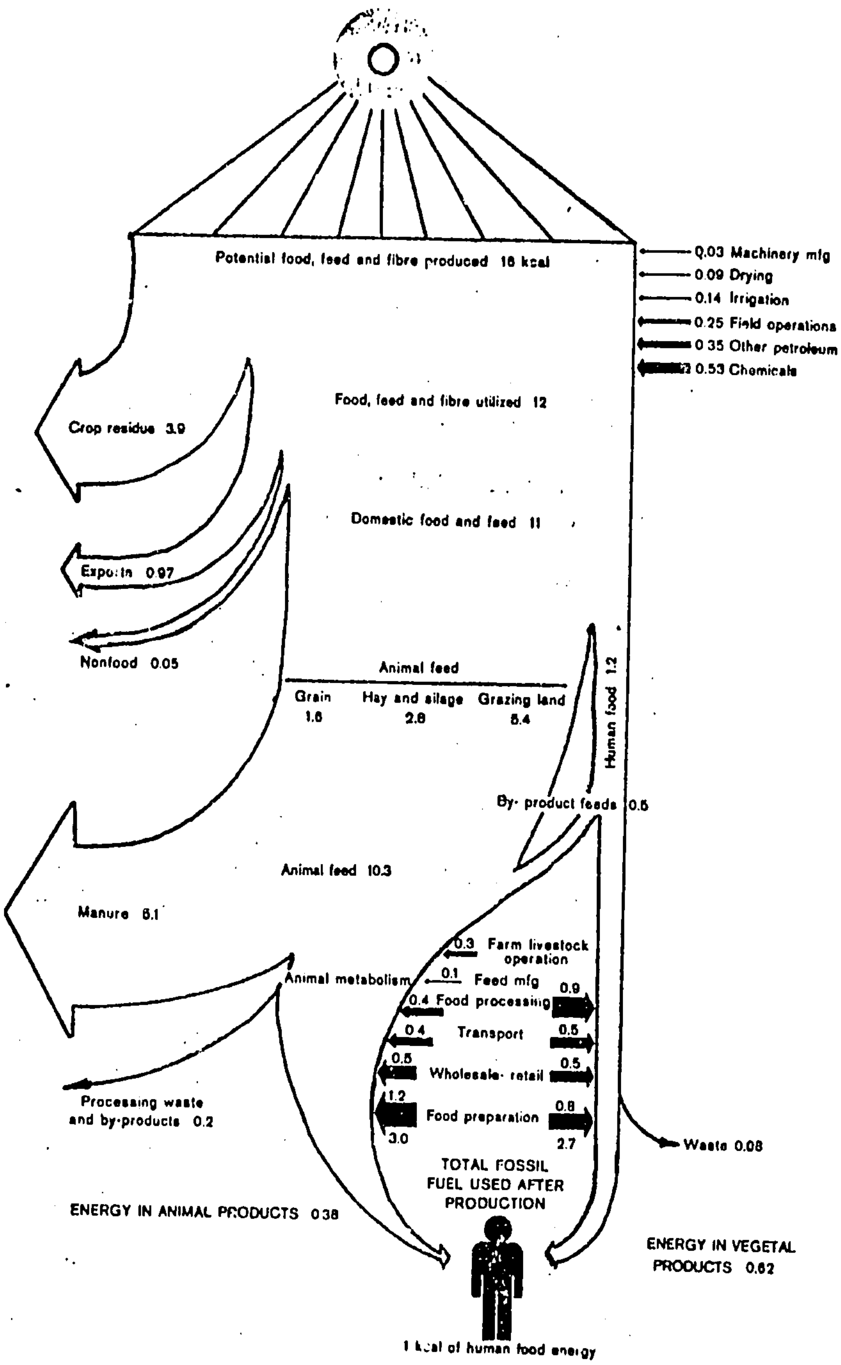
Figure 2 Labor use on farms as a function of energy use in the food system.

The food system in the U.S. uses approximately 16% of the nation's energy. Of this amount only about one-fifth is used for the production of food, i.e. on the arm. The remainder is used for processing, packaging, distribution and marketing.

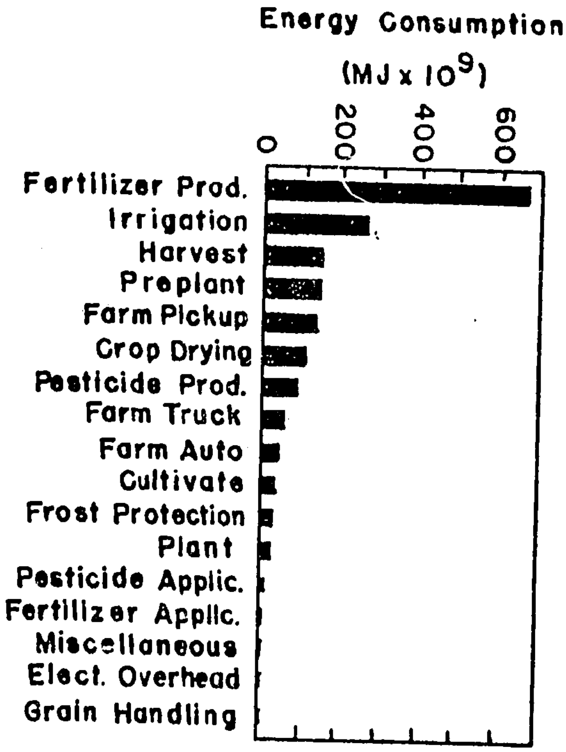
Only a tiny fraction of the incident solar energy is eventually converted to food energy by plants. For example, it takes about 28,000 units of solar energy to produce one unit of food energy [4].

The atmosphere absorbs or reflects all but 16,000 units and less than half of this is in the photosynthetically active range. Much of this either misses the plants or is reflected. Even the portion absorbed is converted only partially to chemical energy in the form of food, feed or fibre. Approximately 16 units of plant energy are ultimately synthesized from the initial 28,000 solar units. Figure 3 shows how these 16 units of energy become one unit of available energy for human consumption. Of the potentially available plant energy, about 31% is in nonfood crops such as cotton and tobacco, in crop residues such as corn stalks or straw, and in exports. Of every 16 kcal of potential plant energy in food, feed, and fibre, only 11 actually enter the U.S. food chain, 1.2 kcal being used directly for human food and 9.8 for animal feed. After processing, transport, marketing and final food preparation, 62% is consumed as vegetal products and 38% as animal products.

FIGURE 3 Energy flow in the U.S. food chain to produce one kilocalorie of human food energy



If one looks only at food production, the use of energy in this part of the food system is shown in Figure 4. The high energy cost of fertilizer production is evident. The characteristic is further indicated by Table 1, which details the energy inputs into a crop that is fairly representative of U.S. agriculture, namely corn (maize).



From CAST (1977)

FIG. 4 CONSUMPTION OF ENERGY IN DIFFERENT ASPECTS OF CROP PRODUCTION IN THE UNITED STATES, 1974

TABLE 1 ENERGY INPUTS ($MJ \cdot ha^{-1}$) IN CORN PRODUCTION[1]

Input	1945	1950	1954	1959	1964	1970
Labor[2]	129	101	96	79	62	51
Machinery	1,862	2,586	3,103	3,620	4,344	4,344
Gasoline	5,621	6,370	7,120	7,494	7,868	8,244
Nitrogen	608	1,303	2,346	3,562	5,039	9,731
Phosphorus	110	157	188	251	283	487
Potassium	54	109	521	624	703	703
Seeds for planting	352	418	195	378	314	652
Irrigation	197	238	279	321	352	352
Insecticides	0	11	34	80	114	114
Herbicides	0	6	11	29	43	114
Drying	103	310	621	1,034	1,241	1,241
Electricity	331	559	1,034	1,448	2,100	3,207
Transportation	207	310	465	621	724	724
Total inputs	9,574	12,478	16,013	19,241	23,187	29,964
Corn yield (output)	35,450	39,620	42,750	56,300	70,900	84,450
Output/input	3.70	3.18	2.67	2.88	3.06	2.82

Source: Pimentel *et al.* (1973).
[1]Authors give a very complete discussion and source of data for this table.
[2]In this analysis, energy for labor is taken as a portion of the food energy consumed by the worker.

From Table 1 it can be noted that over the 25 year period 1945-70, the productivity of corn increased by a factor of 2.4; labour inputs were more than halved, and the use of nitrogen fertilizer, one of the most energy-intensive of all agricultural inputs, increased 16-fold. In 1970, 36.4% of the total energy inputs was for fertilizers; nitrogen alone accounted for nearly one-third of all inputs.

Energy Ratios

It is of interest to compare the food energy available in a crop with the amount of energy required to produce it. The ratio of the energy output (in the food) to the energy inputs represents a kind of energy efficiency. However, there are serious limitations to this type of analysis:

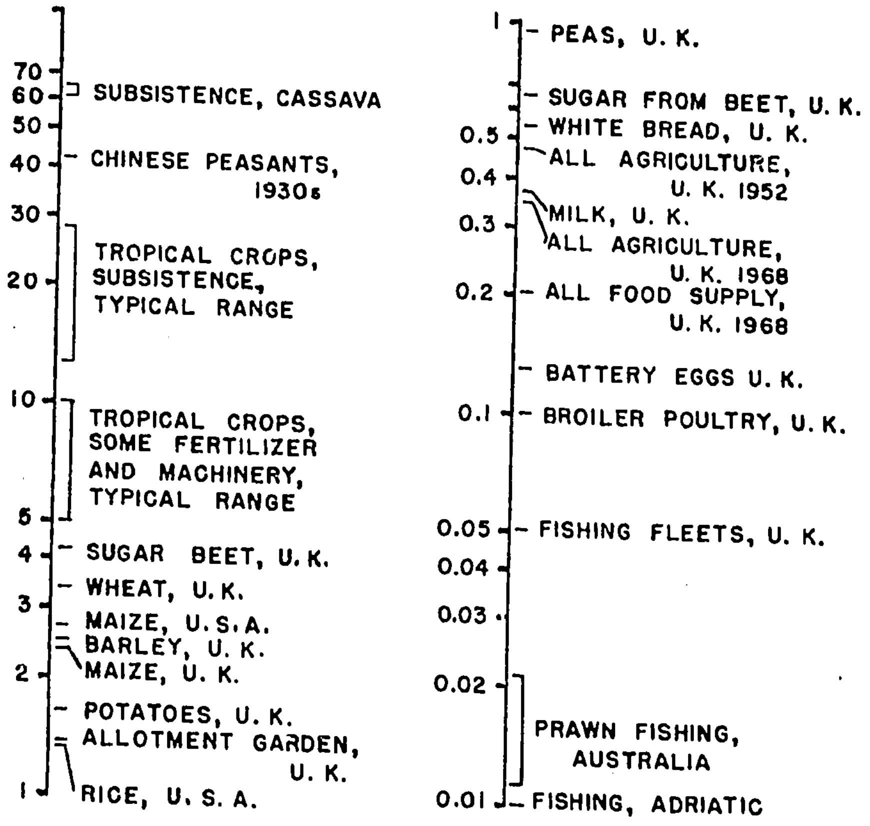
- The comparison is between different kinds of energy.
- The caloric content of food does not fully describe its food value-protein content is also important.
- There is little agreement on what is to be included in the energy inputs. Some analysts include animal and human work, some do not; some include solar energy, some do not.
- The boundary of the system across which inputs are to be measured is usually ill-defined and varies according to who does the analysis.
- Traditional fuel inputs are often ignored or neglected.

Nevertheless, the calculation and examination of energy ratios is of interest, particularly for industrial societies with highly energy-intensive food systems which are threatened by insecurity of energy supply and rising prices for oil and natural gas.

Figure 5 shows energy ratios for certain food production techniques. It can be seen that the energy ratio for primitive and subsistence agricultural systems is very high--there is very little energy input. As agriculture and food production become increasingly mechanized and energy intensive, the ratio falls steadily. This shift is further illustrated by Figure 6 which plots the decline in energy ratio as the socio-technic structure shifts from the agrarian to the industrial.

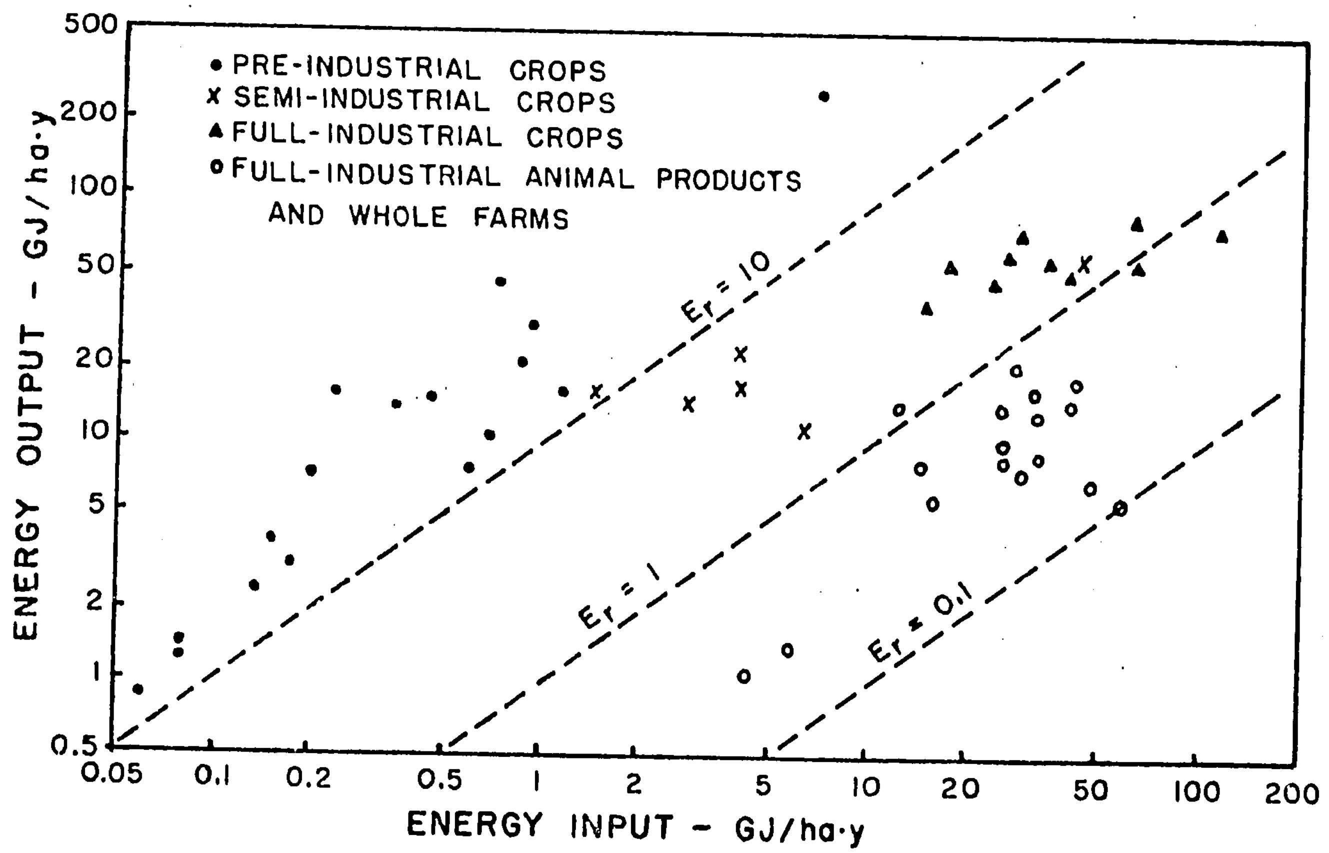
Some analysts, notably the Steinharts [2, 3] use energy subsidy instead of energy ratio. The energy subsidy is the amount of energy required to produce one unit of food energy. It is, therefore, simply the reciprocal of the energy ratio shown in Figures 5 & 6. Figure 7 shows the energy subsidy for a number of widely used foods at different times and by different cultures.

It can be seen that high protein foods such as mild, eggs and meat have a far poorer energy return than plant foods. The position of soybeans is noteworthy. Soybeans possess the best amino-acid balance and protein content of any widely grown crop.



From Leach (1976)

FIG. 5 ENERGY RATIOS FOR FOOD PRODUCTION



From Leach (1976)

FIG. 6 ENERGY INPUTS AND OUTPUTS PER UNIT LAND AREA IN WORLDWIDE EXAMPLES OF FOOD PRODUCTION

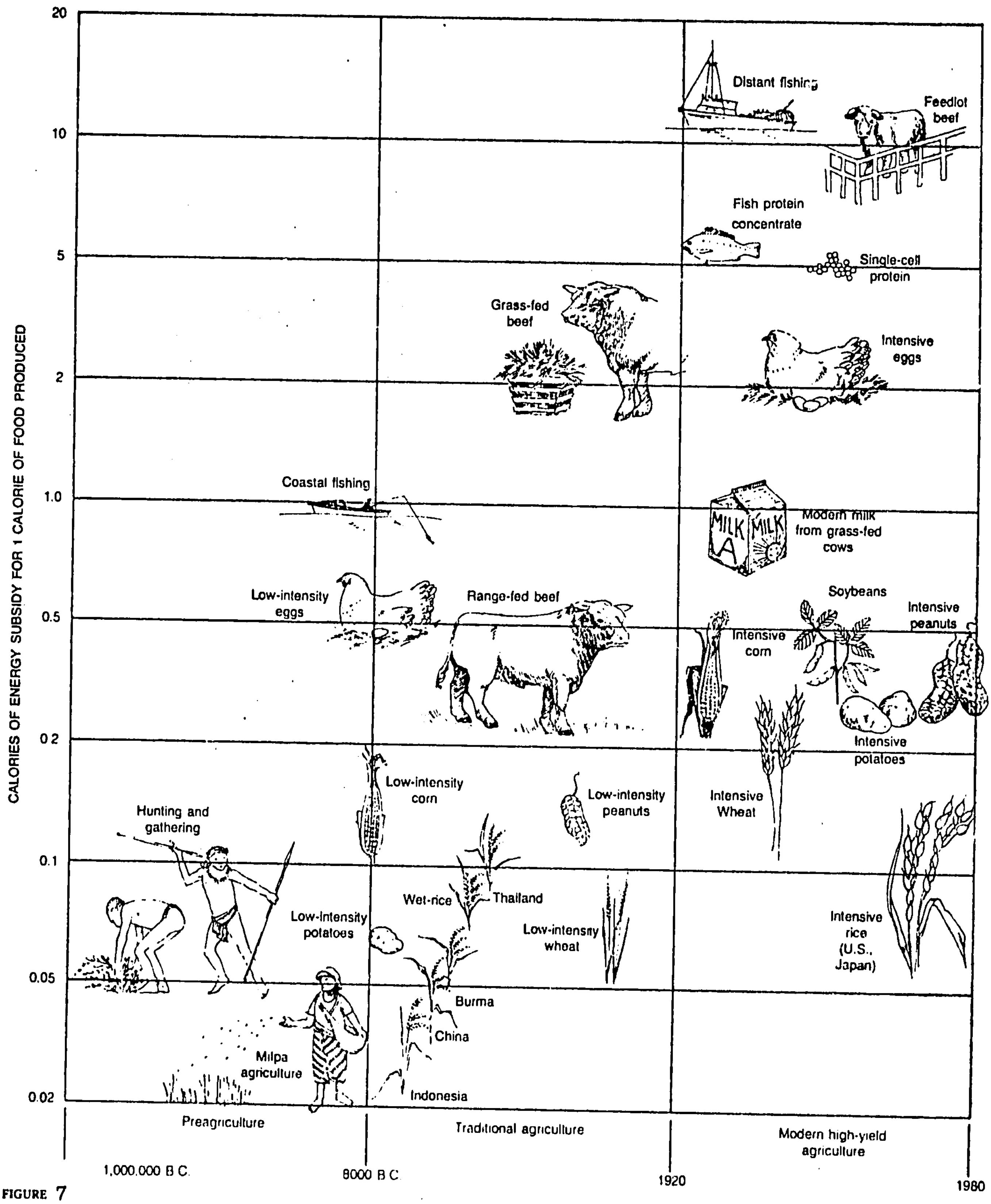


FIGURE 7

Energy subsidies for various food crops. (Data from J. S. Steinhart and C. E. Steinhart, Energy use in the U.S. food system, *Science*, vol. 184, April 19, 1974, p. 312.)

The message of the preceding tables and charts is clear. In primitive societies, 5 to 50 calories of food are obtained for each calorie invested. Some highly civilized cultures have done as well. In sharp contrast, industrialized food production requires an input of 5 to 10 calories of fuel to produce one calorie of food energy.

It would be incorrect to infer that in industrial agriculture a ratio of less than unity is unacceptable. Industrial societies burn fuel to produce food energy. The single important exception are systems designed to produce energy--energy crops and energy plantations--rather than food or other agricultural products. For these systems an energy ratio significantly larger than unity is desirable.

Figure 7 points out the high energy cost of meat, eggs and milk. It takes about nine kcal of food to produce one kcal supplied by beef or mutton. The rich countries (including the USSR) feed more grain to livestock than is consumed by all of the developing countries put together: people, livestock, and all.

TYPE OF FUEL USED

Fifty percent of the energy used in the U.S. food system in 1970 was petroleum--primarily diesel fuel, gasoline and liquified petroleum gas (LPG). Natural gas supplied 30% and electricity another 14% of the total. Table 2 shows the types of fuels used in 1970, including a comparison between production agriculture and the total for the food system. Note that food production per se is almost totally dependent on petroleum. However, when the food system as a whole is examined, natural gas is the principal fuel. In the U.S., natural gas is critical to at least three inputs in production agriculture: fertilizers, irrigation and crop drying. Natural gas is used as a feedstock for almost all of the nitrogen fertilizer produced in the U.S. It has even been suggested that natural gas is so important to fertilizer production that it should be reserved entirely for feedstock, rather than burned as a fuel.

TABLE 2 FUELS AND QUANTITIES USED IN THE UNITED STATES FOOD SYSTEM IN 1970

Fuel	Production TJ	Production %	Food and Fiber Sector[1] TJ	Food and Fiber Sector[1] %
Gasoline	530,837	47.9	809,910	16.4
Diesel	394,500	35.6	1,194,003	24.3
Distillate fuel oil			196,640	4.0
LP gas	130,978	11.8	262,344	5.3
Residual fuel oil			102,890	2.1
Natural gas			1,492,253	30.3
Electric power	52,914	4.8	678,403	13.8
Coal			174,892	3.6
Other			12,196	0.2
Total	1,109,229		4,923,531	

[1]Includes production, farm family living, processing, marketing and distribution, and manufacturing of inputs

AGRICULTURE IN THE DEVELOPING COUNTRIES

In the absence of an industrial infrastructure, food prodution must necessarily rely more on human and animal labour than on the commercial forms of energy. One of the first agricultural practices developed--shifting agriculture--is still widely used. About 30% of the world's arable land is farmed by this method, producing food for about 250 million people [1]. Each hectare of land can provide food for one person. At higher population densities, shorter fallow periods and eventually annual cropping become necessary.

Annual cropping by traditional methods is labour intensive and yields are usually not high. Animal power can help reduce human labour and provide manure for fertilizer; however, draught animals must either be fed some of the crop or pastured, thereby increasing the land area required per capita.

Tables 3 and 4 provide estimates of the commercial energy required for rice and maize production at different levels of technology, as well as the respective yields. The enormous differences in energy use can be easily seen. The U.S. system uses 375 times more commercial energy per hectare than traditional agricultural practice in the Philippines to produce nearly five times the yield. The situation is similar when one examines the production of maize. The U.S. system uses more than 170 times more commercial energy per hectare than traditional agricultural practice in Mexico to produce five times the yield.

TABLE 3 – COMMERCIAL ENERGY REQUIRED FOR RICE PRODUCTION BY MODERN, TRANSITIONAL AND TRADITIONAL METHODS WITH RESPECTIVE YIELDS

	Modern (U.S.A.)		Transitional (Philippines)		Traditional (Philippines)	
	Quantity/ha	*Energy/ha (10^6 joules)*	*Quantity/ha*	*Energy/ha (10^6 joules)*	*Quantity/ha*	*Energy/ha (10^6 joules)*
Inputs:						
Machinery	4.2×10^9 joules	4 200	335×10^6 joules	335	173×10^6 joules	173
Fuel	224.7 litres	8 988	40 litres	1 600	—	—
Nitrogen fertilizer	134.4 kg	10 752	31.5 kg	2 520	—	—
Phosphate fertilizer	—	—	—	—	—	—
Potassium fertilizer	67.2 kg	605	—	—	—	—
Seeds	112.0 kg	3 360	110 kg	1 650	107.5	—
Irrigation	683.4 litres	27 336	—	—	—	—
Insecticide	5.6 kg	560	1.5 kg	150	—	—
Herbicide	5.6 kg	560	1.0 kg	100	—	—
Drying	4.6×10^9 joules	4 600	—	—	—	—
Electricity	3.2×10^9 joules	3 200	—	—	—	—
Transport	724×10^6 joules	724	31×10^6 joules	31	—	—
TOTAL		64 885		6 386		173
Yield (kg/ha)	5 800		2 700		1 250	

SOURCE:

TABLE 4 COMMERCIAL ENERGY REQUIRED FOR MAIZE PRODUCTION BY MODERN AND TRADITIONAL METHODS WITH RESPECTIVE YIELDS

	Modern (U.S.A.)		Traditional (Mexico)	
	Quantity/ha	*Energy/ha (10^6 joules)*	*Quantity/ha*	*Energy/ha (10^6 joules)*
Inputs:				
Machinery	4.2×10^9 joules	4 200	173×10^6 joules	173
Fuel	206 litres	8 240	—	—
Nitrogen fertilizer	125 kg	10 000	—	—
Phosphate fertilizer	34.7 kg	586	—	—
Potassium fertilizer	67.2 kg	605	—	—
Seed	20.7 kg	621	10.4 kg	—
Irrigation	351×10^6 joules	351	—	—
Insecticide	1.1 kg	110	—	—
Herbicide	1.1 kg	110	—	—
Drying	$1\,239 \times 10^6$ joules	1 239	—	—
Electricity	$3\,248 \times 10^6$ joules	3 248	—	—
Transport	724×10^6 joules	724	—	—
TOTAL		30 034		173
Yield (kg/ha)	5 083		950	

SOURCE

However, these tables account only for commercial energy supplies. When the traditional fuel are included, and when the contribution from human and animal labour is factored in, the results are strikingly different. Table 5 shows energy use in rice cultivation when all inputs are included. The traditional agricultural practices of India and China now appear to be even more energy intensive than modern industrial agricultural practice.

Data for six prototypal villages in some developing countries are presented in Table 6. Gross energy input per capita varies from 3.7 million kcal (15.5 GJ) per year in India to 15.5 million kcal (64.8 GJ) per year in Mexico. The efficiency of converting gross energy inputs into useful work was about 5% in India and 25% in Mexico.

The question arises: if the developing countries use in fact more energy per unit of land area to support agriculture than the developed countries, why are their yields not substantially higher? The data of Table 6 suggest part of the answer--energy appears to be used very inefficiently. There are other reasons: most of the energy expended in traditional agricultural practice is human and animate energy used for ploughing, seeding and harvesting; very little is used for fertilizer production and irrigation--vital contributors to higher crop yields.

TABLE 5 – ENERGY INTENSITY OF RICE CULTIVATION IN FOUR MAJOR PRODUCING COUNTRIES

Country	Farm machines and draught animals	Farm operations	Irrigation and nitrogen fertilizer manufacture	Total input	Rice yield	Energy use per metric ton
	HP/ha	.... *10⁶ kcal/ha*	....		*kg/ha*	*10⁶ kcal*
India	0.7	5	2	7	1 400	5
China	0.7	5	3	8	3 000	3
Japan	1.6	3	6	9	5 600	2
U.S.A.	1 5	2	6	8	5 100	2

SOURCE:

TABLE 6 – COMPARISON OF ENERGY USE PER YEAR IN SIX PROTOTYPAL VILLAGES IN DEVELOPING COUNTRIES
(10^3 kilocalories)

Village	Domestic use per caput		Agricultural use for farm work, irrigation, chemical fertilizers				Use per caput for transport, crop processing and other activities		Total per caput use	
			Per caput		Per hectare					
	Useful energy [1]	Energy input	Useful energy	Energy input	Useful energy	Energy input	Useful energy	Energy input	Useful energy	Energy input
Mangaon, India	5	101	13	194	40	645	3	86	20	370
Peipan, China	25	504	35	209	164	1 050	3	81	63	794
Kilombero, Tanzania ...	28	554	2	58	3	96	>1	18	30	630
Batagawara, Nigeria	19	378	4	60	10	184	1	23	23	466
Arango, Mexico	40	428	340	1 030	375	1 150	3	91	383	1 550
Quebrada, Bolivia	43	839	8	169	45	1 010	8	166	58	i 170

SOURCE:

[1] Useful energy is defined as the amount developed at the plough or the shaft of the pump; it depends on the efficiencies of the various technologies of energy use.

FERTILIZERS

Chemical fertilizers are generally credited with about one-half to two-thirds of the production from industrialized agriculture. The primary nutrients are nitrogen, phosphorus, and potassium (N, P, and K). Chemical fertilizers are energy intensive; it requires considerable quantities of energy to produce them on a commercial scale. Of the three nutrients, nitrogen is by far the most important. It is also the most energy intensive: approximately 60 MJ/kg.

There are enormous difficulties barring the development of fertilizer technology on the scale required by the developing countries if they hope to match the yields attained by the industrial nations. The agricultural achievements of Japan and the Netherlands are often cited as offering encouragement, but if India, for example, were to apply inorganic fertilizer as intensively as Holland, India's fertilizer needs would amount to one-third the present world output. In fact, India has tried very hard to increase its production capacity. India had hoped to attain a fertilizer production of 2.4 Mt by 1971. But in 1975 production stood at only 1.3 Mt, some 60% of consumption. Even so, fertilizer production within the developing countries has been doubling about every five years. It would have increased even faster if the cost of fertilizer (which is strongly affected by the price of oil), had not trebled between 1971 and 1974. Figure 8 shows some past trends in fertilizer costs and consumption.

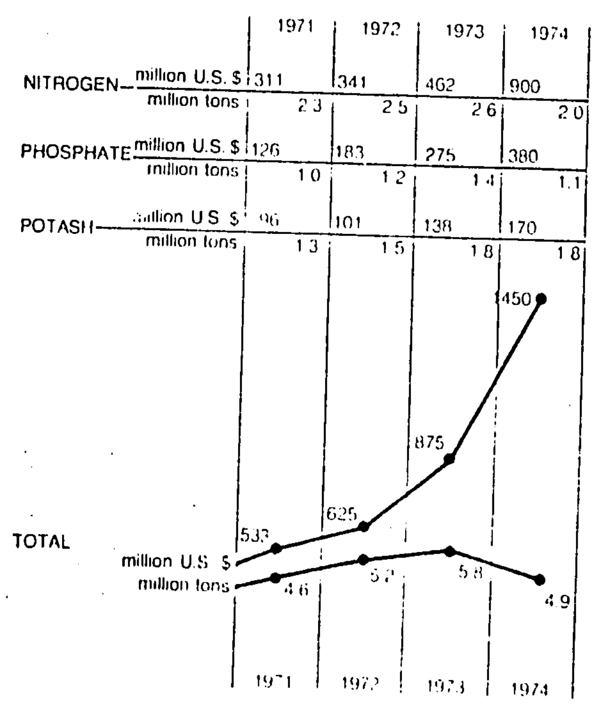


FIGURE 8

The cost of scarcity: The import of manufactured fertilizers by developing countries. The figures for 1972–1974 are estimates. (Adapted from *Ceres*, Food and Agriculture Organization, March–April 1975.)

Nitrogen

Nitrogenous fertilizers are produced by combining hydrogen with atmospheric nitrogen to form synthetic ammonia (the Haber process). Ammonia is the base for at least 90% of all nitrogen fertilizers. In the U.S. the hydrogen is obtained from natural gas-methane. In Europe fuel oil, coal and naphtha are widely used as a hydrogen feedstock. It is interesting to note that approximately 60% of the natural gas produced by the OPEC states in 1972 was flared. This represents a loss equivalent to five times the nitrogen fertilizer consumption of the developing countries in 1978 [1].

In the U.S. about 40% of the anhydrous ammonia produced is used directly as fertilizer, and the remainder is used to produce urea, ammonium nitrate, or compound fertilizers. Urea is rapidly becoming the world's major source of solid nitrogen. This popularity is due to its high analysis (46% N) and its safe handling characteristics.

The greater use of fertilizers in the industrial countries has significantly improved yields per hectare. Figure 9 shows the relationship between the consumption of nitrogenous fertilizers and the yield of cereal grains in both developed and developing countries from 1956 until 1971. Experiments with maize show optimum yields with the application of 225 kg of nitrogen per hectare as suggested by Figure 10. However, the optimum in terms of energy efficiency occurs at about 135 kg per hectare. Figure 11 shows the effect of nitrogen fertilizer on both wheat and rice. Again, maximum yield occurs at around 225 kg of nitrogen per hectare.

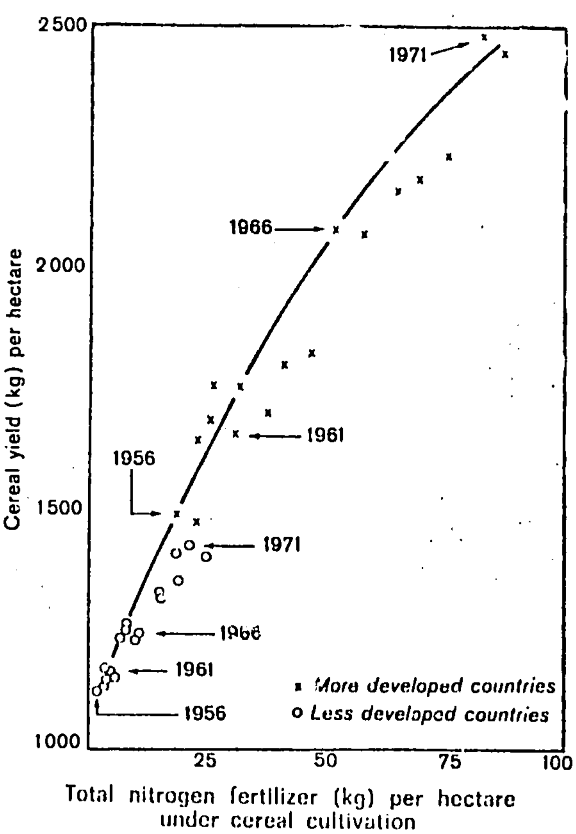


FIGURE 9. Relationship between the use of nitrogen fertilizers and the yield of cereal grains in developed and developing countries, 1956–71. Note that the total nitrogen fertilizer use on all cultivated land is divided by the area under cereal grains; thus the actual application rates would be about one half the rates shown here as only one half of the total land area is under cereal grains

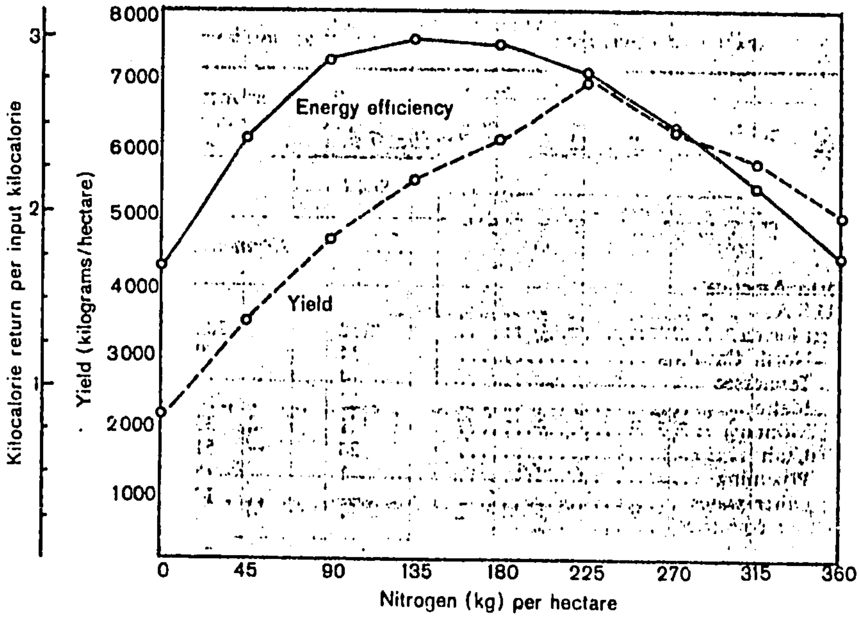
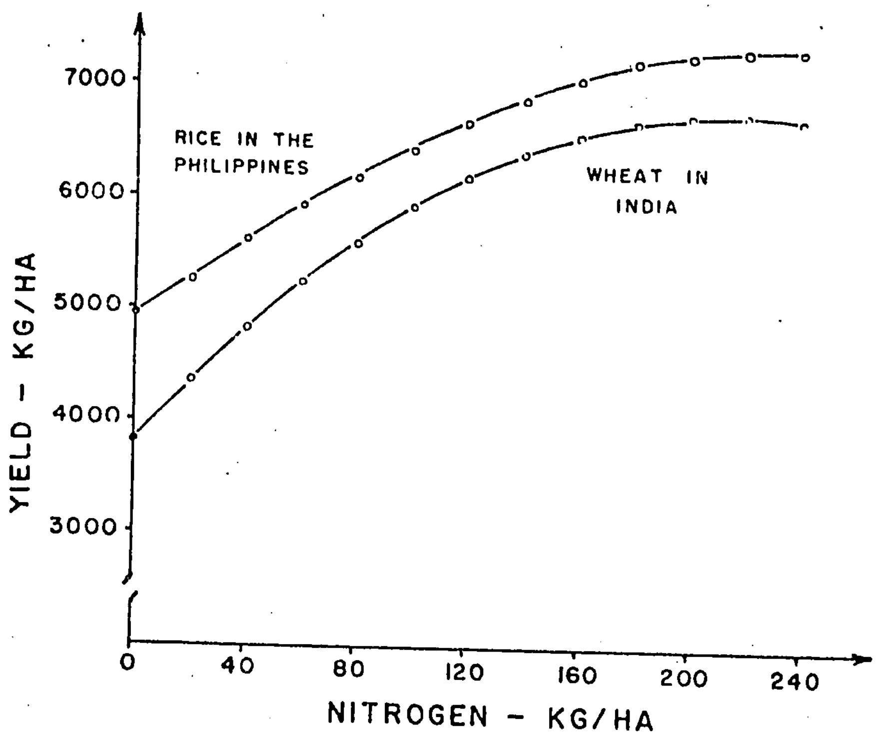


FIGURE. 10. Yield and kilocalorie return per input kilocalorie for maize at different rates of nitrogen fertilizer application. Maize showed optimum yields with the application of about 200 kilograms of nitrogen fertilizer per hectare, whereas the optimum kilocalorie return per input kilocalorie resulted from an application of about 135 kilograms of nitrogen fertilizer per hectare



From Pimentel

EFFECT OF NITROGEN FERTILIZER ON RICE AND WHEAT YIELDS

Phosphates

Phosphorus compounds comprise the second largest group of fertilizers. The consumption and production of phosphate fertilizers increased about 150% between 1960 and 1973--considerably less than the increase in nitrogenous fertilizer production. The principal method of producing phosphate fertilizers is the decomposition of phosphate rock with sulphuric, phosphoric or nitric acid. Sulphuric acid is efficient, but large quantities of sulphate waste are produced (gypsum). The use of nitric acid requires further treatment, usually with ammonia, to produce a suitable fertilizer product. Phosphate fertilizers require about 12-20 MJ of energy per kilogram of fertilizer. Considerable reserves of phosphate rock are available--predominantly in the U.S. and Morocco.

Potassium (Potash)

About 95% of the world output of potash comes from underground mines. Canada possesses almost half of the world's potassium resources followed by Germany and the U.S.S.R. Potash is relatively cheap and plentiful. Its energy intensity is approximately 6 MJ/kg.

Because chemical fertilizers account for such a large and increasing proportion of the total use of commercial energy in agriculture it is important to examine ways in which fertilizers might be used more efficiently. An important factor is the relative proportions of available supplies used in the developed and developing countries. In general, it is the initial applications of nitrogen fertilizer which produce the largest marginal yield. Therefore, the use of additional fertilizer will generate far greater increments in yield in the developing countries since current consumption is low.

There are other ways to improve the efficiency of fertilizer use, including well-timed sowing and better wider management. Improved methods of fertilizer application, including proper placement and timing to coincide with the nutrient demands of crops, will also increase efficiency. Experiments at the International Rice Research Institute in the Philippines show that when fertilizer is placed close to the root zone, application rates can be halved without any reduction in yields.

Chemical fertilizers are a comparatively recent phenomenon in agriculture. Until chilean nitrate and Peruvian guano were introduced into European agriculture in the 1830's and until 'superphosphate' manufacture began in the 1840's, "artificial" fertilizers were united to soot, bones, hoofs and horns, saltpetre and lime. The maintenance and restoration of soil fertility depended on such practices as shifting cultivation, fallowing, crop rotation, catch-cropping (especially with nitrogen-fixing legumes) and the recycling of crop and animal residues.

Because chemical fertilizers have become available at low prices and present the advantages of concentration, portability, and adaptability to different soil conditions and crop requirements, the use of crop and animal residues to maintain soil fertility has steadily declined in the developed countries. In many developing countries, especially where there is no tradition of mixed crop and Westock farming, crop and animal residues have generally been used as fuel rather than as fertilizer. In these countries agricultural modernization has sometimes proceeded directly to the use of energy-intensive chemical fertilizers. A major exception, however, is China where, despite a rapid increase in fertilizer application, the use of crop, animal and human wastes is still substantial.

ORGANIC FERTILIZERS

For centuries man has used organic matter to maintain or increase soil productivity. Manure and fertilizer were practically synonymous terms. In China, the land has been cultivated without a loss of fertility by careful soil management. The use of nitrogen-fixing legumes and the application of animal and human wastes are traditional methods of maintaining the nutrient balance in the soil.

Animal and human wastes contain elements essential to plant growth. The specific nutrient content of animal dung depends on the species and the kind of material the animal eats. In the U.S. manure generally contains about 30 kg of nitrogen, 6 kg of phosphorus, and about 22 kg of potassium per dry metric ton. Table 8 compares the nutrient content of manure and inorganic fertilizer, and further data on manure production are given in Table 9. Human waste contains the highest percentage of nitrogen, and horse, mule, and donkey manure contains the least.

Table 10 shows the results of studies carried out in six villages to determine the nitrogen fertilizer potential of their plant, human, and animal wastes. Careful management and use of organic wastes and could provide a significant amount of nitrogen, but not enough to ideally support the new strains of wheat and rice.

TABLE 8 - COMPARISON OF NUTRIENT CONTENTS OF MANURE AND FERTILIZER, U.S.A., 1973

Nutrient	Total (10^3 metric tons)	Nutrient content	
		Kilograms per dry metric ton	Pounds per wet English ton
Nitrogen			
Manure	7 367	31	10
Fertilizer	7 540	191	—
Phosphorus			
Manure	1 463	6.2	2
Fertilizer	2 222	56	—
Potassium			
Manure	5 342	22	7
Fertilizer	3 836	97	—

SOURCE

TABLE 9. – EXCREMENT PRODUCTION DATA

	Production per 1 000 kg liveweight (kg/yr)	Assumed average liveweight (kg)	Production per head (kg/yr)	Moisture content (percent)	Nitrogen content (percent of dry matter)	
					Solid and liquid wastes	Solid wastes
Cattle	27 000	200	5 400	80	2.4	1.2
Horses, mules, donkeys	18 000	150	2 700	80	1.7	1.1
Pigs	30 000	50	1 500	80	3.75	1.8
Sheep and goats	13 000	40	500	70	4.1	2.0
Poultry	9 000	1.5	13	60	6.3	6.3
Human faeces without urine	—	40-80	50-100	66-80	—	5-7
Human urine	—	40-80	18-25 (dry solids)	—	15-19 (urine only)	—

SOURCE:

TABLE 10. – ANNUAL ORGANIC NITROGEN SUPPLY IN CASE-STUDY VILLAGES

	Hectares of cultivated land	Excluding human excrement			Nitrogen collectable in solid and liquid human excrement[2]	Total nitrogen collectable in solid and liquid wastes	Nitrogen available per hectare cultivated land
		Nitrogen available in net residues[1]	Nitrogen in collectable solid wastes	Nitrogen in collectable solid and liquid wastes			
		Kilograms					
Mangaon, India	300	10 000	4 000	7 000	3 000	10 000	33
Peipan, China[3]	200	13 000	7 000	9 000	4 000	13 000	65
Kilombero, Tanzania	60	1 000	500	800	300	1 100	18
Batagawara, Nigeria	530	13 000	6 000	9 000	4 200	13 200	25
Arango, Mexico[3]	380	7 000	5 000	6 000	1 700	7 700	20
Quebrada, Bolivia (one *parcela*)	1	70	40	50	18	68	68

SOURCE:

[1] Assuming a nitrogen content of 0.2% for crop residues. – [2] Assuming that a maximum of 80% of the nitrogen in human excrement can be collected. For Arango and Peipan, intermediate values for the total nitrogen excreted per person per year have been used, and for the other villages, lower figures have been used. The rationale is that people in Arango and Peipan have more adequate diets, which produce excrement with a higher nitrogen (protein) content. – [3] The nitrogen figures for Peipan and Arango are probably underestimates as crops there are well fertilized. The nitrogen content of crop residues is probably considerably higher, especially because maize, which has a higher nitrogen content in the residue, is a major crop in both areas.

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END USE MATCHING

1. INTRODUCTION

Energy is used to perform certain tasks. At the point of final use, these tasks and activities are often referred to as 'end-uses'. Examples include cooking, lighting, water pumping, drying, and other tasks. Figure 1 indicates schematically the relationship between rural energy use and energy technologies. Since it is relatively easy to convert one form of energy into another by the use of energy conversion technologies, there are a multitude of ways in which a particular end-use activity may be supplied with energy. Figure 2, which shows most of the common routes from energy source to end-use, illustrates how any energy source, when coupled with the appropriate conversion technologies, may be used to supply energy to a specified end-use. But this diagram indicates only that the technology is feasible -- not whether it is desirable, appropriate, efficient, or most importantly, cost-effective.

For a number of reasons it is desirable to carefully match the characteristics of renewable and alternate energy technologies with the characteristics of a specified energy-using activity or end-use. First, the output of most renewable energy systems is intermittent and variable. When the quantity or timing of energy output diverges from the pattern of demand, some form of energy storage must be used. This can adversely affect the cost, reliability, and maintenance of a renewable energy system. A careful matching of the energy output of the supply technology with the energy demand characteristics of the end-use can minimize storage requirements.

Second, each end-use generally requires a particular form of energy: either heat, mechanical shaft power, or electricity. Some tasks can use more than one energy form, i.e. lighting can be provided by electricity, a liquid fuel lantern, or a combustible gas. Other end-uses require a particular energy type. Energy can be changed from one form to another through conversion, but this increases the total cost, decreases the overall system efficiency, and increases the complexity and maintenance requirements of the system.

Third, the acceptance and use of a new technology is likely to be more rapid, particularly in rural areas, if its introduction causes little disruption of the existing practices and customs of the local people. By first carefully investigating end-use characteristics, the matching process can assist the analyst in selecting a technology whose output fits traditional patterns, or at least has minimal impact on the social and cultural norms of the society.

Figure 1 - RELATIONSHIP BETWEEN VILLAGE ENERGY AND ENERGY TECHNOLOGIES

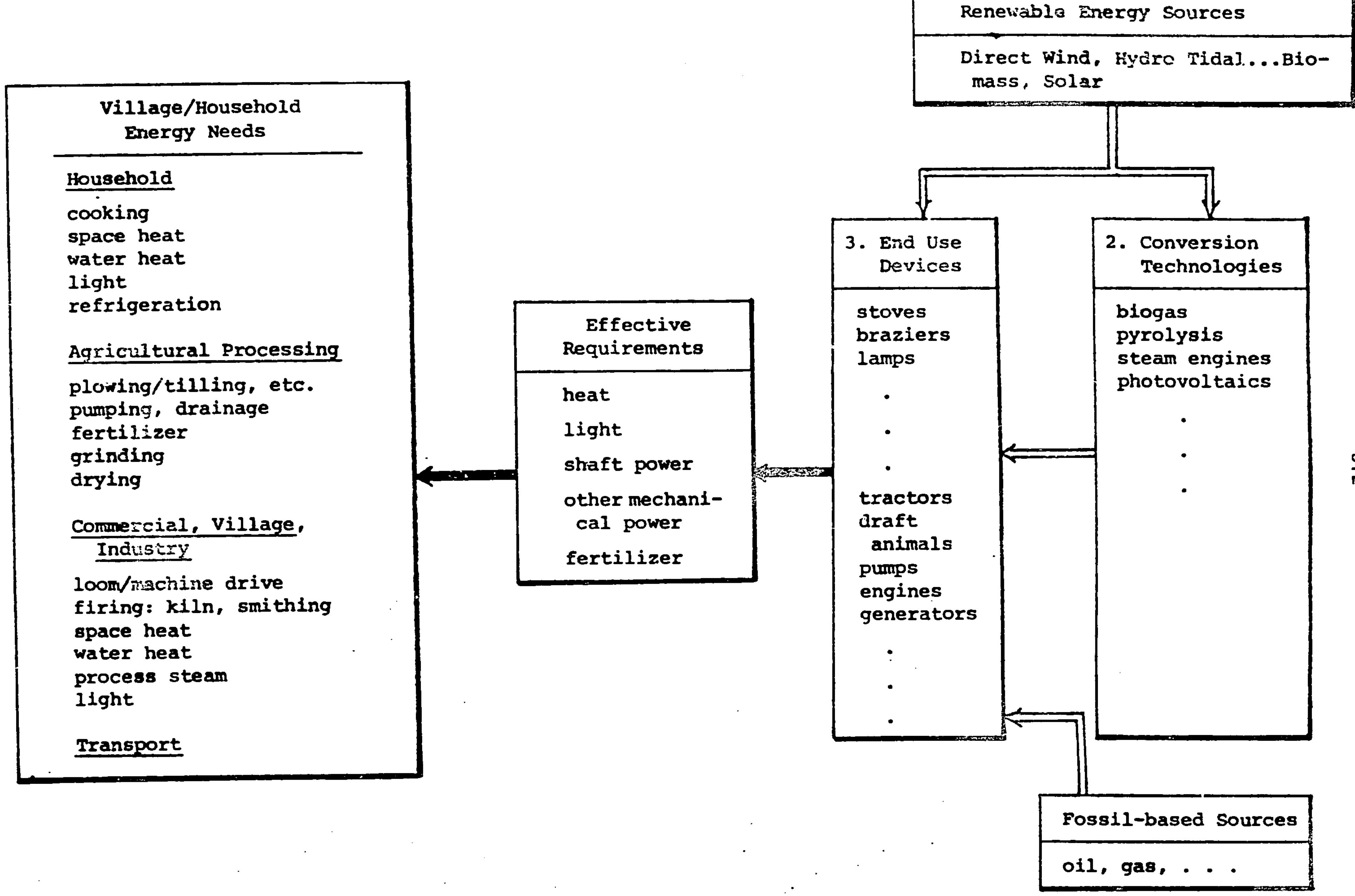
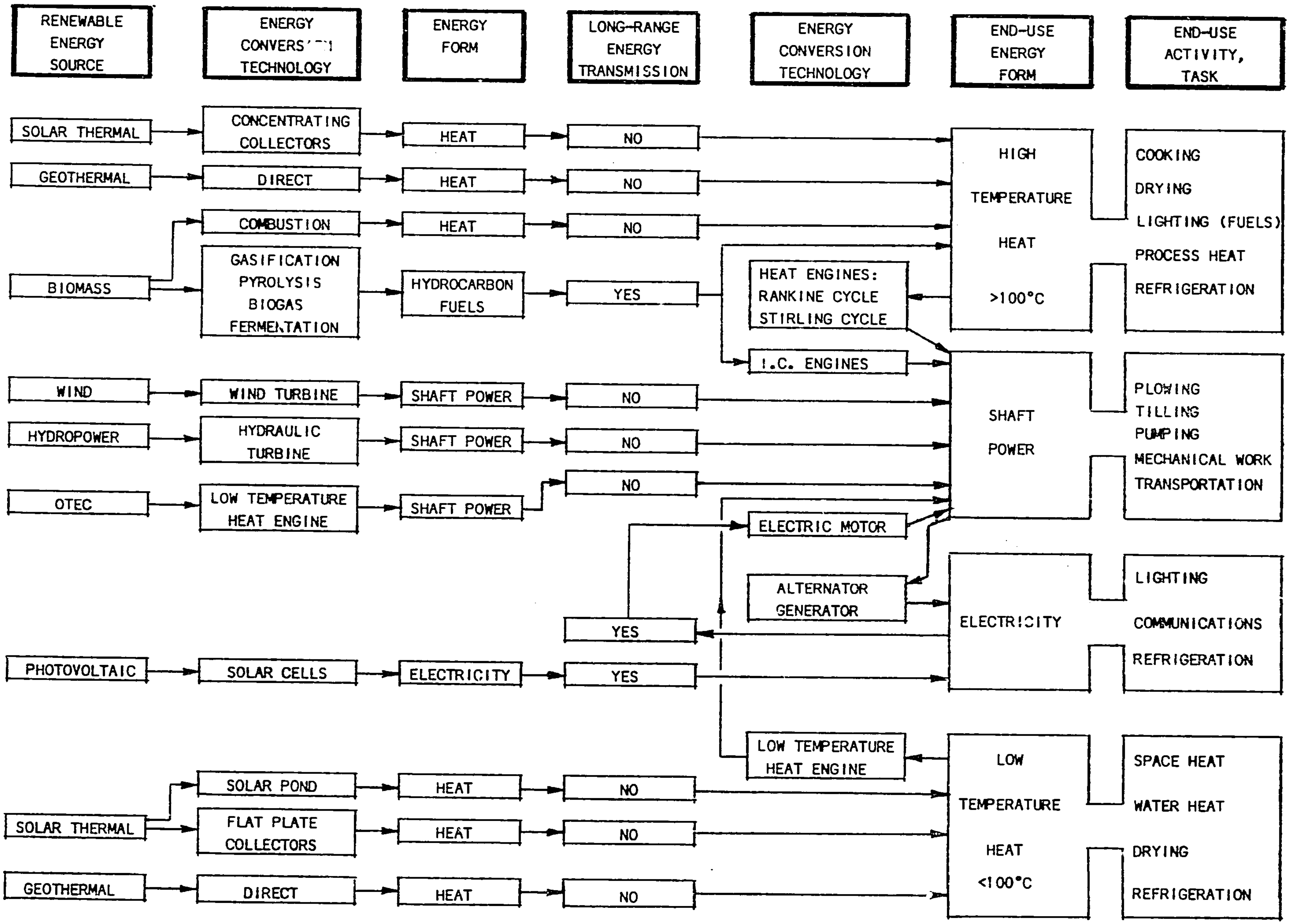


Figure 2 RENEWABLE ENERGY PATHS: FROM SOURCE TO END-USE



2. SYSTEMATIC END-USE MATCHING

In these notes we present a methodology of matching energy sources and conversion technologies to end-uses based on a procedure recently developed by the Solar Energy Research Institute (SERI). A five-step matching process is outlined. The systematic procedure is intended to guide the choice of a renewable energy technology for a given basic rural energy need [2]. The SERI report focuses specifically on village-level energy needs: what the authors refer to as the 'basic human needs' approach; but the methodology is applicable to end-use matching in general, only the program objectives and the ranking of energy needs will change.

The matching procedure is shown schematically in Figure 3.

Step 1: Choice of Project Goals

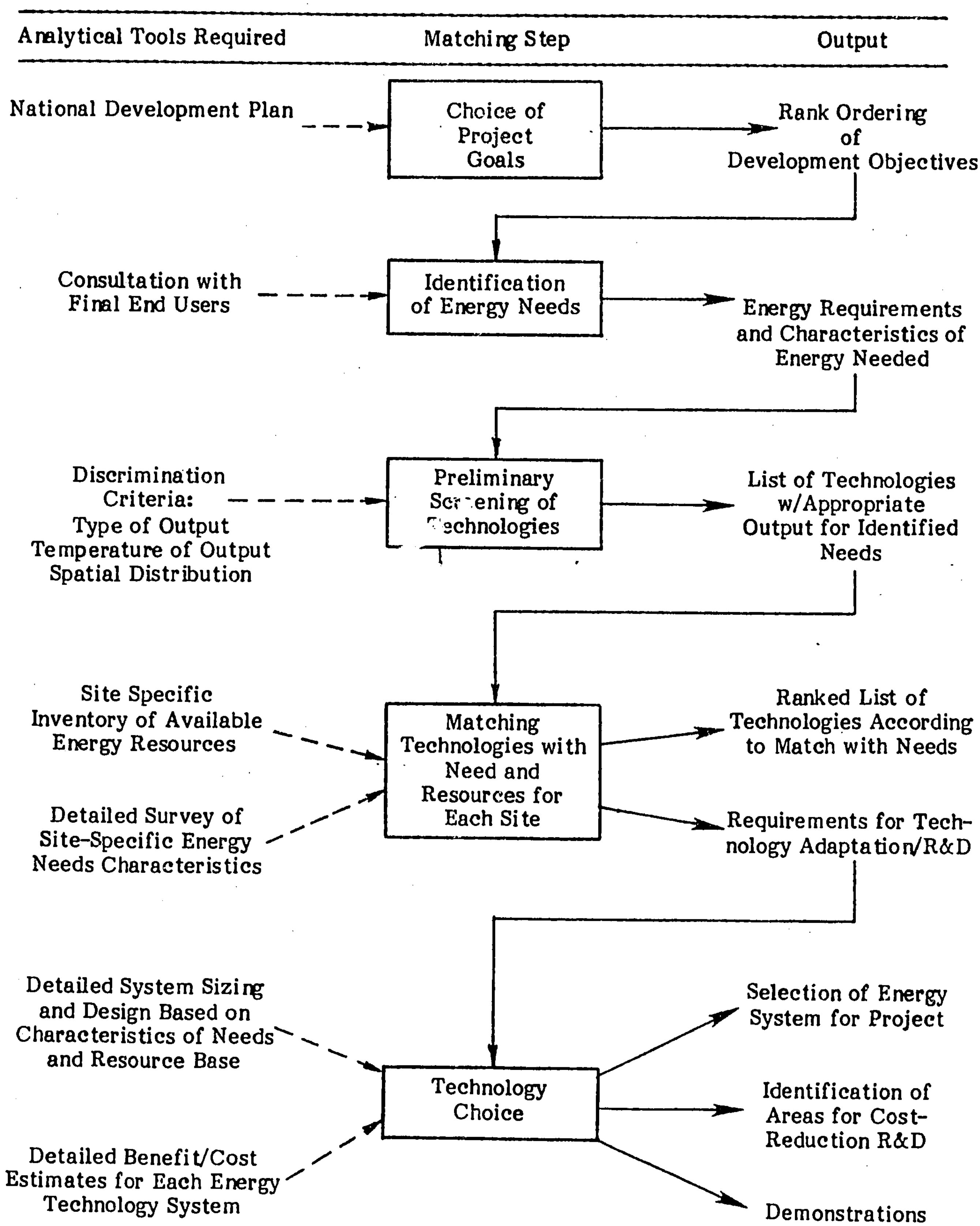
The energy technology selection process must consider a country's development goals and program objectives. National leaders and planners determine the energy needs of the country and from these perceived needs they determine the long-term goals of the development program. The broad objectives of national development plans might include, for example:

- promoting economic growth
- rapid agricultural development
- attaining energy self-sufficiency
- reducing population growth
- reducing foreign exchange expenditures
- eliminating illiteracy
- decreasing urban-rural inequities.

Where the development of renewable and alternate sources of energy has been identified as a component of one or more of these development objectives, it will be necessary for planners to rank energy and other priorities in order of importance.

Since scarce resources such as capital, foreign exchange, skilled labour, management expertise, and raw materials are normally rationed, this weighting of objectives allows the planner to allocate the available resources to the most important tasks first. This process determines what resources will be devoted to the energy projects. However, it is important to remember that energy is an intermediate good, not an end in itself. For each energy project, the planner must keep in mind one central question: to what end am I introducing one or more energy technologies? Without the specification of program objectives it is impossible to fully evaluate the success of a technology introduction. In the final analysis, the important question is not how did the system perform, but how close did it bring the user to the initial development goals.

Figure 3 NEEDS/TECHNOLOGY MATCHING PROCESS



Step 2: Identification of Energy Needs

Having defined the development goals, the next task for the planner is to determine which needs are of prime importance to the target group and what particular activities require energy. Consultation with members of the target group is essential at this stage. All too often, project planners unilaterally decide what is needed, for example: street lights, gas for cooking, or an educational television receiver. If members of the recipient group do not have a substantial role in selecting what needs will be met first, this group is not likely to actively support the technology introduction process, assist in any necessary modifications, or maintain the system once it is installed. In fact, there is substantial project data suggesting that local financial participation in the building or purchase of a system may be crucial to system maintenance and adaptation.

Broad based participation also helps to ensure that the benefits provided by the energy technology are not appropriated by a small number of local people as a result of an existing monopolization of factors such as land, capital, elected or traditional authority, education, etc.

A brief analysis of the fundamental characteristics of the energy needs--how much, when it is required, where it must be delivered, and in what form--should begin at this point to facilitate the preliminary screening of technology options. A much more detailed assessment of the characteristics of the energy needs is undertaken in Step 4.

Step 3: Preliminary Screening of Technologies

Having defined and ranked the energy needs that should be satisfied, the planner and energy analyst can begin to narrow the range of alternate energy technologies capable of producing the required energy.

For this preliminary screening it is useful to identify certain discrimination criteria. These basic criteria are used to characterize energy needs and technologies and are relatively insensitive to site-specific variations or social and cultural patterns. Once the characteristics of the energy need are defined in terms of these criteria, the energy technologies can be examined with reference to the same criteria and preliminary matches can be identified. The advantage of this approach is that the discrimination criteria can assist in eliminating inappropriate technologies before extensive site-specific data have been collected. The three essential discrimination criteria for each need are:

1. type of energy output required
2. quality or quantity of energy required.
3. spatial distribution required.

Each criteria imposes a different screen on the full list of technology options. The first criterion selects those technologies that provide the required energy form to meet the end-use need without energy transformation. The second criterion identifies energy systems that produce a sufficient temperature of power output to perform the identified task. The third criterion, spatial distribution, addresses the problem of providing the energy where it is required. If energy is to be delivered to several sites, the technology options are reduced rapidly to those sources whose output can be moved efficiently from a central location that can be located at each site.

By using these criteria, the planner and energy analyst can reduce the technology options for each identified need. The criterion for the energy type is particularly useful in identifying technologies that do not require energy transformation. The spatial distribution criterion helps ensure that technologies are selected that most easily and efficiently serve the required locations. An ideal technological option includes all three need characteristics. Most feasible technology choices match the characteristic of the energy need on two of the three discrimination criteria.

Step 4: Matching Technologies with Need and Resources for Each Site

This step constitutes the fundamental matching process. It provides the data on energy needs and technologies that determine their compatibility in a particular location. Planners and technologists survey and inventory the local sources of energy, both renewable and non-renewable. The energy resource base is evaluated on the basis of the following general criteria:

- physical availability - size of the resource base and current demands on the resource.
- variability - changes in the resource on a daily, weekly, seasonal, and annual basis; size of swings as a percentage of average resource availability.
- constraints on use - ownership of the resource; historical, cultural, or economic restrictions on use; conflicting or multiple claims on the same resource.

Acquiring accurate resource-availability information covering an entire year can be a formidable task, particularly for renewable energy systems whose sizing is highly dependent on the level of the local resource, such as wind energy conversion systems and small-scale hydroelectric systems. At this stage, preliminary observations can be made and simple measuring instruments installed. Data collected over several months or longer greatly assist in the final design of the selected technologies and in the sizing of any storage and energy conversion devices.

While the resource assessment is being conducted, other members of the project team can detail the characteristics of each of the local energy needs. The criteria used should be virtually identical to those used to characterize the output of the technologies so that energy needs and energy technology options can be easily compared and matched. The focus of the data collection program is the current pattern of energy use: what quantities of fossil and renewable fuels are needed and what human and animal energy is used, in what form, at what time, and for what purposes. In a small rural community, this information can be gathered by various survey methods, community discussions, and the like.[2]

The information and data collected during this phase of the project should be useful not only for the selection of technologies for a specific-project but also for the later evaluation of the social and economic changes produced by the introduction of the new energy systems. If possible, this information should be collected uniformly and systematically and in a manner that permits and facilitates comparison between projects.

With detailed information now at hand on the local resource base and the characteristics of the local energy needs, the project manager and energy technologist can use a systematic process to select the energy systems that most closely match the needs. At the same time the project team can identify any research and development work, field-testing, or technology adaptations that may improve the match between the needs and the candidate technologies.

Step 5: Technology Choice

Using the detailed local data, the planner and technologist can establish a preliminary design for each energy system under consideration. This includes sizing each system, calculating the storage required (if any) to reliably meet the energy needs, designing energy transmission systems, and specifying any changes in existing end-use devices that may be required. For locally designed and fabricated systems, detailed designs incorporating readily available materials should be developed and tested.

Once the detailed specifications of the systems have been completed analytical tools for assessing project feasibility and desirability, such as social cost-benefit analysis and economic analysis, can be employed. Major problems may arise in quantifying certain factors, particularly benefits, because of the lack of information, particularly of site-specific data. Nontheless, even rough estimates of the cost-benefit ratio will assist not only in the final selection but also in comparing these systems with fossil-fueled and tradionally powered systems.

The ranking of the candidate technologies and the final selection of the most appropriate system is by no means an easy task. There is likely to be a number of sources and systems that appear equally matched to the end-use. Moreover, the scale of the system has to be decided, and this variable will introduce an additional degree of complexity into the selection procedure. Each technology, designed for a particular scale, must be treated as a distinct technology option and assessed and evaluated individually.

3. CHARACTERIZATION CRITERIA

It is necessary to develop a set of criteria that are capable of describing the essential characteristics of both the energy technologies and the energy needs. Presented here are thirteen characteristics that describe each energy need based on the physical and temporal requirements for meeting the need, the amounts and kinds of energy that have been and could be used, the traditional social and cultural context of meeting the need, and the cost of fulfilling the need in a new manner. As explained earlier, three criteria were selected as discrimination criteria, since they are less sitespecific than the other characterization criteria. These criteria are used for the preliminary screening of the technology options. The remaining ten characterization criteria are divided into three further categories. In sum, the set of criteria fall into four categories.

1. discrimination criteria.
2. temporal and climatic criteria.
3. cultural/environmental criteria.
4. economic criteria.

Table 1 lists the thirteen criteria, explains how the characteristics apply to energy needs and technologies, and gives the unit of measure for each characteristic.

It should be emphasized that the majority of these criteria are highly site-specific. This is true not only for each characteristic, since the pattern of energy use and the availability of the energy sources will vary between locations, but also for the relative importance of each criterion. In some locations, the time of day may override all other considerations for certain tasks (cooking, for example) while in other places the key consideration may be the spatial distribution. Such priorities can be determined only by direct consultation with the energy consumers.

The criteria presented in Table 1 are by no means an exhaustive list, but they cover most of the important characteristics of energy sources and end-uses that should be considered in the end-use matching procedure. As an illustration of the way that the criteria may be used, Table 2 shows their application to some of the principal alternate energy technologies.

TABLE 1 END-USE MATCHING CHARACTERIZATION CRITERIA

Criteria	End-Use	Energy Technologies	Unit of Measure
I. Discrimination Criteria			
1. Type of output	Form of energy that can satisfy demand.	Form of energy produced.	Not applicable.
2. Quality or quantity	Level of heat or power required.	Range of temperature of energy system output or power available.	°C or Watts.
3. Spatial distribution	The number of locations needed for the task.	Capability to distribute the energy output produced by the technology.	Number of sites required and degree of dispersion.
II. Temporal & Climatic Criteria			
4. Seasonality	Time of year when the energy demand occurs.	Time of year when the resource produces useful energy output.	Growing season, non-growing season, or all year long.
5. Time of day	Time of day when energy is required.	Time of day when the useful energy is produced.	Morning, daytime, night.
6. Duration	Duration of time per day required.	Duration of time the technology provides useful energy during the day.	Number of hours per day.
7. Sensitivity to interruption	Length of time the performance of the task can be halted.	Variablity of output of the energy source.	Can be interrupted or cannot be interrupted. Variable or not variable.

TABLE 1 END-USE MATCHING CHARACTERIZATION CRITERIA (continued)

Criteria	End-Use	Energy Technologies	Unit of Measure
	III. Cultural/Environmental Criteria		
8. Usage by type of person	Persons working with the end-use device or involved with energy supply procedures.	Persons likely to be involved in operating the energy technology and their necessary skills.	By sex, age, and class.
9. End-Use technology	Traditional and commercial sources of energy used and the type of end-use device utilized.	Changes in end-use device or modifications required by new energy source.	Energy consumption per capita and per task. Description of end-use devices.
10. Historical, social and religious factors	Historical, social, and religious influences and customs that affect how the need is met.	Traditional patterns that could create resistance to the use of the energy source.	Description of historical, social and religious customs that influence technology.
11. Environmental and ecological factors	Climatic or environmental factors that influence energy needs.	Factors that influence performance, durability and maintenance of the technology.	Qualitative descriptions
	IV. Economic Criteria		
12. Capital cost	The cost of fulfilling energy need with current practice. The cost of the end-use device.	The projected cost of the technology's local application, including modifications to the end-use device.	Costs given in dollars per unit output, or person-days.
13. Annual cost	Operating and maintenance costs for the current end-use device.	Operating and maintenance costs for the technology and its associated end-use devices.	Dollars, person-days.

TABLE 2 CHARACTERIZATION CRITERIA FOR ALTERNATE ENERGY TECHNOLOGIES

Characterization Criteria	Wind	Photovoltaics	Flat Plate Collectors	Concentrating Collectors	Biogas Generators	Biomass Combustion	Small-scale Hydro
1. Type of output	Shaft power, with conversion: electricity, heat	DC electricity, with conversion: heat, shaft power	Heat, with conversion: electricity, shaft power	Heat, with conversion: shaft power electricity	Heat, with conversion: shaft power electricity	Heat, with conversion: shaft power electricity	Electricity with conversion heat, shaft power
2. Quality or quantity	kW scale	A few kW	Air, max 75°C water, max 90°C	Line focus 300°C, point focus 400°C	100°-400°C	100°-500°C	kW scale
3. Spatial distribution	Good for electricity	Good	Poor for heat, good for electricity	Poor for heat, good for electricity	Moderate for gas, good for electricity	Poor for heat, good for electricity	Good for electricity
4. Seasonality	Varies	Insolation varies	Insolation varies	Insolation varies	Supply may vary	Supply may vary	May vary unless water storage
5. Time of day	Varies	Daytime, but variable	Daytime, but variable	Daytime, but variable	Anytime, if supply is constant	Anytime, if supply is constant	24 hr/day
6. Duration	Highly variable, 8-24 hrs with storage	Varies, 24 hrs if adequate storage	Varies, 24 hrs if adequate storage	Varies, 24 hrs if adequate storage	24 hrs, if adequate storage	24 hrs if supply is constant	24 hr/day
7. Sensitivity to interruption	Interrupted unless adequate storage	Interrupted unless adequate storage	Interrupted unless adequate storage	Interrupted unless adequate storage	May be interrupted if waste supply decreases	Minimal interruption if supply is good	Minimal interruption if water flow is good
8. Use by type of person	Site specific	Site specific	Site specific	Site specific	Site specific	Site specific	Site specific
9. End-use technology	Depends on end-use	Depends on end-use	Depends on end-use	Depends on end-use	Depends on end-use	Depends on end-use	Depends on end-use
10. Historical, Social and Religious factors	Site specific	Site specific	Site specific	Site specific	Site specific	Site specific	Site specific
11. Environmental and ecological factors.	Moderate, system & site dependent	Little or no impact	Little or no impact	Little or no impact	Fertilizer as coproduct	Potential problems if not well managed	Can be serious with water impoundments
12. Capital cost	System & site dependent	System & site dependent	System & site dependent	System & site dependent	System & site dependent	System & site dependent	System & site dependent
13. Annual cost	System & site dependent	System & site dependent	System & site dependent	System & site dependent	System & site dependent	System & site dependent	System & site dependent

4. SCALE

The notion or concept of 'scale' does not refer merely to size--whether a system is large or small in an absolute sense. The scale of an object is a measure of proportion- how that object stands in relation to another object or to its surroundings. In the context of an energy supply technology, the scale of the system refers to the relationship between the output of the supply system and the energy need of its dependent end-use devices.

Clearly, if an energy supply system is to service a single end-use device (for example, an industrial machine) one carefully matches the power output (and other characteristics) of the supply system to the rate at which the end-use device requires energy. However, where an energy supply system is to service many end-use devices at different sites, the analyst or planner is faced with a number of choices. One can scale the supply technology to the site-specific end-use and locate a small supply system at each site thereby obviating the need for energy transmission. Or one can scale the technology to the total projected energy demand and locate a single larger system in the vicinity of a central energy source and transmit power to each locality and site. Clearly, in this case it is necessary to construct, operate, and maintain an energy transmission system the cost of which is likely to be substantial. In between the extremes of the fully decentralized supply system and the large centralized operation, are systems which are a combination of the two. At the lower end of the scale fall community-scale systems serving a small number of sites. At the upper end of the scale fall regional-scale systems consisting of a number of regional energy supply facilities each serving a number of communities.

Economies of Scale

In general, a single large energy system is cheaper to build and to operate per unit of output than a number of small systems with the same total rated output. For example, if you double the size of an electrical generating plant, the capital costs will increase by an amount which is less than twice the cost of the original plant. This non-linear characteristic is often referred to as an economy of scale. Figure 4, which shows the unit capital costs of different kinds of power plants as a function of size, is typical. If we examine the figures for the Light Water Reactor (LWR) it can be estimated that the capital cost of a 600 MWe unit would be about 600 x 700,000 = 420 M$; but doubling the size to 1200 MWe only raises the cost to about 1200 x 565,000 = 678 M$, only 61% above the cost of the 600 MWe unit.

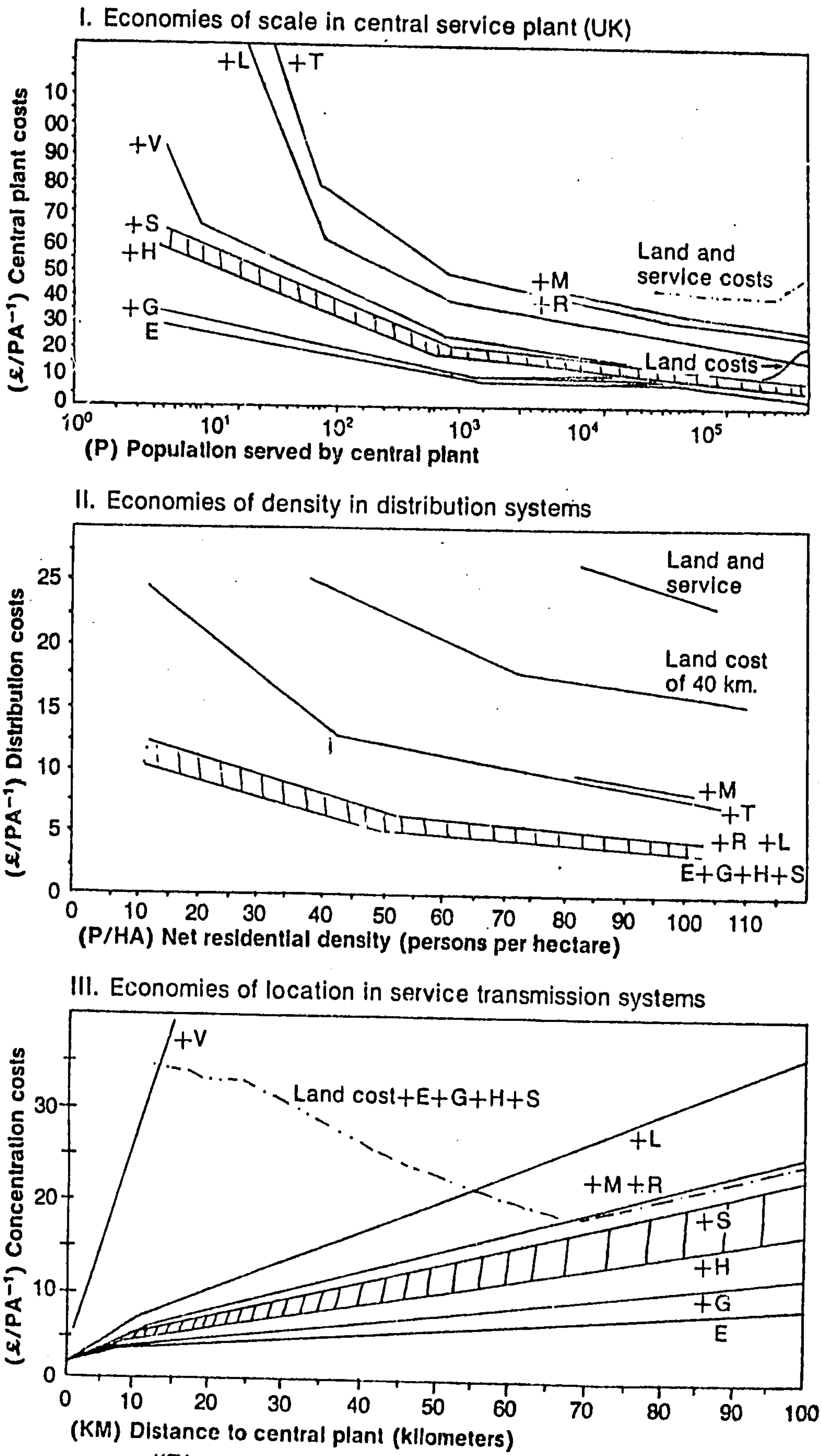
Although the cost of electrical power transmission lines is substantial (about 6,000 $/km in the U.S.), the tendency in the industrialized countries has been to rely on large regionally-centralized energy supply systems, with extensive electrical power transmission systems.

The three graphs shown in Figure 5 show economies of scale in central service plants, economies of density in distribution systems, and economies of location in service transmission systems [4].

Scale also has some bearing on the efficiency with which energy conversion devices operate. Figure 6 indicates the efficiencies that may be generally expected. It is noteworthy that the conversion between electrical and mechanical (rotational) energy is very efficient for large generators and electric motors.

TABLE 3 CHARACTERISTICS OF ENERGY SYSTEM SCALE

SCALE	CHARACTERISTICS
Single-site	Energy system scaled to match end-use device at the site. Generally a small system located at, or close to, the site. Transmission of energy not required between system and end-use except for very short distances. Large number of systems required, therefore possibility for rural industry and employment. Maintenance work is distributed over the area of the sites. Control is at the family level.
Community	Energy supply system scaled to community needs. Generally a larger system located centrally among many end-use sites. Energy transmission over short distances will be required; therefore, energy form must be electricity, or hydrocarbon fuels. Possibility of rural, industry, and employment but fabrication and construction may be difficult. Local infrastructure required for financing, management, operation, maintenance, supervision. Some economies of scale.
Regional	Energy supply system scaled to meet regional energy demand. Probably a a large and complex system located in an urban area. Efficient energy transmission over long distances will be required, therefore energy form must be electricity or liquid fuels. Limited possibility for rural people's participation in construction, operation, and maintenance of the system. Economies of scale may be significant but cost of transmission system and infrastructure may be high. Highly trained personnel required for project management.



KEY

E—Electricity G—Gas H—Water L—Telephone M—Mail
R—Refuse S—Sewage T—Heating V—Transport
(£/PA−1)—Pounds per person per annum
▨—Water plus sanitation

Figure 5.

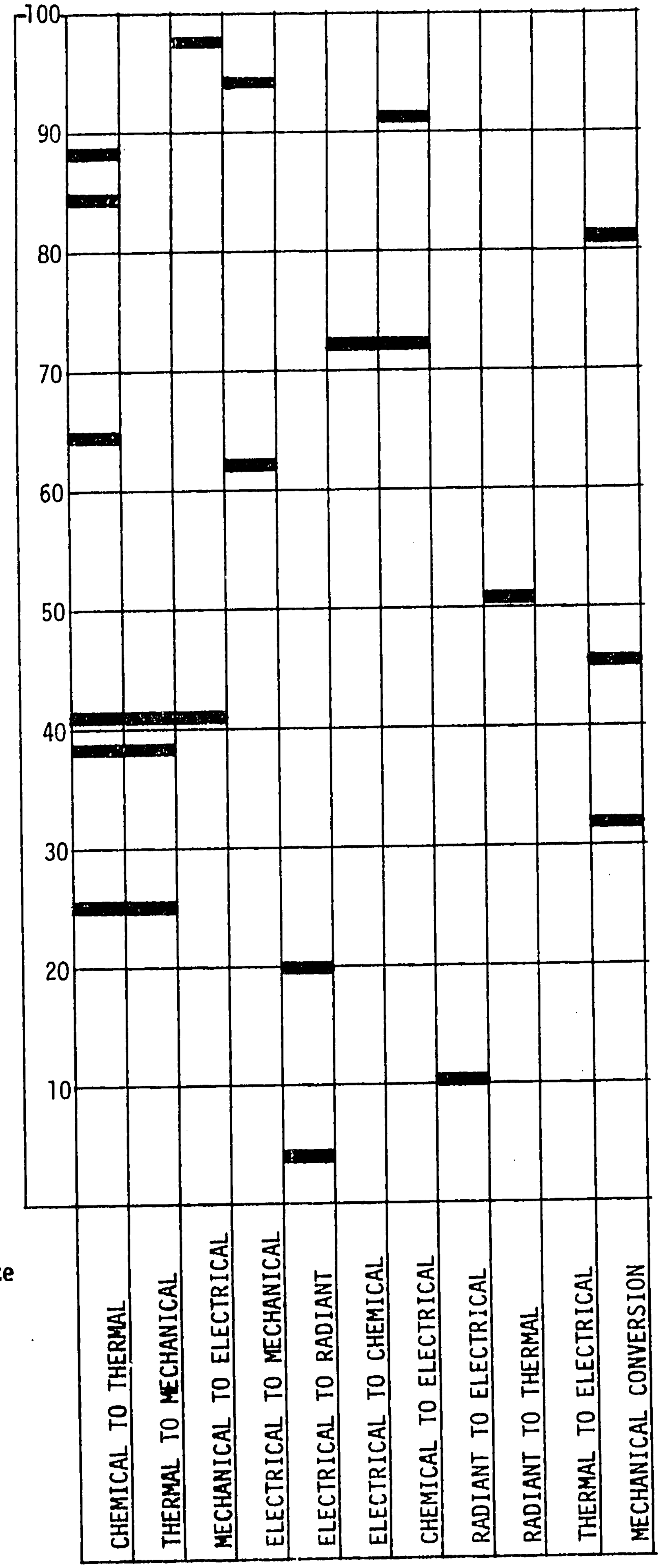


Figure 6. Approximate Energy Conversion Efficiencies. [5]

5. SYSTEM FLEXIBILITY

It may be important to consider the flexibility of the energy technology under consideration. Can the system be used to perform other tasks? Can it be easily adapted to perform other functions? Table 4 below shows the extent to which a renewable energy technology may be potentially utilized to satisfy an alternative end-use.

Table 4. Solar energy applicability matrix. Animals are included as a solar technology (based on photosynthesis). For many villages the use of animals would represent a modernizing step. It includes the use of dung for burning or fertilizing. Source: [6]
Symbols: ++, applicable; +, potentially applicable; −, not applicable.

Solar technology	Energy use													
								Heating						
	Water pumping	Lighting	Cooling	Communications	Water desalting	Spinning	Sawing	Cooking	Space	Domestic water	Grinding	Drying	Transport	Fertilizer
Solar cells (flat plate)	+	+	+	++	−	+	−	−	−	−	+	−	−	−
Flat-plate collectors	+	−	+	−	++	−	−	+	++	++	−	++	−	−
Concentrating collectors	+	+	+	−	+	−	−	+	−	−	−	−	−	−
Solar, Stirling	+	+	+	−	−	+	+	−	−	−	+	−	−	−
Solar, Rankine	+	+	+	−	−	+	+	−	−	−	+	−	−	−
Wind (mechanical)	++	−	−	−	−	+	+	−	−	−	++	−	−	−
Wind generator	++	++	+	++	−	+	+	−	−	−	++	−	−	−
Water (mechanical)	++	−	−	−	−	++	++	−	−	−	++	−	−	−
Hydroelectric	++	++	++	++	−	++	++	−	−	−	++	−	−	−
Bioconversion wood/pyrolysis	−	+	−	−	−	−	−	++	++	++	−	++	−	+
Biogas	+	++	+	−	−	−	−	++	−	+	−	+	+	++
Draft animals	++	−	−	−	−	−	−	+	−	+	++	−	++	++

6. SOCIO CULTURAL FACTORS

It was suggested at the beginning of these notes that one of the reasons for carefully matching a technology with an end-use activity was to try to minimize the disruption of existing practices and customs of the people utilizing the new technology. It is useful, at this stage, to examine this idea rather more closely and to consider some aspects of its complexity.

It should be made clear from the outset that some degree of disruption of traditional practice, custom, and organization is intended. Any intervention in the structure of a village or community has as its objective some kind of change - supposedly for the benefit, one way or another, of all or a sector of the local people. When we say we want to minimize disruption what we mean then is that we want to confine the changes brought about by a shift in technology to those consistent with the project objectives.

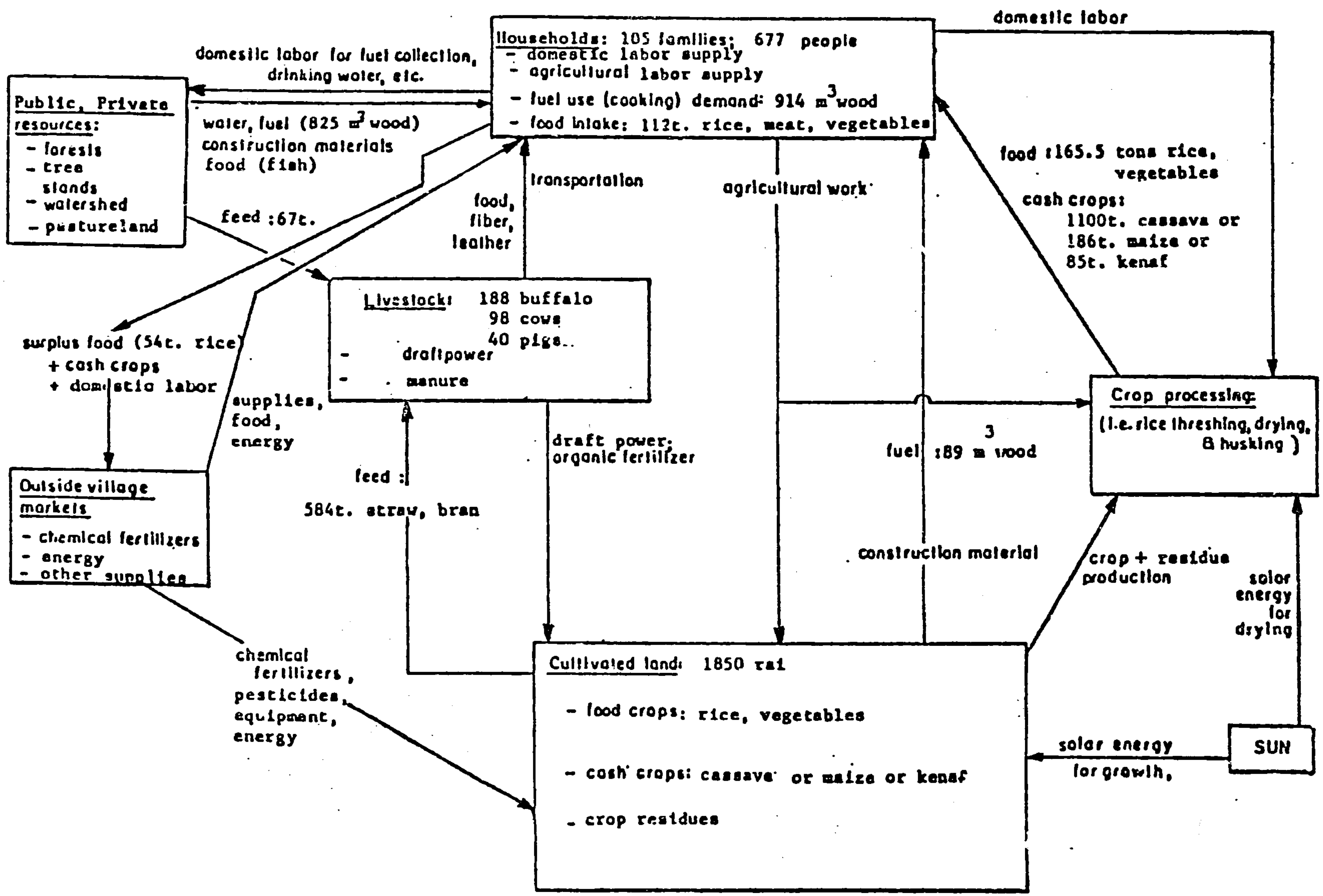
However, viewed as a system, a village economy is a quasi-autonomous, complex, and dynamic structure that functions, like all human communities no matter what the scale, by using energy and other physical resources to create an artificial synthetic environment within which people live and work. It is instructive to map the flow of resources and energy within such a system. Figure 7 shows energy-related resource flows for a village in Thailand [1]; Figure 8 shows resource flows for a village in Bangladesh [7]. Both diagrams illustrate the complexity, the circularity, and the inter-dependency of the resource flows.

So what happens when this system - a system essentially in equilibrium - is disturbed by the introduction of a new technology? What happens to these resource flows when we intervene in the structure?

Consider, for example, the Thai village depicted in Figure 7. There are a number of ways to intervene in this system, the intention being to modify the structure in such a way that benefits accrue to a specified sector of the community - perhaps the poorest class. Figure 9 shows how three technologies might be potentially integrated into the structure. Technology A could be charcoal production, improved stoves, microhydro, or simply a village wood lot. Technology B could be a biogas plant, or a pyrolysis unit producing both charcoal and pyrolytic oils. Technology C could be a solar-powered irrigation pump, a photovoltaic system, a solar dryer, or a solar-driven absorption refrigeration system.

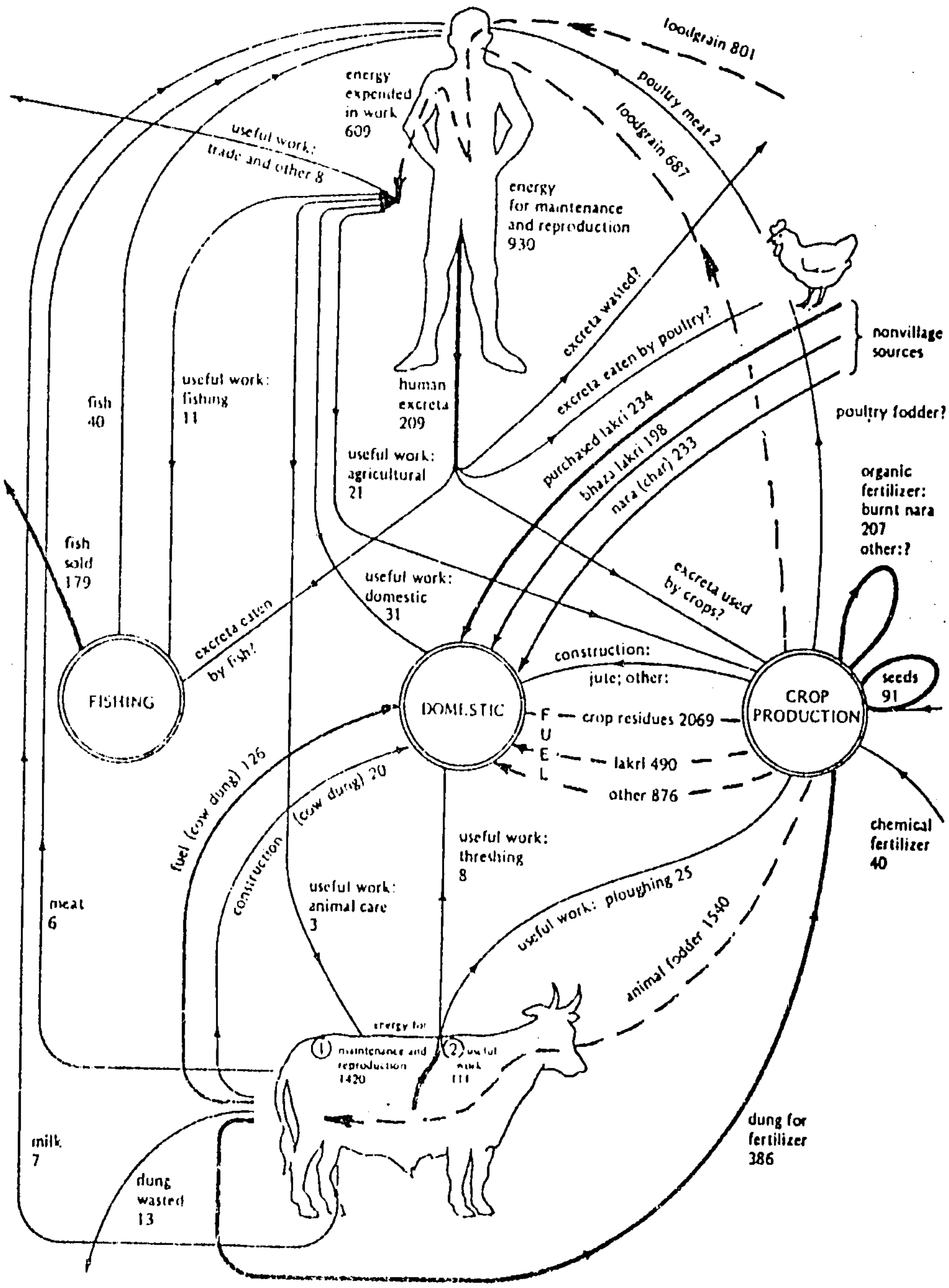
As an example, consider the introduction of a pyrolysis unit using a mixture of crop residues as feedstock. A one-ton per day unit operating for 150 days a year will require a feed of 150 tons of residues annually: about 70 tons of paddy husks and 80 tons of paddy straw. About 37.5 tons of charcoal and 22.5 tons of pyrolytic oils will be produced [1]. Figure 10 summarizes the resource flows affected by the technology. Inputs to the pyrolysis unit divert 80 tons of straw from animal feed, thereby requiring 80 more tons of feed from pasture land. The charcoal produced is equivalent to about 225 m^3 of wood supplied for fuel to households. This diminishes the fuelwood required from the forest land and private land from 825 to 600 m^3.

Figure 7. - VILLAGE ENERGY-RELATED RESOURCE FLOWS: NORTHEAST THAILAND



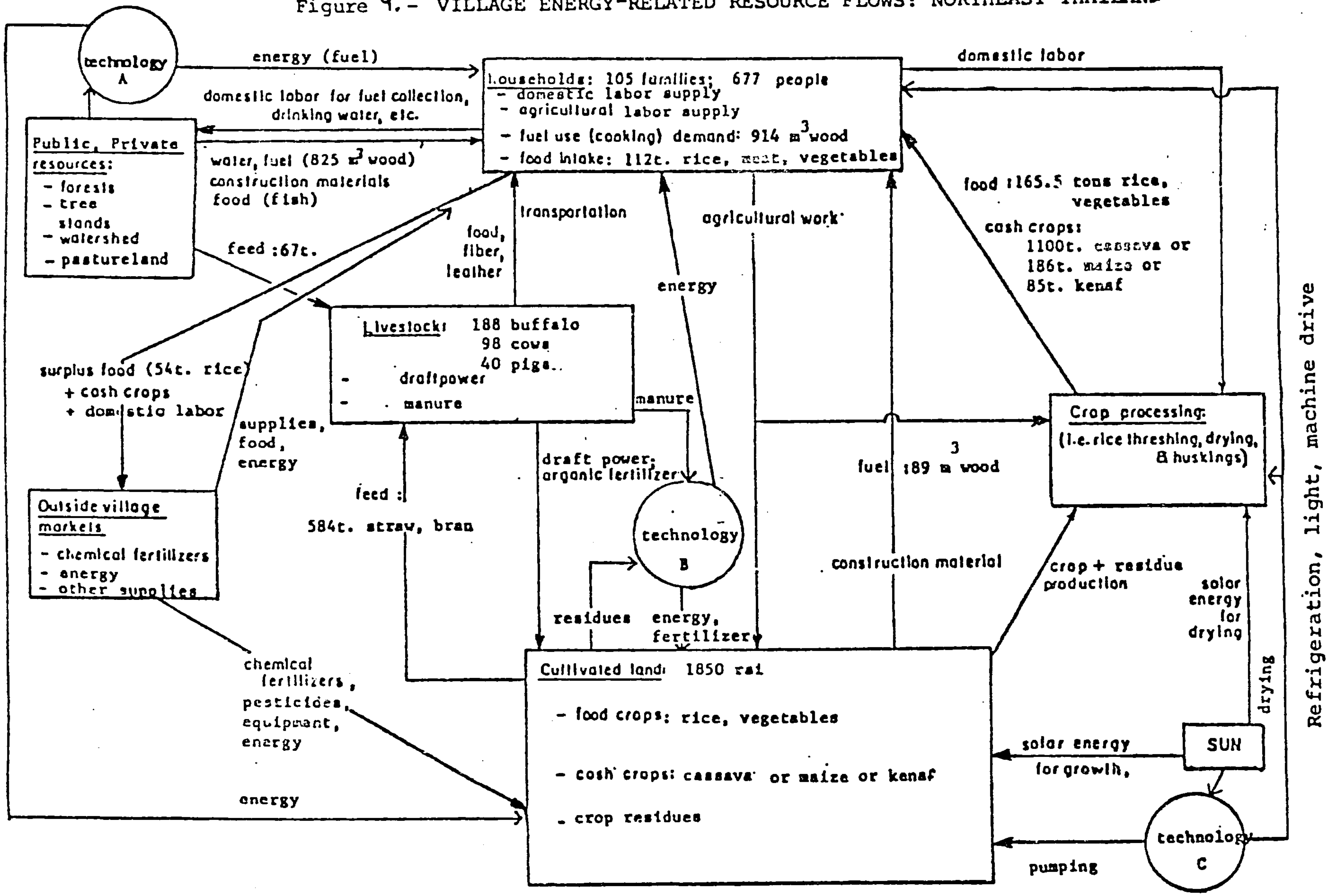
Source: Reference 1.

Figure 8.
Daily Per Capita Energy Flows
in a Village in Bangladesh (in kcals)



Source: Reference 7.

Figure 9.- VILLAGE ENERGY-RELATED RESOURCE FLOWS: NORTHEAST THAILAND



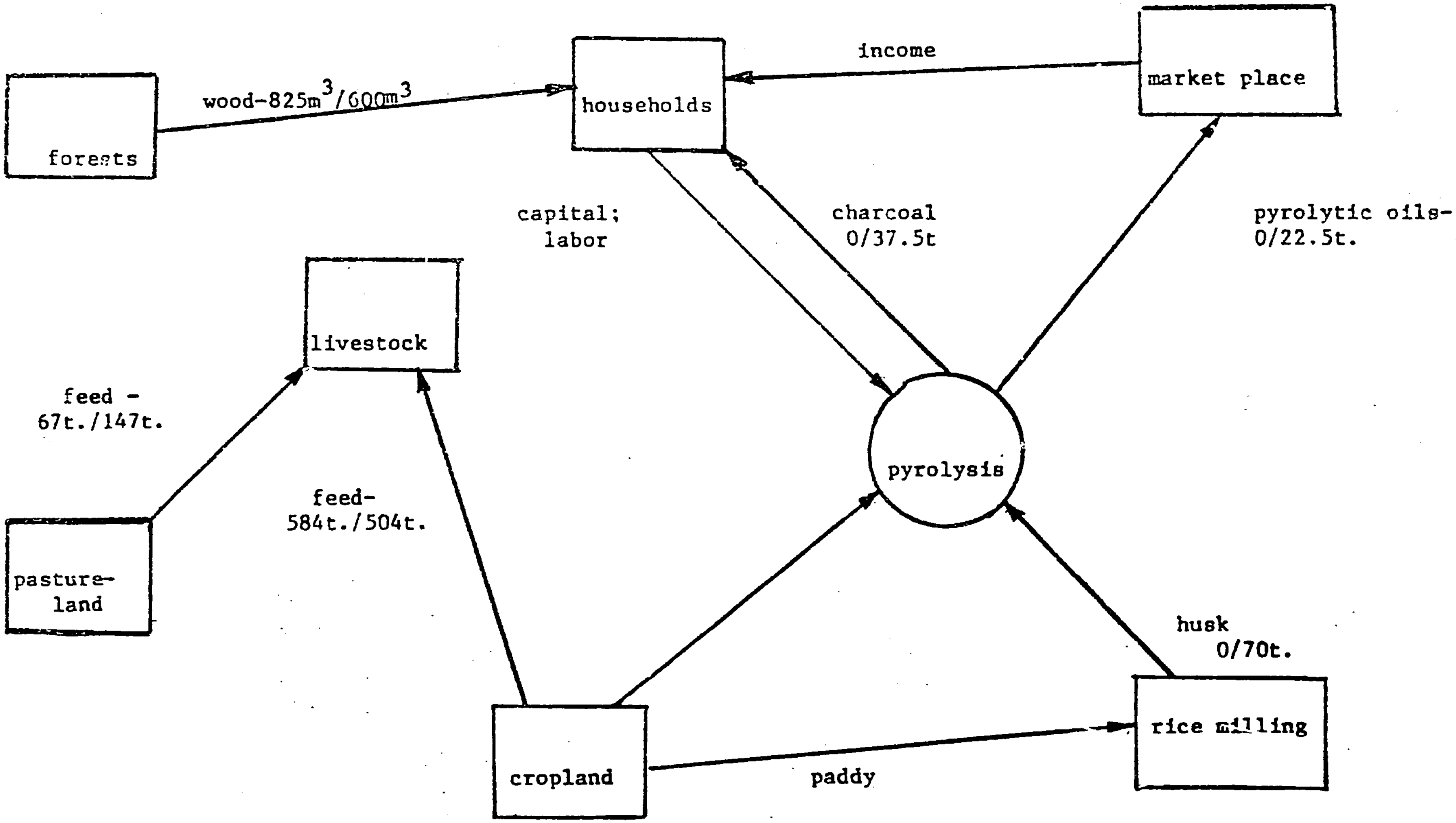


FIGURE 10. ALTERED ANNUAL RESOURCE FLOWS DUE TO PYROLYSIS OF AGRICULTURAL RESIDUES
(Flows before/after introduction of technology)

Moreover, the pyrolysis unit requires capital but the pyrolytic oils may be sold to outside markets, which will generate some cash income. In sum, the introduction of the pyrolysis unit reduces the need for forest resources, but at the expense of an increased demand for pasture land or fodder. Another example is presented in Figure 11 which shows the effect of introducing a biogas unit into the village. In this case the introduction of this technology diverts the use of dung from direct use as fertilizer. The unit requires capital and labour from the household but, in turn, provides fuel gas and fertilizer. In addition, biogas decreases the demand for wood from forest land and increases crop production thereby either raising cash income or reducing food purchases.

These examples illustrate that the introduction of a new technology into the social structure has a broad impact on village resource flows. The impact of the intervention cascades through the system. Resource demands shift, prices change, material that was once free may come to be valued as a resource and denied to the poor and the landless. The relatively wealthy, having greater control over local resources, may benefit significantly from the new technology while the poorer people may become even worse off than before. Figure 12 identifies some of the impacts that may potentially occur.

It is not easy to include these considerations in the process of matching a technology to an end-use. But one should recall the project objectives. If it is intended that, as a result of the project, economic evaluation of the benefits predominantly accrue to a target sector of the community, then the careful evaluation of the long-term socio-economic impacts of the intervention may be crucial to the success of the project.

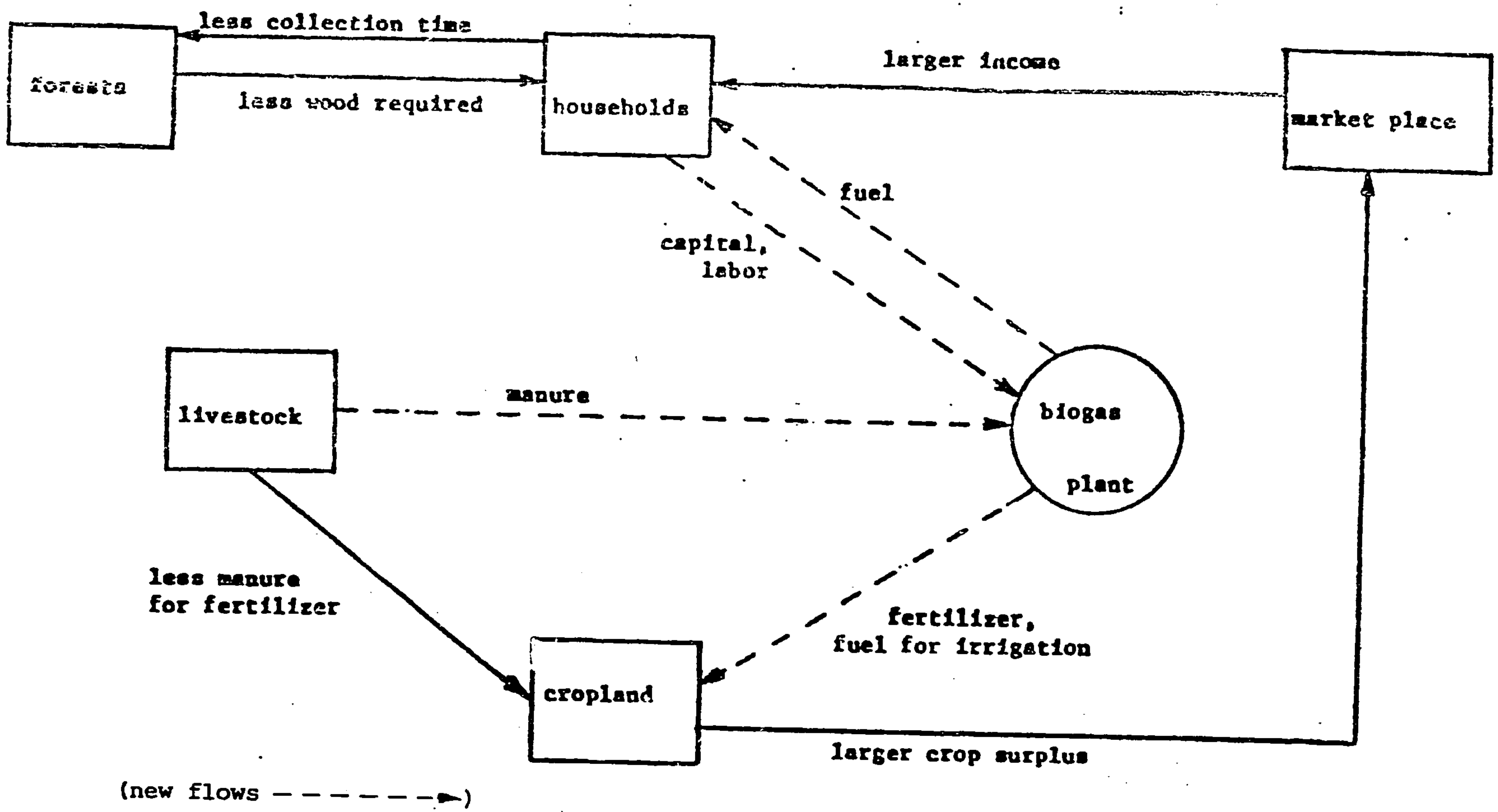


FIGURE 11. ALTERED RESOURCE FLOWS DUE TO BIOGAS PLANT

Figure 12. Potential Impacts Resulting From New Technology.

LEVEL 3

- Increased income
- Village self-reliance and prestige
- Unemployment caused by people being replaced by devices
- Altered socio-economic role of women
- Higher literacy
- Lowered birth rate
- Improved health
- Improved ecological conditions (e.g. slowed desertification

LEVEL 2

- Increased agricultural production
  - Increased acreage under cultivation
  - More intense cropping
- Increased off-farm production
  - Increased small industry
- More energy efficient and/or intensive farming
- Increased educational/recreational activities

LEVEL 1

- More efficient threshing, winnowing, irrigation, cultivation
- Better cooking systems
- Reduced time spent hauling and chopping
- Better lighting (possibility of working, playing and studying at night)
- Relief from onerous tasks
- Better heating/cooling systems
- Clean drinking water
- Washing and bathing in hot water
- stripping of vegetation for fuel

The energy source or technology

Source: Reference 8.

7. EXAMPLES OF END-USE MATCHING [2]

We examine here four end-use activities:

1) Cooking
2) Crop drying/Food preservation
3) Food processing
4) Refrigeration

1. Cooking

Best Matches: Biomass combustion and biogas generation. (Table 5)

Biogas and biomass energy systems do not depend directly on climate variations, but they do require a secure supply of organic materials, annual wastes, or agricultural residues. These materials can be stored, eliminating the need to store output energy. Biomass combustion is probably the most acceptable and convenient technology, providing that sufficient supplies of fuel can be easily obtained without environmental damage. In areas where there is widespread reliance on dung or crops residue combustion, biogas production is probably preferable despite its greater complexity and capital costs. A potential problem with biomass combustion is its general inefficiency although the development of new stoves is improving this aspect of the technology.

2. Crop drying/Food preservation

Best Matches: Flat-plate collectors and biomass direct combustion. (Table 6)

Flat-plate collectors and biomass combustion directly produce low to moder- ate temperature hot air without any energy transformation or storage devices. Both require only small amounts of capital and are simple to maintain and operate. Both technologies are considerably cheaper than the other potentially acceptable technologies, namely biogas generators and concentrating collectors. However, where biomass is scarce, or where its utilization exacerbates an existing environmental problem, biogas systems should be considered.

3. Food Processing (e.g. grinding)

Best Matches: Hydro-mechanical power generation (with water storage), wind turbines (mechanical), and biogas generators with modified internal combustion engines. (Table 7)

All three technologies can provide high torque, low-to-medium rpm shaft power for traditional processing techniques (stone grinding, seed crushing, shelling, etc.). These technologies can also be easily adapted to power higher rpm modern hammer or burr mills. Biogas generators and hydromechanical systems can provide shaft power on demand, providing there is sufficient resource base and adequate storage.

4. Refrigeration

Best Matches: Small-scale hydroelectric generation, biogas generators.
(Table 8)

Refrigeration requires a constant energy source on a daily and seasonal basis. A technology must have either a constant output to fuel the refrigation or sufficient storage to compensate for periods of inadequate or nonexistent output. Small-scale hydroelectric generators can meet these criteria in many locations with moderate inpoundment of water, and will provide electricity for existing DC or AC refrigeration systems. Biogas generators produce a gas capable of driving an absorption chiller.

Table 5. Cooking: Matching Technologies to the End-Use

Basic Need / Characterization Criteria	Cooking	Technologies: Wind	Match	Match w/Storage	Match with Energy Transformation	Photovoltaics	Match	Match w Storage	Match with/ Storage + Energy Transformation	Flat-Plate Collector	Match	Match w/Storage	Match with Energy Transformation	Concentrating Collectors	Match	Match w Storage	Match with Energy Transformation
A. Discrimination Criteria:																	
1 Type of Energy	Heat intermediate form Electrical Chemical	DC Electricity Shaft power	No	No	Yes	DC electricity With energy transformation heat shaft power	No	No	Yes	Heat (hot air and hot water) With energy transformation electricity shaft power	Yes	Yes	Yes	Heat (hot air or water) With energy transformation electricity shaft power	Yes	Yes	Yes
2 Temperature	100°-300°C	N/A unless with energy transformation	No	No	Yes	35°-400°C with energy transformation	No	No	Yes	40°-75°C for hot air 45°-90°C for hot water	No	No	Yes	150°-300°C for line focus unit, 150°-400°C for point focus unit	Yes	Yes	Yes
3 Spatial Distribution	One site per family (most common case)	Good for electricity	Yes	Yes	Yes	Good for electricity	Yes	Yes	Yes	Poor for heat good for electricity	No	No	Yes	Poor for heat, good for electricity	No		Yes
Conclusion on Discrimination Criteria	(If no, no need to continue this option)		No	No	Yes		No	No	Yes		No	No	Yes		Yes	Yes	Yes
B. Site-Specific Temporal & Climatic Criteria:																	
4 Seasonality	All year long	Varies			A	Insolation varies			A	Insolation varies			A	Insolation varies	A	Yes	A
5 Time of Day	Morning Noon Night	Varies			A	Daytime varies because of weather			A	Daytime, varies because of weather			A	Daytime, varies because of weather	No	Yes	A
6 Duration	1-3 hours	Highly variable 8-24 hours of power with storage			A	0-12 hours/day depending on weather, 24 hours if adequate storage			Yes	0-12 hours/day depending on weather, 24 hours if adequate storage			Yes	0-12 hours/day depending on weather, 24 hours if adequate storage	Yes	Yes	Yes
7 Sensitivity to Interruption	Cannot be interrupted	Will be interrupted unless adequate storage			A	Will be interrupted unless adequate storage			A	Will be interrupted unless adequate storage			A	Interrupted unless adequate storage	No	Yes	A
Conclusion on the above criteria					A				A				A		A	Yes	A
C. Site-Specific Social/Cultural/Environmental Criteria:																	
8 Usage by Type of Person	Primarily women	Traditional fuel gatherers replaced				Traditional fuel gatherers replaced				Traditional fuel gatherers replaced					Yes	No	Yes
9 Historical/Social/Religious Influences	Many variations of strong cultural influences on cooking	New cookstoves needed				New cookstoves needed				New cookstoves may be resisted if flavor changes or if outdoors				New cookstoves may be resisted if flavor changes or if outdoors			
10 Traditional Energy Sources Used	Wood Crop residue Animal waste Kerosene																
11 Environmental Costs & Benefits		Wood fuel is reduced			Yes	Wood fuel is reduced			Yes	Wood fuel is reduced			Yes	Wood fuel is reduced	Yes	Yes	Yes
12 Cost Considerations	Cookstoves should be under $25 each for domestic use	1-2 kWe unit with 16-kW storage $5,600-7,000 plus shipping			No	Cost of cells very high, plus storage			No	For electricity with storage $60,000-70,000 for 1-kW system			No	$16-35 for 150-300 W factory-built units	Yes	No	No
Conclusions must be made on a site by site basis																	

A if adequate resource B if adequate resource & storage C if one site D not needed

Cooking (continued)	Biogas Generators	Match	Match w/Storage	Match with Energy Transformation	Biomass Direct Combustion	Match	Match w/Storage	Match with Energy Transformation	Small Scale Hydro	Match	Match w/Storage	Match with Energy Transformation
A. Discrimination Criteria:												
1 Type of Energy	Heat from methane gas With energy transformation electricity shaft power	Yes	N/A	No	Heat With energy transformation shaft power electricity (steam turbine)	Yes	N/A	No	Electricity Shaft power With energy transformation heat	No	No	Yes
2 Temperature	100°-400°C	Yes	N/A	No	100°-500°C	Yes	N/A	No	As heat 100°-400°C	No	No	Yes
3 Spatial Distribution	Moderate for gas, good for electricity	Yes	N/A	No	Poor for heat good for electricity	Yes	N/A	No	Poor for shaft power, good for electricity	Yes	Yes	Yes
Conclusion on Discrimination Criteria		Yes	N/A	No		Yes	N/A	No		No	No	Yes
B. Site-Specific Temporal & Climatic Criteria:												
4 Seasonality	Supply of waste varies	Yes			Supply of biomass varies	Yes	N/A		Stream flow may vary, unless large water storage			A
5 Time of Day	Anytime when waste is available	Yes			Anytime	Yes	N/A		24 hours/day			A
6 Duration	3 or more hours, 24 hours if adequate storage	Yes			24 hours if input fuel is available	Yes	N/A		24 hours/day			A
7 Sensitivity to Interruption	May be interrupted if waste supply decreases	Yes			Minimal interruptions if adequate fuel is available and fed to stove	Yes	N/A		Will not be interrupted if stream flow is constant or water storage is available			A
Conclusion on the above criteria		Yes				Yes						A
C. Site-Specific Social/Cultural/ Environmental Criteria:												
8 Usage by Type of Person	Requires laborers to collect the waste fuels				Less wood gathering required	Yes	N/A		Traditional fuel gatherers replaced			
9 Historical/Social/ Religious Influences	New cookstoves needed				More efficient cookstoves required		N/A		New cookstoves required			
10 Traditional Energy Sources Used												
11 Environmental Costs & Benefits	Less biomass required plus fertilizer produced, water is required	Yes			Less biomass plus less wood gathering required		N/A		Wood fuel is reduced if reservoir made may use valuable land			Yes
12 Cost Considerations	$4 825 for 5 000 ft^3/day $10 000 for electricity	Yes			Cookstoves should be under $25 for domestic use	Yes	N/A	No	1 kW $4 525 5 kW $8 950 10 kW $12 550 plus electric cookstoves	No	No	No
Conclusions must be made on a site by site basis												

Table 6. Crop Drying: Matching Technologies to the End-Use

Characterization Criteria	Basic Need: Crop Drying Food Preservation	Technologies: Wind	Match	Match w/Storage	Match with Energy Transformation	Photovoltaics	Match	Match w/Storage	Match with/Storage + Energy Transformation	Flat-Plate Collector	Match	Match w/Storage	Match with Energy Transformation	Concentrating Collectors	Match	Match w/Storage	Match with Energy Transformation
A. Discrimination Criteria:																	
1 Type of Energy	Heat	Electricity Shaft power With energy transformation heat	No	No	Yes	Electricity Shaft power With energy transformation heat	No	No	Yes	Heat With energy transformation electricity shaft power	Yes	Yes	D	Heat With energy transformation electricity shaft power	Yes	Yes	D
2 Temperature	30°-60°C	-35°-400°C from heat	N/A	N/A	Yes	-35°-400°C with energy transformation into heat	N/A	N/A	Yes	45°-75°C from hot air	Yes	Yes	N/A	150°-300°C	Yes	Yes	N/A
3 Spatial Distribution	Depends on tradition: either one site per community or one per family	Good for electricity	Yes	Yes	Yes	Good for electricity	Yes	Yes	Yes	Moderate to poor for heat	Yes	Yes	Yes	Poor for heat good for electricity	Yes	Yes	Yes
Conclusion on Discrimination Criteria			No	No	Yes		No	No	Yes		Yes	Yes	D		Yes	Yes	D
B. Site-Specific Temporal & Climatic Criteria:																	
4 Seasonality	During the harvest season	Varies			A	Insolation varies			B	Insolation varies	A	A		Insolation varies	A	A	
5 Time of Day	Anytime	Varies			Yes	Daytime varies because of weather				Daytime varies because of weather	Yes	Yes		Daytime varies because of weather	Yes	Yes	
6 Duration	Over 6-12 hrs/day need to preserve before food spoils	Highly variable 8-24 hours/day with storage			B	0-12 hours depending on weather, 24 hours with storage			B	0-12 hours depending on weather 24 hours with storage	A	Yes		0-12 hours depending on weather, 24 hours with storage	A	Yes	
7 Sensitivity to Interruption	Can be interrupted for short periods of time	Will be interrupted unless adequate storage			Yes	Will be interrupted unless adequate storage				Will be interrupted unless adequate storage	Yes	Yes		Will be interrupted unless adequate storage	Yes	Yes	
Conclusion on the above criteria					B				B		A	A			A	A	
C. Site-Specific Social/Cultural/Environmental Criteria:																	
8 Usage by Type of Person	Women and children Farm families																
9 Historical/Social/Religious Influences	Techniques vary with custom, religion, and taste preference	Requires electric device more complex than traditional				Electrical device must be socially acceptable and easy to operate				Similar to traditional sun drying method				Similar to traditional sun drying method			
10 Traditional Energy Sources Used	Sunlight Wood fuel burned for drying/smoking																
11 Environmental Costs & Benefits	Some food spoils because drying too slow Wood fuel depleted	Wood not needed Drying occurs faster with less spoilage				Wood not needed Drying occurs faster with less spoilage				Wood not needed Drying occurs faster with less spoilage				Wood not needed Drying occurs faster with less spoilage			
12 Cost Considerations		1 kW with storage $5,500-6,400 plus cost of heat transformer				High cost of photovoltaic cells plus transformer and storage				$170-500/m² of collector plus the cost of storage				$25-100 for 1-1.5 m manually tracked units			
Conclusions must be made on a site by site basis																	

(A) if adequate resource (B) if adequate resource & storage (C) if one site (D) not needed

Crop Drying (continued)

	Biogas Generators	Match	Match w/Storage	Match with Energy Transformation	Biomass Direct Combustion	Match	Match w/Storage	Match with Energy Transformation	Small Scale Hydro	Match	Match w/Storage	Match with Energy Transformation
A. Discrimination Criteria:												
1. Type of Energy	Heat from methane gas With energy transformation electricity shaft power	Yes	Yes	D	Heat With energy transformation electricity (steam turbine) shaft power	Yes	N/A	D	Shaft power Electricity With energy transformation heat	No	No	Yes
2. Temperature	100°-400°C	Yes	Yes	N/A	100°-500°C direct heat, lower temperature hot air for drying	Yes	N/A	N/A	25°-400°C from heat	N/A	N/A	Yes
3. Spatial Distribution	Good for gas through pipelines	Yes	Yes	Yes	Poor for heat but could have several stove sites	Yes	N/A	N/A	Good for electricity, can transform to heat at each site	Yes	Yes	Yes
Conclusion on Discrimination Criteria		Yes	Yes	D		Yes	N/A	D		No	No	Yes
B. Site-Specific Temporal & Climatic Criteria:												
4. Seasonality	Supply of waste may vary	A	A		Supply of biomass may vary	A			Stream flow may vary unless water storage or pump			A
5. Time of Day	Anytime if waste is available	Yes	Yes		Anytime	Yes			Anytime			Yes
6. Duration	3-24 hours/day depending on supply of waste	A	A		24 hours if biomass fuel is available	Yes			24 hours/day if stream flow is constant or water storage			Yes
7. Sensitivity to Interruption	May be interrupted if waste supply decreases	A	A		Minimal interruption if adequate fuel is available and fed to stove	Yes			May be interrupted if water supply is not constant			A
Conclusion on the above criteria		A	A			A						A
C. Site-Specific Social/Cultural/Environmental Criteria:												
8. Usage by Type of Person												
9. Historical/Social/Religious Influences	Some religions may find use unacceptable for food drying				Similar to drying methods by open burning, although less smoke flavor				Electrical device must be socially acceptable and easy to operate			
10. Traditional Energy Sources Used												
11. Environmental Costs & Benefits	Fertilizer by-product Drying occurs with less spoilage				Less wood fuel used because stove is more efficient				Drying is faster, especially in areas with moisture, slow sun drying, and food spoilage			
12. Cost Considerations	$4 825 for 5 000 ft³/day plus cost of gas-fueled crop drying device				$15-35/unit, stoves need to be cheap to replace open burning method				1-kW Hopps unit, $4 525 plus cost of devices for heat transforming and drying			
Conclusions must be made on a site by site basis												

Table 7. Food Processing: Matching Technologies to the End-Use

Characterization Criteria / Basic Need: Food Processing (grinding, etc.)	Food Processing (grinding, etc.)	Technologies: Wind	Match	Match w/Storage	Match with Energy Transformation	Photovoltaics	Match	Match w/Storage	Match with Storage + Energy Transformation	Flat-Plate Collector	Match	Match w/Storage	Match with Energy Transformation	Concentrated Collectors	Match	Match w/Storage	Match with Energy Transformation
A. Discrimination Criteria:																	
1. Type of Energy	Mechanical shaft power intermediate form electrical	Shaft power Electricity			(D)	DC electricity With transformation shaft power	No	No	Yes	Heat With energy transformation shaft power electricity	No	No	Yes	Heat With energy transformation shaft power electricity	No	No	Yes
2. Temperature	Not Applicable	Not Applicable	N/A	N/A	N/A	N/A				45°-90°C with heat	N/A	N/A	N/A	150°-300°C	N/A	N/A	N/A
3. Spatial Distribution	Varies with village custom at central sites or each dwelling	Poor for shaft power, good for electricity	(C)	(C)	Yes	Good for electricity, then transform power at each site	Yes	Yes	Yes	Poor for heat or shaft power	No	No	(C)	Poor for heat or shaft power	No	No	(C)
Conclusion on Discrimination Criteria			(C)	(C)	(C)		No	No	Yes		No	No	(C)		No	No	(C)
B. Site-Specific Temporal & Climatic Criteria:																	
4. Seasonality	After the crop season	Varies	(A)	(A)		Insolation varies			(B)	Insolation varies			(A)	Insolation varies			(B)
5. Time of Day	Anytime	Varies	Yes	Yes		Daytime, varies because of weather			(B)	Daytime, varies because of weather			Yes	Daytime, varies because of weather			Yes
6. Duration	As long as needed	Varies widely 8-24 hours with storage	(A)	Yes		0-12 hours/day depending on weather and season			(B)	0-12 hours/day depending on weather and season			(A)	0-12 hours/day depending on weather and season			(B)
7. Sensitivity to Interruption	Can be interrupted	Will be interrupted unless adequate storage	Yes	Yes		Will be interrupted unless adequate storage			Yes	Will be interrupted			Yes	Will be interrupted unless adequate storage			Yes
Conclusion on the above criteria			(A)	(A)					(B)				(A)				(B)
C. Site-Specific Social/Cultural/Environmental Criteria:																	
8. Usage by Type of Person	Primarily women and children	Labor may be replaced				Labor may be replaced				Labor may be replaced				Labor may be replaced			
9. Historical/Social/Religious Influences	Religious and cultural patterns may affect how food is processed	New processing device must be accepted and feasible to operate				New processing device must be accepted and feasible to operate				New processing device must be accepted and feasible to operate				New processing device must be accepted and feasible to operate			
10. Traditional Energy Sources Used	Human power Animal power																
11. Environmental Costs & Benefits																	
12. Cost Considerations		Sailwing windmill and grinder $350-500; 1-2 kWe wind turbine with 16 kW storage $5,600-7,000	Yes	Yes		1.8 kW system $50,400 plus mill hammer			No	$170-500/m2 collector with grinder and storage			No	17 m2 system with 1 hour storage $4,000			No
Conclusions: must be made on a site by site basis																	

(A) if adequate resource (B) if adequate resource & storage (C) if one site (D) not needed

Food Processing (continued)

	Biogas Generators	Match	Match w/Storage	Match with Energy Transformation	Biomass Direct Combustion	Match	Match w/Storage	Match with Energy Transformation	Small Scale Hydro	Match	Match w/Storage	Match with Energy Transformation
A. Discrimination Criteria:												
1. Type of Energy	Heat from methane gas With energy transformation shaft power electricity	No	No	Yes	Heat With energy transformation shaft power electricity (steam turbine)	No	No	Yes	Electricity Shaft power With energy transformation heat	Yes	Yes	Ⓓ
2. Temperature	N/A				N/A				N/A			
3. Spatial Distribution	Good for gas; poor for shaft power	Yes	Yes	Yes	Poor for heat or shaft power	No	No	Ⓒ	Poor for shaft power; good for electricity	Ⓒ	Yes	Yes
Conclusion on Discrimination Criteria		No	No	Yes		No	No	Ⓒ		Ⓒ	Yes	Ⓓ
B. Site-Specific Temporal & Climatic Criteria:												
4. Seasonality	Supply of waste may vary			Ⓐ	Supply of biomass may vary			Ⓐ	Stream flow may vary without water storage or pump	Ⓐ	Ⓐ	
5. Time of Day	Anytime if waste is available			Yes	Anytime			Yes	Anytime	Yes	Yes	
6. Duration	24 hours/day if waste supply constant			Ⓐ	Anytime if biomass fuel is constant			Ⓐ	24 hours/day if stream flow is constant or water storage available	Ⓐ	Yes	
7. Sensitivity to Interruption	May be interrupted if waste supply decreases			Yes	Minimal interruption if adequate fuel is available and fed to stove			Yes	Will not be interrupted if stream flow is constant or water storage available	Yes	Yes	
Conclusion on the above criteria				Ⓐ				Ⓐ		Ⓐ	Ⓐ	
C. Site-Specific Social/Cultural/Environmental Criteria:												
8. Usage by Type of Person	Labor for processing replaced, labor for fuel collection needed				Labor for processing replaced, labor for fuel collection needed				Labor for processing replaced			
9. Historical/Social/Religious Influences	New processing method must be acceptable and feasible to operate				New processing method must be acceptable and feasible to operate				New processing method must be acceptable and feasible to operate			
10. Traditional Energy Sources Used												
11. Environmental Costs & Benefits	Fertilizer by-product				Wood fuel required				Possible land lost to reservoir			
12. Cost Considerations	$10 000 for 3 000 ft³/day of gas with engine for shaft power			Yes	Cost of boiler and turbine to produce shaft power			Yes	1 kW $4 525 2 kW $8 950 plus grinding equipment		Yes	
Conclusions: must be made on a site by site basis												

Table 8. Refrigeration: Matching Technologies to the End-Use

Characterization Criteria / Basic Need	Refrigeration	Technologies: Wind	Match	Match w/Storage	Match with Energy Transformation	Photovoltaics	Match	Match w/Storage	Match with/Storage + Energy Transformation	Flat-Plate Collector	Match	Match w/Storage	Match with Energy Transformation	Concentrating Collectors	Match	Match w/Storage	Match with Energy Transformation
A. Discrimination Criteria:																	
1. Type of Energy	Electricity Heat	Electricity Shaft power With energy transformation heat	Yes	Yes	Yes	Electricity With energy transformation heat	Yes	Yes	Yes	Heat (hot water or air) With energy transformation electricity shaft power	Yes	Yes	Yes	Heat With energy transformation electricity shaft power	Yes	Yes	Yes
2. Temperature	Cooling 0°-10°C Heat 100°-300°C	35°-400°C with energy transformation	N/A	N/A	Yes	35°-400°C with energy transformation	N/A	N/A	Yes	Heat 45°-90°C	No	No	N/A	Heat 150°-300°C	Yes	Yes	Yes
3. Spatial Distribution	1-50 locations per village, one central site likely	Good for electricity	Yes	Yes	Yes	Good for electricity poor for heat	Yes	Yes	Ⓒ	Poor for heat good for electricity	Ⓒ	Ⓒ	Yes	Poor for heat good for electricity	Ⓒ	Ⓒ	Yes
Conclusion on Discrimination Criteria			Yes	Yes	Yes		Yes	Yes	Ⓓ		No	No	Yes		Yes	Yes	Yes
B. Site-Specific Temporal & Climatic Criteria:																	
4. Seasonality	All year long	Varies	No	Ⓐ	No	Insolation varies	No	Ⓐ		Insolation varies			Ⓑ	Insolation varies	No	Ⓑ	Ⓑ
5. Time of Day	24 hours/day	Varies	No	Ⓐ	No	Daytime, but varies because of weather	No	Ⓐ		Daytime, but varies because of weather			Ⓑ	Daytime, but varies because of weather	No	Ⓑ	Ⓑ
6. Duration	24 hours/day	Highly variable 0-24 hours of power with adequate storage	No	Ⓐ	No	0-12 hours/day but varies with weather 24 hours/day with adequate storage	No	Ⓐ		0-12 hours/day, but varies with weather. 24 hours/day with adequate storage			Ⓑ	0-12 hours/day, but varies with weather. 24 hours/day with adequate storage	No	Ⓑ	Ⓑ
7. Sensitivity to Interruption	Cannot be interrupted	Will be interrupted unless adequate storage	No	Ⓐ	No	Will be interrupted unless adequate storage	No	Ⓐ		Will be interrupted unless adequate storage			Ⓑ	Will be interrupted unless adequate storage	No	Ⓑ	Ⓑ
Conclusion on the above criteria			No	Ⓐ	No		No	Ⓐ					Ⓑ		No	Ⓑ	Ⓑ
C. Site-Specific Social/Cultural/Environmental Criteria:																	
8. Usage by Type of Person	Medical staff. Persons who need food storage																
9. Historical/Social/Religious Influences																	
10. Traditional Energy Sources Used	None, food drying used																
11. Environmental Costs & Benefits	Improved health Can preserve perishable crops																
12. Cost Considerations		1-2-kW unit with 18-kW storage $5,600-7,100				1-kW system with storage and DC refrigerator $32,000	No	No	No	3-kW refrigerator with 100-m² collector $12,800			No	3-kW system with heat storage and absorption refrigerator $14,800			No
Conclusions must be made on a site by site basis																	

Ⓐ if adequate resource Ⓑ if adequate resource & storage Ⓒ if one site Ⓓ not needed

Refrigeration (continued)

	Biogas Generators	Match	Match w/Storage	Match with Energy Transformation	Biomass Direct Combustion	Match	Match w/Storage	Match with Energy Transformation	Small Scale Hydro	Match	Match w/Storage	Match with Energy Transformation
A. Discrimination Criteria:												
1 Type of Energy	Heat from methane gas With energy transformation electricity shaft power	Yes	Yes	Yes	Heat With energy transformation electricity (steam turbine) shaft power	Yes	N/A	Yes	Electricity Shaft power With energy transformation heat	Yes	Yes	Yes
2 Temperature	Heat 100°-400°C	Yes	Yes	N/A	Heat 100°-500°C	N/A	N/A	N/A	-35°-400°C with energy transformation	N/A	N/A	Yes
3 Spatial Distribution	Good as gas through pipe-lines or as electricity, poor for shaft power	Yes	Yes	Yes	Poor for heat, good for electricity	C	N/A	Yes	Good for electricity poor for heat	Yes	Yes	C
Conclusion on Discrimination Criteria		Yes	Yes	D		Yes	N/A	Yes		Yes	Yes	D
B. Site-Specific Temporal & Climatic Criteria:												
4 Seasonality	Supply of waste may vary	A	A		Supply of biomass may vary	A		B	Stream flow may vary without water storage or pump	No	A	
5 Time of Day	Anytime when waste is available	Yes	Yes		Anytime	A		B	Anytime	Yes	Yes	
6 Duration	24 hours/day if fuel supply is constant	A	A		24 hours if biomass fuel supply is constant	A		B	24 hours/day	Yes	Yes	
7. Sensitivity to Interruption	Will be interrupted if waste supply decreases	A	A		Minimal inter-ruption if adequate fuel is available and fed to stove	A		B	Will not be interrupted if stream flow is constant or water storage available	No	A	
Conclusion on the above criteria		A	A			A		B		No	A	
C. Site-Specific Social/Cultural/Environmental Criteria:												
8 Usage by Type of Person												
9 Historical/Social/Religious Influences												
10 Traditional Energy Sources Used												
11 Environmental Costs & Benefits												
12 Cost Considerations	$4 825 for 5 000 ft day of gas plus waste or gas storage	Yes	Yes	No	Combustion unit plus device to use heat for refrigeration or energy transformation				1 kW Hoppes unit $4 525 plus battery and or water storage		Yes	N/A
Conclusions must be made on a site by site basis												

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THE COST OF RENEWABLE ENERGY SYSTEMS:

SOME CASE STUDIES [1]

SMALL-SCALE WATER PUMPING

Water pumping for irrigation is one of the major agricultural energy uses in the developing countries. For small cultivated plots in the 1 to 5 hectare range, the most common pumping methods rely on traditional manual or animal-powered devices or on small internal combustion pumpsets. Where there is no supply of electricity, and the power required for irrigation is modest, renewable energy powered pumping methods may be cost competitive with present practice.

We will compare the economics of six alternative systems for pumping water from shallow sources. The technologies are:

1) 3 HP gasoline engine and pump
2) 250 Wp photovoltaic water pump
3) U.S. manufactured wind system
4) Locally built sail-wing wind pump
5) 1 kW SOFRETES solar irrigation pump
6) Biogas plant with pumpset

It is assumed that we wish to pump 15,000 gallons per day. The dynamic head is assumed to be approximately 20 feet. This quantity of water pumped in one day (57 m^3) would cover a one hectare plot with just over one-half centimeter of water.

1. 3 HP Gasoline Engine and Pump

A Briggs and Stratton 3 HP pump has been selected (model MFE-30). It is one of the smallest readily available gasoline-fueled pumpsets. Although small diesel pumps are available in this size or slightly larger, they are not so common and they are more costly. This pumpset has a capacity of 110 gpm and can pump to a head of 100 feet. It is therefore oversized for this application, but smaller pumps are not readily available.

The capital cost of $490 includes the cost of the engine, pump, hoses, and mounting frame. We estimate that at the reduced head of 20 ft. the pump will deliver about 200 gpm. The fuel consumption is 1.1 litres per hour.

To deliver 15,000 gallons per day the pump must operate for

$$\frac{15,000}{200 \times 60} = 1.25 \text{ hrs/day}$$

The pump will consume, therefore, 1.375 litres of gasoline each day. If it operates for 200 days each year, the annual fuel consumption will be 275 litres of gasoline a year.

The following data therefore apply:

Capital cost	\$490
Transportation	\$300
Annual maintenance	75 \$/yr.
Fuel use	275 litres/yr.
Useful life	5 years

If we amortize the cost of the pumpset over its useful life (5 years), discounting at 10% per annum, we can determine a capital recovery factor (CRF) as:

$$CRF(0.1,5) = \frac{0.1}{1-(1+0.1)^{-5}} = 0.2638$$

So the annual charges resulting from capital cost and transportation charges are equal to 0.2638 x 790 = 208.40 \$/yr. Gasoline is assumed to cost 60¢/litre in 1980 and escalates at 5% per year in real terms. The 1990 cost will therefore be 98¢/litre. We can therefore arrive at the following estimates of present and future costs:

	1980	1990
Fuel costs (¢/litre)	60	98
Annual fuel cost	165 \$/yr.	270 \$/yr.
Annual maintenance	75 \$/yr.	75 \$/yr.
Capital charges	208 \$/yr.	208 \$/yr.
Total	448 \$/yr.	553 \$/yr.

The output of water is 3 million gallons per year. So the cost of irrigation is estimated as

1980: 15 ¢/1000 gallons
1990: 18 ¢/1000 gallons

2. 250 Watt (Peak) Photovoltaic System

The system under consideration here is the Tri-Solar Corporation Model SEI-15M Portable Sun Pump with a 250 Wp array. It will pump about 12-15,000 gallons per day to a height of 8-15 feet, using a submersible centrifugal pump. No electrical storage is provided. The present price is \$5,250. The expected lifetime is 15 years.

The following data are assumed to apply:

	1980	1990
Capital cost	$5250	$1813
Transport	800	500
Maintenance	75	75
Useful life - pump	10 yrs	10 yrs
- array	15 yrs	15 yrs
Salvage value after 10 yrs	$50	$50

The salvage value should be discounted over ten years at 10% and deducted from the capital cost. The capital costs therefore become:

1980: $5231
1990: $1794

We amortize the cost of the system over its lifetime of ten years.

CRF(0.1, 10) = 0.16275

We therefore have the following estimates for present and future system costs.

	1980	1990
Capital charges	982 $/yr	373 $/yr
Maintenance	75 $/yr	75 $/yr
Total	1057 $/yr	448 $/yr

If the system pumps 3 million gallons of water each year, the cost is given as:

1980: 35¢/1000 gallons
1990: 15¢/1000 gallons

3. U.S. Manufactured Wind System

We make the optimistic assumption that we have a site with, on average, 15 mph winds. The machine selected is an Aeromotor WECS with 8 ft diameter swept area. It is mounted on a 27 ft tower and is expected to pump an average of 1875 gallons per hour for 8 hours each day. The cost of the wind system is $2670 FOB Dallas, Texas. The system weighs 900 pounds. It is further assumed that by 1990 a similar system will be available locally at a lesser price. The following data then apply:

	1980 (U.S. made)	1990 (locally made)
Capital cost		
Wind turbine	$1136	
Tower	980	
Cylinder	437	
Misc. equipment	117	
Total capital	$2670	$1781
Transport	$ 800	$ 400
Installation	$ 450	$ 450
Salvage value	$ 700	$ 400
Maintenance	100 $/yr	100 $/yr
Useful life	10 yrs	10 yrs

We need to discount the salvage value back to the present (at 10% per year) and subtract that amount from the capital cost. The discount factor is 2.5937. The capital costs then become:

1980: $2400
1990: $1627

We know CRF(0.1, 10) = 0.16275

So we have the following cost data:

	1980	1990
Capital charges	594 $/yr	403 $/yr
Maintenance	100 $/yr	100 $/yr
Total	694 $/yr	503 $/yr

The cost of water supply (3 million gallons a year) is therefore given by:

1980: 23¢/1000 gallons
1990: 17¢/1000 gallons

4. Locally Built Sail-Wing Wind Pump

The system taken here as typical is based on the machines described in reference 2. Since the output of the wind pumper is well below 15,000 gallons per day (assuming the 15 mph - average wind) we have taken costs for 3 wind systems. The costs are estimated as follows.

Capital cost (3 machines)	$2940
Maintenance	100 $/yr
Useful life	10 yrs

Since CRF(0.1, 10) = 0.16275

we have capital charges of	478 $/yr
and maintenance costs of	100 $/yr
	578 $/yr

The cost of water supplied is therefore 19¢/1000 gallons.

The cost of the locally fabricated sail-wing wind pumper is expected to remain relatively constant.

5. 1 kW SOFRETES Solar Irrigation Pump

The basic system developed by SOFRETES (Societé Francaise d'Etude Thermiques et d'Energie Solaire) uses flat plate collectors to heat water which passes through a heat exchanger (evaporator) in which an organic liquid such as butane or Freon is vaporized. The vapour drives a Rankine cycle reciprocating engine or turbine. The vapour is condensed in another heat exchanger using the pumped irrigation water at coolant. The first 1 kW SOFRETES system cost as much as $50,000. The company now claims to be able to provide a similar system for $25,000 [3]. We make the additional optimistic assumption here that the capital cost of this system will fall by a further 50% by 1990. The following data therefore apply.

	1980	1990
Capital cost	$25,000	$12,500
Maintenance	100 $/yr	100 $/yr
Useful life	15 yrs	15 yrs

$$\text{CRF}(0.1, 15) = \frac{0.1}{1 - (1.1)^{-15}} = 0.13147$$

	1980	1990
Capital charges	3287 $/yr	1643 $/yr
Maintenance	100 $/yr	100 $/yr
	3387 $/yr	1743 $/yr

The cost of irrigation water as produced by a SOFRETES pump is therefore given by:

1980: 1.13 $/1000 gallons
1990: 0.58 $/1000 gallons.

6. Biogas Plant with Pumpset

Gasoline engines will run on biogas at about 80% of rated output. The engine usually needs to be started on gasoline and then switched over to biogas. We assume the pump will now deliver 0.8 x 200 = 160 gpm. To deliver 15,000 gallons per day the pump must operate for

$$\frac{15,000}{160 \times 60} = 1.56 \text{ hrs/day}$$

The consumption of biogas should be approximately 0.5 m^3 per HP-hr rated. Using the same 3 HP Briggs and Stratton pumpset that was analyzed before, we can estimate the daily biogas consumption as

$$1.56 \text{ (hr)} \times 3 \text{ (HP)} \times 0.5 \text{ (m}^3\text{/HP-hr)} = 2.34 \text{ m}^3$$

We have good cost data on a Chinese-type biogas system that generates about 4 m^3/day of gas (see Chapter 8). Although the system is over-sized it will provide a conservative estimate of the cost of pumping water using such a system.

The following data apply:

3 HP Pumpset

Capital cost	$490
Transportation	$300
Annual maintenance	75 $/yr
Useful life	5 yrs

4m^3/day Biogas Plant

Capital cost	$454
Operation and maintenance	50 $/yr
Useful life	10 yrs

Amortizing the cost of the pumpset over 5 years and the biogas plant over 10 years, both at 10%, gives the following estimates for annual costs.

Capital charges - pumpset	208 $/yr
Maintenance - pumpset	75 $/yr
Capital charges - biogas plant	74 $/yr
Maintenance - biogas plant	50 $/yr
	407 $/yr

The cost of pumping water (3 million gallons a year) is therefore 14¢/1000 gallons.

3. The value of wind-pumping is highly site-specific. For areas with average winds better than 15 mph it is possible that locally-produced wind systems should be considered. They have the advantage of minimal foreign-exchange costs.

4. An important conclusion from this analysis concerns the question of scale. As we have mentioned, the 3 HP gasoline pumpset is oversized for this application. One of the advantages of many renewable energy technologies is that they can be sized for small-scale applications such as irrigating the one hectare area of land used in this example. As the task becomes larger-scale, however, the gasoline and diesel-powered systems become increasingly economic. For example, if the area of land were to increase by a factor of four, the same 3 HP engine could accomplish the same task simply by being operated for 5 hours a day instead of the present 1.25 hours. The annual fuel costs would rise to 660 $/yr; total yearly expenses to 943 $/yr, but because the system now pumps 12 million gallons of water a year the unit cost falls to 8¢/1000 gallons. Even at the 1990 gasoline price of 98¢/gallon the cost of irrigation is still only 11¢/1000 gallons.

   With the exception of the biogas system, all the other pumping technologies would not be expected to show any significant decline in the cost of pumping.

   With the biogas plant, it will be necessary to scale up the size of the digester but the pumpset can remain the same. A digester about three times as large should be sufficient for the task. The annual charges would now appear as

Capital charges - pumpset	208 $/yr
Maintenance - pumpset	75 $/yr
Capital charges - biogas plant	222 $/yr
Maintenance - biogas plant	150 $/yr
	655 $/yr

This is equivalent to 5.5¢/1000 gallons -- a substantial reduction over the previous cost.

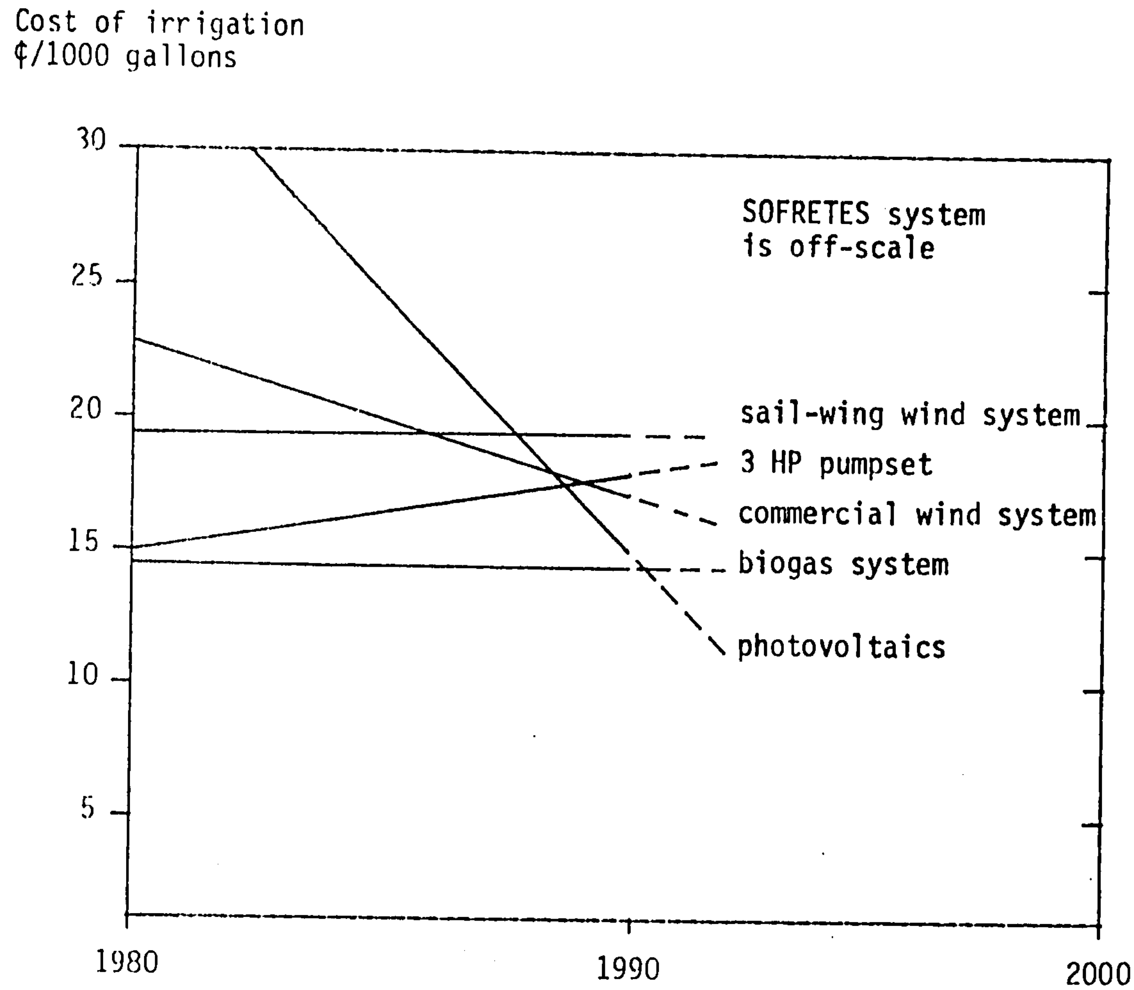

Figure 1 Base Case (3 million gallons/year)

Conclusions

1. Because of its rapidly declining capital costs and low recurrent costs, the photovoltaic system appears to be economically competitive with gasoline powered pumpsets by 1990. However, this scenario is based on several precarious assumptions concerning the future price of photovoltaic systems. It remains to be seen whether these cost projections will prove to be accurate.

2. The biogas system appears to be an attractive option. Its viability will depend on a number of factors one of which is the availability of sufficient dung.

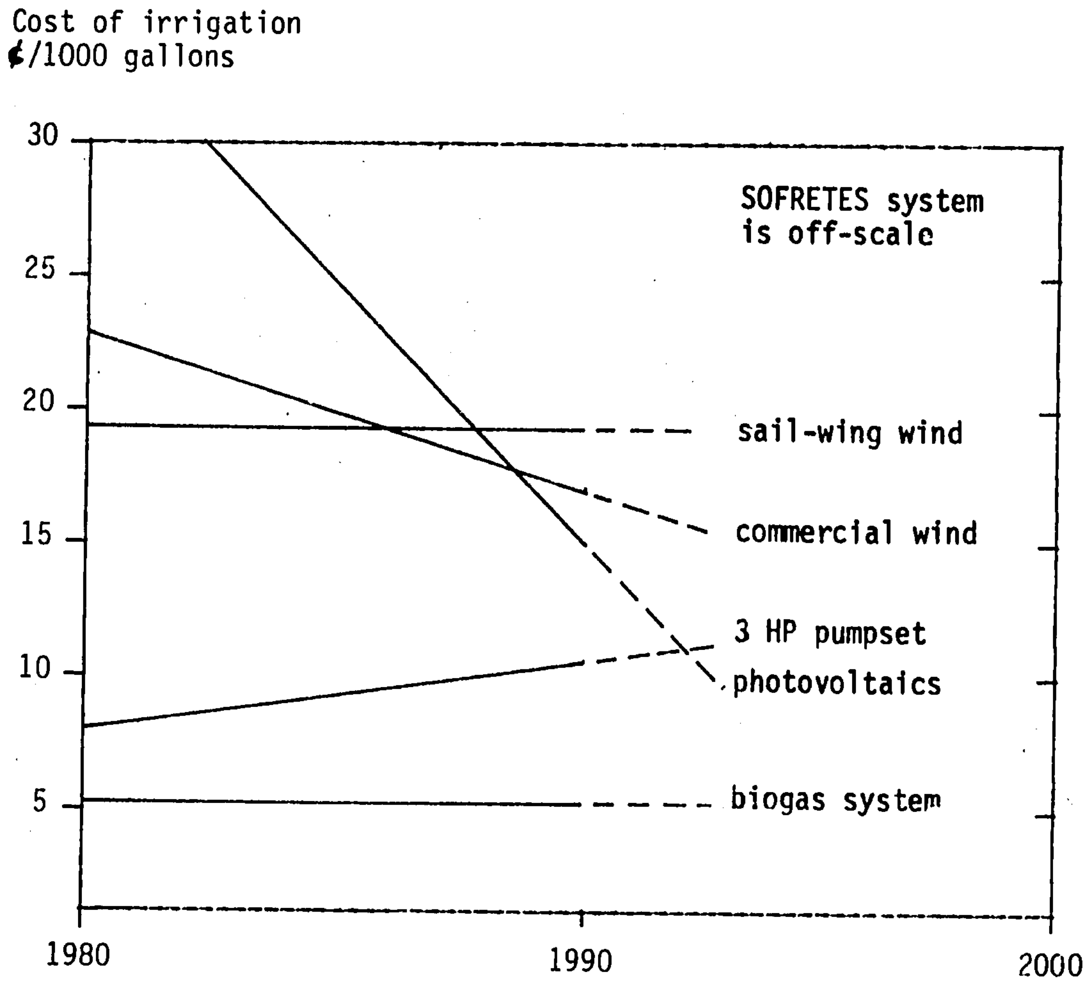

Figure 2 12 million gallons/year

VILLAGE ELECTRIFICATION

This case study examines five technologies for providing electricity to rural areas. Traditionally, rural economies have opted for national grid extensions, where these have not been prohibitively expensive, or have employed small-scale diesel generators to generate electricity. As petroleum fuels become increasingly expensive, particularly in the rural areas of developing countries, there is strengthening interest in generating electricity from small-scale renewable energy systems.

We will compare the economics of five alternative systems for generating electrical power. The technologies are:

1) 4 kW diesel generator
2) 3.5 kWp photovoltaic system
3) vertical axis wind system
4) small-scale hydropower
5) community-scale biogas system

The base load we consider here is relatively small-scale, providing approximately 15 kWh/day. This amount of electrical energy would be sufficient to provide lighting for 20 - 30 homes or to run perhaps 10 refrigerators.

1. 4 kW Diesel Generator

Winco Division of Dyna Technology, Inc. (Minneapolis) produces a variety of diesel generator systems available in a wide range of sizes. A small unit is rated at 4 kW and weighs 624 pounds. In this application, it has been estimated that the generator will be run for 5 hours each day at 80% efficiency for 350 days each year to give

$$5 \left(\frac{\text{hr}}{\text{day}}\right) \times 350 \text{ days} \times 4 \text{ kW} \times 0.8 = 5600 \text{ kWh/yr}$$

Pertinent costs for the system are shown below. These costs assume a 10 year useful lifetime with oil changes every 60 hours, tune-ups every 1000 hours, and overhauls every 5 years.

Capital cost (including overhaul)	\$4743
Shipping	\$1000
Installation	\$1000
Maintenance	700 \$/yr
Fuel use	2782 litres/yr
Useful life	10 years

We amortize the cost of the generator over its useful life discounting at 10% per annum.

Since CRF (0.1, 10) = 0.16275, capital charges will be 0.16275 x 6743 = 1097 \$/yr. The cost of diesel is taken as 50 ¢/litre and escalates at 5% per year to reach 81 ¢/litre by 1990. The oil change is estimated to cost \$163 in 1980, \$199 in 1990. The annual expenses associated with this system can therefore be estimated as follows.

	1980	1990
Capital charges	1097 \$/yr	1097 \$/yr
Maintenance	700 \$/yr	700 \$/yr
Fuel	1391 \$/yr	2253 \$/yr
Oil changes	163 \$/yr	199 \$/yr
Total annual costs	3351 \$/yr	4249 \$/yr

The cost of the generated electrical energy (5600 kWh/yr) is therefore:

1980: 60¢/kWh
1990: 76¢/kWh

2. 3.5 kWp Photovoltaic System

Costs and other data associated with this system are shown below:

	1980	1990
Module cost (\$/Wp)	\$10.00	\$2.00
Balance of system (\$/Wp)	\$16.50	\$8.00
Invertor	\$5,500	(DC system)
3.5 kW cost	\$98,250	\$35,000
Transport	\$4,500	\$2,500
Operation and maintenance	1000 \$/yr	350 \$/yr
Useful life - array	15 yrs	15 yrs
Useful life - battery	5 yrs	5 yrs
Salvage value - array (\$/Wp)	1.00	0.25
Salvage value - battery (\$/Wp)	1.75	1.00

Although the balance of system (BOS) costs indicated above include the cost of the battery, the battery cost is estimated as 3.30 \$/Wp (1980) and 2.75 \$/Wp (1990). We now need to deduct the present value (PV) of the salvage values from the system capital cost. The future income is discounted at 10% per year.

	1980	1990
PV of array salvage value	\$838	\$209
PV of battery salvage value	\$3,803	\$2,173
Adjusted system cost ex-battery	\$85,862	\$25,166
Adjusted cost of battery	\$7,747	\$7,452
Transportation	\$4,500	\$2,500

The cost of the battery and the cost of the rest of the system are amortized over different time periods. The battery cost is amortized over 5 years at 10% per year.

$$\text{CRF } (0.1,\ 5) = \frac{0.1}{1 - (1.1)^{-5}} = 0.2638$$

So the annual charges due to the battery are given by

1980: 0.2638 x 7747 = 2044 \$/yr
1990: 0.2638 x 7452 = 1966 \$/yr

The rest of the system has a lifetime of 15 years. Since

$$CRF\ (0.1,\ 15) = \frac{0.1}{1 - (1.1)^{-15}} = 0.13147$$

the annual charges due to the rest of the system (plus transportation) are therefore given by:

1980: 0.13147 x 90,362 = 11,880 \$/yr
1990: 0.13147 x 27,666 = 3,637 \$/yr

Annual expenses therefore amount to:

	1980	1990
Battery	2,044 \$/yr	1,966 \$/yr
Rest of system	11,880 \$/yr	3,637 \$/yr
Operation and maintenance	1,000 \$/yr	350 \$/yr
	14,924 \$/yr	5,953 \$/yr

The cost of electrical energy (5600 kWh per year) is then:

1980: 2.67 \$/kWh
1990: 1.06 \$/kWh

3. Vertical Axis Wind System

An option for remote village electrification is the use of a wind driven generator. Wind machines are available from several manufacturers in the United States. The Pinson Cycloturbine (Marston Mills, Massachusetts) has been selected here as an example of the modern wind technology. The machine is a vertical axis wind turbine (VAWT) with 10 foot blades. The machine is rated at 4 kW in a 20 mph wind. The following technical data apply.

	1980	1990
Capital cost	\$7200	\$5800
Invertor	\$2500	(DC system)
Batteries	\$4000	\$4000
Installation	\$1000	\$1000
Transport	\$3500	\$1500
System weight	4600 lbs	4600 lbs
Annual output	5412 kWh	5412 kWh
Useful life - VAWT	10 yrs	10 yrs
Useful life - batteries	5 yrs	5 yrs
Salvage value - VAWT	\$1500	\$1500
Salvage value - batteries	\$1000	\$1000
Operation and maintenance	\$685	\$615

We need to adjust the capital cost to reflect to present value (PV) of the salvage values. We have

	1980	1990
PV of VAWT salvage value	\$578	\$578
PV of battery salvage value	\$621	\$621
Adjusted VAWT cost	\$6622	5222
Adjusted battery cost	\$3379	\$3379

We amortize the batteries over 5 year and the rest of the system (VAWT + invertor + installation + transport) over 10 years. Using CRF (0.1, 5) = 0.2638 and CRF (0.1, 10) = 0.16275 we arrive at the following estimate of annual expenses.

	1980	1990
Batteries	891 \$/yr	891 \$/yr
Rest of system	2217 \$/yr	1257 \$/yr
Operation and maintenance	685 \$/yr	615 \$/yr
Total annual expenses	3793 \$/yr	2763 \$/yr

The cost of electrical energy (5412 kWh per year) is therefore given by

1980: 70¢/kWh
1990: 51¢/kWh

4. Small-Scale Hydropower

The cost of small-scale microhydro installations are site-specific and highly variable. The cost per installed kilowatt ranges from about \$350 to around \$2000. As an approximate estimation for a 4 kW system we will use the following figures.

Capital cost	\$8000
Operation and maintenance	800 \$/yr
Useful life	15 years
Capital charges	1052 \$/yr
Operation and maintenance	800 \$/yr
Total annual expenses	1852 \$/yr

Assuming the same energy usage as with the previous systems, 5600 kWh/yr, we can estimate the cost of this energy as 1852/5600 = 33¢/kWh.

5. Community-Scale Biogas System

The last technology we shall consider is a biogas system coupled with a diesel generator converted for dual-fuel use. We assume biogas can be substituted for 80% of the diesel fuel, and that approximately 4 m^3 of biogas will substitute for 1 litre of diesel fuel [4].

Using the same 4 kW diesel generator as before we estimate fuel use as follows:

diesel fuel	0.2 x 2782 litre/yr	= 556 litre/yr
biogas	0.8 x 2782 x 4	= 8902 m^3/yr

The system operates for 350 days each year so the output from the biogas plant should be 8902/350 = 25 m^3/day. It is estimated that a 4m^3/day biogas plant (Chinese type) costs $454 or 124 $/yr in capital charges and maintenance costs. Neglecting any economies of scale, it will be assumed that a 25 m^3/day system will cost 124 x 25/4 = 775 $/yr.

Annual charges for the combined system therefore become

	1980	1990
Capital charges - generator	1097 $/yr	1097 $/yr
Capital charges - biogas	775 $/yr	775 $/yr
Maintenance	700 $/yr	700 $/yr
Fuel (diesel)	278 $/yr	450 $/yr
Oil changes	163 $/yr	199 $/yr
Total annual costs	3013 $/yr	3221 $/yr

These expenses indicate the cost of electrical energy (5600 kWh/yr) as

1980: 54¢/kWh
1990: 58¢/kWh

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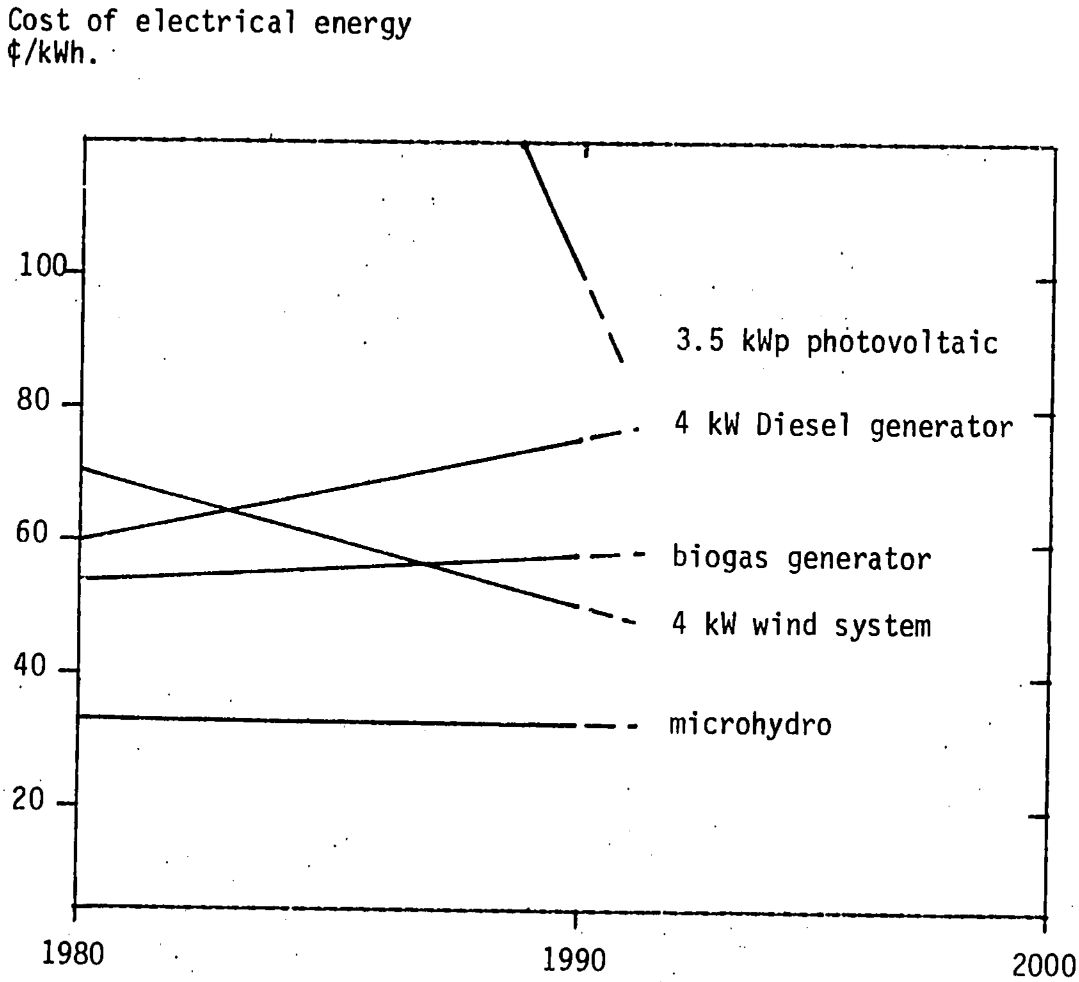

Figure 3 Projected Cost of Electricity

Conclusions

1. Small-scale hydroelectric installations appear to be the most cost-effective technology for supplying electricity. However, many countries do not have an adequate resource base.

2. The vertical axis wind system appears attractive but the analysis is based on very optimistic assumptions about both the wind regime and future capital costs. This cost projection must be viewed as extremely uncertain.

3. The photovoltaic system, although often touted as a potentially viable technology, still does not look like a near-term option, except in very specialized applications.

4. The biogas system is an interesting option. Many developing countries manufacture diesel engines. Foreign exchange costs with this technology would be minimal.

5. A further step in this analysis would be to examine the cost of supplying electricity by extending a powerline to the village from a grid system. The cost of powerline extensions is often quoted at about 6000 $/km. We make the following assumptions:

Cost of power generation	10¢/kWh
CRF (10%, 20 yrs)	0.11746

The cost per unit of electricity is shown in the table below for increasing extension distances and increasing energy consumption.

ENERGY COSTS PER UNIT $/kWh

kWh/yr consumption \ extension	20 km	40 km	60 km
5600	2.62	4.13	7.65
10000	1.51	2.92	4.33
20000	0.80	1.51	2.21
30000	0.57	1.04	1.51
40000	0.45	0.80	1.16

Power line extension can only compete with decentralized sources when the electrical energy consumption is substantial. This is not usually the case with rural communities in developing countries.

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References

1. Most of the cost data used in these analyses may be found in: "The Economic Costs of Renewable Energy" Final Report, Development Sciences Inc., Massachusetts, 1981.

2. Fraenkel, P., "Food From Windmills", Intermediate Technology Development Group, London, England, 1975.

3. Walton, J.D., et al., "A State of the Art Survey of Solar Powered Irrigation Pumps, Solar Cookers, and Wood Burning Stoves for Use in Sub-Sahara Africa", Georgia Institute of Technology, 1978.

4. Biogas Newsletter #12 and #13, 1981.

TABLE 1

(A) APPROXIMATE PROPERTIES OF SOME GASES

Gas	Density ρ at 20°C, 1.013 bar kg/m^3	Gas Constant R J/kg K	Isentropic Exponent k	Kinematic Viscosity ν at 20°C, 1.013 bar m^2/s
Air	1.204	287.1	1.40	1.486×10^{-5}
Ammonia	0.718	481.5	1.32	1.533
Carbon dioxide	1.841	187.8	1.30	0.845
Methane	0.667	518.5	1.32	1.793
Nitrogen	1.165	296.8	1.40	1.589
Oxygen	1.329	260.1	1.40	1.589
Sulfur dioxide	2.720	127.1	1.26	0.520×10^{-5}

(B) SOME PROPERTIES OF AIR AT ATMOSPHERIC PRESSURE

Temperature °C (°F)	Density ρ kg/m^3	Kinematic Viscosity ν m^2/s	Dynamic Viscosity μ Pa s
−17.8 (0)	1.382	1.171×10^{-5}	1.57×10^{-5}
− 6.7 (20)	1.326	1.263	1.68
+ 4.4 (40)	1.274	1.356	1.73
15.6 (60)	1.222	1.468	1.79
20.0 (68)	1.202	1.486	1.80
26.7 (80)	1.176	1.570	1.84
37.8 (100)	1.135	1.672	1.90
48.9 (120)	1.109	1.756×10^{-5}	1.95×10^{-5}

(C) MECHANICAL PROPERTIES OF WATER AT ATMOSPHERIC PRESSURE

Temperature °C (°F)	Density kg/m^3	Dynamic Viscosity Pa s	Surface Tension N/m	Vapor Pressure Pa	Elastic Modulus N/m^2
0 (32)	1000	1.796×10^{-3}	0.0756	552	1.98×10^9
4.4 (40)	1000	1.550	0.0750	827	2.04
10.0 (50)	1000	1.311	0.0741	1 170	2.10
15.6 (60)	1000	1.130	0.0735	1 790	2.16
21.1 (70)	1000	0.977	0.0725	2 480	2.20
26.7 (80)	995	0.862	0.0718	3 520	2.24
32.2 (90)	995	0.761	0.0709	4 830	2.27
37.8 (100)	995	0.680	0.0699	6 620	2.28
48.9 (120)	990	0.560×10^{-3}	0.0680	11 700	2.29×10^9

TABLE 2

SPECIFIC GRAVITY AND KINEMATIC VISCOSITY OF CERTAIN LIQUIDS

(Kinematic Viscosity = tabular value × 10^{-6})

	Water†		Commercial Solvent		Carbon Tetrachloride		Medium Lubricating Oil	
Temp. °C (°F)	Relative Density	Kin. Visc. m^2/s	Relative Density	Kin. Visc. m^2/s	Relative Density	Kin. Visc. m^2/s	Relative Density	Kin. Visc. m^2/s
4.4 (40)	1.000	1.550	0.728	1.50	1.621	0.752	0.905	443
10.0 (50)	1.000	1.311	0.725	1.37	1.608	0.697	0.900	260
15.6 (60)	0.999	1.130	0.721	1.27	1.595	0.650	0.896	175
21.1 (70)	0.998	0.984	0.717	1.17	1.582	0.604	0.891	116
26.7 (80)	0.997	0.864	0.713	1.09	1.569	0.564	0.888	87.4
32.2 (90)	0.995	0.767	0.709	1.02	1.555	0.520	0.885	64.1
37.8 (100)	0.993	0.687	0.705	0.96	1.542	0.492	0.882	45.7
43.3 (110)	0.991	0.620	0.702	0.89	1.520	0.465	0.874	34.8
48.9 (120)	0.990	0.567					0.866	27.2
65.6 (150)	0.980	0.441					0.865	15.0

	Dust-Proofing Oil*		Medium Fuel Oil*		Heavy Fuel Oil*		Regular Gasoline*	
Temp. °C (°F)	Relative Density	Kin. Visc. m^2/s	Relative Density	Kin. Visc. m^2/s	Relative Density	Kin. Visc. m^2/s	Relative Density	Kin. Visc. m^2/s
4.4 (40)	0.917	75.2	0.865	6.08	0.918	412	0.738	0.752
10.0 (50)	0.913	52.5	0.861	5.16	0.915	300	0.733	0.711
15.6 (60)	0.909	37.9	0.858	4.41	0.912	205	0.728	0.678
21.1 (70)	0.905	28.4	0.854	3.83	0.908	146	0.724	0.641
26.7 (80)	0.902	21.7	0.851	3.39	0.905	106	0.719	0.613
32.2 (90)	0.898	17.2	0.847	2.96	0.902	77.7	0.715	0.585
37.8 (100)	0.894	13.8	0.843	2.58	0.899	58.2	0.710	0.557
43.3 (110)	0.890	11.3	0.840	2.11	0.895	44.6	0.706	0.530

Some Other Liquids

Liquid and Temperature	Relative Density	Kin. Visc. m^2/s
Turpentine at 20°C	0.862	1.73
Linseed oil at 20°C	0.925	35.9
Ethyl alcohol at 20°C	0.789	1.53
Benzene at 20°C	0.879	0.745
Glycerin at 20°C	1.262	661
Castor oil at 20°C	0.960	1030
Light machinery oil at 16°C	0.907	137

†ASCE Manual 25.

*Keuler & Lenz, University of Wisconsin, Madison.

TABLE 3 Properties of Dry Air at Atmospheric Pressures between 250 and 1000 K[a]

T^b (K)	ρ (kg/m³)	c_p (kJ/kg · K)	μ (kg/m · sec × 10⁵)	ν (m²/sec × 10⁶)	k (W/m · K)	α (m²/sec × 10⁴)	Pr
250	1.4128	1.0053	1.488	9.49	0.02227	0.13161	0.722
300	1.1774	1.0057	1.983	15.68	0.02624	0.22160	0.708
350	0.9980	1.0090	2.075	20.76	0.03003	0.2983	0.697
400	0.8826	1.0140	2.286	25.90	0.03365	0.3760	0.689
450	0.7833	1.0207	2.484	28.86	0.03707	0.4222	0.683
500	0.7048	1.0295	2.671	37.90	0.04038	0.5564	0.680
550	0.6423	1.0392	2.848	44.34	0.04360	0.6532	0.680
600	0.5879	1.0551	3.018	51.34	0.04659	0.7512	0.680
650	0.5430	1.0635	3.177	58.51	0.04953	0.8578	0.682
700	0.5030	1.0752	3.332	66.25	0.05230	0.9672	0.684
750	0.4709	1.0856	3.481	73.91	0.05509	1.0774	0.686
800	0.4405	1.0978	3.625	82.29	0.05779	1.1951	0.689
850	0.4149	1.1095	3.765	90.75	0.06028	1.3097	0.692
900	0.3925	1.1212	3.899	99.3	0.06279	1.4271	0.696
950	0.3716	1.1321	4.023	108.2	0.06525	1.5510	0.699
1000	0.3524	1.1417	4.152	117.8	0.06752	1.6779	0.702

[a]From *Natl. Bureau Standards (U.S.) Circ. 564*, 1955.

[b]Symbols: K = absolute temperature, degrees Kelvin; $\nu = \mu/\rho$; ρ = density; c_p = specific heat capacity; $\alpha = c_p\rho/k$; μ = viscosity; k = thermal conductivity; Pr = Prandtl number, dimensionless. The values of μ, k, c_p, and Pr are not strongly pressure-dependent and may be used over a fairly wide range of pressures.

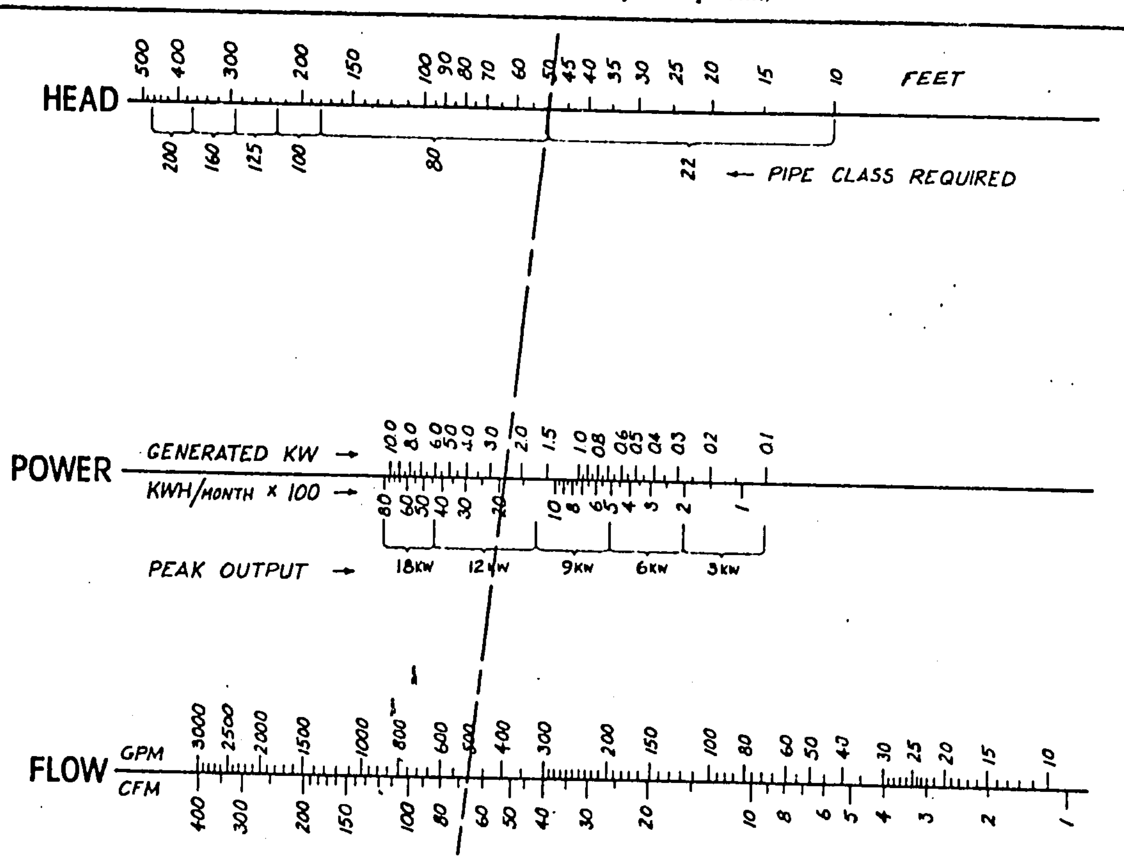
TABLE 4 Properties of Water (Saturated Liquid) between 273 and 533 K[a]

T K	T °F	T °C	c_p (kJ/kg · °C)	ρ (kg/m³)	μ (kg/m · sec)	k (W/m · °C)	Pr	$\frac{g\beta\rho^2 c_p}{\mu k}$ (m⁻³ · °C⁻¹)
273	32	0	4.225	999.8	1.79×10^{-3}	0.566	13.25	
277.4	40	4.44	4.208	999.8	1.55	0.575	11.35	1.91×10^{9}
283	50	10	4.195	999.2	1.31	0.585	9.40	6.34×10^{9}
288.6	60	15.56	4.186	998.6	1.12	0.595	7.88	1.08×10^{10}
294.1	70	21.11	4.179	997.4	9.8×10^{-4}	0.604	6.78	1.46×10^{10}
299.7	80	26.67	4.179	995.8	8.6	0.614	5.85	1.91×10^{10}
302.2	90	32.22	4.174	994.9	7.65	0.623	5.12	2.48×10^{10}
310.8	100	37.78	4.174	993.0	6.82	0.630	4.53	3.3×10^{10}
316.3	110	43.33	4.174	990.6	6.16	0.637	4.04	4.19×10^{10}
322.9	120	48.89	4.174	988.8	5.62	0.644	3.64	4.89×10^{10}
327.4	130	54.44	4.179	985.7	5.13	0.649	3.30	5.66×10^{10}
333.0	140	60	4.179	983.3	4.71	0.654	3.01	6.48×10^{10}
338.6	150	65.55	4.183	980.3	4.3	0.659	2.73	7.62×10^{10}
342.1	160	71.11	4.186	977.3	4.01	0.665	2.53	8.84×10^{10}
349.7	170	76.67	4.191	973.7	3.72	0.668	2.33	9.85×10^{10}
355.2	180	82.22	4.195	970.2	3.47	0.673	2.16	1.09×10^{11}
360.8	190	87.78	4.199	966.7	3.27	0.675	2.03	
366.3	200	93.33	4.204	963.2	3.06	0.678	1.90	
377.4	220	104.4	4.216	955.1	2.67	0.684	1.66	
388.6	240	115.6	4.229	946.7	2.44	0.685	1.51	
399.7	260	126.7	4.250	937.2	2.19	0.685	1.36	
410.8	280	137.8	4.271	928.1	1.98	0.685	1.24	
421.9	300	148.9	4.296	918.0	1.86	0.684	1.17	
449.7	350	176.7	4.371	890.4	1.57	0.677	1.02	
477.4	400	204.4	4.467	859.4	1.36	0.665	1.00	
505.2	450	232.2	4.585	825.7	1.20	0.646	0.85	
533.0	500	260	4.731	785.2	1.07	0.616	0.83	

[a]Adapted from Brown, A. I., and S. M. Marco, "Introduction to Heat Transfer," 3d ed McGraw-Hill Book Company, New York, 1958.

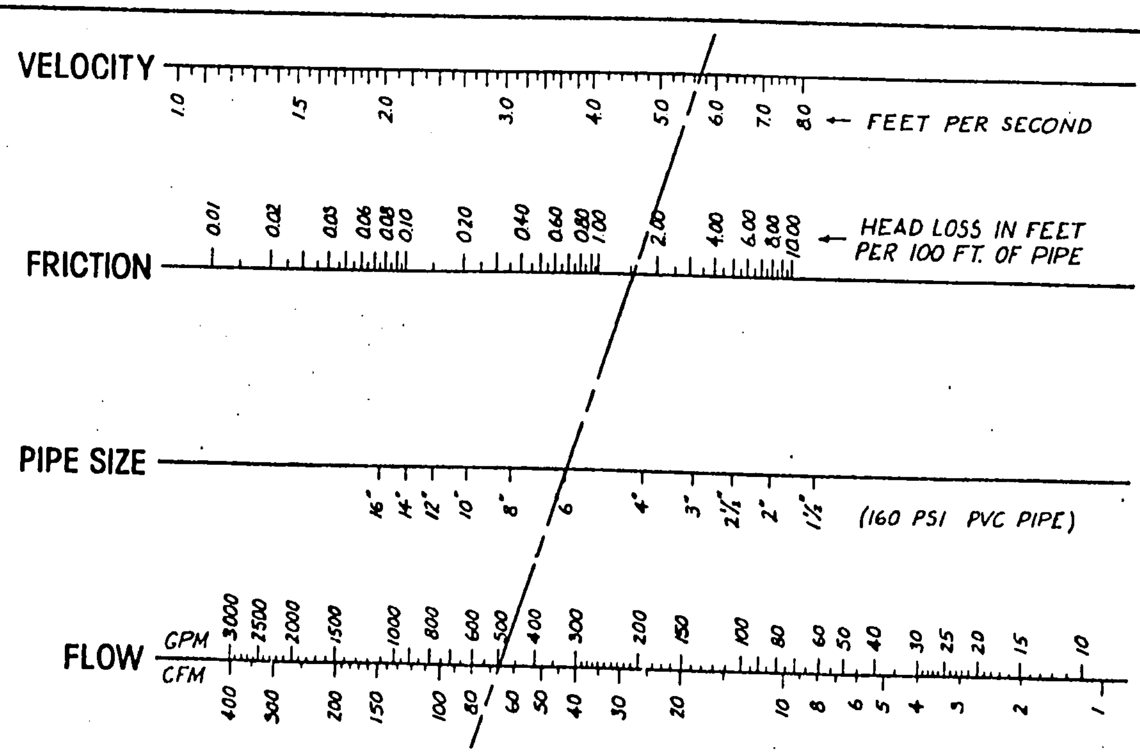
NOMOGRAPH TO DETERMINE TYPICAL OUTPUT POWER FROM A MICRO-HYDRO SYSTEM

(Assumed system efficiency of 53 percent)



Adapted from: Independent Power Developers' brochure "Hydroelectric Power".

NOMOGRAPH TO DETERMINE LOSSES DUE TO FRICTION IN PVC PIPE



Adapted from: Independent Power Developers' brochure "Hydroelectric Power".

Physical Quantity	Conversion Factor	
Area	1 ft^2	= 0.0929 m^2
	1 m^2	= 10.764 ft^2
	1 ha	= 2.471 acres
	1 acre	= 4074 m^2
Density	1 lb/ft^3	= 16.018 kg/m^3
Energy	1 Btu	= 1055.1 J
	1 cal	= 4.1868 J
	1 kWh	= 3.6 MJ
	1 hp-hr	= 2.685 MJ
	1 MW-yr	= 31.5576 TJ
Enthalpy	1 Btu/lb	= 2326.0 J/kg
Force	1 lb-force	= 4.4482 N
Heat transfer coefficient	1 Btu/hr ft^2 °F	= 5.678 W/m^2K
Length	1 ft	= 0.3048 m
	1 inch	= 2.54 cm
	1 mile	= 1.6093 km
Mass	1 lb-mass	= 0.4536 kg
Power	1 hp	= 745.7 W
	1 Btu/hr	= 0.2931 W
	1 Btu/hr ft^2	= 3.1548 W/m^2
Pressure	1 psi	= 6894.8 Pa
	1 in. W. G	= 0.0361 psi
	1 atm	= 101325 N/m^2 (Pa)
	1 atm	= 14.696 psi
Radiation	1 Langley	= 41860 J/m^2
	1 kWh/m^2	= 316.95 Btu/ft^2
Specific heat	1 Btu/lb°F	= 4187 J/kgK
Thermal conductivity	1 Btu/hr ft°F	= 1.731 W/mK
Thermal resistance	1 hr °F/Btu	= 1.8958 K/W
Time	1 year	= 8766 hr
	1 year	= 31.5576 Ms
Velocity	1 ft/s	= 0.3048 m/s
	1 mph	= 0.44703 m/s
Viscosity (dynamic)	1 lb/ft sec	= 1.488 Ns/m^2
	1 cp	= 0.001 Ns/m^2
(kinematic)	1 ft^2/s	= 0.09029 m^2/s
Volume	1 ft^3	= 28.317 litre
	1 gallon (US)	= 3.785 litre
	1 m^3	= 35.314 ft^3
	1 ft^3	= 7.481 gallon (US)
Volumetric flow	1 ft^3/min	= 0.472 litre/s
	1 gpm	= 0.06308 litre/s
	1 gpm/ft^2	= 0.67903 $litre/m^2s$
	1 gal/ft^2	= 40.742 $litre/m^2$
	1 ft^3/lb	= 62.30 litre/kg